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Preliminary Economic Assessment
Marmato Project
Colombia

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Report Prepared for

Gran Colombia Gold Marmato S.A.S.

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1 Summary

This report was prepared as a Preliminary Economic Assessment-level (PEA) National Instrument 43-101 (NI 43-101) Technical Report (Technical Report) for Gran Colombia Gold Marmato S.A.S. (GCM or Company and formerly Mineros Nacionales S.A.S.) by SRK Consulting (U.S.), Inc. (SRK) on the Combined Marmato Mine (Upper Mine) and Marmato Deeps Mine (MDZ or Project). GCM is an indirect wholly-owned subsidiary of Gran Colombia Gold Corp. (Gran Colombia).

1.1 Property Description and Ownership

The Marmato Concession is located between latitudes and longitudes 5°28’24”N and 5°28’55”N, and 75°35’57”W and 75°38’55”W, respectively; with altitudes ranging from approximately 1,156 to 1,705 m. Marmato is made up of three separate areas within a historic mining district, named Zona Alta (upper zone), Zona Baja (lower zone) and Echandia (northern portion of the Project), all of which are 100% owned by GCM. GCM is currently in the process of extending the duration of the Zona Baja mining contract for which the current 30-year term expires in October 2021.

On October 4, 2019, Gran Colombia entered into a letter of intent with Bluenose Gold Corp. (Bluenose). A TSXV-listed company, in respect of the proposed acquisition by Bluenose of certain mining assets (the Mining Assets) held by the Company at Marmato. The Mining Assets principally comprise the existing producing underground gold mine, including the right to mine in the lower portion of the Echandia license area, the existing 1,200 tonnes per day (tpd) processing plant and the area encompassing the Deeps mineralization, all located within the mining license area referred to as Zona Baja. The acquisition will be completed through a reverse takeover transaction (RTO Transaction) with the newly created entity, to be named Caldas Gold Corp. (Caldas), owning the Zona Baja Mining Assets. The proposed mining plan contained herein focuses exclusively on the Mineral Resources in Zona Baja. Gran Colombia will retain ownership of the areas at the Marmato Project referred to as Zona Alta and Echandia.

SRK notes within the transfer of licenses from the previous owner, there is a gap between the existing licenses for Zona Baja and Echandia. This ground is under application from the Company with the Colombia government for formal approval to continue mining. SRK has reviewed the application within the government website and notes that the status is defined as “in progress”, which has been in place since September 30, 2009. The Company is taking steps to get the approval finalized.

Mining by GCM within this area has historically been conducted through the current operations and it has been reported to SRK that this area is under application for adjustment of the status for inclusion in the current mining operations of GCM. As the area represented by this gap contains approximately 8.7% of Measured & Indicated resources in Zona Baja and 1.9% of Inferred resources in Zona Baja, SRK recommends that GCM accelerate its effort to obtain government resolution on this gap as it could potentially result in loss of Mineral Resources (and future Reserves). Only approximately 2.3% of the total LOM gold production included in the proposed mine plan for Zona Baja contained herein is sourced from the area represented by this gap.

1.2 Geology and Mineralization

The mineralization in the current mine consists of three distinct phases, a first phase characterized by the mesothermal vein/veinlet mineralization which defines the MDZ, followed by an epithermal low sulfidation style, superimposed by an epithermal intermediate sulfidation phase. Gold-silver mineralization is mainly hosted by a pyrite+sphalerite vein to veinlet system fitting in a sinistral transpressional shearing system, associated with intermediate argillic alteration within the host porphyryitic rocks. Approximately 92% of the gold/silver-bearing particles are intergrown with sulphides, or occur at sulphide – gangue grain boundaries. Current mining in the area is via narrow underground stoping of the higher-grade vein mineralization.

The MDZ mineralization consists of a network of thin, less than 5 centimeters, sulfide veinlets, mainly pyrrhotite+chalcopyrite, hosted in weak argillic and deeper potassic alteration which is related to a previous event and rimmed by a thin sodium-calcitic alteration halo, which is related to the mineralization. Recent geological reports on MDZ (Sillitoe, 2019) concluded:

- Gold grade distribution in the Zona Baja (MDZ) mineralized orebody is unrelated to the presence of distinct porphyry phases and is entirely dependent on the intensity of structurally localized veinlets;
- Potassic alteration, represented chiefly by biotite, is progressively better preserved at depth in the Zona Baja, raising the possibility that early potassic alteration could also be gold bearing, but further work is required to confirm this theory;
- Gold distribution appears to be exclusively a product of veinlet intensity and orientation related to structural controls during orogenesis. The veinlets responsible for much of the Zona Baja gold are those containing quartz, pyrrhotite and traces of chalcopyrite and having prominent albite alteration halos; and
- The presence of visible gold is also noted in the core and, as expected, relates to increased assay values when present.

The mineralization occurs in parallel, sheeted and anastomosing veins (“vein domain”), all of which follow a regional structural control, with minor veins forming splays of the main structures (“splays”) which often have limited strike or dip extent. The upper vein domain intersects broader zones of intense veinlet mineralization (termed “porphyry domain” in this Technical Report) that is hosted by a lower grade mineralized porphyry stock. In addition, a discrete, relatively high-grade core (“feeder zone”) to the main deeper mineralization (MDZ) identified by GCM.

The upper portion of the MDZ has been exposed in Level 21 of the existing GCM mining operations, while deeper sections have been observed in drillcore, both of which have been confirmed as different styles of mineralization. The lowest levels of the mine have currently intersected a combination of the porphyry domain, where the gold is associated with pyrite veinlets, and the MDZ where gold is associated with pyrrhotite. There is a transition zone exists between the two domains, which is observed to some extent in the current mine workings with overprinting of the epithermal system on the MDZ. The vertical extent of the transition is not clearly defined from the current drilling. Currently, underground mining at the GCM-operated mine remains focused on the vein structures located in the central portion (Zona Baja) of the Marmato deposit.

The updated drilling database indicates that the veins typically range between 0.5 and 5 m wide and extend for 250 to 1,000 m along strike, and 150 to 750 m down dip. These observations are supported by underground mining which has confirmed that individual vein structures have good geological continuity and can extend for 100 to 800 m along strike and 100 to at least 300 m down dip. Between 2017 and 2019, GCM has worked on updating the quantity of the underground channel sampling captured in the database, which has increased the information available to model the vein domains.

The broad zones of veinlet mineralization in the porphyry domain modelled initially by SRK in 2017 typically varied from 10 to 230 m wide, reaching up to 340 m wide in areas of significant veinlet accumulation, while extending with good geological continuity for between 200 m and approximately 950 m along strike, and between 100 and 900 m down dip. SRK has updated these domains during the 2019 geological modelling process using more discrete zones and application of an indicator grade shell approach using a 0.5 grams per tonne (g/t) gold (Au) cut-off grade (CoG).

At depth within the central portion of the deposit, SRK has noted a zone of elevated grades which has been referred to as the higher grade MDZ (more than 2.0 g/t Au). This zone is indicated to be continuous along strike for approximately 500 m and has a confirmed down dip extent that reaches up to 800 m, with a thickness that varies between 35 and 150 m. It is possible that the main MDZ mineralization is bounded within a series of faults but limited drilling at the edges of the deposit make confirmation difficult to assess at this stage. To avoid the potential for volumetric “blow-outs”, SRK has used the faults as a hard boundary in the geological domaining process.

1.3 Status of Exploration, Development and Operations

There has been a significant increase (100%) in the channel sampling and underground drilling completed by the Company operated mine as part of routine grade control procedures, since the previous model. There are an additional 12,765 channel samples as part of the on-going validation and capture of historical sampling, and current grade control practices. In addition to the channel sampling, there has been an increase in the drilling database of 152 additional holes, which is split between 35 exploration holes (16,076 m) and 117 mine holes (9,172 m) inclusive of grade control sampling.

The latest sampling has comprised selective infill drilling targeting the MDZ to a spacing of 50 to 100 m, and additional underground channel sampling within the Company operated mine, which extends from Levels 16 to 21.

Drillholes have been drilled from four purpose-built underground drilling stations with two contractor rigs being used to date (a third rig started drilling in early September 2019 to assist in completing the designed program). Three of the drilling stations are located on Level 20 with a single station established on Level 21. Drillholes have been drilled in a fan pattern and dip -60 and -75 degrees predominantly to the southwest.

All samples were prepared and fire assayed by SGS Laboratories in their facility in Medellin. The results of the drilling have validated aspects of the previous interpretation, but also provided additional information which has led to further confirmation of the geological model initially created during the 2017 Mineral Resource Estimate.

1.4 Mineral Processing and Metallurgical Testing

A metallurgical program was conducted by SGS Lakefield on test composites from the MDZ. The metallurgical program included comminution testwork, mineralogical studies and an evaluation of several different flowsheet options including:

- Whole-ore cyanidation;
- Gravity concentration followed by cyanidation of the gravity tailing; and
- Gravity concentration followed by gold flotation from the gravity tailing and cyanidation of the flotation concentrate.

The following significant metallurgical and mineral processing factors have been identified:

- Metallurgical testwork was conducted on a test composite representing the average material from the MDZ as well as variability composites from East, Central and West Zones of the MDZ deposit;
- Native gold was by far the predominant gold carrier and the majority (more than 99%) of the gold particles occurred within mineral structures that would be readily accessible by leaching solutions. Gold particles were not often in direct contact with sulfides, yet very commonly pyrrhotite, chalcopyrite, and bismuth minerals were found in close vicinity to the gold mineralization;
- The SAG mill comminution (SMC) tests were conducted on the East, West and Center MDZ and the reported Axb values ranged from 28 to 31 and averaged 29, indicating that the material is very hard with respect to SAG mill impact grinding. Bond ball mill work indices (BWi) conducted on the MDZ composites ranged from 19.0 to 20.7 kilowatt hour per tonne (kWh/t) indicating that the MDZ material is very hard with respect to ball mill grinding. The MDZ material is much harder than the current Marmato material which was reported to have a BWi of 15.7 kWh/t;
- The MDZ material responded very well to each of the process flowsheet options tested with gold recoveries estimated at 94 to 96% and silver recoveries estimated at 38 to 44%;
- Cyanidation leach residues can be detoxified to less than 10 milligrams per liter (mg/L) Cyanide Weak Acid Dissociable (CNWAD) using the industry-standard Sulphur dioxide (SO2)/Air process; and
- Additional testwork should be conducted during the next phase of study to fully define the process design criteria and operating parameters.

1.5 Mineral Resource Estimate

The Mineral Resource model presented herein represents an updated resource evaluation prepared for the Marmato Project. The resource estimation methodology involved the following procedures:

- Database compilation and verification;
- Construction of wireframe models for the fault networks and centerlines of mining development per vein;
- Definition of resource domains;
- Data conditioning (compositing and capping) for statistical and geostatistical analysis;
- Variography;
- Block modelling and grade interpolation;

- Resource classification and validation;
- Assessment of “reasonable prospects for economic extraction” and selection of appropriate reporting CoGs; and
- Preparation of the Mineral Resource Statement.

The resource evaluation work was completed by Mr. Benjamin Parsons, MAusIMM (CP#222568), with assistance from Mr Giovanny Ortiz, FAusIMM (#304612). The effective date of the Mineral Resource Statement is July 31, 2019, which is the last date assays were provided to SRK.

The Mineral Resource estimation process was a collaborative effort between SRK and GCM staff. GCM provided SRK with an exploration database with flags of the main veins as interpreted by GCM. In addition to the database, GCM has also supplied a geological interpretation comprising preliminary 3D digital files (DXF) through the areas investigated by core drilling for each of the main veins.

SRK imported the geological information into Seequent Leapfrog® Geo (Leapfrog®) to complete the geological model. Leapfrog® has been selected due to the ability to rapidly create accurate geological interpretations, which interact with a series of geological conditions and data types.

SRK has produced block models using Datamine™ Studio RM Software (Datamine™). The procedure involved import from Leapfrog™Geo of wireframe models for the fault networks, veins, definition of resource domains (e.g. high-grade sub-domains), data conditioning (compositing and capping) for statistical analysis, geostatistical analysis, variography, block modelling and grade interpolation followed by validation. Grade estimation for the veins has been based on block dimensions of 5 m by 5 m by 5 m for the Porphyry and MDZ units. Sub-blocking to 0.5 m by 1 m by 1 m has been allowed to reflect the narrow nature of the geological model. The block size reflects the relatively close-spaced underground channel sampling and spacing within veins compared to the wider drilling spacing, with the narrower block size used in the MDZ at depth to reflect the proposed geometry of the mineralization (i.e. steeply dipping feeder zone).

SRK reviewed and updated the geostatistical properties of the domains. Gold grades have been interpolated using nested three-pass estimates within Datamine™, using an Ordinary Kriging (OK) routine. SRK has also run Inverse Distance Weighted (IDW2) and Nearest Neighbor (NN) estimates for validation purposes.

The search ellipses follow the typical orientation of the mineralized structures, and where appropriate, were aligned along the mineralized veins, as detailed below:

- Dynamic searches were used for the vein mineralization domains. Within these domains, the true dip and true dip direction has been calculated on a block by block basis;
- In comparison, given the relatively short strike and dip of the splay, SRK has elected to use an average dip and strike for each structure;
- For the porphyry domain, SRK has generated a default dip and dip-direction to orientate the search volume along the main regional trend;
- For the MDZ, a single dip and strike has been used with the search ranges orientated along the main dip and strike of the domain;
- All contacts between the veins have been treated as hard boundaries for domaining with only coded samples from any given vein used in the estimation of that domain; and

- Statistical characteristics such as search volume used, kriging variance, and number of samples used in an estimate, were also computed and stored in each individual block for descriptive evaluations.

SRK has validated the block model using a combination of visual checks, statistical comparison of composite grades to all three estimation methods, and via swath plot analysis. SRK considered the estimates to be representative of the underlying data.

Block model quantities and grade estimates for the Project were classified according to the CIM Definition Standards for Mineral Resources and Reserves (CIM, 2014). SRK developed a classification strategy which considers the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Data quality, drillhole spacing and the interpreted continuity of grades controlled by the veins have allowed SRK to classify portions of the veins in the Measured, Indicated and Inferred Mineral Resource categories.

Measured: Measured Resources are limited to vein material within the current levels being mined by the Company and estimated within the first search volume which required a minimum of five composites and a maximum of 20 composites. These areas are considered to have strong geological knowledge as they have been traced both down-dip and along strike via mapping, plus underground channel samplings provided sufficient data populations to define internal grade variability.

Indicated: SRK has delineated Indicated Mineral Resources at Marmato primarily between Level 16 to 21 currently in operation at Mineros Nacionales license zone. Indicated Mineral Resources have been given at the following approximate data spacing, as a function of the confidence in the grade estimates and modelled variogram ranges. SRK has expanded the limits of the Indicated classification to also cover areas within Echandia Licence where:

- 50 m by 50 m (XY) from the nearest drillhole;
- Enabled multiple holes to be used during the estimation process; and
- Support from both diamond drilling and channel sampling present.

Inferred: In general, Inferred Mineral Resources have been limited to within areas of reasonable grade estimate quality and sufficient geological confidence, and are extended no further than 150 m from peripheral drilling on the basis of modelled variogram ranges.

SRK has defined the proportions of Mineral Resource to have potential for economic extraction for the Mineral Resource based on different CoGs relating to the mineralization style (i.e. vein versus porphyry) and potential differences in selective underground mining methods.

The estimation domains have been grouped for the Mineral Resource statement into veins, porphyry and MDZ mineralization. The veins account for the veins, halos and splay material and have used a 1.7 g/t Au cut-off, while all other domains (grade-shell, Deeps, porphyry) have used a lower cut-off of 1.2 g/t to account for the larger bulk mining methods involved. The Mineral Resource statement for the Project is shown in Table 1-1.

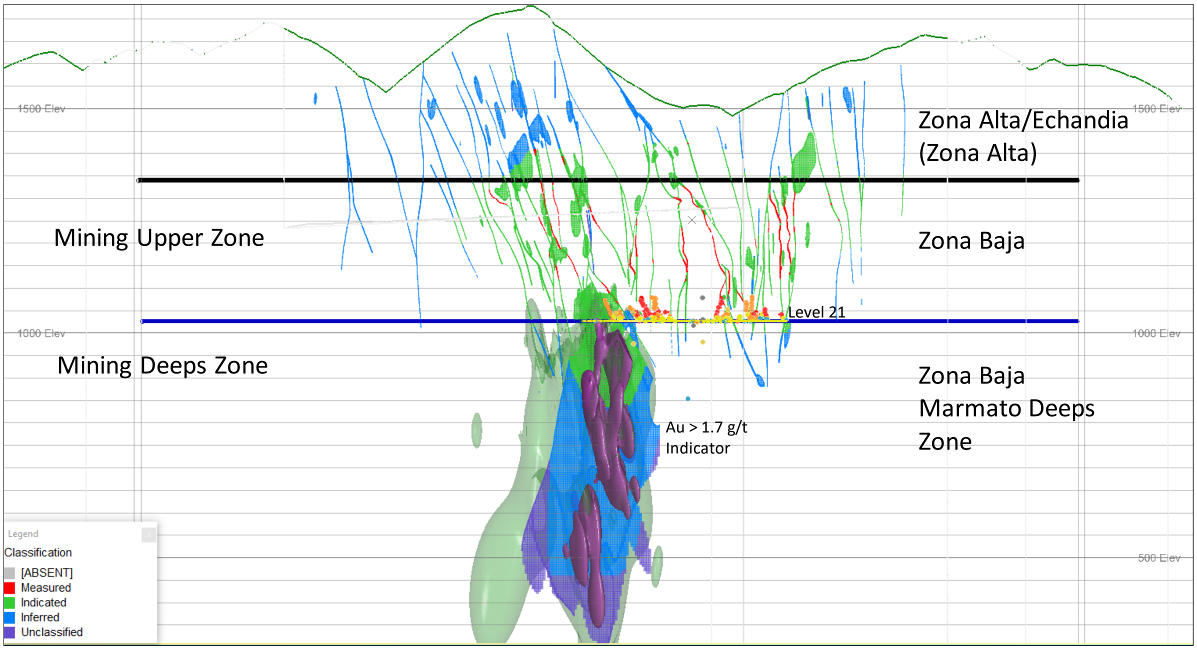
Table 1-1: SRK Mineral Resource Statement for the Marmato Project, Dated July 31, 2019*, Within All Licences

Category	Quantity	Grade		Metal	
		Au	Ag	Au	Ag
	Mt	gpt	gpt	000'oz	000'oz
Underground Vein**					
Measured	2.7	5.0	25.5	433	2,190
Indicated	10.8	4.5	20.5	1,579	7,133
Measured and Indicated	13.5	4.6	21.5	2,013	9,323
Inferred	8.6	4.2	19.3	1,154	5,330
Underground Porphyry**					
Measured					
Indicated	3.7	2.9	26.4	350	3,149
Measured and Indicated	3.7	2.9	26.4	350	3,149
Inferred	3.0	4.3	48.0	420	4,680
Underground Deeps***					
Measured					
Indicated	6.4	2.6	4.7	537	978
Measured and Indicated	6.4	2.6	4.7	537	978
Inferred	41.2	2.1	2.7	2,812	3,609
Underground Combined					
Measured	2.7	5.0	25.5	433	2,190
Indicated	21.0	3.7	16.7	2,466	11,260
Measured and Indicated	23.6	3.8	17.7	2,899	13,450
Inferred	52.9	2.6	8.0	4,387	13,619

* Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate.
** Porphyry and Vein mineral resources are reported at a CoG of 1.9 g/t. CoGs based on a price of US$1,500 per ounce of gold, suitable benchmarked technical and economic parameters and gold recoveries of 95 percent for underground resources, without considering revenues from other metal.
*** Deeps mineral resources are reported at a CoG of 1.3 g/t. CoGs based on a price of US$1,500 per ounce of gold, suitable benchmarked technical and economic parameters and gold recoveries of 95 percent for underground resources, without considering revenues from other metal.

The proposed mining plan is predicated on splitting the above Mineral Resources into three styles of mineralization within two distinct areas. The three styles of mineralization are based on the key geological types defined in the Mineral Resources of Veins, Porphyry and Deeps (MDZ). The mining areas are split into Zona Alta and Zona Baja based on the presence of the existing mining infrastructure, with Zona Alta representing material above the existing mines, and Zona Baja as all material, including the Echandia Licences, below an elevation of 1,340 masl. The proposed mining plan focuses exclusively on the Mineral Resources in Zona Baja. This PEA does not concern itself with an evaluation of the Zona Alta mining zone.

A summary of the breakdown for the Mineral Resources as used in the current study is shown in Figure 1-1, and with the tonnages, grades, Au metal and Ag metal presented in Table 1-2. It is important to note that pursuant to the RTO Transaction described in Section 1.1, the Mining Assets which encompass the Mineral Resources in Zona Baja in Figure 1-1 and Table 1-2 are being spun out to Caldas. Gran Colombia will retain ownership of the mining titles encompassing the Mineral Resources in Zona Alta in Figure 1-1 and Table 1-2.



Source: SRK, 2019

Figure 1-1: Summary Breakdown of Mining Areas for Study

Table 1-2: SRK Mineral Resource Statement for the Marmato Project, Dated July 31, 2019*, Breakdown by Mining Areas

Licence Grouping			Zona Alta (1)			Zona Baja (2) and Deeps Zone				Total
Type			Veins	Porphyry	*Total*	Veins	Porphyry	Deeps	*Total*	
Measured	Tonnes	(kt)	0.6		*0.6*	2.1			*2.1*	**2.7**
	Grade	Au (g/t)	5.6		*5.6*	4.9			*0.1*	**5.0**
		Ag (g/t)	33.2		*5.6*	23.2			*4.9*	**25.5**
	Metal	Au (koz)	109		*109*	325			*325*	**433**
		Ag (koz)	648		*648*	1,543			*1,543*	**2,190**
Indicated	Tonnes	(kt)	3.6	2.1	*5.7*	7.2	1.6	6.4	*15.2*	**21.0**
	Grade	Au (g/t)	4.6	3.1	*4.1*	4.5	2.7	2.6	*3.5*	**3.7**
		Ag (g/t)	25.3	38.9	*4.1*	18.1	10.1	4.7	*3.5*	**16.7**
	Metal	Au (koz)	542	210	*752*	1,037	140	537	*1,714*	**2,466**
		Ag (koz)	2,966	2,622	*5,588*	4,167	527	978	*5,672*	**11,260**
Measured & Indicated	Tonnes	(kt)	4.2	2.1	*6.3*	9.2	1.6	6.4	*17.3*	**23.6**
	Grade	Au (g/t)	4.8	3.1	*4.2*	4.6	2.7	2.6	*3.7*	**3.8**
		Ag (g/t)	4.8	3.1	*4.2*	4.6	2.7	2.6	*3.7*	**17.7**
	Metal	Au (koz)	650	210	*860*	1,362	140	537	*2,039*	**2,899**
		Ag (koz)	3,614	2,622	*6,236*	5,709	527	978	*7,214*	**13,450**
Inferred	Tonnes	(kt)	5.3	2.7	*7.9*	3.3	0.3	41.2	*44.9*	**52.9**
	Grade	Au (g/t)	4.1	4.5	*4.2*	4.4	3.1	2.1	*2.3*	**2.6**
		Ag (g/t)	4.1	4.5	*4.2*	4.4	3.1	2.1	*2.3*	**8.0**
	Metal	Au (koz)	688	386	*1,074*	466	34	2,812	*3,312*	**4,387**
		Ag (koz)	3,753	4,573	*8,326*	1,577	107	3,609	*5,293*	**13,619**

Source: SRK, 2019

1) Zona Alta includes mineral resources from the Echandia license above 1,340 masl.
2) Zona Baja includes mineral resources from the Echandia license below 1,340 masl and above 1,025 masl and are accessible from the current mining operation.

1.6 Mineral Reserve Estimate

No Mineral Reserves have been estimated for the Project.

1.7 Mining Methods

The Project has been in operation in various forms since the mid-1500s. GCM (formerly known as Mineras Nacionales (MN) was awarded the contract for the concessions in 1989. The Project was originally developed as a 300 tpd underground project in 1997 and has expanded through the years to the existing 1,200 tpd capacity operation. The mine is currently developed to the 1,025 m elevation. A transition is occurring from narrow vein mineralization to large porphyry mineralized areas (gold associated with pyrrhotite veinlets). For this PEA, there are three different mining methods, separated into three distinct zones.

- The first zone is the mineralized material between 1,025 m (meters) elevation to 1,350 m elevation, referred to as the Veins. This is the current mine and will be mined using the current conventional cut and fill stope method.
- The second zone is the wider porphyry material between 1,025 m elevation and 1,070 m elevation, referred to as Level 21. This material is on Level 21 with some development accesses already in place. A modified longhole stoping method will be used in this area. The stope size is 10 m wide by 15 m high with varying length of up to 26 m. 5 m dip pillars are left due to the use of unconsolidated hydraulic fill.
- The third zone is the porphyry material below 1,025 m elevation, referred to as MDZ. There is a sill pillar left in-situ between the MDZ and Level 21. The MDZ material is mined using a longhole stoping method with stope sizes that are 10 m wide by 25 m high with varying of up to 20m. The MDZ area is currently not developed.

The first two zones are considered the Upper Mine of Zona Baja, and the material is processed in the existing processing facility. The third zone is considered the MDZ and the material is envisioned to be sent to a new processing facility. Separate mine plans are presented for the Upper Mine and MDZ areas of Zona Baja.

1.7.1 Upper Mine

Mining is being conducted in the Zona Baja area that extends approximately 300 m vertically and 900 m along the vein structure. The mine layout has development typically on 50 m levels and there are currently six production levels. Portal access is utilized with a main haulage level established with rail haulage, ore passes, and apiques (inclined shaft) hoist systems used to move personnel and materials.

Mine design using Vulcan software was completed based on an AuEq cut-off grade (CoG). Stope optimizations were performed based on a calculated CoG and served as the basis for the design. Parameters were assumed similar to that currently being mined, with the exception of Level 21, where the design was modified to utilize dip pillars to allow mining with the sand backfill available from the current plant.

The underground mine design for the Marmato Zona Baja Upper Mine (Veins and Level 21 blocks) results in a mine plan resource of 5.5 Mt (diluted) with an average grade of 3.82 g/t Au and 15.39 g/t Ag. This estimate is based on a 2.81 g/t AuEq cut-off, a 90% mining recovery and a 20% dilution for the veins. Level 21 blocks are based on a 2.05 g/t AuEq cut-off, a 90% mining recovery and a 15% dilution. A 5% allowance is added to the development for overbreak. The mined material summary is detailed in Table 1-3.

Table 1-3: Upper Mine - Mine Plan Resource Classification – Vein and Level 21 Material (1)

Description	Tonnes (kt)	Au (g/t)	Ag (g/t)	Contained Au Oz (koz)	Contained Ag Oz (koz)
Measured	802	3.94	17.95	102	463
Indicated	4,308	3.84	14.64	532	2,028
Inferred	433	3.34	18.06	46	251
Total	**5,543**	**3.82**	**15.39**	**680**	**2,742**

Source: SRK, 2019

(1) Includes Measured, Indicated and Inferred material based on CoG of 2.81g/t AuEq for the veins, and 2.05 g/t AuEq for Level 21 material.

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, in that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that the PEA will be realized.

A production schedule, shown in Table 1-4, was developed using iGantt from the mine design. The production schedule is based on 350 days per year, with two eight hour shifts per day. Scheduling targeted 700 tonnes per day (tpd) (245,000 tonnes per year [tpy]) from the vein stopes from 2019 to 2021, then increasing to 1,000 tpd (350,000 tpy) for the rest of the mine life. Level 21 stopes are producing at 50 tpd (17,500 tpy), 300 tpd (105,000 tpy) and 450 tpd (157,500 tpy) for years 2019, 2020 and 2021 respectively.

Table 1-4: Upper Mine Production Schedule – Vein and Level 21 Material

Marmato (Unit)	Mineralized Tonnes (kt)	Waste Tonnes (kt)	Au (g/t)	Ag (g/t)	Development Length (m)	Backfill Volume (m^3)
2019	109	28	3.94	14.32	1,385	36,876
2020	350	43	3.77	12.16	2,718	118,997
2021	402	30	3.53	10.95	2,247	140,420
2022	350	21	3.64	14.41	1,984	120,942
2023	350	19	3.63	15.28	1,811	120,772
2024	350	17	3.64	15.06	1,685	120,489
2025	350	25	3.75	14.81	2,269	120,603
2026	350	22	3.92	15.29	2,068	120,301
2027	350	11	4.08	15.57	1,280	120,015
2028	350	16	4.08	16.42	1,790	119,101
2029	350	29	3.89	17.68	2,671	120,204
2030	350	17	3.58	16.33	1,730	120,978
2031	350	13	3.82	17.04	1,431	120,163
2032	350	4	4.04	18.08	660	119,090
2033	350	4	4.03	17.28	618	119,050
2034	350	4	3.95	16.14	521	119,180
2035	130		3.54	13.57	28	44,045
Totals	**5,543**	**301**	**3.82**	**15.39**	**26,895**	**1,901,227**

Source: SRK, 2019

Mining is completed by a cut and fill (CAF) stope mining method with sand backfill from tailing material from the processing plant. The CAF panels are typically 35 m long by 50 m high, with varying thickness depending on the vein. The panels are accessed from 2.2 m by 2.2 m haulage levels on the top and bottom. Raises are developed along the vein to break the panel into discreet mining stopes as well as to provide ventilation. Sub-levels are then driven horizontally along strike. Once the sublevel is opened, vertical holes are drilled up at a length of approximately 1.7 m to 2.3 m over the width of the vein. After blasting, the mineralized material is mucked using either slushers, bobcats or mini-scoops and loaded into trains and hauled out. Once mucking is complete, concrete walls are built on either end of the stope and the stope is filled with hydraulic fill. When the fill is sufficiently drained, the next slice of mineralized material can be mined.

At the bottom of Zona Baja on level 21, a transition zone has been encountered with material consistent with MDZ. A modified stoping method is being employed that is planned to increase productivity reduce costs. The stopes will be 15 m wide by 15 m high and vary in length depending on mineralized material geometry. The design uses a bottom up stoping method with a primary/secondary stoping sequence. Access drifts are planned to be 3.5 m by 3.5 m with a 3.5 m by 3.5 m decline driven at -13% for access to the different levels.

There are currently approximately 777 people working on the underground mining operation with and increase to 910 people expected by 2023.

The mine has a ventilation system that moves approximately 139 thousand cubic feet per hour (kCFM) of fresh air through the portals from Level 17 and Level 18 via 50 hp fans. The fresh air moves down the apiques (inclined shafts) to the levels and out to raises at the end of the level drifts where it is exhausted on Level 16.

A staged pumping system is installed that removes groundwater and backfill water from the mine at rates that average 37 L/s with a range from 26.8 L/s to 46.4 L/s.

1.7.2 Marmato MDZ Project

The MDZ area of Zona Baja is currently in the exploration phase and has not been developed. Mineralization is located approximately 600 to 1,200 m below the surface (530 m elevation to 1,015 m elevation). Based on geomechanical information and mineralization geometry an underground longhole stoping method (LHS) is suitable for the deposit and vein mining can continue to occur above the MDZ area.

The stopes will be 10 m wide and stope length will vary based on mineralization grade. A spacing of 25 m between levels has been used. The deposit is mined in blocks where mining within a block occurs from bottom to top with the use of paste backfill. Sill pillars are left in situ between blocks. The backfill will have sufficient strength to allow for mining adjacent to filled stopes, thus eliminating the need for dip pillars. In the top mining block a higher grade core is extracted first, mined from bottom to top. Subsequently additional stopes are mined from the bottom of the block up, mining adjacent to (but not underneath) backfilled stopes.

The mine will be drift access, mineralization will be transported from stopes to the surface by underground trucks. Internal intake and exhaust raises will be developed using raisebore machines and air will flow into dedicated intake and exhaust ventilation drifts to surface.

The mine design process involved using stope optimization within Vulcan™ software to determine the potentially mineable areas calculated CoG. The underground mine design process for the MDZ area results in mine plan resources of 20.8 million tonnes (Mt) (diluted) with an average grade of 2.5 grams per tonne (g/t) gold (Au) and 3.38 g/t silver (Ag). This estimate is based on a mine design using a 1.75 g/t Au CoG for the design of stopes and applying a 1.5 g/t Au CoG to material within the design. These numbers include a 93% to 100% mining recovery based on type of opening (stope, development, etc.) to the designed wireframes in addition to a 0% to 7% unplanned waste dilution. An additional development allowance of 15% was applied to main ramps to account for detail currently not in the design. Dilution for stopes was applied at zero grade (Table 1-5).

Table 1-5: MDZ Mine Plan Resource Classification (1)

Description	Tonnes (kt)	Au (g/t)	Ag (g/t)	Contained Au Oz (koz)	Contained Ag Oz (koz)
Indicated	3,360	2.80	4.66	302	503
Inferred	17,462	2.45	3.14	1,374	1,761
Total	**20,821**	**2.50**	**3.38**	**1,677**	**2,264**

Source: SRK, 2019
(1) Includes Indicated and Inferred material reported using a 1.5g/t Au cut-off.

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that the PEA will be realized.

A process facility of 4,000 tpd (1.4 million tonnes per year [Mtpy]) is used for the target mine production scheduling. The annual production rate schedule is based on unit rates developed by SRK consistent with industry norms. Backfill has been sequenced and typically includes a 14-day lag prior to mining stopes adjacent to backfill. The backfill sequence needs further refinement and detail at the next level of study.

The mining operation schedule is based on 365 days per year, seven days per week, with two 12 hour shifts each day. A production rate of 4,000 tpd was targeted with ramp-up to full production as quickly as possible. Development drifts begin in January of 2021 and production commences mid-year 2023. Table 1-6 shows the annual production schedule completed using iGantt scheduling software based on these assumptions.

Table 1-6: MDZ Annual Mining Schedule

Year	Mineralized Tonnes (kt)	Waste Tonnes (kt)	Mineralization Au (g/t)	Mineralization Ag (g/t)	Development Length (m)	Backfill Volume (m^3)
2021	-	180	-	-	2,790	-
2022	52	375	2.96	3.17	6,987	-
2023	831	318	3.21	3.79	9,966	222,594
2024	1,400	172	3.08	3.73	4,339	528,620
2025	1,401	67	3.48	4.09	2,273	536,495
2026	1,400	129	3.16	4.65	4,490	519,407
2027	1,397	242	2.59	3.57	5,968	514,815
2028	1,342	292	2.30	4.20	7,143	494,940
2029	1,400	214	2.26	5.19	6,272	511,787
2030	1,400	189	2.16	4.67	5,842	504,879
2031	1,400	175	2.05	2.49	5,780	502,542
2032	1,400	86	2.09	1.64	3,757	520,709
2033	1,400	92	2.18	1.79	3,871	516,445
2034	1,400	77	2.18	2.02	3,531	517,480
2035	1,400	55	2.22	2.82	3,261	522,577
2036	1,400	76	2.36	3.27	4,090	502,255
2037	1,205	2	2.47	3.06	122	488,351
2038	592	-	2.57	3.14	-	246,480
Totals	**20,821**	**2,739**	**2.50**	**3.38**	**80,482**	**7,650,376**

Source: SRK, 2019

Stope lengths vary throughout the deposit ranging from 5 m to a maximum of 20 m giving a range of approximately 3,000 to 12,500 t per stope. After bottom and top accesses are established a slot raise will be developed at the far end of the stope (hangingwall side). Drilling will continue with the longhole drill using a fan shaped pattern. Holes will be loaded with bulk emulsion and stope blasting will commence in the slot and subsequently rings will be blasted retreating toward the level access. The mining will commence in a primary/secondary stoping sequence that includes time for the paste backfill to be placed and cure to a minimum strength requirement. The lag between backfill and production on the adjacent stope is typically 14 days. Mine development will be ongoing with development of the main ramps and footwall access continuing ahead of mining.

The mine ground support plan will include bolting per the ground support plan supplementing with wire mesh, shotcrete, and additional support where required. Cable bolts are expected to be utilized on the brows of the stopes and in certain larger intersections or infrastructure locations where warranted.

The ventilation system will include fresh air down the main haulage decline and a new fresh air decline located near the existing Marmato operation. The air will flow across the main access levels and out through air raises at the ends of the stope level. The exhaust air will then exit the mine through two new exhaust declines located near the existing Marmato operation. Fans will be located in the two exhaust declines. Total air requirements are estimated at 370.5 m^3/s.

The mine pumping system will include a portable staged system for development and permanent pumping stations located at the base of the MDZ upper zone and the bottom of the MDZ bottom zone. The pumping systems will be designed to handle 220 L/s and operate at 110 L/s.

The mine will have emergency egress through the fresh air intake at the Marmato operation site, through the existing Marmato mine and an emergency hoist located at the bottom of the MDZ bottom zone. A stench gas system will also be available for use in the MDZ mine. Additionally, multi-person refuge chambers are included that will be located in active working areas over the life of mine (LoM).

A new MDZ Project main project substation will provide power to feed a new mine substation located on the surface near the MDZ portal. The mine substation will feed the underground electrical systems, infrastructure, and equipment power down the main haulage decline via a 13.2 kV feed line. The 13.2 kV power will feed throughout the mine to main load centers where the power will be stepped down to 480 V for underground equipment use. Feeds will be provided at 220 V for auxiliary use in the shops and for smaller loads such as fans, pumps, and auxiliary lighting. A diesel backup generator at the surface will supply backup power for the required ventilation systems to maintain minimum ventilation requirements in the case of emergency.

There will be a staff of approximately 324 people operating the mine. These include approximately 25 technical and management personnel and 299 mine operating and shift personnel. There will be approximately 99 people on site during a typical day shift and 74 during a night shift.

The mine will operate with a large mechanical fleet of equipment including the typical equipment for a longhole stoping operation including longhole drills, two boom jumbos, 17t LHDs, 50 t trucks, explosives charges, cable and mechanical bolters, and transmixer truck and shotcrete equipment. Support equipment includes a grader, small skidsteers, telehandlers, maintenance equipment, scissor lifts, and employee transport equipment.

1.7.3 Hydrogeology

The mine area is located in the hydrogeological regional area of Magdalena Cauca, specifically in The Cauca River catchment (Celdas Department). The region is comprised of igneous and metamorphic rocks with limited groundwater storage capacity and hydraulic conductivity (IDEAM, 2013). The porphyry units represent the main hydrogeological units in the mine area, with a low hydraulic conductivity and limited groundwater storage capacity. Groundwater flow is compartmentalized within structural blocks with limited hydraulic communication across fault boundaries due to fault gouge, weathering, or an offset of geological units (Knight Piésold, 2012).

A hydrogeological field investigation was conducted in 2011 to 2012 (Knight Piésold, 2012). The investigation included hydraulic tests performed in the porphyry intrusion units and the construction of three piezometers in the underground mine and 11 in the ground surface.

Water levels recorded thru 2012 show elevations from to 661 to 2,022 m elevation, following the topography at 100 m depth. A depressurization zone was detected in the underground piezometers where the water levels have a horizontal trend. The shape or extent of the depressurization zone is currently unknown. In a more regional scale, the groundwater flows West to East, following the topographical gradient to the Cauca River, located at 668 m elevation, which represents the main discharge for the hydrogeological system.

105 and 41 Lugeon test in porphyry units were conducted in underground angled holes and vertical peripherical piezometers respectively, also 9 Le Franc tests were performed in saprolitic units (Knight Piésold, 2012). As a result, the geometrical mean of hydraulic conductivity values ranges from 1.4 $x10^{-2}$ m/d to 6.4 $x10^{-2}$ m/d in the porphyry units and approximately 5.9 $x10^{-2}$ m/d in the saprolite. It is

important to note several of the hydraulic tests mentioned above were completed in unsaturated conditions and/or into the shallow portion of the intrusive units or into the development mine zone area, consequentially the hydraulic conductivity values could be considered overestimated and not representative of the groundwater flow conditions toward the underground mine.

Mine Dewatering

The measured monthly average of total dewatering in the Marmato mine is 37 L/s, varying from 26.8 L/s to 46.4 L/s. Strong seasonal trends were not observed, however a decrease of approximately 20 L/s can be observed in the last 12 months. A major structure zone with significant water flow (7 to 8 L/s) was detected at levels 17 and 21 to the north of the Criminal Fault.

The dewatering flow is a combination of groundwater inflows and water content in the backfill material (50% of water). According to Marmato operational personnel, the contribution of the back-fill material is 7 to 14 L/s, depending on the number of hydraulic backfill equipment in operation. Therefore, the average fresh groundwater inflow into the mine could vary from 23 to 30 L/s.

A preliminary analytical model was used for estimating groundwater flow into the future mine. A Theis solution method corrected for an unconfined aquifer was implemented and calibrated using the current water discharge condition. As a result, a total of 70 L/s and 110 L/s were estimated from mine sequence from level 765 and 430 respectively.

The bottom of the mine will be located 262 m below and 2.5 km to the east of the Cauca River. There is a risk of water intrusion from the river bed. Structural features similar to those detected to the north of the Criminal Fault could connect mine developments with the river. Further hydrogeological investigations of this area are required to evaluate potential significant increments in groundwater inflow.

1.7.4 Geotechnical

Preliminary indications are that the rock mass characteristics are similar between the upper vein mining areas with the lower MDZ mining areas and thus, for this PEA SRK treated them as similar. These two mining areas are planned to utilize different mining methods because of the nature of the mineralization, however, the same geotechnical design methodologies would apply given the similar nature of the rock mass.

At the time of this report, GCM has drilled 35 drillholes, totaling close to 8,800 m, to characterize the MDZ area. For the vein zone, a rock mass characterization program was completed by Knight Piésold in March 2012 (Knight Piésold, 2012), including 15 coreholes drilled from the surface for a total of about 6,300 m of core. Additional drillholes are in progress. The field investigation included geotechnical core logging and core sample collection for laboratory strength testing.

Based on the rock mass strength parameters determined from the characterization for the PEA study, SRK completed an empirical stope stability assessment, sill pillar design and ground support requirements for the MDZ area. Based on Mathews' method, the recommended stope dimensions for all levels are:

- Stope height: 25 m;
- Stope width: 10 m; and
- Stope length: 20 m.

Dilution into the stopes was estimated using an empirical design chart for Equivalent Linear Overbreak/Slough (ELOS). For the stope parameters specified above, the ELOS chart estimates about 0.35 m of sloughing. Assuming dilution in the near vertical stopes comes from both the hangingwall and footwall rock.

To determine adequate sill pillar dimensions, SRK considered two different empirical methodologies; the Scaled Span Method (1990) and the Lunder and Pakalnis Empirical Method (1997). Based on these results, SRK recommends a sill pillar thickness of 10 m, assuming short-term mine exposure and that the stopes will be tightly backfilled.

Ground support requirements were estimated using empirical support charts developed by Barton (1974). SRK considers that empirical methods are acceptable for a PEA level study. However, for further stages of the Project, SRK recommends that a numerical model is built to reevaluate stope stability for the different portions of the mine, and to verify sill pillar dimensions and ground support requirements. The proposed stope and sill pillar designs are valid for a PEA level only and cannot be applied for construction.

1.8 Recovery Methods

GCM operates a 1,200 tpd process plant to recover Au and Ag values from material produced from current Marmato mining operations in the Upper Zone. In addition, GCM is evaluating the development of the MDZ, which is below the current mining operations, and the construction of a new 4,000 tpd plant to process material soley from the MDZ.

The existing Marmato process plant flowsheet incorporates unit operations that are standard to the industry and includes:

- Three-stage crushing;
- Closed circuit ball mill grinding;
- Gravity concentration;
- Flotation;
- Flotation and gravity concentrate regrind;
- Cyanidation of the flotation and gravity concentrates;
- Counter-current-decantation;
- Merrill-Crowe zinc precipitation; and
- Smelting of precipitates to produce final dore' product.

The current Marmato process plant performance is summarized in Table 1-7 for the period from 2013 to 2019 (January to July). During this period mineralized tonnes processed has increased from 274,191 to 338,902 tpy while grades have declined slightly from 2.90 g/t Au in 2013 to 2.45 g/t Au in 2019 and silver grades have ranged from 12.36 to 9.13 g/t Ag. Overall gold recovery has ranged from 89.0 to 83.7% and has averaged about 86.5% over the past three years. Silver recovery has ranged from 41.1 to 31.5% and has averaged 33.2% over the past three years. Gold production has increased from 22,566 ounces in 2013 to 24,909 ounces in 2018.

Table 1-7: Summary of Marmato Plant Operating Performance

Parameter	2013	2014	2015	2016	2017	2018	2019 (Jan to Jul)
Mineralized Tonnes	274,191	295,023	303,279	341,309	365,119	338,902	211,817
Mineralization Grade							
Au (g/t)	2.90	2.85	2.79	2.56	2.48	2.67	2.45
Ag (g/t)	12.36	9.13	9.33	9.24	9.61	10.53	10.41
Metal Recovery							
Au (%)	88.6	89.0	88.0	83.7	86.8	85.5	87.2
Ag (%)	36.6	41.1	37.9	35.8	34.9	33.2	31.5
Metal Produced							
Au (Ounces)	22,566	24,113	23,954	23,449	25,163	24,909	14,538
Ag (Ounces)	39,916	34,753	34,490	36,318	39,524	37,522	22,878

Source: GCM, 2019

Metallurgical testwork was conducted to evaluate three different process flowsheet options for processing material from the MDZ in the new plant including:

- Whole-ore cyanidation;
- Gravity concentration followed by cyanidation of the gravity tailing; and
- Gravity concentration followed by Au and Ag flotation from the gravity tailing and cyanidation of the flotation concentrate.

After conducting a trade-off study, a process flowsheet that includes gravity concentration followed by cyanidation of the gravity tailing was selected for processing MDZ material as this flowsheet option offers higher overall Au recoveries along with lower estimated capital and operating costs. The process flowsheet will incorporate process unit operations that are standard to the industry, including: primary crushing; SAG mill/ball mill grinding; agitated cyanide leaching; Au and Ag carbon-in-pulp (CIP) adsorption onto activated carbon; gold and silver desorption; and electrowinning and refining.

Process operating costs for the new 4,000 tpd process plant are estimated at US$13.34/t processed. The major contributors to operating cost are labor, reagents, comminution consumables and power.

The capital cost for the 4,000 tpd process plant is estimated at US$65.6 million and is considered at a PEA level with a +/-50% level of accuracy.

1.9 Project Infrastructure

The existing Marmato Project has a mature and functioning infrastructure system including all the necessary facilities and supporting utilities to produce at the planned production levels. The current facilities include a security checkpoint that provides access to the office and administrative office area. The facilities include employee motorcycle parking, meeting area, cafeteria, multiple shops and warehouses, a camp with cafeteria, exercise and sports field, equipment storage yards, compressor station, welding shop, a 500 kW backup generator, processing plant, underground mine, explosives storage a short distance from the mine that is managed by the military, main power substation and distribution powerlines with motor control centers at key loads. The site has three portals that access the mine workings. The site is considering additional water supply from the Cauca River to supplement water availability during the dry season.

The MDZ Project infrastructure will be developed on a separate greenfield site approximately 2.7 km north of the existing Project. The new site will require a new access road off the existing El Llano access road to access a new processing facility and mine portal with access to the MDZ. The new infrastructure will include a substation that ties into local power system and local distribution to the mine substation and processing facility. Mine surface facilities include the mine portal, truck shop, tire pad, fuel storage and fuel distribution system, mine waste rock storage, paste backfill plant, and run of mine (RoM) stockpile. Explosives storage is planned to be at the existing mine explosives storage. The mill will have a crushing area with a surge stockpile feeding the main processing plant. Support facilities will include warehousing, shops, offices, and laydown yards. Parking will be provided near the entrance to the MDZ site and a security gate for restricted access will be constructed at the entrance to the facility. An expansion camp near the existing camp is planned to house the additional employees at the MDZ Project. Water will be supplied by mine dewatering, recycled water from the tailings storage facilities, and from supplemental water drawn from the Cauca River as needed.

The area already supports a significant mining population and skilled labor will be available from the region.

1.10 Tailings Management Facilties

The existing Marmato tailings storage facility is estimated to provide three to four years of additional capacity, after which new dry stack tailings storage facilities (DSTF) would need to accommodate tailings through the currently-estimated 16-year LoM. Currently the tailings are sent to two unlined settling ponds. The underdrain water from these ponds are treated with flocculant and either sent back to the plant for user in the process or discharged under permit to the nearby stream. The tailings are excavated from the settling ponds once sufficiently dewatered and hauled via trucks to the existing tailings storage facilities (TSF).

To provide capacity for continued mining in the Upper Zone and expansion into the MDZ and based on the results of a preliminary siting study completed as part of PEA preparation, two separate DSTFs would be required to provide capacity for tailings produced through the currently estimated 16-year LoM. DSTF-1 and DSTF-2 were identified as the most feasible locations for DSTF construction and consideration in PEA preparation. GCM is currently evaluating additional siting options that may require evaluation at later project stages.

DSTF-1 would be constructed and receive dewatered tailings from both the Upper and the MDZ for the first one to two years after commissioning of the facility. Filter presses were incorporated into the design to achieve a target moisture content of 15% and facilitate tailings management within the relatively steep and confined valleys at the site. After the filter plant installation, tailings generated from mining in the MDZ would be filtered, trucked and stacked at DSTF-1 until it reaches capacity at approximately year 7, at which time filtered tailings would be hauled to DSTF-2 adjacent to the plant location.

Rock starter embankments would be constructed at DSTF-1 and DSTF-2 using waste rock generated from underground mining or from a nearby suitable borrow source. Filtered tailings would be trucked to the DSTF, spread using a dozer, mixed with 0.5%-1% cement and compacted in thin lifts with vibratory or sheepsfoot compaction equipment.

The DSTF would be constructed with underdrain systems consisting of central header pipes and connecting transverse drains to manage any seeps, springs or upwelling groundwater within the proposed DSTF footprints. It is currently assumed that the DSTF’s would not be lined with geomembrane due the tailings being dewatered via filter presses and amended with cement. Stormwater channels and geomembrane-lined top-deck transfer ponds would be constructed to manage SRK and GCM are currently coordinating a geotechnical field program to characterize foundation conditions, potential rock starter embankment construction materials, and filtered tailings. Laboratory testing will also be completed to develop design criteria and determine required cement amendment requirements.

1.11 Environmental Studies and Permitting

The existing Marmato Project predates the regulatory requirements to prepare an environmental impact assessment as part of the permitting process. Instead, the operations were authorized through the approval of an Environmental Management Plan ("*Plan de Manejo Ambiental*") (PMA). The PMA for Marmato was approved by the regional environmental authority (*Corporación Autónoma Regional del Caldas* or Corpocaldas) on October 29, 2001 under Resolution 0496, File No. 616. The site-specific PMA covers environmental studies and required management procedures and practices.

Routine monitoring is conducted on permitted domestic wastewater discharges, industrial wastewater discharges, and air quality emissions from stationary sources. The results of this monitoring is reported to Corpocaldas. Environmental management programs for the current Marmato operations total approximately US$425,200 annually.

With respect to environmental geochemistry at Marmato:

- Acid-generating sulfide minerals identified in the deposit include pyrite, arsenopyrite, iron-bearing sphalerite, pyrrhotite, and chalcopyrite (SRK, 2017);
- Samples of groundwater discharging into the underground are predominantly acidic;
- The underground water samples contain elevated metal(loid) concentrations; and
- Geochemical properties of current and future waste rock and tailings are currently unknown.

Very little waste rock is generated by the underground operations at Marmato. What little waste rock is generated is used as structural backfill in the underground workings or on the surface for small construction projects, such as maintenance of roads. Since so little, if any, waste rock is brought to the surface, a comprehensive geochemical characterization program is probably not warranted on this surface located material, and only opportunistic sampling and testing of construction materials is probably necessary. At the moment, Corpocaldas does not require the testing of mine waste materials; only effluent discharges. This is likely to change in the future, as source control becomes more of the norm in this jurisdiction.

Tailings from the cyanide leach circuit are detoxified before being sent to dewatering ponds for later drystacking disposal. The clarified overflow water from these ponds is pumped back to the plant for use in the process. Excess water, not needed at the plant, is discharged under permit to the adjacent stream, Quebrada Cascabel. Monitoring of the residual tailings to determine whether or not they are classifiable as 'hazardous' is accomplished through Corrosive, Reactive, Explosive, Toxic, Inflammable, Pathogen [biological] (CRETIP) analyses. Toxicity analyses were carried out by the Universidad Pontificia Bolivariana on cyanides (CN) and metals (chromium, mercury and lead). The results support the classification of the tailings as non-toxic for the metals based on comparisons to the maximum concentration thresholds established by Decree 4741 of 2005. The analyses also showed that total CN was below the threshold allowed in Decree 1594 of 1984 for water discharges.

For the MDZ expansion of the Marmato Project, GCM intends to continue with a similar approach to tailings and tailings water management. However, rather than using less efficient settling ponds, the tailings will be filter pressed. The “dry” tailings will be transported to the new disposal facility, mixed with cement, and stacked in a configuration that minimizes surface runoff. To the extent practicable, contact water collected on the deck of the DSTF will be reused in the process; excess water will be discharged under permit. A critical driver of environmental impacts from tailings is whether contact water will be contained. The current and predicted future quality of contact water needs to be determined. Two components of potential chemical loading need to be estimated:

- Loading to surface water due to TSF runoff; and
- Loading to groundwater through seepage from the base of the TSF.

The Marmato Project is authorized under a number of resolutions issued by Corpocaldas in the name of GCM (previously named Mineros Nacionales S.A.S.) These include, among others:

- Environmental Management Plan or PMA (Resolution No. 0496);
- Various water concessions; and
- Discharge permits (Resolutions 270 and 255).

According to GCM, all of the resolutions for operation of the existing Marmato Project are current and valid, or in the process of being renewed. Most will need to be amended as part of the authorization of the deep zone expansion of the Marmato Project to include the increases in tonnage to be mined, new process facilities, and tailings management area.

The current PMA authorizes the mining and processing of up to 1,500 tpd of ore. The current processing plant has capacity of 1,200 tpd. The PMA will need to be modified to allow for the proposed MDZ expansion project, which envisions a rate of an extra 4,000 tpd in a second processing plant to be constructed. By regulation, the total of mined material (including waste and material) cannot exceed 2 Mtpy in order for Corpocaldas (regional Environmental Authority) to remain as the permitting authority. If more than 2 Mtpy is mined, then the PMA will need to be submitted to and authorized by the federal authority, ANLA (Environmental License National Authority). During construction, Channel Occupancy Permits will most likely need to be obtained for the new tailings site, the tailings pipeline corridor, the process plant site, and the site of the underground portal (bocamina). Likewise, a Forest Exploitation Permit may be needed for areas of proposed surface disturbance with trees (diameter at breast height [DBH] more than 10 cm).

Additional baseline data collection will need to be completed on the new areas proposed for the plant site, portal, ancillary facilities, and tailings disposal area(s). GCM is currently in the process of bidding out the baseline programs, and also the modifications to the PMA to an environmental consultant (under a single contract). The plan is to award this contract in Q4 2019, with an anticipated completion by end of Q1 2020 and submittal to Corpocaldas.

The 2001 PMA for Marmato specifically requires the management of the social component of the Project. GCM is required to maintain records of all community activities (including number of participants, topics, duration, etc.), which is to be turned over to Corpocaldas as part of the ongoing monitoring programs. As part of the social management and monitoring program, GCM has developed a social investment model which seeks to promote the development of communities in the area of influence, with the purpose of contributing to the consolidation of society and fostering economic development (Economic Development), guaranteeing the care and respect for the environment (Environmental Development), and supporting and participating in actions aimed at improving the quality of life and well-being of its inhabitants (Social Development and Promotion of Solidarity Actions).

Article 209 of Law 685 of 2001 requires that the concession holder, upon termination of the agreement, shall undertake the necessary environmental measures for the proper reclamation and closure of the operation. To ensure that these activities are carried out, the Environmental Insurance Policy shall remain in effect for three years from the date of termination of the contract. Little else regarding the specifics of mine closure is provided in the Law. Decree 2820 Article 40 Paragraph 2 of 2010 specifically indicates that the concession holder must submit a plan for dismantling and abandonment of the Project.

Reclamation and closure costs for the current operation are provided in the May 2019 reclamation and closure plan. These costs are based on percentages of costs to build the facilities. The plan does not provide the basis for the percentages. The reclamation and closure cost estimate provided totals 20,128,000,000 pesos (US$5.8 million based on exchange rate of 3,455 to 1). A requirement for long-term post-closure water treatment, if any, would significantly increase this estimate.

Given the conceptual design nature of the MDZ expansion of the Marmato Project, a detailed closure cost assessment cannot be made at this time. However, it is not unreasonable to assume that the closure cost would be 1.5 to 2 times the current estimate, given the increase in production anticipated for the new operations and the construction of a new plant and tailings disposal area(s). A cost of US$6.1 million was included in the technical economic model to account for a second tailings disposal facility. These costs will need to be revisited more accurately during the PFS phase of project development.

1.12 Capital and Operating Costs

1.12.1 Marmato Upper Zone Capital Costs

As the Marmato Upper Zone is a currently operating underground mine, the estimate of capital includes only sustaining capital to maintain the equipment and all supporting infrastructure necessary to continue operations until the end of the projected production schedule. The cost estimate is based on budgetary estimates prepared by Marmato and reviewed by SRK. The estimate indicates that the Project requires sustaining capital of US$40.5 million to support the projected production schedule throughout the LoM. Table 1-8 summarizes the LoM sustaining capital estimate and Table 1-9 and Table 1-10 present the same estimate by year.

Table 1-8: Marmato Upper Zone Sustaining Capital (LoM)

Description	LoM ($000's)
Drilling	1,450
Development	19,501
Mine Equipment	11,545
Other Mine	2,690
Surface	4,330
Plant	1,010
Total	**40,526**

Source: Gran Colombia/SRK, 2019

Table 1-9: Marmato Upper Zone Sustaining Capital (2019 to 2026) ($000's)

Description	2019	2020	2021	2022	2023	2024	2025	2026
Drilling	100	100	100	100	100	100	100	100
Development	1,186	2,014	1,374	1,545	1,395	1,268	1,873	1,623
Mine Equipment	920	2,155	1,640	725	580	515	450	645
Other Mine	280	550	720	250	330	300	80	-
Surface	220	480	300	330	250	250	350	250
Plant	50	50	50	50	50	50	50	260
Total	**2,756**	**5,349**	**4,184**	**3,000**	**2,705**	**2,483**	**2,903**	**2,878**

Source: Gran Colombia/SRK, 2019

Table 1-10: Marmato Upper Zone Sustaining Capital (2027 to 2034) ($000's)

Description	2027	2028	2029	2030	2031	2032	2033	2034
Drilling	100	100	100	100	100	100	50	-
Development	816	1,159	2,096	1,284	966	315	289	297
Mine Equipment	410	540	470	550	565	590	390	400
Other Mine	130	-	-	50	-	-	-	-
Surface	350	250	250	150	250	250	150	250
Plant	50	50	50	50	50	50	50	50
Total	**1,856**	**2,099**	**2,966**	**2,184**	**1,931**	**1,305**	**929**	**997**

Source: Gran Colombia/SRK, 2019

1.12.2 MDZ Capital Costs

The MDZ is a lower part of the deposit that is undeveloped. Before Marmato can exploit this part of the deposit it will have to expand the existing operation. The expansion is planned to be executed between the years of 2021 and 2022 and the following areas will require an investment of capital for the following areas:

- Exploration Drilling;
- Mine Development;
- Mining Equipment;
- Surface Facilities and Equipment;
- Underground Facilities and Equipment;
- Power Supply;
- Access Road;
- Camp;
- Other Supporting Infrastructure;
- Mineral Processing Plant; and
- Tailings Storage Facility.

The capital cost estimates prepared for the expansion into this mining area also include estimates for Engineering, Procurement and Construction Management (EPCM) and the owner's cost to manage it. The cost estimate is based on cost models prepared by SRK with site specific inputs from Marmato. The estimate indicate that the expansion will require an investment of US$268.9 million, including an estimated capital of US$215.1 million plus 25% contingency of US$53.8 million. Table 1-11 summarizes the expansion capital estimate.

Table 1-11: Marmato Deeps Zone Construction Capital ($000's)

Description	Total	2021	2022
Exploration Drilling	2,600	1,300	1,300
Development	52,021	14,790	37,232
Mining Equipment	26,956	923	26,033
Surface Facilities and Equipment	12,114	1,264	10,850
UG Facilities and Equipment	499	-	499
Power Supply	675	169	506
Access Road	225	169	56
Camp	750	188	563
Pump Station	200	-	200
Process Plant	65,551	26,221	39,331
Tailings Storage Facility	35,953	-	35,953
EPCM	7,562	268	7,294
Owners	10,000	5,000	5,000
Sub-Total	**215,107**	**50,291**	**164,816**
Contingency (25%)	53,777	12,573	41,204
Total	**268,884**	**62,864**	**206,020**

Source: Gran Colombia/SRK, 2019

Additionally, the MDZ will require sustaining capital to maintain the equipment and all supporting infrastructure necessary to continue operations until the end of its projected production schedule. The estimates indicate that the Project requires a sustaining capital of US$140.7 million to support the projected production schedule through the LoM. Table 1-12 summarizes the LoM sustaining capital estimate Table 1-13 and Table 1-14 present the same estimate by year.

Table 1-12: Marmato Deeps Zone Sustaining Capital (LoM)

Description	LoM ($000s)
Drilling	14,500
Development	55,848
Mine Equipment	22,245
UG Facilities and Equipment	8,150
Surface	280
Plant	7,000
Tailings	22,804
Other Sustaining	3,812
Project Closure	6,100
Total	**140,739**

Source: Gran Colombia/SRK, 2019

Table 1-13: Marmato Deeps Zone Sustaining Capital (2023 to 2030)($000's)

Description	2023	2024	2025	2026	2027	2028	2029	2030
Drilling	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000
Development	9,701	5,878	829	877	8,385	10,434	6,486	5,866
Mine Equipment	5,727	800	3,564	3,536	935	2,318	2,352	1,004
UG Facilities and Equipment	4,603	1,440	463	375	200	75	920	75
Surface	280	-	-	-	-	-	-	-
Plant	-	500	500	500	500	500	500	500
Tailings	842	842	842	871	842	842	7,608	5,730
Other Sustaining	219	237	242	247	250	251	262	263
Project Closure	-	-	-	-	-	-	-	-
Total	**22,371**	**10,696**	**7,440**	**7,406**	**12,112**	**15,419**	**19,128**	**14,438**

Source: Gran Colombia/SRK, 2019

Table 1-14: Marmato Deeps Zone Sustaining Capital (2031 to 2038) ($000's)

Description	2031	2032	2033	2034	2035	2036	2037	2038
Drilling	1,000	1,000	1,000	1,000	1,000	1,000	500	-
Development	6,010	1,029	353	-	-	-	-	-
Mine Equipment	1,004	1,004	-	-	-	-	-	-
UG Facilities and Equipment	-	-	-	-	-	-	-	-
Surface	-	-	-	-	-	-	-	-
Plant	500	500	500	500	500	500	500	-
Tailings	1,245	871	842	842	585	-	-	-
Other Sustaining	263	263	263	263	263	263	263	-
Project Closure	-	-	-	-	-	-	-	6,100
Total	**10,023**	**4,668**	**2,958**	**2,605**	**2,348**	**1,763**	**1,263**	**6,100**

Source: Gran Colombia/SRK, 2019

1.12.3 Marmato Operating Costs

SRK and Marmato prepared the estimate of operating costs for the PEA's production schedule. These costs were subdivided into the following categories:

- Mining Operating Expenditure;
- Processing Operating Expenditure; and
- Site G&A Operating Expenditure.

Marmato Upper Zone LoM cost estimate is presented in Table 1-15 and MDZ LoM cost estimate is presented in Table 1-16.

Table 1-15: Marmato Upper Zone Operating Costs Summary

Description	LoM (US$/t-Ore)	LoM (US$000's)
Mining	46.71	258,918
Process	15.88	88,003
G&A	11.85	65,667
Total Operating	**74.44**	**412,587**

Source: Gran Colombia/SRK, 2019

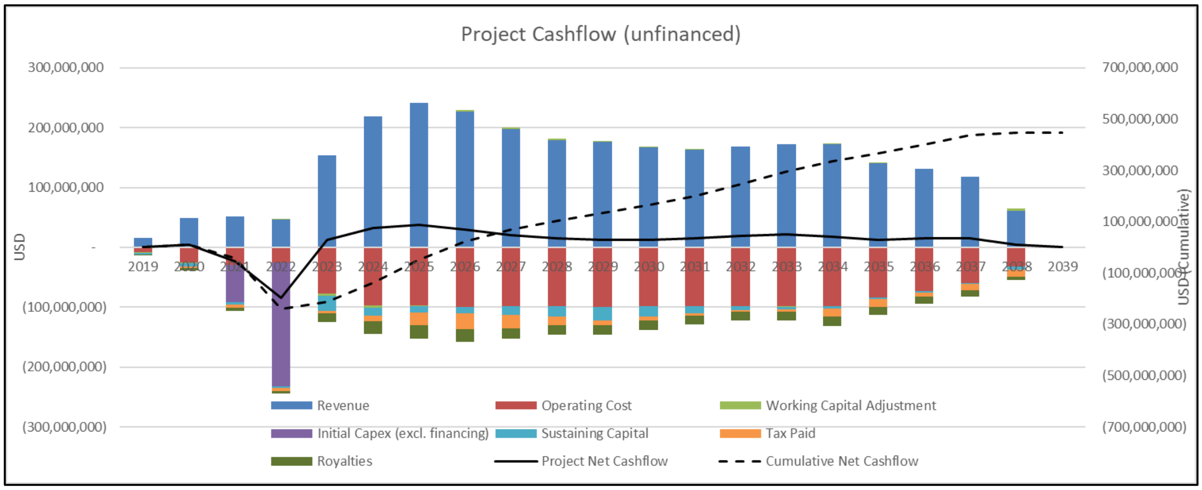
Table 1-16: Marmato Deeps Zone Operating Costs Summary

Description	LoM (US$/t-Ore)	LoM (US$000's)
Mining	32.43	675,150
Process	16.14	336,005
G&A	3.46	72,000
Total Operating	**52.02**	**1,083,155**

Source: Gran Colombia/SRK, 2019

1.13 Economic Analysis

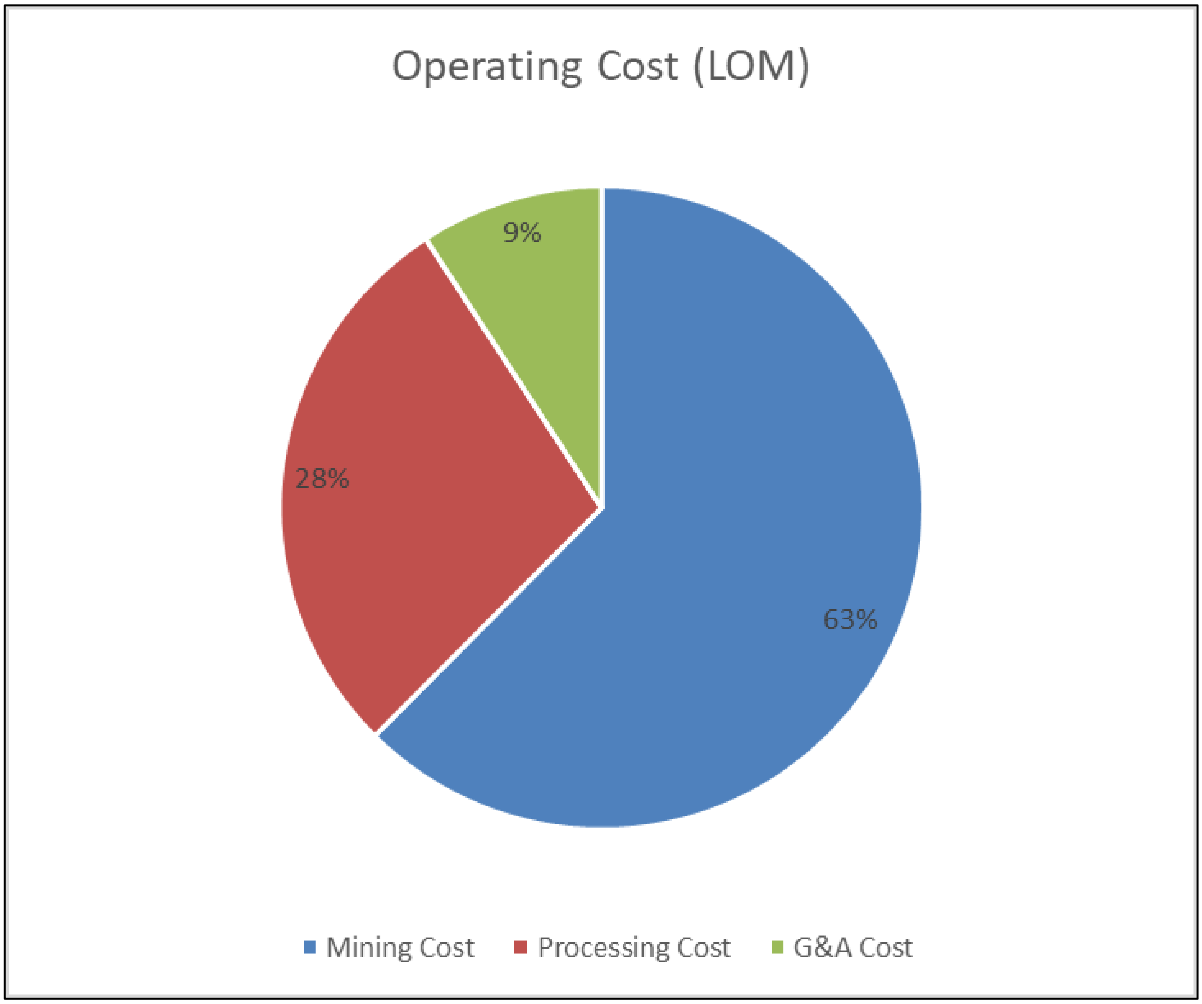
The valuation results of the Marmato Project indicate that the Project has an after-tax Net Present Value (NPV) of approximately US$206.8 million, based on a 5% discount rate, gold price of US$1,300/oz and silver price of US$17.00/oz. The operation is projected to have negative cash flows during the years 2021 and 2022, when the MDZ is installed, with payback for the expansion expected by 2026. The annual free cash flow profile of the Project is presented in Figure 1-2. The full annual TEM is in Appendix D.



Source: SRK, 2019

Figure 1-2: Marmato After-Tax Free Cash Flow, Capital and Metal Production

Indicative economic results are presented in Table 1-17. The Project is a gold operation with a sub-product of silver, where gold represents 99% of the total projected revenue and silver the remaining 1%. The underground mining cost is the heaviest burden on the operation representing 63% of the operating cost, as presented in Figure 1-3.



Source: SRK, 2019

Figure 1-3: Marmato Operating Cost Break-Down

Table 1-17: Marmato Indicative Economic Results

LoM Cash Flow		
Total Revenue	USD	2,851,475,063
Mining Cost	USD	(934,068,009)
Processing Cost	USD	(424,007,603)
G&A Cost	USD	(137,666,667)
Total Opex	USD	(1,495,742,278)
Operating Margin	USD	1,355,732,785
Operating Margin Ratio	%	48%
Taxes Paid	USD	(195,035,875)
Free Cashflow	USD	717,096,368
LoM Capital		
Expansion CAPEX	USD	(268,884,037)
Sustaining CAPEX	USD	(181,264,836)
Total LOM CAPEX	USD	(450,148,872)
Before Tax		
Free Cash Flow	USD	643,248,206
NPV @ 5%	USD	322,995,835
NPV @ 8%	USD	214,836,234
NPV @ 10%	USD	163,296,934
IRR	%	28%
After Tax		
Free Cash Flow	USD	448,212,331
NPV @ 5%	USD	206,821,360
NPV @ 8%	USD	126,595,938
NPV @ 10%	USD	88,893,940
IRR	%	20%
Payback	Year	2026

Source: SRK, 2019

The estimated All-in Sustaining Costs (AISC), including sustaining capital, is US$882/Au-oz. Table 1-18 presents the breakdown of the Marmato cash costs.

Table 1-18: LOM All-in Sustaining Cost Breakdown

Description	Unit	Value
Mining	USD/Au	428
Processing	USD/Au	194
G&A	USD/Au	63
Refining	USD/Au	6
Royalty	USD/Au	120
Sustaining Capital	USD/Au	83
Silver Credit	USD/Au	(14)
AISC	**USD/Au**	**882**

SRK's standard Cash Cost reporting methodology for NI 43-101 reports includes smelting/refining costs; whereas Gran Colombia's basis of reporting treats these costs as a reduction of realized gold price (the refinery discounts the selling price by a factor to cover these charges) and excludes them from its reported "total cash cost per ounce".
Source: SRK, 2019

1.14 Conclusions and Recommendations

1.15 Conclusions and Recommendations

The Project requires a PFS level design effort to complete tradeoff studies and further evaluate the feasibility and viability of the Project.

1.15.1 Mineral Resources

SRK is currently working with GCM's geologists to optimize the remainder of the 2019 drilling program, with a focus on increasing the confidence in the MDZ. The technical studies will aim to infill the drilling spacing to a 50 by 50 m grid in the upper portions of the MDZ, and potentially increase the Inferred Resources at the end. SRK is also working on a number of engineering studies to support the future development of potential maiden Mineral Reserves for the Project. SRK anticipates the drilling to be completed during Q4 2019 with an updated Mineral Resource produced during the same time period.

1.15.2 Geotechnical

The current geotechnical design parameters are based on characterization data from 35 diamond drillholes, for which six were drilled deep in the orebody. Continued geotechnical work to further characterize the geotechnical characteristics of the deposit in preparation for PFS is recommended.

1.15.3 Tailings Management Facilities

During the next phase of study, the DSTF footprints need to be finalized and detailed characterization of the proposed DSTF foundations and the tailings material itself needs to be completed, in addition to identifying and testing a suitable borrow source for rock starter embankment construction and outer slope cladding for erosion protection. The ability to filter the tailings and achieve a target moisture content of about 15% by weight must be established. The depth to groundwater and the source(s) of perennial water within each DSTF footprint must be determined, and the ability to permit the DSTFs without a base liner in natural drainages at the site must be established.

1.15.4 Mining

SRK recommends review of the mining parameters for the project in order to optimize the mining plan for the deposit. Work to increase resolution on ventilation, geotechnical and hydrogeological aspects of the project is required for completion of a PFS. This information will provide a basis for a PFS level mine design.

1.15.5 Metallurgy and Mineral Processing

Additional metallurgical studies will need to be conducted during the next phase of study. This work has commenced and is being done at SGS Canada. This work is being conducted on master composites and variability composites that are reprensentative of the MDZ. These studies are directed at optimizing process parameters for the selected process flowsheet and establish the process design criteria for the process plant.

1.15.6 Recovery Methods

During the next phase of study, process plant engineering and design will need to be conducted by a qualified engineering design based on process design criteria developed during the metallurgical program. This work has commenced and is being conducted by Ausenco. Process engineering and design will be conducted to a level of detail that will support capital and operating cost estimates at a +/-25% level of accuracy.

1.15.7 Infrastructure

GCM is currently working on retaining a third party engineering firm for completion of PFS level infrastructure engineering. GCM has estimated that a third party firm will be selected and contracted by the end of 2019.

1.15.8 Hydrogeology

A work program for development of PFS level hydrogeological parameters is planned as part of the next phase of work.

1.15.9 Environmental Studies and Permitting

The next phase of work should include the development of a detailed closure plan to inform closure cost estimate. Hydrogeological, surface water and geochemical work programs should be pursued to inform the development of the next level of study for the project. Baseline studies and the regulations impacting the proposed operations should be completed and reviewed in detail.

1.15.10 Project Economics

Preparation of a PFS level estimate of capital and operating costs will occur in the next phase of work. In addition, a program of tradeoff studies should be completed in an effort to optimize the economic result of the project.

2 Introduction

2.1 Terms of Reference and Purpose of the Report

This report was prepared as a PEA level NI 43-101 Technical Report for GCM by SRK on Marmato mine.

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in SRK's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by GCM or its assignee of the Project, subject to the terms and conditions of its contract with SRK and relevant securities legislation. The contract permits GCM or its assignee of the Project to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with GCM. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

The PEA is preliminary in nature, that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

This report provides Mineral Resource estimates, and a classification of resources prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 10, 2014 (CIM, 2014).

2.2 Qualifications of Consultants (SRK)

The Consultants preparing this technical report are specialists in the fields of geology, exploration, Mineral Resource and Mineral Reserve estimation and classification, underground mining, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the Consultants or any associates employed in the preparation of this report has any beneficial interest in GCM, The Consultants are not insiders, associates, or affiliates of GCM. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between GCM and the Consultants. The Consultants are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered Qualified Persons (QP), as defined in the NI 43-101 standard, for this report, and are members in good standing of appropriate professional institutions. QP certificates of authors are provided in Appendix A. The QP's are responsible for specific sections as follows:

- Ben Parsons, Principal Consultant (Resource Geologist) is the QP responsible for data verification, preparation of the geological model and the mineral resource estimate. Sections 4 through 12, 14, 23 and portions of Sections 1, 24, 25 and 26 summarized therefrom, of this Technical Report.
- Eric Olin, Principal Consultant (Metallurgy) is the QP responsible for Metallurgy Sections 13, 17, and portions of Sections 1, 24, 25 and 26 summarized therefrom, of this Technical Report.
- Fernando Rodrigues, Principal Consultant (Mining Engineering) is the QP responsible for Upper Zone Mining and Economics and related portions Sections 16.1, 16.3 (except for section 16.3.2), 19 and 22, and portions of Sections 1, 24, 25 and 26 summarized therefrom, of this Technical Report.
- Jeff Osborn, Principal Consultant (Mining Engineering) is the QP responsible for Infrastructure and Cost Estimation Sections 18.1, 18.2, 18.3, 18.5 and 21, and portions of Sections 1, 24, 25 and 26 summarized therefrom, of this Technical Report.
- Joanna Poeck, Principal Consultant (Mining Engineering) is the QP responsible for Section 15 and Section 16.4, and portions of Sections 1, 24, 25 and 26 summarized therefrom, of this Technical Report.
- John Tinucci, Principal Consultant (Geotechnical Engineering) is the QP responsible for Geotechnical Section 16.2 and portions of Sections 1, 24, 25 and 26 summarized therefrom, of this Technical Report.
- Joshua Sames, Senior Consultant (Civil Engineering) is the QP responsible for Tailings Section 18.4, and portions of Sections 1, 24, 25 and 26 summarized therefrom, of this Technical Report.
- Cristian Pereria, Senior Consultant (Hydrogeology) is the QP responsible for Hydrogeology Sections 16.3.2 and 16.4.2, and portions of Sections 1, 24, 25 and 26 summarized therefrom, of this Technical Report.
- David Hoekstra, Principal Consultant (Water Resource Engineering) is the QP responsible for Hydrology Section 20.2.4, and portions of Sections 1, 24, 25 and 26 summarized therefrom, of this Technical Report.
- David Bird, Associate Principal Consultant (Geochemistry) is the QP responsible for Geochemistry Sections 20.1.6, and portions of Sections 1, 24, 25 and 26 summarized therefrom, of this Technical Report.
- Mark Willow, Principal Consultant (Environmental) is the QP responsible for Environmental Section 20 (except section 20.1.6), and portions of Sections 1, 24, 25 and 26 summarized therefrom, of this Technical Report.

2.3 Details of Inspection

Table 2-1: Site Visit Participants

Personnel	Company	Expertise	Date(s) of Visit	Details of Inspection
Ben Parsons	SRK	Mineral Resources	June 11 to June 13, 2019 August 17, 2017 and March 12 to March 14, 2012	Underground Site visit levels 18 – 21, review latest drilling intersections, Underground Site visit levels 17 – 20, review latest drilling intersections, Underground Site visit, review latest drilling intersections.
Jeff Osborn	SRK	Mining/Infrastructure	July 16 to July 18, 2019	Surface Facilities and New MDZ location
			August 22 and 23, 2017	Underground and Surface Facilities including TSF as well as core shack area
Fernando Rodrigues	SRK	Mining/Reserves	August 22 and 23, 2017	Underground and Surface Facilities including TSF as well as core shack area
John Tinucci	SRK	Geotechnical	August 22 and 23, 2017	Underground and Surface Facilities including TSF as well as core shack area
Mark Willow	SRK	Environmental	December 1, 2016	Environmental Impact review
Cristian Pereira	SRK	Hydrogeology	August 12 to August 13, 2019	Hydrogeology review

Source: SRK, 2019

2.4 Sources of Information

The sources of information include data and reports supplied by GCM personnel as well as documents cited throughout the report and referenced in Section 27.

SRK has been supplied with numerous technical reports and historical technical files. SRK's report is based upon:

- Numerous technical review meetings held at GCM's offices in Medellín, Colombia;
- Discussions with directors, employees and consultants of the Company;
- Data collected by the Company from historical exploration on the Project;
- Access to key personnel within the Company, for discussion and enquiry;
- A review of data collection procedures and protocols, including the methodologies applied in determining assays and measurements;
- Gran Colombia Gold Marmato S.A.S. for the site-specific closure plan and cost estimate presented in Plan de Cierre y Abandono de Mina La Maruja (May 2019);
- Site Environmental Manager, Ing. Adrián Quintero Jiménez, and corporate Environmental Manager, Erwin Wolff Carreño, for information on permits, monitoring programs and data, and the environmental management budget estimate;
- Knight Piésold (2012) for information on the geochemistry of the deposit;

- Existing reports provided to SRK, as follows:
  - NI 43-101 Mineral Resource Estimate on the Marmato Project, Colombia, June 21, 2012;
  - NI 43-101 Mineral Resource Estimate on the Marmato Project, Colombia, June 16, 2017;
  - Geochronology, Geochemistry and Magmatic-Hydrothermal Oxide Characterization of the Marmato Gold Deposit, Colombia;
  - Lead isotopic compositions of the gold mineralization of Marmato, Colombia: Characterization of the transition domain in epithermal - porphyry systems; and
  - Further Geological Observations on The Lower Zone Gold Deposit at Marmato, Colombia, Richard H Sillitoe, July 2019.
- Data files provided by the Company to SRK as follows:
  - Topographic grid data in digital format;
  - Drillhole database, including collar, survey, geology, and assay;
  - QA/QC data including details on duplicates, blanks and certified reference material (CRM); and
  - DXF files, including geological interpretation, vein domain digitized 2D section interpretations, stope outlines and mined depletions.

2.5 Effective Date

The effective date of this report is July 31, 2019.

2.6 Units of Measure

The metric system has been used throughout this report. Tonnes are metric of 1,000 kg, or 2,204.6 lb. All currency is in U.S. dollars (US$) unless otherwise stated.

3 Reliance on Other Experts

The Consultant's opinion contained herein is based on information provided to the Consultants by:

- Site Environmental Manager, Ing. Adrián Quintero Jiménez, and corporate Environmental Manager, Erwin Wolff Carreño, for information on permits, monitoring programs and data, and the environmental management budget estimate;
- Knight Piésold (2012) and SRK Geochemist, David Bird, for information on the geochemistry of the deposit; and
- Gran Colombia Gold Marmato S.A.S. for the site-specific closure plan and cost estimate presented in *Plan de Cierre y Abandono de Mina La Maruja* (May 2019).

SRK has relied upon the work of other consultants in the project areas in support of this Technical Report.

SRK has not performed an independent verification of land title and tenure as summarized in Section 4 of this report. SRK did not verify the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between third parties, but have relied on the Company and its legal advisor for land title issues.

The Consultants used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending. This report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the Consultants do not consider them to be material.

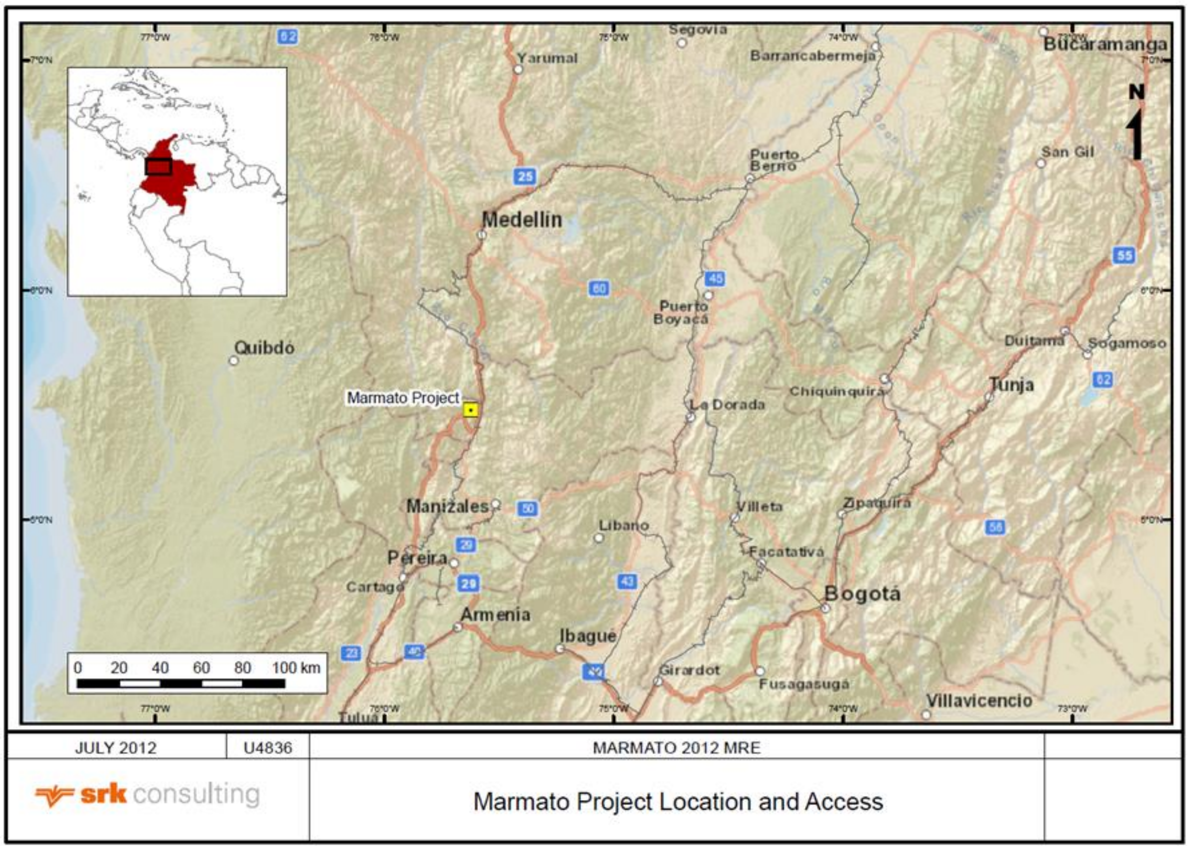
These items have not been independently reviewed by SRK and SRK did not seek an independent legal opinion of these items.

4 Property Description and Location

4.1 Property Location

The Marmato Project is located in the Municipality of Marmato, Department of Caldas, Republic of Colombia and is approximately 125 kilometers (km) due south of the city of Medellín, the capital of the Department of Antioquia (Figure 4-1).

The property sits between latitudes and longitudes 5º28’24”N and 5º28’55”N, and 75º35’57”W and 75º38’55”W respectively. The Project can be accessed from Medellín via paved roads on the Medellín to Cali highway which forms part of the Pan America Highway.

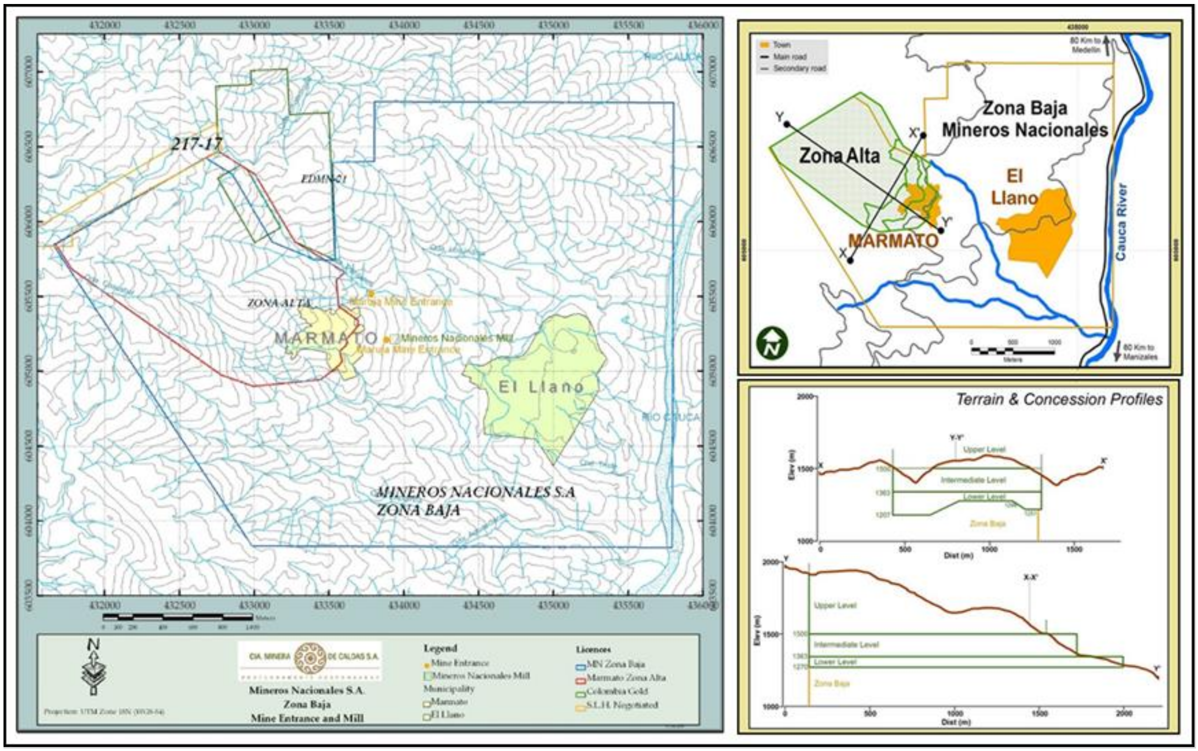


Source: SRK, 2012

Figure 4-1: Location Map

4.2 Mineral Titles

The current Marmato Project area has historically been divided into three main zones with numerous license boundaries defined within. The Project is made up of three separate concessions (Figure 4-2), named Zona Alta, Zona Baja and Echandia. Prior to ownership by GCM each of these licenses have changed owners a number of times. The following section briefly describes the three license areas.



Source: Mineros Nacionales, 2010

Figure 4-2: Land Tenure Map(s)

The horizontal division (Zona Alta and Zona Baja) of mining rights at Marmato is unique in Colombia and was created in 1954 by Decree 2223 to enable concession contracts to be defined by horizontal mine levels. This is defined as an Aporte Minero Mine (Mining Contribution 1017 for precious metals), which was granted in 1981. The Zona Alta concession is further subdivided into three levels. The Upper Level (Nivel Superior) is located above the 1,500 m contour to the hilltop at about 1,705 m; the Intermediate Level (Nivel Intermedio) is between the 1,500 m and 1,363 m contours; and the Lower Level (Nivel Inferior) is between the 1,363 m contour and the top of the Zona Baja (Lower Zone). The top of the Zona Baja is defined in Contract 014-89 with Mineros Nacionales S.A. (Mineros Nacionales) and coincides with the road and varies from 1,207 m to 1,298.3 m in elevation.

The Zona Alta portion of the Marmato Project hosted a large number of individual small mines. During 2009 to 2010 the Company focused work on consolidating the various licenses (including surface rights) to create a single project area for the entire Marmato deposit, which has now been completed. SRK has not reviewed all the legal documents to confirm the completion of this process at this given time, however GCM senior management confirmed that any outstanding consolidation would not impact the three main licenses. Some minor licenses have lapsed but this will not materially impact the current Mineral Resource or the ability to continue the current mining process.

The Zona Baja license lies below the Marmato Zona Alta property and is adjacent to Echandia. Zona Baja extends east to the River Cauca. The license is bounded vertically by the Zona Alta and Cerro El Burro in Marmato, but in the other parts it continues to surface. The license continues vertically to depth in all parts.

The Zona Baja contract was owned by Mineros Nacionales, a private Colombian corporation which was owned 94.5% by Mineros S.A. (Mineros), a Colombian corporation whose shares are traded on the Colombian stock exchange (BVG – Bolsa de Valores de Colombia). The remaining 5.5% of Mineros Nacionales was owned by a number of private and juridical persons. The contract registration number is 014-89M and the mining title registration number is GAFL-11. It covers a surface area of 952.5830 ha. The Zona Baja contract was awarded to Mineros Nacionales, since renamed GCM, in October 1991 and is valid for 30 years until October 2021. In October 2017, GCM commenced the process to renew the contract for another 30-year term.

On February 15, 2010 Medoro Resources Ltd. (Medoro) acquired all of the issued and outstanding ordinary shares of Mineros Nacionales S.A. from Mineros S.A., for total cash consideration of US$35 million. With this acquisition, Medoro acquired 100% of Mineros Nacionales' interests in the Zona Baja concession (the Zona Baja property). Medoro merged into Gran Colombia Gold in 2011.

The Echandia property lies to the north east of the Zona Alta limit, and extends to depth (Figure 4-2). The Echandia has contract number RPP 357 and registry number EDMN-01. The Reconocimiento de Propiedad Privada (RPP) type of contract translates as Recognition of Private Property. RPPs were created by Law 20 of 1969. The law respected prior mining and land rights and required that proof be submitted of mining. Echandia is an old freehold property dating from the 19th century. The RPP titles grant surface and subsurface rights in perpetuity.

Exploitation is required in order to maintain the validity of an RPP license. Mining on a relatively small scale is being maintained in the area of contract number RPP 357

There is an overlap of mining license applications in a triangular area at Cien Pesos in the south-east part of the Echandia license. This was the result of a surveying error in the Mining Registry which erroneously excluded the triangular area from Echandia, and is understood to have been later corrected. 25 mining license applications have been made in this area by small miners. Some of these were granted contracts by Minercol and its predecessors (described below), but they have not been signed and cannot be registered due to the overlap with the Echandia license. Medoro, though Minera de Caldas, has bought 10 of these licenses. SRK has not viewed the legal documents in line with these licenses but in discussions with senior management it was confirmed that this would not impact the three main licenses.

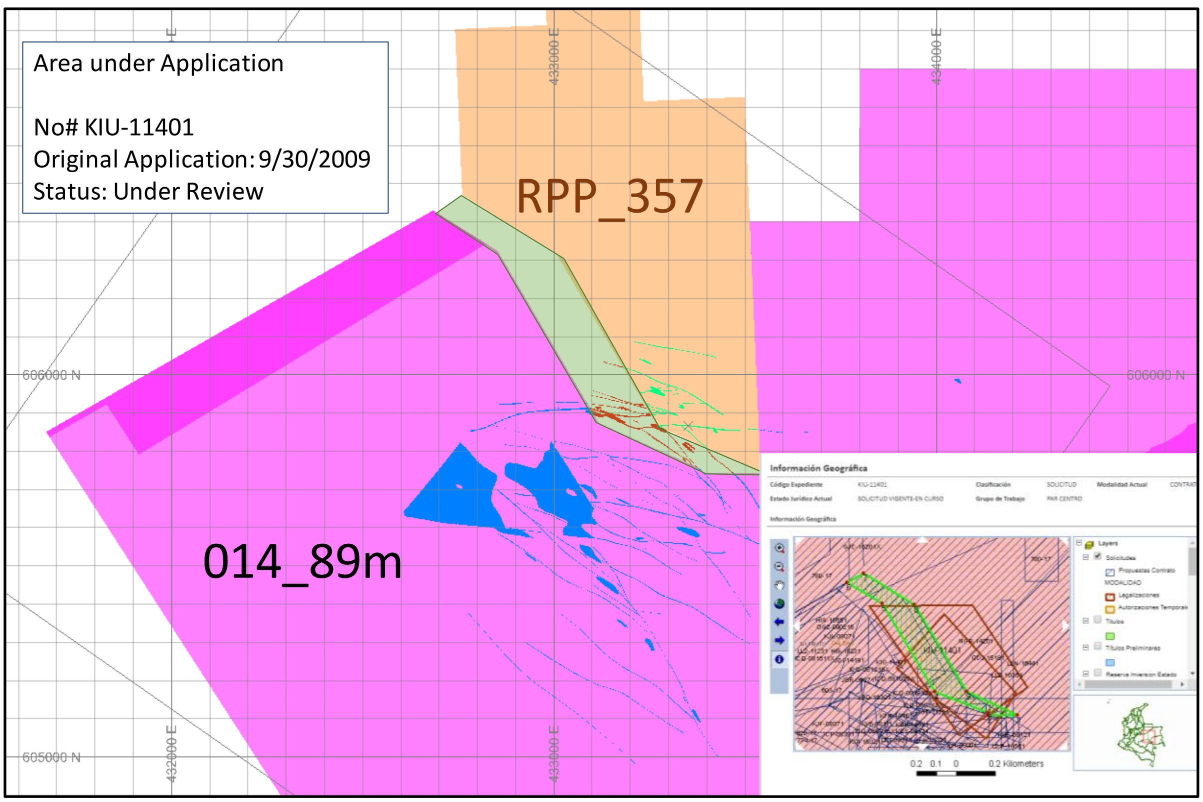
On February 8, 2010, Medoro (previous owners), announced that it has acquired all of the issued and outstanding ordinary shares of Colombia Gold plc (Colombia Gold). A total of 33,333,238 common shares of Medoro were issued to former shareholders of Colombia Gold pursuant to the offer. With this acquisition, Medoro acquired 100% of Colombia Gold interests in the Echandia concession (the Echandia property) which adjoins the Zona Alta property.

Effective June 10, 2011, GCM completed a merger with Medoro and the combined company continued under the name Gran Colombia Gold Corp. As a result, GCM acquired 100% of Medoro’s interest in the Marmato Project.

SRK notes within the transfer of licenses from the previous owner, there is a gap between the existing licenses for Zona Baja and Echandia. This ground is under application from the Company with the Colombia government for formal approval to continue mining. SRK has reviewed the application within the government website and notes that the status is defined as "in progress", which has been in place since September 30, 2009. The Company is taking steps to get the approval finalized.

Mining by GCM within this area has historically been conducted through the current operations and it has been reported to SRK that this area is under application for adjustment of the status for inclusion in the current mining operations of GCM. As the area represented by this gap contains approximately 8.7% of Measured & Indicated resources in Zona Baja and 1.9% of Inferred resources in Zona Baja, SRK recommends that GCM accelerate its effort to obtain government resolution on this gap as it could potentially result in loss of Mineral Resources (and future Reserves). Only approximately 2.3% of the total LOM gold production included in the proposed mine plan for Zona Baja contained herein is sourced from the area represented by this gap.

A summary of the location of the area of concern is shown in Figure 4-3. It is expected that this will not limit the current mining operation. SRK estimates within this area on a global basis for the Mineral Resources approximately 6% of the Measured and Indicated Mineral Resources and approximately 1% of the Inferrred material.



Level 1,050 m, area in green is under application but has been historically mined.
Source: SRK, 2019

Figure 4-3: Summary of Gap in Licenses Within the Current Operations, with Associated Applications

4.2.1 Nature and Extent of Issuer's Interest

In regard to surface rights within Zona Alta, GCM compiled a GIS database of surface rights ownership within a 6 km radius of Marmato. Each of the properties was reviewed to determine discrepancies between legal descriptions and actual ownership. The Mining Law allows for expropriation of land if negotiations among subsurface and surface owners are unsuccessful.

SRK is not aware of any other restrictions which impact the current mining operations within the Zona Baja (Mineros Nacionales) and Echandia areas of the Project.

4.3 Royalties, Agreements and Encumbrances

In 1991, Mineros Nacionales entered into an agreement with Ecominas (a State Industrial and Commercial Organization) for the exploration and exploitation of Mining Title No.014-89M. The mentioned title was previously granted by the Colombian State to Ecominas. It was agreed by the parties that Mineros Nacionales would pay a royalty to Ecominas (now referred to as Agencia Nacional de Mineria) equal to 6% on gold revenue and 8% on silver revenue as economic compensation.

Besides that, Mineros Nacionales is bound by law to pay the Colombian State a 4% royalty.

GCM also pays a royalty of 4% on gold and silver revenue to an associated company owned by Gran Colombia, Minera Croesus S.A.S. ("Croesus"), in respect of production sourced from the neighbouring Echandia mining title owned by Croesus. This royalty obligation will remain in place after the RTO Transaction described in Section 1.1.

4.4 Environmental Liabilities and Permitting

The main environmental details for the Project are covered in Section 20 of this report.

4.4.1 Environmental Liabilities

The existing Marmato Project is authorized through the approval of a PMA. The PMA for Marmato was approved by Corpocaldas on October 29, 2001 under Resolution 0496, File No. 616. The PMA and its requisite environmental management procedures and practices amount to approximately US$425,200 annually. This amount is likely to increase with the proposed expansion project.

The 2001 Mining Code requires the concession holder to obtain an Insurance Policy to guarantee compliance with mining and environmental obligations which must be approved by the relevant authority, annually renewed, and remain in effect during the life of the Project and for three years from the date of termination of the concession contract. The value to be insured will be calculated as follows:

- During the exploration phase of the Project, the insured value under the policy must be 5% of the value of the planned annual exploration expenditures;
- During the construction phase, the insured value under the policy must be 5% of the planned investment for assembly and construction; and
- During the exploitation phase, the insured value under the policy must be 10% of the value resulting from the estimated annual production multiplied by the pithead price established annually by the Government.

According to the Code, the concession holder is liable for environmental remediation and other liabilities based on actions and/or omissions occurring after the date of the concession contract, even if the actions or omissions occurred at a time when a third-party was the owner of the concession title. The owner is not responsible for environmental liabilities which occurred before the concession contract, from historical activities, or from those which result from non-regulated mining activity, as has occurred on and around the Marmato Project site.

As noted above, given the tenure of the current operations, the Environmental Insurance Policy is not required for the Marmato operation. Current liabilities at the site are generally associated with the reclamation and closure of the mining facilities and tailings disposal areas. Given the extensive impacts associated with artisanal mining in the area, a clear delineation between possible environmental liabilities attributable to the GCM and those from unregulated mining activities is not possible; however, GCM has been making a concerted effort to deal with legacy environmental issues in order to better make that separation. The social issues related to mining in Colombia, especially the interactions between mining companies and artisanal operators, have been violent at times, and could continue to pose a health and safety liability for GCM employees and the neighboring communities.

4.4.2 Required Permits and Status

Discussion related to mining in Colombia, the Mining and Environmental Codes, as well as the permits and authorizations necessary for mineral exploration and exploitation is provided in Section 20.3.

4.5 Other Significant Factors and Risks

There are no legal restrictions that affect access, title or right or ability to perform work on the property with the exception of the pending license approval. In light of past experience, social issues may cause issues from time to time within the Zona Alta region of the Project.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Topography, Elevation and Vegetation

The Marmato Project is located in an area of steep mountainous terrain with a relief of approximately 1,600 m (Figure 5-1). The Project is bound to the east by the Cauca River at 600 m elevation, and otherwise surrounded by the peaks of the nearby mountains that reach up to 2,200 m elevation and are commonly incised through landslide activity. Despite the abrupt relief, the landscape is in general well vegetated and supports crop cultivation and livestock. The dominant land use in the area of Marmato is cattle grazing, coffee, sugar cane, citrus fruit, bananas, and mining. The Middle Cauca region, where Marmato is located, was occupied for two thousand years before the Spanish conquest by farmers, potters, gold miners and goldsmiths of the Quimbaya culture (500 BC to 1600 AD).

The ecological zones defined on the Holdridge Life Zone climatic classification system are zoned by elevation (Municipio de Marmato, 2004; Correa, 2006; Cia Minera de Caldas, 2008):

- Premontane (subtropical) wet forest transitional to tropical moist forest and dry forest; defined as temperatures >24°C, annual rainfall of 1,500 to 2,800 mm, and elevation of 700 to 1,000 m. This area includes the Cauca River valley and the lower part of El Llano town.
- Premontane (subtropical) wet forest defined as temperatures of 18°C to 24°C, rainfall of 2,000 mm to 4,000 mm, and elevation of 1,000 to 1,900 m. The main areas of mining and exploration are in this zone.
- Lower montane (warm temperate) wet forest defined as temperatures of 12°C to 18°C, rainfall of 2,000 to 4,000 mm, and elevation of 1,900 m to 2,900 m.

Much of the original forest cover has been cleared for agriculture and grazing, especially at lower elevations. Land is used for cattle grazing, coffee, sugar cane, citrus fruit, bananas, and mining in Marmato.

5.2 Accessibility and Transportation to the Property

The Project is in the Municipality of Marmato in Caldas. The concessions of the Marmato Project are located on the eastern side of the Western Cordillera (Cordillera Occidental) of Colombia on the west side of the Cauca River.

Marmato is 200 km west of the Pacific Ocean and 300 km south of the Caribbean Sea and Atlantic Ocean. The nearest port is Buenaventura on the Pacific Ocean (320 km by the Pan American Highway).

The property is a three-hour drive from Medellín, via the Medellín to Cali highway which is part of the Pan American Highway. The route from Medellín is via Itaguí (7 km), Caldas (12 km), Alto de Minas (13 km), Santa Barbara (27 km), La Pintada (26 km), La Guaracha del Rayo (32 km), and then a turn onto a secondary road to an 8 km long partially asphalted road to Marmato. There is an international airport located in Medellín with flights to the USA, Panamá, Venezuela, Spain and Peru, and a national airport in Manizales with flights to Medellín and Bogotá.

Source: SRK, 2012

Figure 5-1: Marmato Project, Looking Northwest Towards Cerro El Burro

5.3 Climate and Length of Operating Season

Climate at the site is typical of the equatorial zone, with the region falling within the Köppen classification zone of Am, described as moist tropical climates with high temperatures year-round and short dry seasons in a monsoon cycle. Average annual precipitation was estimated as 1,889 millimeters per year (mm/y) (Knight Piésold, 2012) with two drier periods around January and July and wetter periods around April-May and October-November. Temperatures are warm year-round, with maximum temperatures ranging from 28.7°C to 31.6°C and minimum temperatures in the range of 17.4°C to 18.7°C (Knight Piésold, 2012). Relative humidity at the site is typically in the 70 to 80% range. The climate allows year-round operations.

5.4 Sufficiency of Surface Rights

Refer to Section 4.2 of this report.

5.5 Infrastructure Availability and Sources

5.5.1 Power

Power is available through the Colombian power supplier Central Hidroeléctrica de Caldas (CHEC), a subisidiary of Empresas Públicas de Medellín (EPM) through existing local substations. The power system feeding the Project has a 40 MVA capacity. Substantial transmission capacity is available in the region around the Project, with energy provided over the transmission system by the third largest electricity producer in Colombia, ISAGEN.

5.5.2 Water

External water supplies are available from both groundwater and surface water sources. Dewatering for the underground mine is currently being utilized as a water supply for the existing operations, and withdrawals from the nearby Cascabel river have been utilized as well. Additional dewatering flows are expected to be produced as a result of dewatering the MDZ Project. Water is also available from the nearby Cauca River.

5.5.3 Mining Personnel

The region has historically and currently had a strong mining presence with around 1,200 people working at the current Marmato Project and a substantial number of artisanal miners in the area close to the mine. Skilled personnel should be available from the local miners as well as supplemented from the other nearby areas to support the workforce as needed.

Field personnel for the exploration program have been employed from the towns of Marmato and El Llano and neighboring municipalities. In the long term, personnel currently working on the large number of small scale mines and from the surrounding region would be able to supply the basic workforce for any future mining operation.

5.5.4 Existing and Potential Tailings Storage Facilities

The existing Marmato tailings storage facility is estimated to provide three to four years of additional capacity, after which new dry stack tailings storage facilities (DSTF) would need to accommodate tailings through the currently-estimated 16-year LoM. A siting study was completed as part of PEA preparation and two preferred locations were evaluated to satisfy this requirement.

5.5.5 Potential Waste Disposal Areas

Waste is typically returned to the stopes in the existing mine. At the MDZ Project, a waste rock storage area is designated near the portal. Other locations are available if needed.

5.5.6 Potential Processing Plant Sites

The Project is an operating mine with an existing plant site. An additional plant location has been identified for use for the MDZ Project approximately 2.7 km from the existing plant with minimal population on the site or in close proximity to the site.

6 History

Colombian gold production between 1514 and 1934 has been estimated at 49 million ounces (Moz) which makes Colombia number one in South America with 38% of the total historical production (Emmons, 1937). Two-thirds of the gold production was from placer deposits. Subsequent Colombian production is estimated at 30 Moz by the Banco de la Republica (Shaw, 2000), which gives Colombia a total recorded historical gold production of approximately 80 Moz. 75% of this production came from the Departments of Antioquia and Caldas, with the Marmato Project located near the border between the two departments.

6.1 Prior Ownership and Ownership Changes

Marmato is one of the most important historical gold properties in Colombia and lies in the heart of the main historical gold producing region (dating back to 500 BC). The location name is derived from "marmato" or "marmaja", an old Spanish term for pyrite. The property has a long and complex ownership history, summarized in Table 6-1.

Table 6-1: Ownership History at Marmato

Date	Ownership History
1525	Colonization of Colombia and first references to Marmato
1634	First larger scale workings begin; and first gold mill
1798	Silver mines located at Echandia, with two near surface veins exploited
1819 to 1925	Various English companies mine gold at Marmato
1925 to 1938	Mines were expropriated and initially remained closed, then later leased to contractors
1954	Marmato was divided into two zones (Decree 2223), Alta (Upper) and Baja (Lower)
1981 to 2004	Marmato becomes part of the Aporte Minero scheme and was managed by a succession of state mining companies
1984 to 1985	Minera Phelps Dodge de Colombia S.A. (Minera Phelps Dodge) explores the Zone Baja of Marmato
1991	Contract for the Zona Baja is awarded to Mineros Nacionales in October 1991 for a period of 30 years by the state entity Empresa Colombiana de Minas (Ecominas); the contract is now administered by Agencia Nacional de Mineria (National Mining Agency or ANM)
1996 to 2000	Conquistador Mines Ltd. (Conquistador), a Vancouver listed junior company (now called Orsa Ventures Corp), explored the Project through its Colombian subsidiary Corona Goldfields S.A. (Corona Goldfields). Conquistador had an option to explore the Zona Baja over 4 years and to acquire 50.1% of Mineros Nacionales (it bought 13.15% which it later sold in 2001), and acquired several mines in the Zona Alta.
1995 to 1997	Gran Colombia Resource Inc. (unrelated to GCM and now defunct) carried out exploration at Echandia and Chaburquia properties on the northern portion of the Marmato System
2005 - 2008	Minera de Caldas began exploration of Marmato and surrounding areas with the aim of identifying bulk mineable targets of low grade gold and silver. Colombia Goldfields Limited (CGL), began acquisition of property within Zona Alta, plus completed 46,000 m of drilling
2009 - 2010	Medoro purchased CGL (Zona Alta), Colombia Gold (Echandia) and Mineros Nacionales (Zona Baja), which consolidated the three primary gold properties at Marmato
2011 - Present	GCM and Medoro Resources Ltd merged to create the largest underground gold and silver producer in Colombia, under the name of Gran Colombia Gold Corp..

Source: SRK, 2019

6.2 Exploration and Development Results of Previous Owners

Modern exploration at Marmato began in the mid-1980s and has continued into the present day under various entities. The exploration and development results of prior owners are listed below:

1984 to 1985: Minera Phelps Dodge explored the Zona Baja of Marmato, with the objective of defining a 300 tpd underground operation. It completed surface and underground sampling and drilled seven underground core holes, and defined a Proven reserve of 102,900 t at 7.83 g/t gold and 24 g/t silver, and a total reserve (Proven, Probable and Possible) of 754,600 t at the same grade.

1993: Mineros Nacionales began mining the Maruja mine via a 300 tpd underground operation under contract (No. 041-89M). Mineros S.A. acquired 51.75% of Mineros Nacionales and upgraded the mine and mill. Mineros subsequently increased ownership of Mineros Nacionales to 94.5%. Further exploration was completed through the 1990s with 24 underground core holes drilled and three reverse circulation (RC) holes drilled. The plant was expanded to a capacity of 800 tpd.

1996 to 2000: Conquistador drilled 44 holes (14,873 m), 30 from surface (11,496 m) and 14 underground diamond holes (3,377 m), plus 1,147 channel samples totaling 2,847 m from surface trenches and underground cross-cuts. Conquistador also commissioned MRDI to complete a resource estimate and scoping study in 1998, but carried out no further work on the Project due to the expiration of the option contract.

1995 to 1997: Gran Colombia Resource Inc. conducted soil surveys, surface magnetic and geophysical surveys, channel samples (La Negra, La Felicia and La Palma adits) and completed 75 diamond drillholes (surface and underground) totaling 15,000 m. A scoping study was made by Geosystems International, Denver, in 1997 which concluded that there was not sufficient grade continuity for a bulk-tonnage resource and mining operation, and no further work was carried out.

2005: Minera de Caldas began exploration of Marmato and surrounding areas with the aim of identifying bulk mineable targets of low grade gold and silver. CGL carried out underground sampling, surveying and mapping, preliminary metallurgical test work and diamond drilling to define a mineral resource. CGL carried out 46,000 m of drilling in 2007 and 2008.

2010: Medoro commenced infill drilling of the Project via surface and underground diamond drilling with view of producing a prefeasibility study in 2011.

2011 to 2017: GCM completed further infill drilling from surface and underground locations, plus channel sampling of existing cross-cuts.

2017 to 2019: GCM exploration has focused drilling on defining and infilling the MDZ, plus on-going exploration within the current mining operations and cross-cuts on levels 20 and 21.

6.3 Historic Mineral Resource and Reserve Estimates

A qualified person has not done sufficient work to classify the historical estimate as a current Resource Estimate or Mineral Reserve and the issuer is not treating the historical estimate as a current Resource Estimate.

6.4 Historic Production

Production has occurred from the Marmato property since pre-colonial times, but there are no published historical records of the actual gold and silver production for all periods since mining commenced, however sporadic records for different periods have been noted.

To give an indication of the current mining activity at the deposit SRK has reproduced a summary (Table 6-2) of the total produced gold and silver at Marmato on an annual basis between 2004 and 2011. The figures also represent only the official declared gold recovered and does not include illegal mining which persists at Marmato even to present times.

Table 6-2: Gold Production from the Municipality of Marmato 2004 to December 2018

Year	Ore Tonnes (t)	Grade Au (g/t)	Au Produced (oz)
2004	186,330	3.60	21,583
2005	231,540	3.30	24,541
2006	262,517	3.10	26,171
2007	300,756	3.22	31,127
2008	254,474	2.95	24,138
2009	250,638	3.51	24,372
2010	252,136	3.39	23,318
2011	250,553	3.19	22,715
2012	268,137	2.85	21,717
2013	274,190	2.90	22,566
2014	295,023	2.85	24,116
2015	303,279	2.79	23,954
2016	341,308	2.55	23,447
2017	366,485	2.46	25,162
2018	340,052	2.67	24,951

Source: GCM, 2019

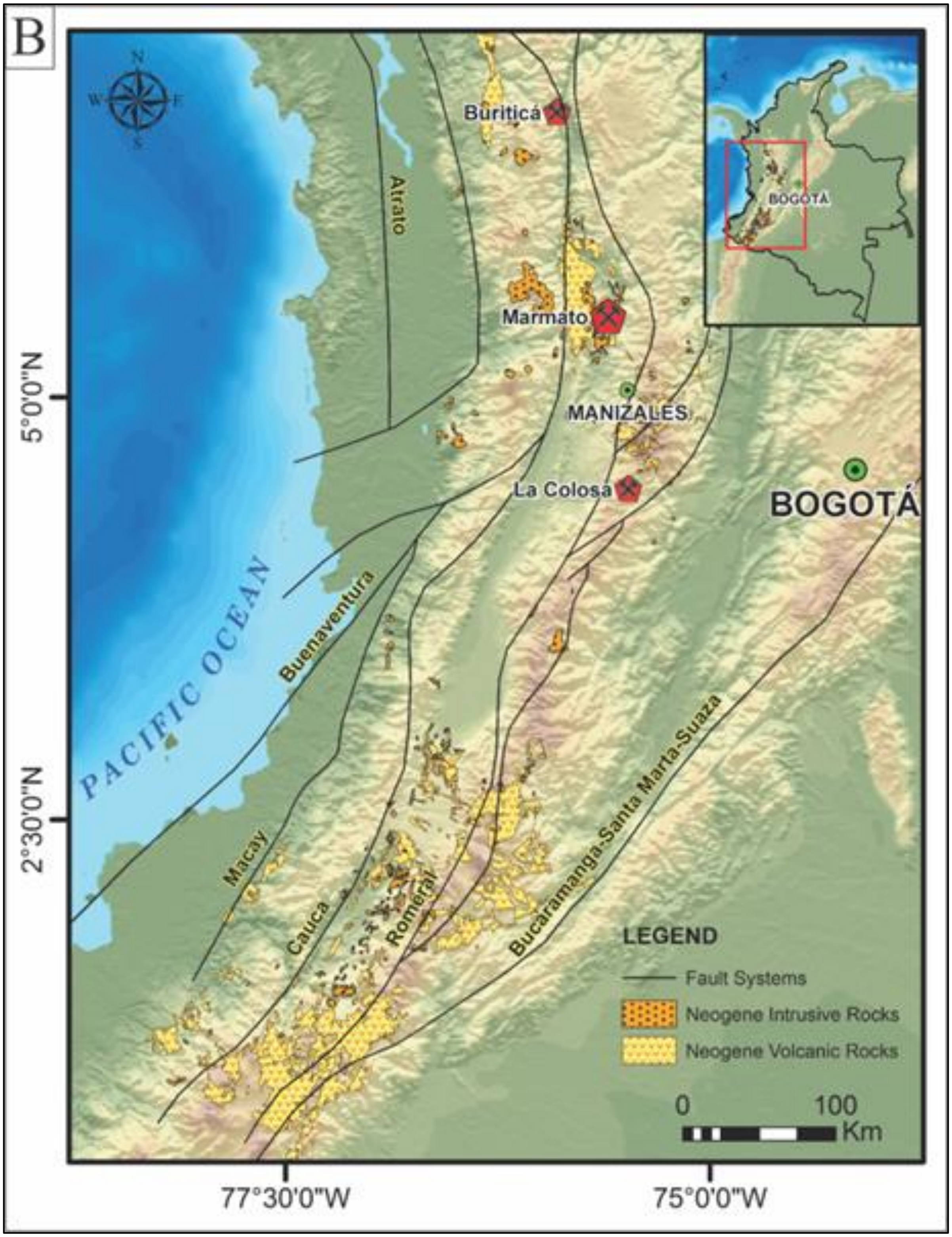
7 Geological Setting and Mineralization

7.1 Regional Geology

The Colombian Andes are part of the Northern Andean Block which includes the Northern Volcanic Zone of the Andes (Gansser, 1973; Shagam, 1975). They are formed of three N to NNE trending mountain ranges, the Western, Central and Eastern Cordilleras, separated by two major intermontane basins, the Cauca-Patía Depression and the Magdalena Depression, which represent terrane boundaries. The Colombian Andes have a complex history of volcanism, subduction, accretion and faulting, represented by the juxtaposition of metamorphic, igneous and sedimentary rocks of various ages from the Precambrian to the present (Aspden et al., 1987; Restrepo and Toussaint, 1988). Cediel et al. (2011) have defined nine principal tectonic terranes in Colombia which are:

- Guyana shield;
- Maracaibo sub-plate;
- Central continental sub-plate;
- Pacific terranes;
- Caribbean terranes;
- Choco-Panama arc;
- Guajira terrane;
- Caribbean Plate; and
- Nazca Plate.

Marmato is located on the eastern side of the Western Cordillera which is separated from the Central Cordillera by the River Cauca. It lies within the Romeral terrane which is bounded by the Cauca Fault on the west side and the Romeral Fault to the east, and is part of the Pacific terranes realm. The recent tectonic setting of the Colombian Andes is characterized by the subduction of young (less than 20 Ma) oceanic crust beneath relatively thin continental crust (less than 40 km; Cediel and Caceres, 2000; Cediel et al., 2003). The Benniof zone is located at around 140 to 200 km depth below the volcanic belt of the Colombian Andes which has slightly migrated to the east during the last 10 Ma (Pennington, 1981; Vargas & Mann, 2013). The Marmato stock is part of the Miocene magmatism characterized by calc-alkalic subvolcanic intrusions and volcanic rocks of the Combia Formation. The Miocene magmatism cross-cuts the units of the Romeral terrain, the plutonic units of the Albian and early Cenozoic, and the siliciclastic sequences of the Amagá Formation (Cáceres et al. 2003; Tassinari et al, 2008). Miocene gold related magmatism in Colombia has been well-recognized in the Western and Central cordilleras associated with stocks (Sillitoe et al., 1982; Toussaint and Restrepo, 1988; Lodder et. al, 2010; Lesage et al., 2013). In addition, late Miocene-Pliocene magmatism with gold mineralization has also been recognized in the Santander Massif in the northern part of the Eastern Cordillera (Mantilla et al., 2009).

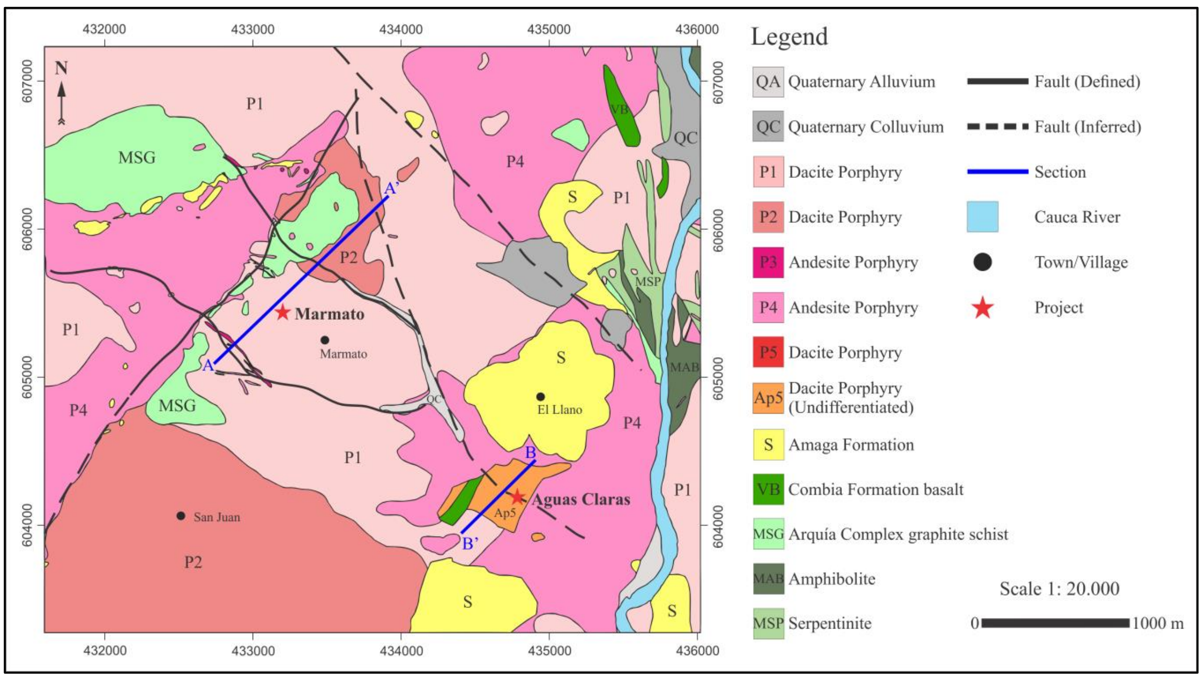


Source: Modified from the Geological Map of Colombia, 1:1 million scale, Colombian Geological Survey, 2015

Figure 7-1: Regional Geology Map

7.2 Local Geology

The Marmato gold deposit is hosted by the porphyritic andesitic to dacitic Marmato stock which is 18 km long and 3 to 6 km wide and is elongated north to south (Calle et al., 1984). It intrudes the Arquía Complex and Amagá Formation on the east side in the Cauca Valley, and the Combia Formation on the west side. The Marmato gold deposit is hosted in a multiphase porphyry suite, the Marmato Porphyry Suite, which is about 3.0 km long by 1.6 to 2.5 km wide and is located near the southern end of the larger Marmato Stock. Five main porphyry pulses have been identified in the Marmato Porphyry Suite by cross-cutting relationships in core logging and named P1 to P5 from oldest to youngest, respectively. The ages of the intrusions have been reported recently between 6.58 ± 0.07 Ma to 5.74 ± 0.14 Ma by U-Pb LA-ICP-MS of zircon (GCM, 2016, dating carried out by the Brasilia University Isotope Geochronology Laboratory, Brasil). The Aguas Claras Porphyry Suite is located 3 km southwest of the Marmato Porphyry Suite and also has five porphyry pulses identified from cross-cutting relationships in core logging named AP1 to AP5 from oldest to youngest. Two intrusions of the Marmato Porphyry Suite, P3 and P5, cross-cut the Aguas Claras Porphyry Suite as dikes. There is no previous dating of the Aguas Claras Porphyry Suite. The local geology map is presented in Figure 7-2.



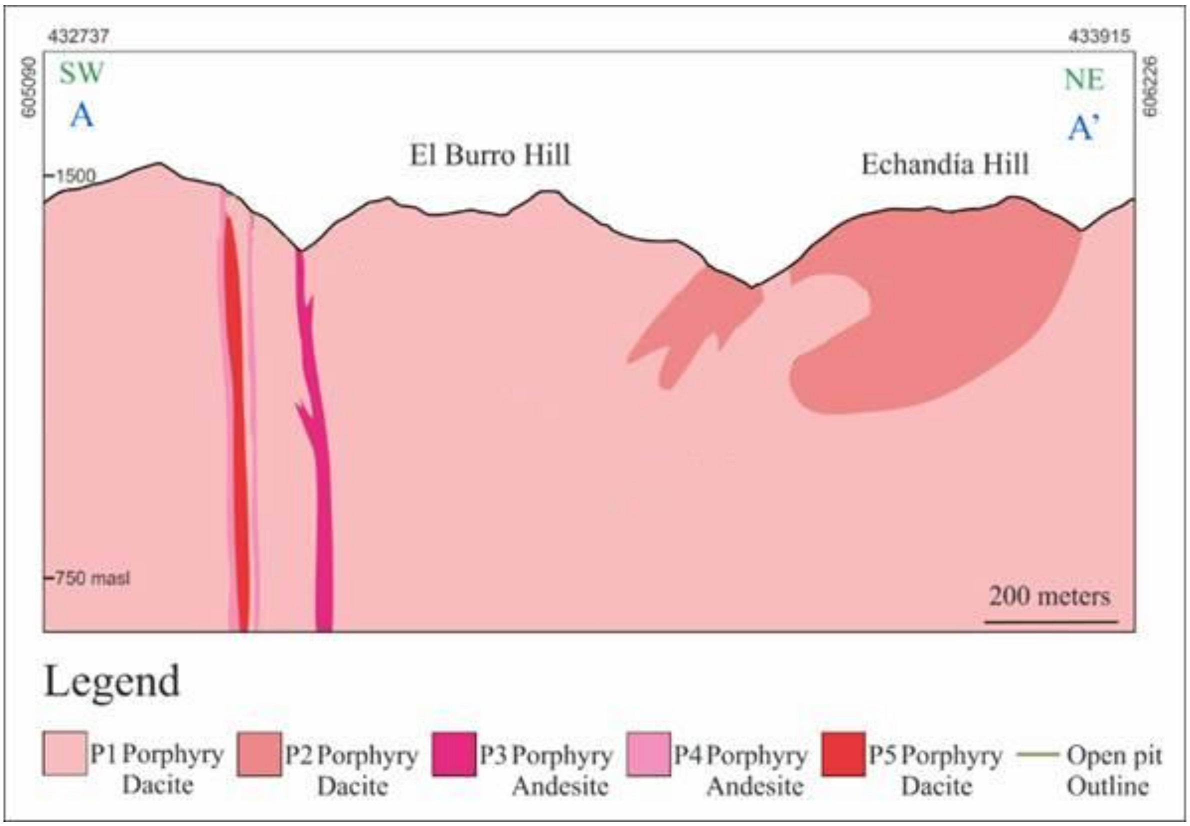
Source: GCM, 2017

Figure 7-2: Local Geology Map

7.3 Property Geology

The Marmato gold deposit consists of a structurally-controlled epithermal vein system with a mineral assemblage dominated by pyrite, arsenopyrite, black Fe-rich sphalerite (the type locality for "marmatite", Boussingault, 1830), pyrrhotite, chalcopyrite and electrum in the upper zone, and a mesothermal veinlet system with a mineral assemblage dominated by pyrrhotite, chalcopyrite, bismuth minerals and visible gold in the MDZ.

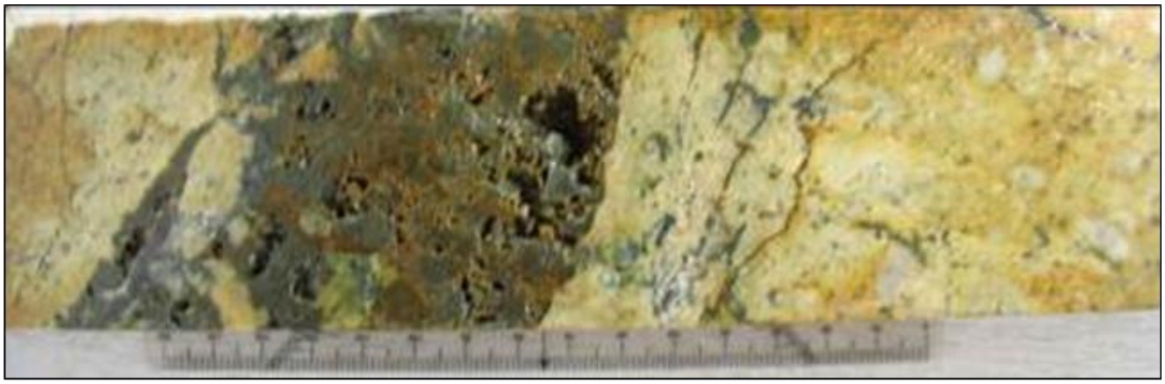
Dacitic and andesitic intrusions at Marmato are characterized by quartz, hornblende, biotite and zoned plagioclase phenocrysts in a finely crystalline quartz-plagioclase groundmass, with variations in phenocryst proportion and sizes between intrusions. Intrusion P1 is a main dacitic porphyry stock in the Marmato Porphyry Suite and is characterized by large β quartz phenocrysts >7 mm. It is cross-cut by intrusion P2 which corresponds to a porphyry dacite intrusion with fewer and smaller phenocrysts. Intrusion P3 forms dikes of andesitic porphyry with plagioclase megacrysts >10 mm, and cross-cuts intrusions P1 and P2 (Figure 7-3). Intrusion P4 is an andesitic porphyry stock which cross-cuts P1, P2 and P3, and is characterized by smaller plagioclase phenocrysts. The youngest porphyry P5 is dacitic and forms dikes cross-cutting P1. It is characterized by large quartz phenocrysts and elongate plagioclase phenocrysts. Mineralization is hosted mainly by stocks P1 to P4, while is absent in P5.



Source: GCM, 2019
The deposit outcrops on El Burro Hill and Echandía Hill

Figure 7-3: Cross-Section of the Marmato Gold Deposit Looking NW Showing the Intrusions P1 to P5

Gold mineralization occurs in veins and veinlets with dominant NW and W/NW trends. The deposit mainly comprises sulfide-rich veinlets and veins composed of minor quartz, carbonate, pyrite, arsenopyrite, Fe-rich sphalerite (i.e. marmatite), pyrrhotite, chalcopyrite and electrum in the epithermal upper zone, and quartz, pyrrhotite, chalcopyrite, bismuth sulfide and telluride minerals and free gold in the mesothermal lower zone. Pervasive early propylitic alteration is over-printed principally by phyllic and intermediate argillic alteration related to the gold mineralized veins of low to intermediate sulfidation epithermal type, with weak and patchy potassic (biotite) alteration at depth.



MT-1210, 0.25m @ 17.5 g/t Au, 19 g/t Ag

(a) Low to intermediate sulfidation epithermal style mineralization

P1 dacite porphyry with stockwork of veinlets of quartz-pyrrhotite-minor chalcopyrite with narrow illite halo, overprinting biotite alteration.
MT-1498 at 854.0 m. Sample 2.0 m at 17.3 g/t Au, 3.5 g/t Ag and 330 ppm Cu.

Veinlet of quartz-pyrrhotite-chalcopyrite with halo of green sericite-smectite in P1 dacite porphyry with weak biotite alteration.
MT-1498 at 847.3 m. Sample 2.0 m at 2.88 g/t Au, 0.7 g/t Ag, and 225 ppm Cu.

(b) Mesothermal style mineralization

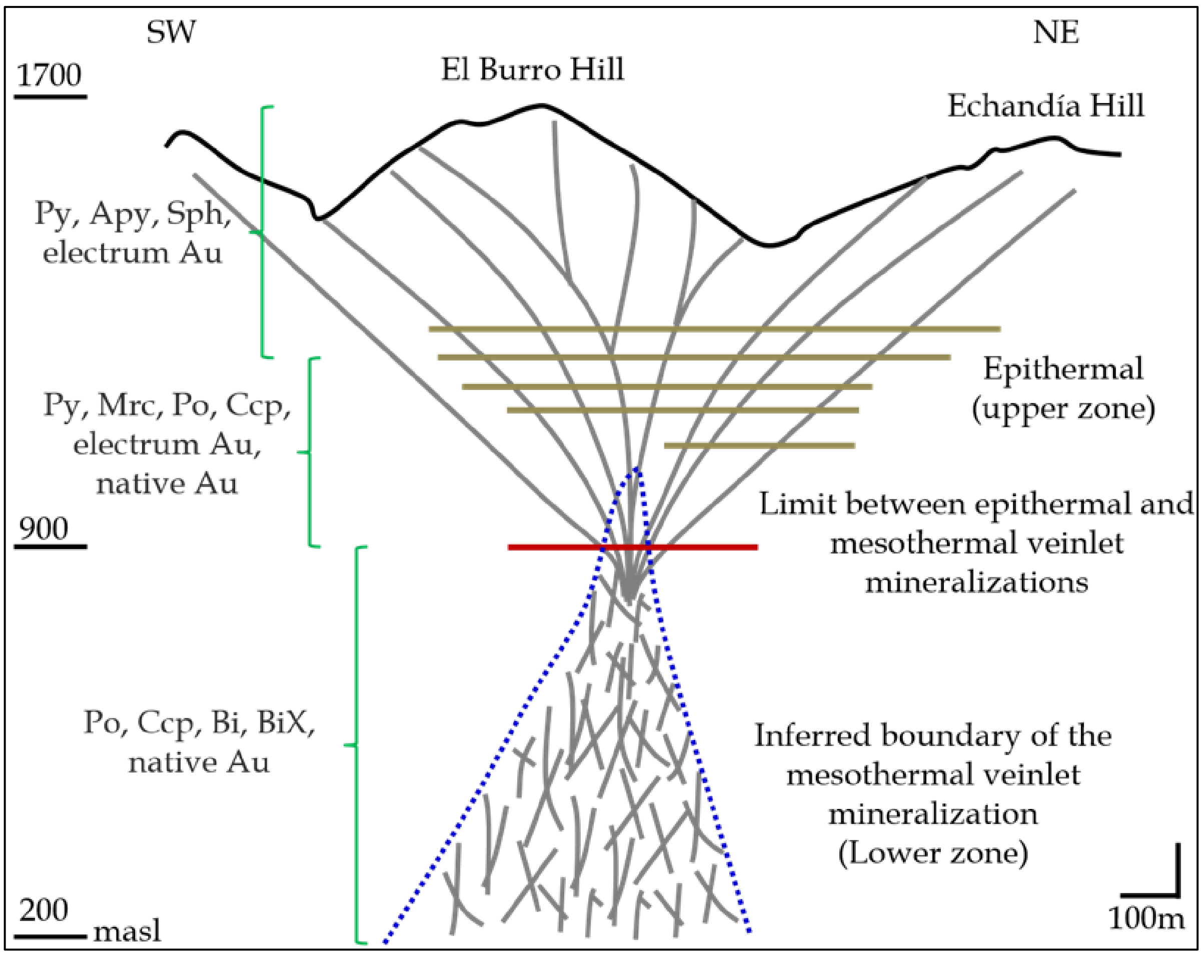
Source: GCM, 2017

Figure 7-4: Examples from Drill Core of the Different Mineralization Styles

SRK considers the change in the mineralization style between the higher epithermal style of mineralization (gold-silver mineralization mainly hosted by a sheeted pyrite±sphalerite veins) to the deeper mesothermal veinlet style (i.e. pyrrhotite±chalcopyrite) to be an important distinction (Figure 7-5). Further drilling and study of the transition zone and deeper areas will be required to increase the understanding and confidence in the mesothermal geological model to aid estimation to higher levels of confidence (Indicated and Measured).

In November 2018, a geological site visit was completed to review the MDZ mineralization (Sillitoe, 2019), which concluded the following key aspects:

- Gold grade distribution in the Lower Zone orebody is unrelated to the presence of distinct porphyry phases and is entirely dependent on the intensity of structurally localised veinlets;
- The only geological parameter than can be used to constrain the grade model is veinlet intensity, although the presence of visible native gold also acts as a useful grade indicator;
- Potassic alteration, represented chiefly by biotite, is progressively better preserved at depth in the Lower Zone orebody, raising the possibility that early potassic alteration could also be gold bearing; however, this possibility remains to be demonstrated; and
- The exclusive structural control of the Lower Zone orebody implies that additional examples could exist elsewhere within the P1 stock and that they represent a priority exploration target.



Source: GCM, 2017

Figure 7-5: Schematic Cross-Section of the Marmato Gold Deposit, Showing the Two Principal Zones and the Vertical Zonation of Mineralization

7.3.1 Structure

The dominant northwest and east-west trends of the veins are interpreted to be due to regional tectonic forces and may have formed as tension fractures related to northwest-southeast compression and sinistral strike-slip movement on the north-south trending Cauca and Romeral Faults which lie on either side of the deposit.

In April 2010, the Company commissioned Telluris Consulting Ltd (TCL) to complete a review of the local and regional geology to define a structural-hydrothermal model for the Marmato deposit. TCL defined the Marmato deposit as a series of north-northwest to east-west trending steep to moderately dipping, gold-bearing, sulfide-rich veins hosted in a north-south trending late Miocene porphyry complex. TCL noted that the porphyry complex was emplaced in folded and thrusted Paleozoic and Mesozoic metamorphic and sedimentary sequences adjacent to the eastern margin of the broadly north-south trending Cauca-Romeral terrane accompanied by east northeast to northwest-southeast compression. This resulted in north-south trending thrust and transpressional structures along with steep northwest and northeast conjugate fault zones.

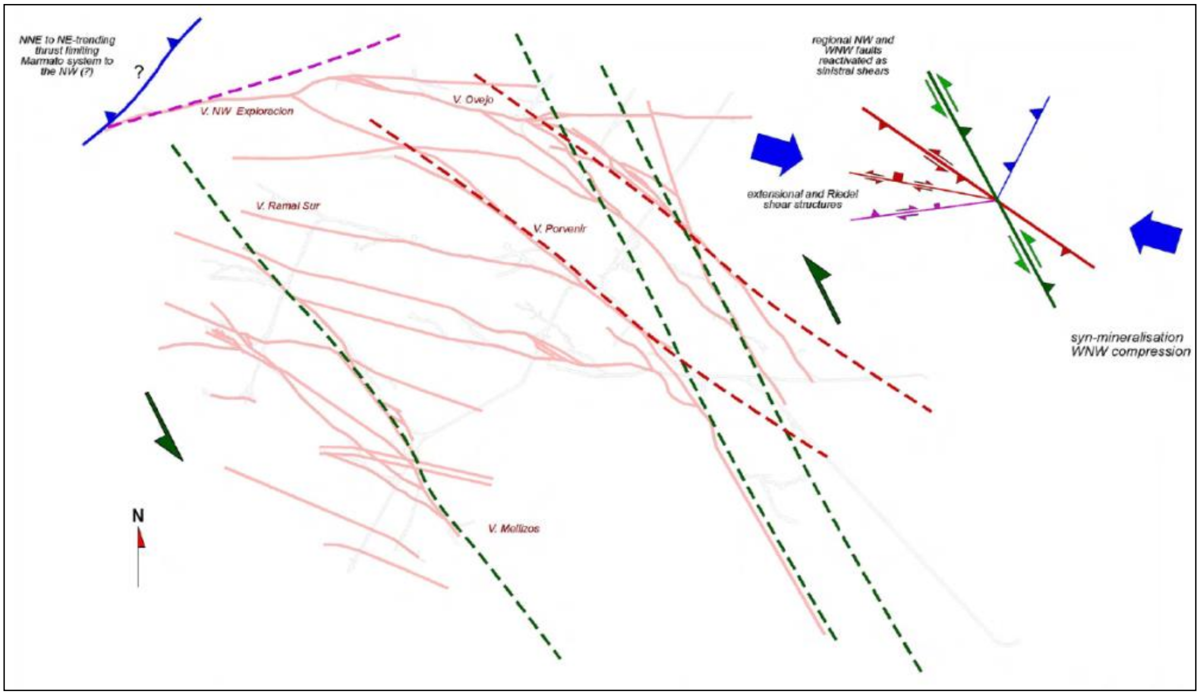
Within the relatively young intrusive rocks of the Marmato deposit there are principally two deformation stages recognized from the TCL study:

- Syn-mineralization west northwest-east southeast compression that reactivated some of the basement structures as well as generating a range of second order shear and extensional structures along north northwest to east-west trends as well as north northeast trending thrusts; and
- Continued post-mineralization compression into the late-Pliocene, (approximately 2 Ma) that resulted in uplift due to renewed thrusting along the main terrane boundaries forming thrust-bounded intermontane basins such as the Cauca-Patia depression.

Within the Marmato area, there four principal trends of mineralized structures:

- Northwest trending steep to sub-vertical faults/fractures (140° to 150°N);
- West-northwest trending steep to moderately inclined structures (110° to 120°N);
- East-west trending structures (100° to 090°N) that tend to have moderate to relatively low-angle dips; and
- East-northeast to northeast-trending structures (065° to 080°N) that show a range of dips.

In addition to these ore-bearing structures, there is a set of north-northeast trending structures of varying dips that appear to represent different components of a reverse/thrust fault system. Both the west-northwest and east-west veins tend to splay from the main northwest structures which is consistent with extensional and Riedel shear components to a sinistral shear system. TCL reported that kinematic indicators show that mineralization accompanied a phase of west-northwest to east-southeast orientated compression (Figure 7-6). The north-northeast trending reverse faults and conjugate fractures reflect this compression component.



Source: TCL, 2010

Figure 7-6: Telluris Consulting Interpretation of Vein Orientations at Mineros Nacionales

The TCL report suggests the presence of north northeast trending thrust structure at the northwest limit of the Mineros Nacionales workings which needs further work to be confirmed where possible via underground access to the structure or considered when interpreting drilling data.

Post-mineral faulting is observed on the margins of some veins and veinlets with alteration to soft, white clay gouge with ground pyrite (logged as fault gouge, FLG). In some places there is later, coarse euhedral pyrite in the clay gouge. Brittle fault breccias with no clay gouge are also observed (BXF). In the Mineros Nacionales mine workings of the Zona Baja it is observed that the northwest trending veins have competent wall rocks and require no mine support, whereas the east-west trending veins have faulted wall rock with soft clay fault gouge and require support.

Supergene white clay (intermediate argillic) alteration occurs in the superficial parts of the deposit as a result of the weathering of propylitic alteration.

Unconsolidated Quaternary sedimentary deposits have been noted from drill data but not modelled in the current update. These comprise rock scree and landslides, and coarse alluvial gravel in creeks which include waste rock from artisanal mining. Landslides and alluvial flows are a geological hazard, especially on the steep southeast face of Cerro El Burro above the town of Marmato. There is saprolitic weathering and little rock exposure on the higher ground to the north west of Cerro El Burro.

7.3.2 Alteration

Two stages of pervasive alteration have been recognized: early propylitic and later intermediate argillic. These affect all types of porphyry, although alteration is weak in P5. The propylitic alteration is characterized by epidote replacement of plagioclase cores, albite replacement of plagioclase rims and matrix, chlorite replacement of mafics, with disseminated pyrite and pyrrhotite, and varies in intensity from veinlet-halo to pervasive. Calcite partially replaces plagioclase where propylitic alteration is weakly developed. Cross-cutting relationships show evidence for multiple events of propylitic alteration related to each phase of intrusion.

Intermediate argillic alteration overprints the propylitic alteration and varies in intensity from vein-/veinlet-halo to pervasive, associated to the intermediate sulfidation mineralization style and replaces epidote, chlorite and albite. There is a strong but generally narrow halo of white to green illite or sericite alteration related to veins and veinlets of the mesothermal mineralization event which grades outwards to pervasive illite, with smectite in distal parts. The main disseminated sulfide is pyrite, although pyrrhotite and iron-rich sphalerite also occur, which to some extent formed the basis for the previous model domains.

Additionally, weak and patchy potassic alteration, represented chiefly by biotite occurs at depth in the MDZ. Progressively better preservation of early potassic alteration at depth may indicate the possibility of early gold-bearing phases.

7.4 Significant Mineralized Zones

Gold and silver mineralization within the MDZ has been confirmed based on the 2018 and 2019 drilling programs. The drilling completed to date has defined mineralization at depth within the central portion of the deposit. SRK has noted a zone of elevated grades which has been referred to as the higher grade MDZ (more than 2.0 g/t Au). This zone is indicated to be continuous along strike for approximately 500 m and has a confirmed down dip extent that reaches up to 800 m, with a thickness that varies between 35 and 150 m. The higher grade MDZ is still open to the east and at depth. It is possible that the main MDZ mineralization is bounded within a series of faults but limited drilling at the edges of the deposit make confirmation difficult to assess at this stage

8 Deposit Type

8.1 Mineral Deposit

The alteration and mineralization in the Upper Zone at Marmato evolved through early stage, higher-temperature propylitic alteration to later, lower-temperature intermediate argillic alteration, with most of the gold and silver being deposited in the later stage.

The gold-silver and base metal association in the Upper Zone at Marmato is typical of the intermediate sulfidation epithermal type. The veins lack typical epithermal textures and the mineralization has a relatively high depth and temperature of formation, and straddles the deep epithermal to mesothermal transition as defined by the original classification of Lindgren (1922) and by estimates of formation temperature of 300°C (Heald et al., 1987). The Marmato deposit lacks known shallow and surface epithermal features such as lithocaps, sinters and crustiform banded veins.

Mineralization is interpreted to be genetically related to the host porphyritic rocks, as shown by the inter-mineral timing of the porphyry phases cross-cutting earlier stages of propylitic alteration, the late-mineral timing of the final dacite P5, and miarolitic cavities lined with propylitic-stage minerals. The veins and veinlets are structurally controlled and did not form a multi-directional porphyry stockwork or breccia related to hydro-brecciation. In this model, the host stocks might be considered as late-mineral intrusions with respect to a postulated porphyry gold-copper-molybdenum centers.

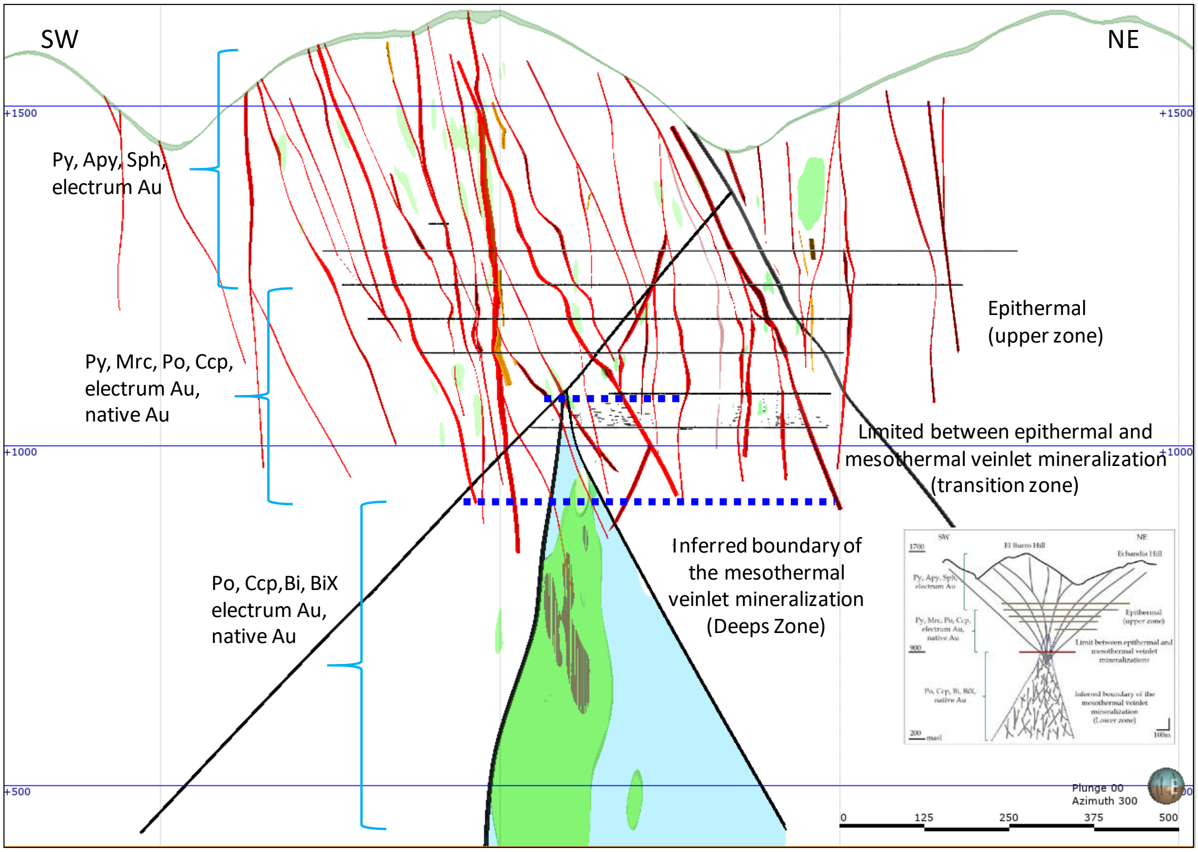
The upper portion of the MDZ has been exposed in Level 21 of the existing GCM mining operations, while deeper sections have been observed in drillcore, both of which have been confirmed as separate styles of mineralization. The lowest levels of the mine have currently intersected a combination of the porphyry domain, where the gold is associated with pyrite veinlets, and the MDZ where gold is associated with pyrrhotite. Gold grade distribution in the Lower Zone orebody is unrelated to the presence of distinct porphyry phases and is entirely dependent on the intensity of structurally localised veinlets. Sillitoe (2019) concluded, that the only geological parameter than can be used to constrain the grade model is veinlet intensity, although the presence of visible native gold also acts as a useful grade indicator.

8.2 Geological Model

As part of the updated Mineral Resource, GCM and SRK initially focused on the creation of a more complete geological model (i.e., one encompassing the major geological features inclusive of the current veins being mined). The main geological units and entities identified are:

- Vein;
- Porphyry intrusions (P1 to P4);
- Major fault network; and
- Schist country rock.

Drilling during 2011/2012, along with check logging of the drill core by the Company and SRK, has confirmed the presence of a high-grade feeder zone at the core of the main mineralization (termed the MDZ). Recent drilling supports the presence of the MDZ and this zone was also incorporated in the updated model; a schematic diagram of the conceptual model is shown in Figure 8-1.



Source: SRK, 2019

Figure 8-1: Conceptual Model for the Marmato Deposit, Showing Two Principal Mineralization Zones

9 Exploration

This section summarizes the exploration work completed at Marmato to date which varies for the different zones of the Project due to the different ownership prior to being amalgamated by the Company. No major regional exploration has been completed since the previous NI 43-101 Technical Report and therefore this section provides a summary of the historical work completed.

9.1 Relevant Exploration Work

9.1.1 Imagery and Topography

A high-resolution Ikonos satellite image and detailed topographic map with 2 m contour intervals was produced in early 2007. This map provided a detailed base map for improved accuracy when plotting the results of the exploration programs. An extension to the Ikonos image and topographic map was commissioned in 2008 and received in late 2008, and was stitched to the original image and map to provide seamless products. The extensions were to cover areas for evaluation for possible mining infrastructure for the Marmato Project, such as waste rock and tailing storage areas, and exploration drill targets in the Caramanta Project.

9.1.2 Surface Geochemistry

GCM collected 1,880 rock chip samples and 700 soil samples on surface in Echandia, for a total of 2,580 samples. The geochemical samples identified anomalies coincident with low magnetic anomalies covering an area of about 800 m by 1,100 m in size.

9.1.3 Geophysics

During 2007 and 2008 a helicopter survey which included both magnetic and radiometrics was completed.

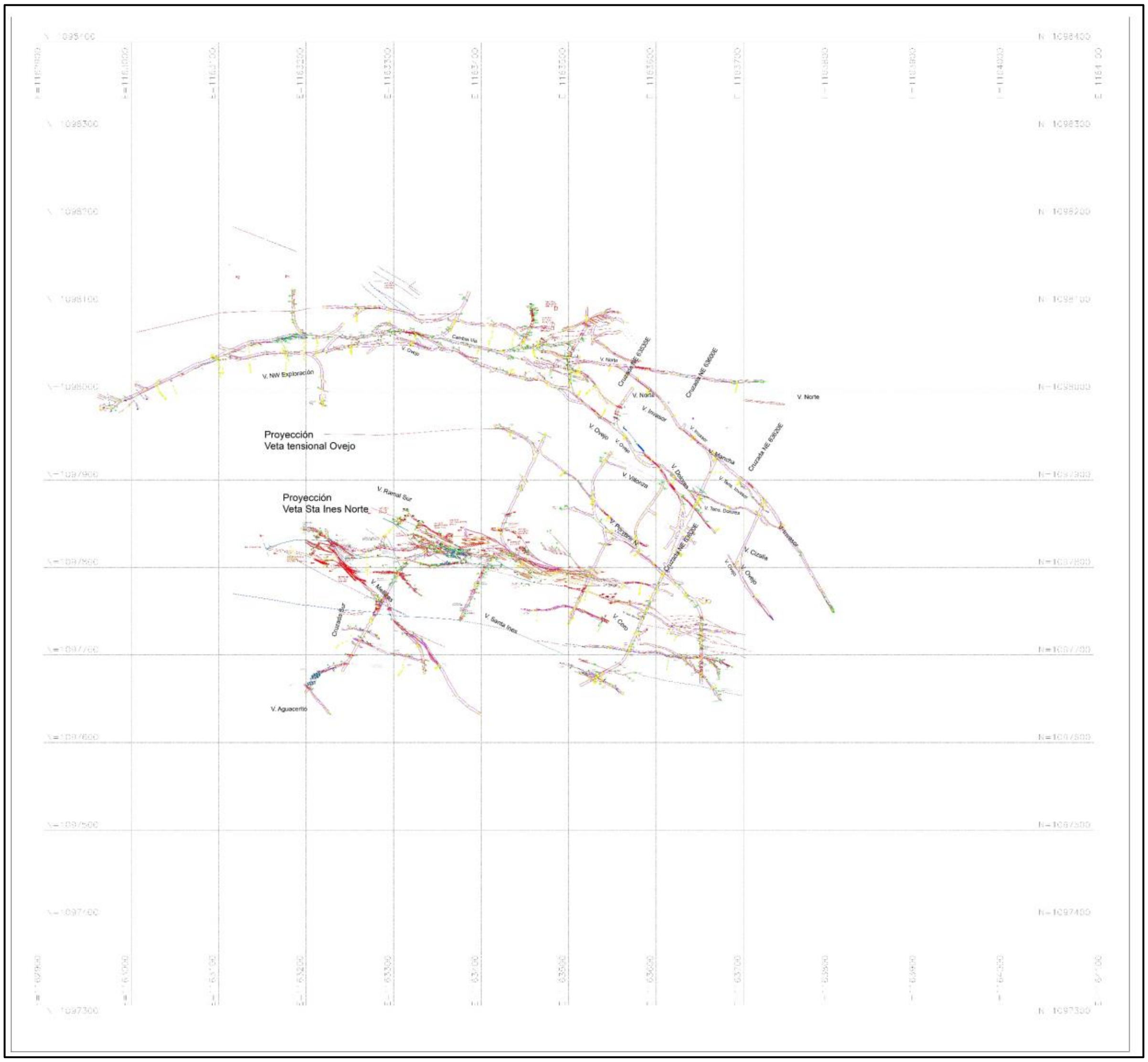
9.1.4 Geological Mapping

Geological mapping at 1:1,000 scale has been carried out on surface, although outcrop exposures are limited away from the steep face of Cerro El Burro above Marmato.

9.1.5 Underground Mapping

Detailed surveying (total station or theodolite) and geological mapping of the accessible underground workings within the Zona Alta has been completed where access was available.

GCM which is currently working within the Zona Baja area supplied AutoCAD drawings of mine level plans and sections for all veins currently being mined. SRK has been supplied with this information for the current update and utilized the information during the construction of the geological model. The level plans and information have a degree of time lag as they are not updated on a routine basis (6 monthly) but based on the current production levels at GCM. It is not anticipated that any changes will have a significant impact on the Mineral Resource Estimate. SRK has used these underground level maps (Level 16 – Level 21) as the basis for the current interpretation of the veins, which has been supplemented with information from mining where available. An example of a Level plan is shown in Figure 9-1.



Source: GCM, 2019

Figure 9-1: Example of Level Plan from Gran Colombia Gold Marmato (Level 20)

9.2 Sampling Methods and Sample Quality

Two stages of exploration have been completed at Zona Alta. The initial phase used basic hammer and chisel techniques to cut channels 2 m long and 5 cm wide by 1 cm deep. Due to the relatively poor quality of sampling, many of these samples were repeated during a second stage of exploration using a hand-held core saw.

Where access was not possible due to poor ground conditions, the original hand-cut samples have been retained in the database, but are considered of lower quality for classification purposes. The Zona Alta area is currently mined by small multiple small-scale mining operations, and the Company does not have input into the sampling processes user by the miners, or compile an active database of any sampling within these areas.

The Company has carried out detailed mapping and sampling of a number of crosscuts within the current GCM operation. Continuous channel samples were taken where possible. Samples were initially marked up on the face using paint into 0.5 to 2 m sample intervals. A rock-saw was then used to create two horizontal parallel cuts across the length of the sample about 6 to 8 cm apart and 2 cm deep. Then a series of vertical cuts 10 cm apart were made to facilitate breaking the rock. After these cuts were completed the rock sample was broken using a hammer and chisel, and collected in a sample bag. The sample width and depth were designed to give a sample weight similar to a split HQ drill core sample. All samples were logged using the same logging codes as utilized in the diamond drilling procedures.

SRK reviewed the sampling locations of a large continuous cross-cut during an underground site visit and is satisfied that the sampling procedures used are in line with industry best practice and no evidence of selective sampling of higher grade vein material was evident. SRK has therefore accepted the results from the channel sampling program as acceptable for the definition of Mineral Resources at Marmato. In the updated database, SRK notes some channel samples from the mine have been limited to the vein only and are not supported by other channels. It is SRK's view that these could impact on the geological model and estimation and therefore should be considered as having lower confidence. In any underground operation routine sampling compared to selective sampling will provide the best confidence for the geological models, as it is just as important to know where low grade exists for mine planning requirements.

All GCM verification sample points were surveyed using either total station of theodolite. A total of 4,285 samples (over 6,699 m) have been taken over 1,431 channels. SRK has integrated these channels into the database and treated them as horizontal drillholes, with samples cut to sufficient size to relate to that of a diamond drillhole. SRK considers this approach to be acceptable and has used this data in producing the resource estimates presented here.

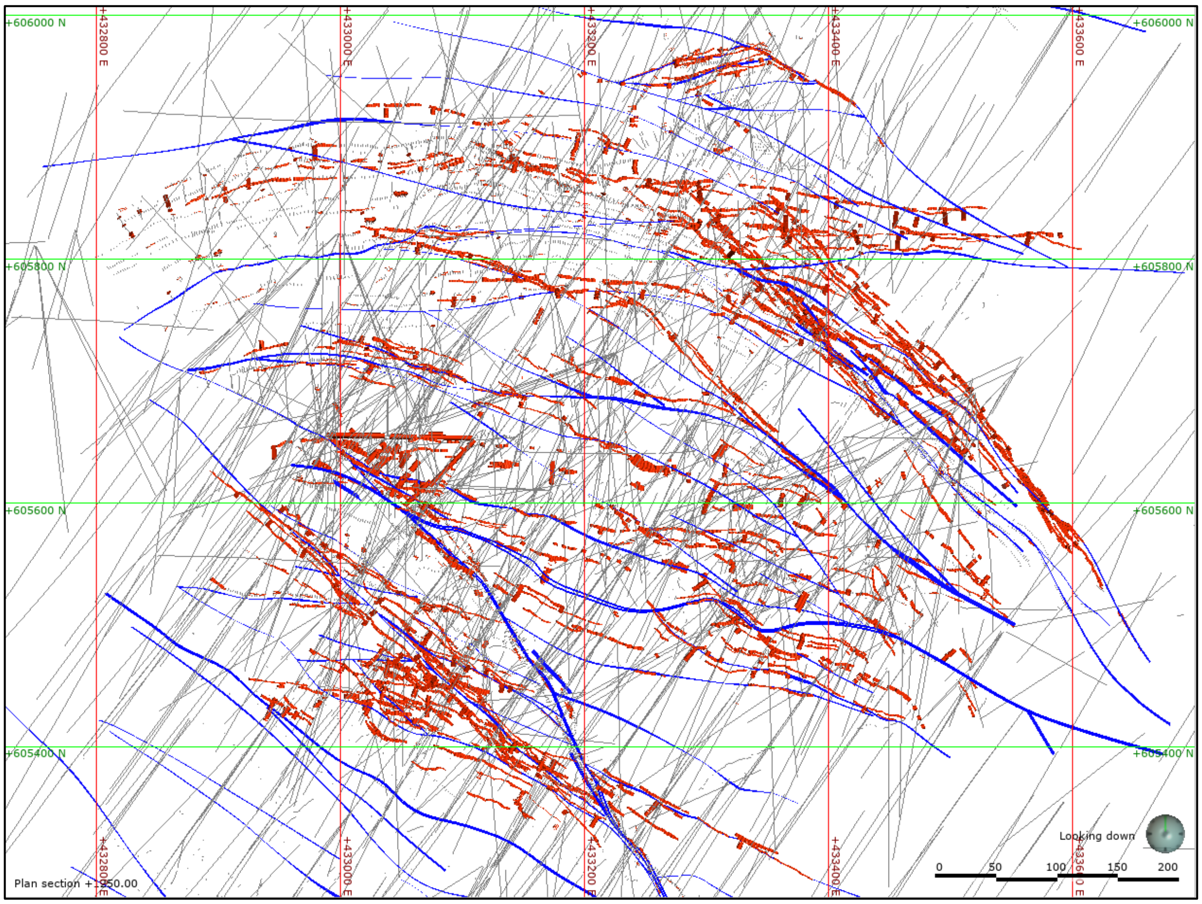
9.3 Significant Results and Interpretation

SRK has reviewed the sampling methods and sample quality for the Marmato Project and is satisfied that the results are representative of the geological units seen and that acceptable minimal biases have been identified.

SRK has reviewed the methods employed by the Company during the underground sampling of Zona Baja which showed clearly marked sampling intervals and associated check sampling. It is SRK's view that the sampling intervals and density of samples are adequate for the definition of a compliant Mineral Resource Estimate. SRK recommends the Company continue with the current underground sampling program on the lower levels of the GCM mine as per the current exploration program.

There has been a significant increase (100%) in the channel sampling and underground drilling completed by the Company operated mine as part of the routine grade control procedures, since the previous model. There are an additional 12,765 channel samples as part of the on-going validation and capture of historical sampling, and current grade control practices. The increased database allows for improved details in the geological definition of the veins models, for which it has formed the basis (Figure 9-2).

The inclusion of the grade control channel samples has aided in the estimation of the vein domains but has caused a number of issues in the previous methodology used to define the porphyry domain (within the pyrite zone). The impact of additional short sampling which have been logged typically with the lithology "P1", within the larger indicator-based grade shells could potentially result in the over-estimation of grade and therefore require further restriction. SRK tested a number of scenarios and has made additional adjustments for the estimation procedures within the porphyry sampling by applying filters on the information used during geological modelling and the estimation process. The adjustments have resulted in a reduction of both tonnage and contained metal within this zone. SRK has discussed these issues with the GCM geology team and will work on a method to improve the modelling of these domains in future estimates.



Source: SRK, 2019

Figure 9-2: 2D Plan View of Sampling Data Versus Vein Interpretations, Showing New Sample Data Highlighted in Red, Versus Plan Section of Veins in Blue (Level 1250 M)

10 Drilling

10.1 Type and Extent

Drillholes, where regularly spaced, are inclined -60 and -75 degrees predominantly to the southwest, with occasional scissor holes towards the northeast. Fan drilling has been utilized both at surface and from underground, which are also typically orientated towards the southwest, with a small number of less extensive fans orientated towards the northeast.

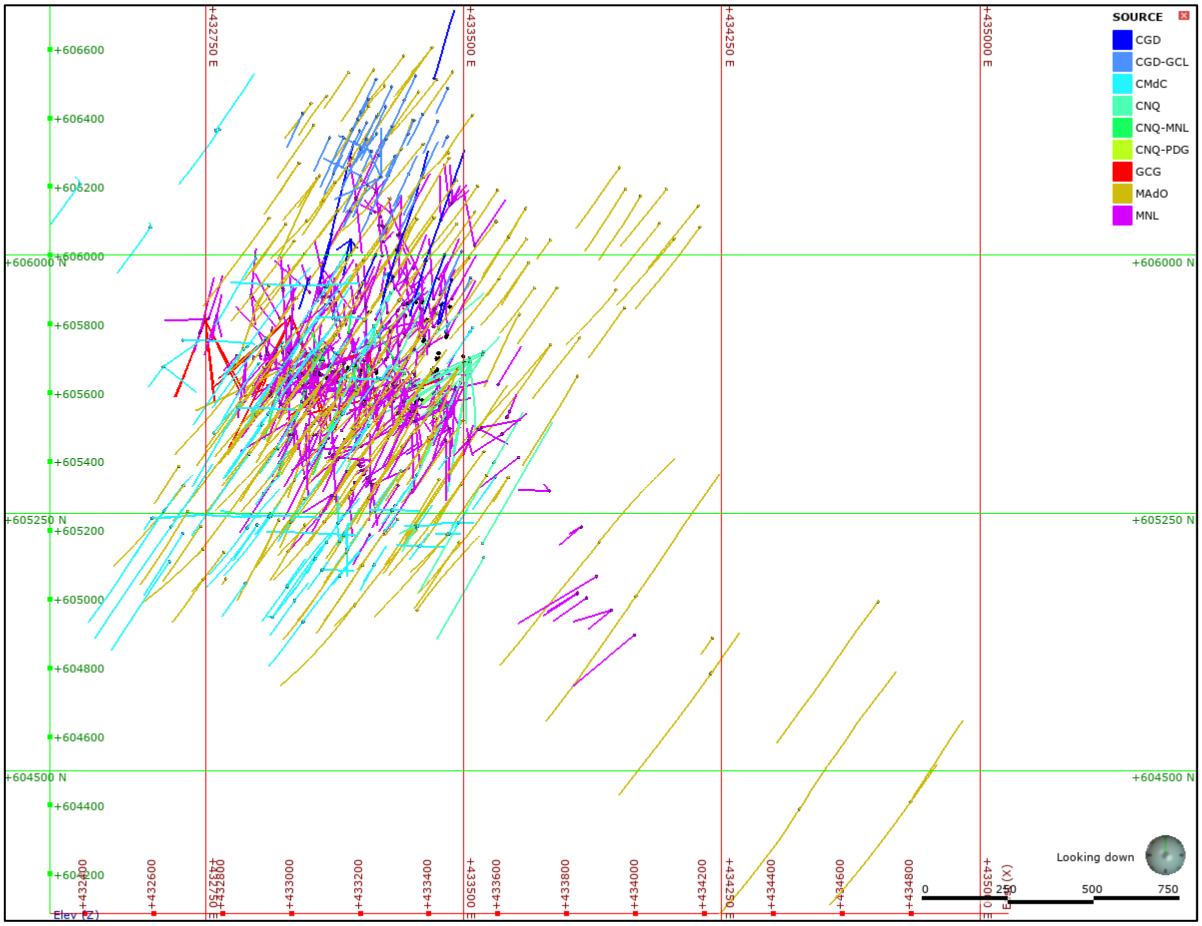
A technical report completed by SRK on September 4, 2011, titled “A NI 43-101 Mineral Resource Estimate on the Marmato Project, Colombia”, provides in depth detail on the historic drilling programs (Table 10-1).

For the purpose of the current drilling program the Zona Alta, Zona Baja and Echandia licenses are referred to collectively as the Marmato license, given the understanding these areas have now largely been consolidated into a single license (Figure 10-1). New sample data is shown in Figure 10-2.

Table 10-1: Summary of Drilling Completed by Company

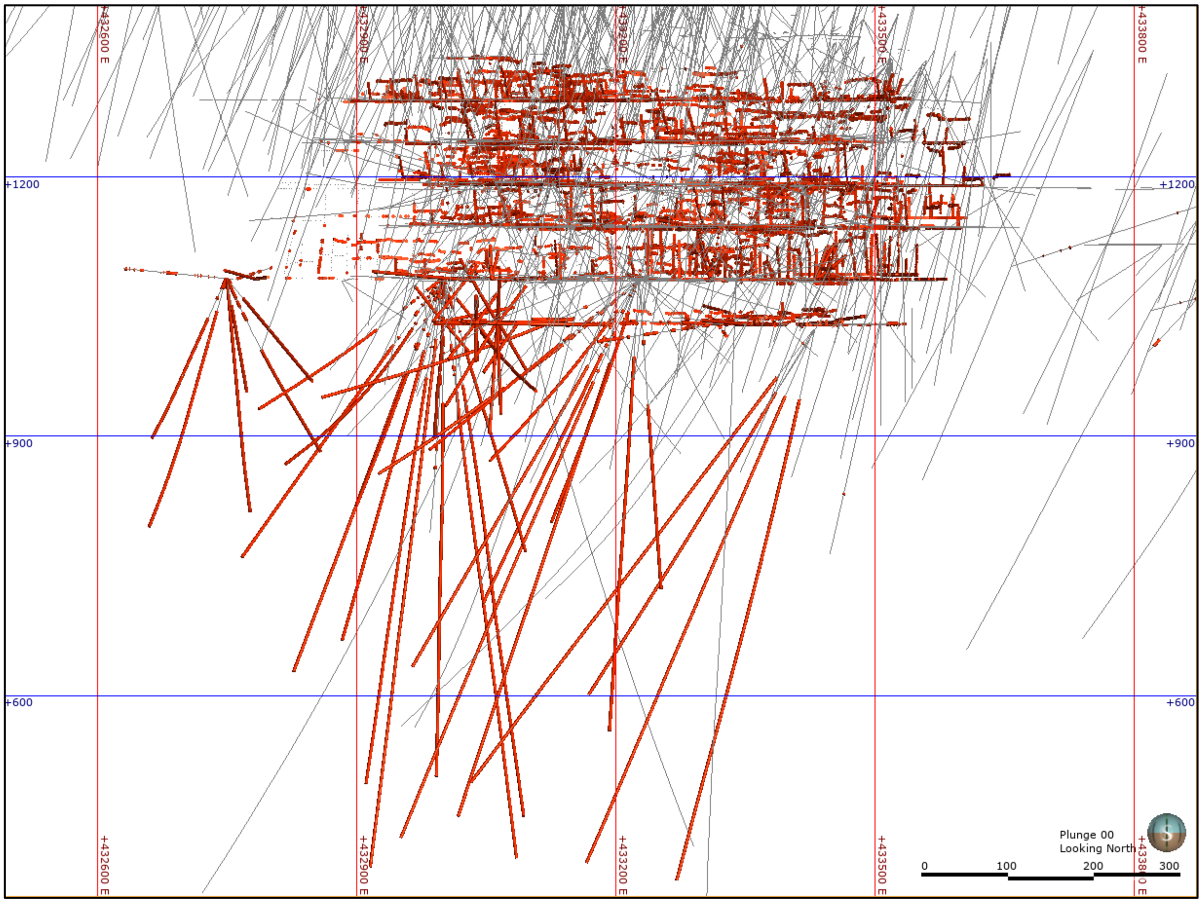
	Company	Series	Number of Holes	Total Meters
Zona Alta	Compañía Minera de Caldas	CMdC	205	46,518.8
	Minerales Andinos de Occidente	MAdO	150	47,733.6
	Zona Alta Subtotal		**355**	**94,252.4**
Echandia	Colombia Gold	CGD	20	5,937.1
	Gran Colombia Resources	CGD-GCL	75	11,186.7
	Minerales Andinos de Occidente	MAdO	65	25,770.2
	Echandia Subtotal		**160**	**42,893.9**
Zona Baja	Mineros Nacionales	CNQ	47	14,873.6
	Mineros Nacionales	CNQ-MNL	25	1,803.4
	Minera Phelps Dodge	CNQ-PDG	6	696.0
	Minerales Andinos de Occidente	MAdO	127	48,286.3
	Mineros Nacionales	MNL	557	47,078.4
	Gran Colombia Exploration	GCM	32	14,531.5
	Zona Baja Subtotal		**794**	**127,269.0**
Grand Total			**1,309**	**264,415.3**

Source: SRK, 2019



Source: SRK 2019

Figure 10-1: Location Map Showing Drillholes Completed at Marmato by Company



Source: SRK, 2019

Figure 10-2: 3D View of Sampling Data, Showing New Sample Data Highlighted in Red (Looking North)

10.2 Procedures

10.2.1 Summary

All surface hole collars have been surveyed using a Differential Global Positioning System (DGPS) and have been surveyed to a high degree of confidence in terms of the XY location. Underground drilling collars have been surveyed by the mines survey department and verified against existing development.

Down-hole directional surveys were conducted using a GyroSmart digital gyro tool, manufactured by Flexit Navigation A.B. (Flexit) and Imego A.B. (Imego) of Sweden, which was purchased from Ingetrol. Prior to the purchase of this instrument in 2007, down-hole directional surveys were conducted using a Flexit Multishot tool supplied by Terramundo.

GCM constructed a core storage facility at Marmato during 2010 (Figure 10-3), and acts as the main exploration facility with logging and offices setup also located at the facility. SRK has visited the core storage facility during multiple site visits and found the facility to be organized and clean, with sufficient space for the ongoing exploration.

Source: SRK, 2019

Figure 10-3: Core Storage Facility at Marmato Constructed in 2010, and Current Status 2019

During the drilling campaigns and sample preparation phases several procedures to ensure sample integrity were employed, including:

- A geological staff member was assigned to each diamond drilling rig;
- A trained technician was assigned to each coring rig to record core recoveries and RQD and to ensure that core was properly handled and packed after each core run;
- All transport of samples from drill site to the Marmato sample Logging, Preparation and Storage facility was undertaken by a staff member;
- All splitting and sampling were supervised;
- Prior to sending samples to the laboratory, all sample bags and number strings were checked for continuity and sample bag integrity;
- All diamond cores were photographed as a routine documentation of samples; and
- Drill core was stored in locally constructed wooden core boxes with painted labels on the end of each core box detailing box number, drillhole number and sampling intervals.

To confirm that no bias has been introduced, a test program was completed whereby samples of the cuttings were taken from the core saw tray and sludge samples were taken from the sumps used to settle out fine solids from the drill water. The settling tanks were installed as part of the environmental management plan for drilling. The core intervals to which these samples correspond were recorded so that the cuttings and sludge sample grades can be compared with the average grade for the interval.

10.2.2 Collar Surveys

All hole collars have been surveyed using a DGPS and have been surveyed to a high degree of confidence in terms of the XY location. Data has been provided to SRK in digital format using UTM grid coordinates.

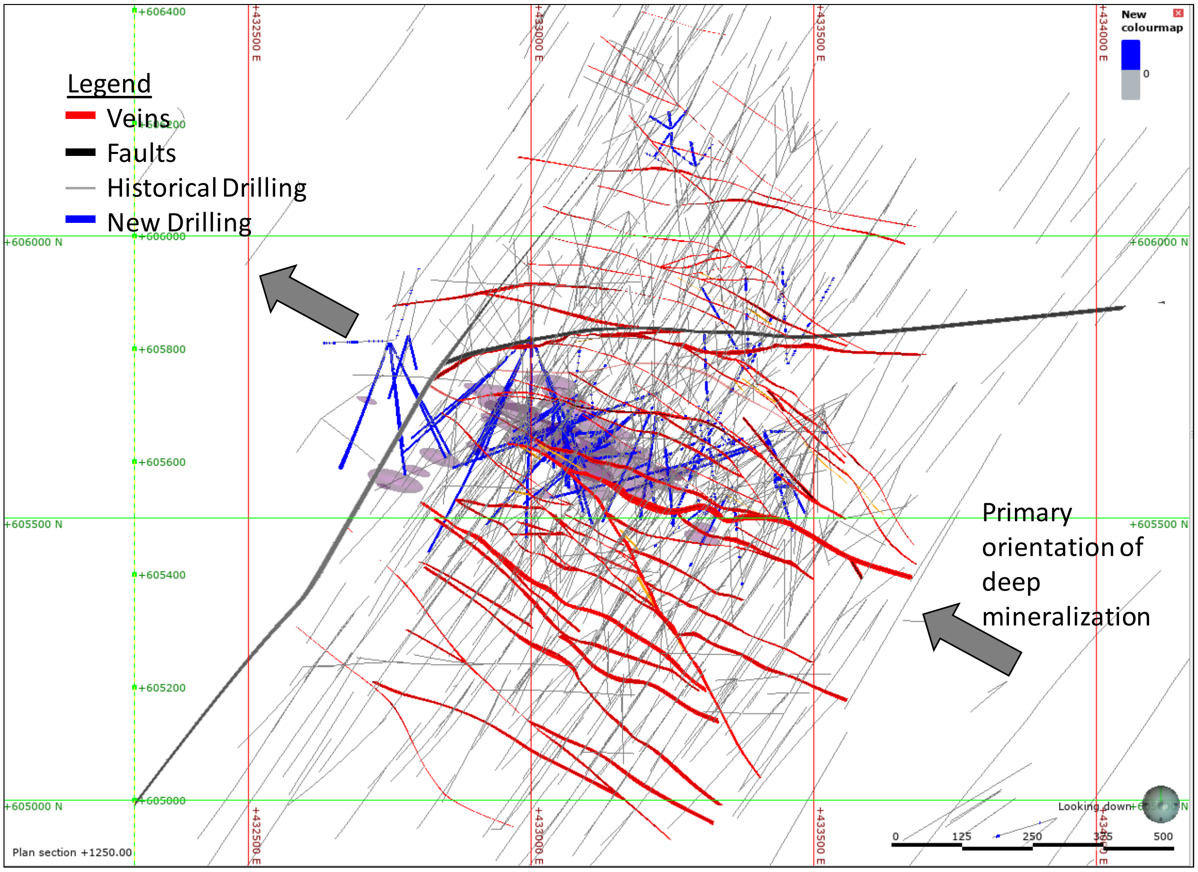
Drillhole collar elevations have been adjusted for errors based on projections on a digital terrain model (dtm) surveys based on the Ikonos satellite imagery, which gives contour levels every 2 m. It is SRK's view that even given the extreme topography found at Marmato that the current procedures site the collar locations with a sufficient degree of confidence

10.2.3 Drilling Orientation

During the initial exploration, drillholes were regularly spaced and orientated -60 and -75 degrees predominantly to the south west, with occasional scissor holes towards the north east. More recently with the focus on the MDZ, the Company has used fan drilling from underground adits, which are also typically orientated towards the south-west, with a smaller number of less extensive fans orientated towards the north-east.

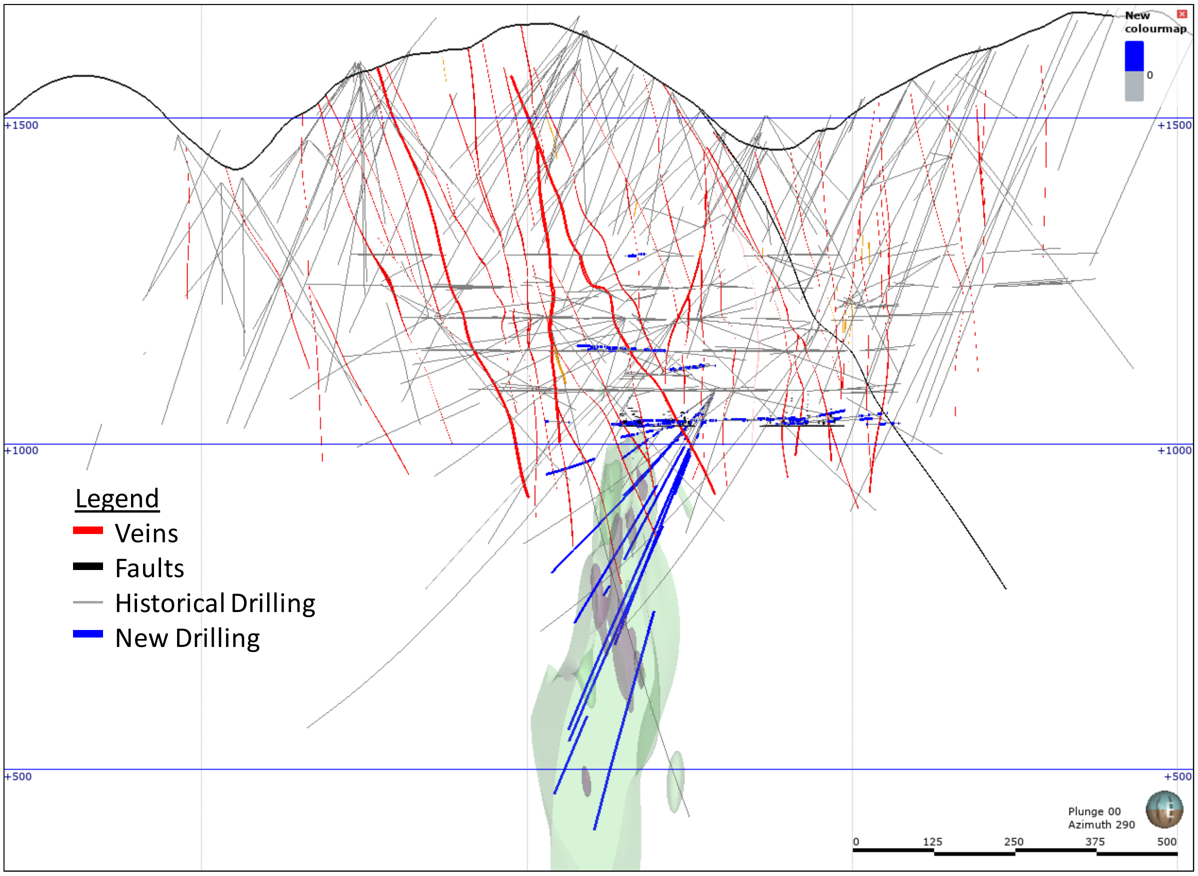
Drillholes have been drilled from four purpose-built underground drilling stations with two contractor rigs being used to date. Three of the drilling stations are located on Level 20 with a single station established on Level 21. Drillholes have been drilled in a fan pattern and dips ranging from -20 to -75 degrees predominantly to the southwest (ranging from 151º to 255º). Holes targeting the upper portions of the MDZ have shallower angles while the deeper targeted holes longer (>400 m) and steeper (Figure 10-4 and Figure 10-5).

In addition to the drilling completed by the GCM exploration team (MT-IU series), the current operating mine has also completed routine exploration drilling ahead of mining. The routine exploration is typically completed using horizontal drilling from the existing drives to aid in the mapping and delineation of the known veins prior to mining.



Source: SRK, 2019

Figure 10-4: Plan Showing Primary Drilling Orientation to the South and Southwest Relative to the Main Mineralization Orientation at Depth



Source: SRK, 2019

Figure 10-5: Cross Section (Orientated Looking Northeast), Showing Orientation of Drilling Relative to the Deep Mineralization, and Horizontal Drilling in the Current Operation

10.3 Interpretation and Relevant Results

The updated drilling database indicates that the veins typically range between 0.5 and 5 m wide and extend for 250 to 1,000 m along strike, and 150 to 750 m down dip. This is supported by underground mining which has confirmed that individual vein structures have good geological continuity and can extend for 100 to 800 m along strike and 100 to at least 300 m down dip. Between 2017 and 2019, GCM has worked on updating the quantity of the underground channel sampling captured in the database, which has increased the information available to model the vein domains.

The broad zones of veinlet mineralization in the porphyry domain modelled initially by SRK in 2017 typically varied from 10 to 230 m wide, reaching up to 340 m wide in areas of significant veinlet accumulation, while extending with good geological continuity for between 200 m and approximately 950 m along strike, and between 100 and 900 m down dip. SRK has updated these domains during the 2019 geological modelling process using more discrete zones and application of an indicator grade shell approach using a 0.5 g/t Au CoG.

At depth within the central portion of the deposit, SRK has noted a zone of elevated grades which has been referred to as the higher grade MDZ (more than 2.0 g/t Au). This zone is indicated to be continuous along strike for approximately 500 m and has a confirmed down dip extent that reaches up to 800 m, with a thickness that varies between 35 and 150 m. It is possible that the main MDZ mineralization is bounded within a series of faults but limited drilling at the edges of the deposit make confirmation difficult to assess at this stage.

11 Sample Preparation, Analysis and Security

11.1 Security Measures

The Chain of Custody procedures for sample security were set up for the Company by Dr. Stewart Redwood in December 2005 (with the latest update in August 2009). During the initial exploration (2005 to 2006), sample numbers were created in the field based on a combination of the sample location, sample type and sample point, with descriptions of each sample noted in a field book and later transcribed. While providing useful information, the decision was taken in 2006 to change to a sequential numbering system based on preprinted sample tickets. In both cases the sample numbers have been transcribed on the sample bags to avoid errors (e.g. lost tickets).

At the drill rig, the drilling contractors are responsible for removing the core from the core barrel (using manual methods), and placing the core in prepared core trays (3 m length). The core is initially cleaned to remove drilling additives, but attempts are made to ensure fine material is not lost. Once completed, the core tray is closed with a wooden lid, hammered shut, and GCM geologists or technicians take possession. The drill core is then transported to the core shed for selection of sampling intervals and initial sample preparation. On receipt at the core shed, GCM geologists and technicians follow the logging and sampling procedures laid out in Section 11.2. Once completed and the half core has been photographed, the core boxes are again sealed and then transported to the onsite core storage facility. The core storage facility is within a secure area with a single access gate controlled by a 24-hour security guard.

In preparation for shipment, samples were packed into nylon rice sacks with approximately five samples per rice sack. The shipments were accompanied with the laboratory submittal forms and were transported to Medellín. Samples were accumulated at sample dispatch (in the case of historical holes this was a warehouse in Medellín) until a hole was completed. Drillholes were only submitted in their entirety once sampling was completed. The samples were transported by GCM employees to the preparation facilities. Upon reception at the sample preparation facility, the laboratory company checked that the samples received matched the work order and signed that it had accepted the samples.

Once the sample preparation was completed, the laboratory dispatched the sample pulps by courier to selected overseas laboratories. The laboratories were instructed to retain excess sample pulps after analysis which can be used in the event that check analyses are requested by GCM.

The coarse sample rejects and sample pulps from the preparation facilities in Medellín were picked up by GCM technicians during routine sample shipments to the preparation facilities. The coarse rejects and pulps were returned to the GCM core shed at Marmato for long-term storage.

11.2 Sample Preparation for Analysis

11.2.1 Historical Sample Preparation (Pre 2010)

Prior to the opening of the Inspectorate and SGS sample preparation laboratories in Medellín in August 2006 and November 2007, respectively, there were no internationally certified sample preparation laboratories for mineral exploration in Colombia. At the start of exploration work by GCM, all samples had to be sent to other countries to be prepared and analyzed. Entire rock samples were sent by air to Inspectorate in Sparks, Nevada (ISO 9001:2000 and ISO 9002:1994 certified) for preparation and analysis, or to ALS Chemex in Quito, Ecuador (ISO 9001:2000 and ISO 17025:2005 certified) for preparation, with analysis by ALS Chemex in Lima. ALS Chemex in Reno, Nevada was also used for some check analyses.

The Sparks analytical laboratory was used until late 2007; however, considerable QA/QC problems were experienced during 2007 as well as long delays in turnaround time, and since late 2007 to 2010 the Lima analytical laboratory was used. The analyses from Inspectorate's Sparks laboratory which failed QA/QC were repeated. Other samples analyzed initially at Inspectorate's Sparks laboratory were re-analyzed at Inspectorate's Lima, Peru laboratory. Only sample batches that passed QA/QC were accepted and stored in the final database.

The secondary laboratory used was SGS (ISO 9001 certified) at a sample preparation facility in Medellín, and at their analytical laboratory run by SGS del Perú S.A.C., El Callao, Lima. Inspectorate is used as the laboratory for check on any SGS submissions and replicate assays of samples analyzed initially at SGS del Peru S.A.C in Callao, Lima, Peru.

The sample preparation at the Inspectorate laboratory in Medellín consisted of drying the entire sample and crushing it to >70% passing -10 mesh by jaw crusher and roll mill. This was later changed to >85% passing -10 mesh using a TM Terminator Jaw Crusher. A split of 250 to 500 g was then obtained using a Jones splitter and was pulverized to >80% passing -150 mesh with Labtech LM2 pulverizing ring mill. Tested barren silica sand was used as a clean wash between each sample in pulverization.

The sample preparation procedures at the SGS laboratory in Medellín and SGS Colombia S.A. facility in Barranquilla, comprised drying the sample, crushing the entire sample in two stages to -6 mm and -2 mm by jaw crusher (>95% passing), riffle splitting the sample to 250 to 500 g, and pulverizing the split to >95% passing -140 mesh in 800 cm^3 chrome steel bowls in a Labtech LM2 pulverizing ring mill (preparation code 321).

The sample preparation method at the Inspectorate laboratory in Sparks, Nevada was to dry and crush the entire sample to >85% passing -10 mesh by TM Terminator Jaw Crusher, spilt 250 g to 300 g using a Jones splitter and pulverize this to >90% passing -150 mesh with a Labtech LM2 pulverizing ring mill. Tested barren silica sand was used as a clean wash between each sample in pulverization (rock chip 0 to 10 lb method).

The sample preparation procedure at the ALS Chemex laboratory in Quito was to log the sample into the tracking system, weigh, dry, crush the entire sample to >70% passing 2 mm, split off up to 1.5 kg and then pulverize the split to >85% passing 75 microns (code PREP-32).

11.2.2 Sample Preparation (2010 – 2017)

The sample preparation method at the internal GCM mine laboratory comprise drying the sample, crushing the entire sample to minus 5 mm by jaw crusher (>95% passing), riffle splitting a sub-sample of 200 to 300 g, and pulverizing the sub-sample in a disc mill to >80% passing -200 mesh. SRK visited the facility during the 2017 site inspection, and noted new equipment had been purchased and was not currently in use at the time of visit (Figure 11-1). The new equipment was consistent with those used at the third-party commercial laboratory. One issue noted during the site inspection is the stacking of sample trays (full) prior to pulverizing, which SRK does not consider to be best practice as this could result in cross contamination of samples. Ideally sample should be stacked on individual trays on a trolley as shown in Figure 11-1.

Since January 2010, the primary laboratory used for the exploration samples in the drill and underground sampling programs was ACME Laboratories for sample preparation in Medellín, and analytical laboratories in Sparks, Nevada, USA and Lima, Peru. The 2011 drill program utilized the ACME sample preparation laboratory in Medellín and the ACME assay laboratory in Santiago, Chile. In addition, the SGS laboratory in Lima, Peru was used as a check laboratory.

SRK visited the ACME sample preparation facilities on November 4, 2010. The sample preparation method at ACME, Medellín was to dry the sample in large controlled and crush the entire sample to >85% passing -10 mesh by TM Terminator Jaw Crusher (Figure 11-2).

The sample is then spilt to 250 to 300 g using a Jones splitter and pulverized to >90% passing -150 (75 µm) mesh with a Labtech LM2 pulverizing ring mill.

Tested barren silica sand was used as a clean wash between each sample in the crushing and pulverization stages (rock chip 0 to 10 lb method).

Source: SRK, 2017

Figure 11-1: Sample Preparation at Mine Laboratory Showing New Equipment (Crusher and Pulverizer)

Showing:
(a) Terminator Jaw Crusher;
(b) Jones Riffle Splitter;
(c) LM2 Mill; and
(d) Final Bar-Coded Sample Pulp.
Source: SRK, 2010

Figure 11-2: Sample Preparation Facilities at ACME Laboratories in Medellín

11.2.3 Sample Preparation (2017 – Current)

Since 2017 GCM has used SGS Laboratories in Medellin is the primary labatory for both sample preparation and analyzing all exploration drilling core samples. All GCM mine drilling have undergone preparation and analysis at ALS Laboratory, to ensure sample quality. GCM has incorporated routine check analysis on each laboratory with secondary assays at ALS for the SGS submissions and vice-versa.

Samples sent to SGS were prepared using method PRP93, which involved drying in oven at 100°C followed by jaw crushing. The sample was crushed to 90% passing -10 mesh size. The crusher was cleaned with compressed air between samples. A 250 g split, using a Jones splitter, was pulverized to 95% passing -140 mesh using a ring and puck pulverizer

11.3 Sample Analysis

The ACME laboratory in Santiago analyzed the samples (from the 2010-2012 drill programs) for gold by fire assay (FA) with atomic absorption spectrophotometer (AAS) finish. Samples over 10 g/t Au were assayed by FA with gravimetric finish. Silver was assayed by aqua regia digestion and AAS finish. Silver samples above 100 g/t were assayed by FA with gravimetric finish.

The historical samples by Conquistador (one of the previous owners) have been assayed by Barringer for gold by FA with atomic absorption (AA) finish, and checks by gravimetric finish for some high grade samples. Silver was determined by acid digestion with AA finish.

A detailed description of the sample analytical procedure undertaken for the 2011 SRK Mineral Resource Estimate (January 2011) and is provided, given the incorporation of these samples in to the current estimate:

The Inspectorate laboratory in Lima analyzed the samples for gold by FA with an AA finish (detection limits 0.005 ppm to 3 ppm, method FA/AAS). Silver was analyzed by aqua regia digestion and AA finish (method AA, detection limits 0.2 to 200 ppm). Over-limit gold assays (above 3,000 ppb or 3.0 ppm) were repeated by FA (1 assay ton, 29.2 g) with gravimetric finish (method Au FA/GRAV). Samples above a 200 g/t silver upper limit of detection were repeated by FA (1 assay ton, 29.2 g) with gravimetric finish (method Ag FA/GRAV). Samples were analyzed for multiple elements by aqua regia digestion and inductively coupled plasma (ICP) finish (32 Element ICP Package for Ag, Al*, As, Ba*, Bi, Ca*, Cd, Co, Cr*, Cu, Fe, Hg, K*, La*, Mg*, Mn, Mo, Na*, Ni, P, Pb, S*, Sb*, Se, Sn*, Sr*, Te*, Ti, Tl*, V, W, Zn). Inspectorate states that for elements marked * the digestion is partial in aqua regia in most silicate matrices and the analysis is partial. Over-limit zinc and lead analyses (more than 10,000 ppm) were rerun by aqua regia digestion and AA. Multi-element analyses were not carried out on the final batches of samples.

The Inspectorate laboratory in Sparks, Nevada analyzed samples for gold and silver by FA with an AA finish for gold (detection limits 2 ppb to 3,000 ppb) and AA finish for silver (detection limits 0.1 ppm to 200 ppm) (method Au, Ag FA/AA/AAS). Over-limit gold assays (above 3,000 ppb or 3.0 g/t) were repeated by FA with gravimetric finish (method Au FA/GRAV). Samples above a 200 ppm silver upper limit of detection were repeated by FA with gravimetric finish (method Ag FA/GRAV). Samples were analyzed for multi-elements by aqua regia digestion and ICP finish (30 Element ICP Package for Ag, Al*, As, B*, Ba*, Bi, Ca*, Cd, Co, Cr*, Cu, Fe, Hg, K*, La*, Mg*, Mn, Mo, Na, Ni, P, Pb, Sb*, Se, Sr*, Ti, Tl*, V, W, Zn). Inspectorate states that for elements marked * the digestion is partial in aqua regia in most silicate matrices and the analysis is partial. Over-limit zinc and lead analyses (>10,000 ppm) were rerun by aqua regia digestion and AA.

SGS del Perú S.A.C. analyzed samples for gold by FA (30 g sample) with an AA finish (code FAA313; detection limits 0.005 ppm to 10 ppm), and for silver with an aqua regia digestion and an AAS finish (code AAS12CP), or three acid digestion with AAS finish (code AAS42C); detection limits in both are 0.3 ppm to 500 ppm). Multi-element geochemical analyses were done by two different methods. One method (ICM40B) uses a four acid digestion and both ICP-AES and ICP-MS for 50 elements (Ag, Al, As, Ba*, Be*, Bi, Ca*, Cd, Ce, Cr*, Co, Cs, Cu, Fe*, Ga*, Ge*, Hf*, In*, K*, La*, Li*, Lu*, Mg*, Mn*, Mo, Na*, Nb*, Ni*, P*, Pb, Rb*, S*, Sb, Sc*, Se, Sn*, Sr*, Ta*, Tb, Te*, Th*, Ti*, Tl*, U*, V*, W*, Y*, Yb*, Zn*, Zr*), elements marked * the digestion is partial.

The second method (ICP12B) uses a two acid (HNO3 and HCl) digestion and both ICP-AES and ICP-MS for 38 elements (Ag, Al, As, Ba*, Be*, Bi, Ca*, Cd, Co, Cr*, Cu, Fe*, Ga*, Hg, K*, La*, Mg*, Mn*, Mo, Na*, Nb*, Ni*, P*, Pb, S*, Sb, Sc*, Sn*, Sr*, Ti*, Tl*, V*, W*, Y*, Zn*, Zr*), elements marked * the digestion is partial. SGS indicates that the analysis is partial for elements marked * and depends on the mineralogy. Over limit gold values were repeated by FA with a gravimetric finish (method FAG303) and a lower limit of detection of 0.02 g/t. Silver grades above 100 ppm and zinc grades above 1% were repeated by four acid digestion and AA (method AAS41B). Gold and silver for some samples was by FA with gravimetric finish on 30 g (method FAG323 with lower limit of detection of 0.03 g/t gold and 0.03 g/t silver).

Since 2017 GCM has used SGS Laboratories in Medellin is the primary labatory for both sample preparation and analyzing all exploration drilling core samples. Samples have been analysed at each laboratory for gold and silver by FA. At SGS the assaying using a Au 30 g AAS (method FAA313). All GCM mine drilling have undergone preparation and analysis at ALS Laboratory, to ensure sample quality. GCM has incorporated routine check analysis on each laboratory with secondary assays at ALS for the SGS submissions and vice-versa.

The GCM channel samples have been assayed at an onsite internal mine laboratory for gold and silver by FA with gravimetric finish. SRK reviewed the laboratory and noted some areas of improvement relating to the state of the equipment. The mine has recently purchased new sample preparation equipment, which should result in improved assay quality. SRK recommends GCM complete routine check analysis between the Mine Laboratory and an independent commercial laboratory in Medellín. SRK recommends that all exploration samples are kept clear of the mine laboratory to avoid any potential contamination.

11.4 Quality Assurance/Quality Control Procedures

SRK completed a detailed review of the Quality Assurance/Quality Control (QA/QC) procedures and results as part of the 2012 MRE. The results are summarized in the current report as limited drilling has been completed between 2012 and 2017.

The routine QA/QC program at Marmato comprises certified standard reference materials (CRM), quartz blanks, preparation duplicates (PD), Coarse Duplicates (CD), field duplicates (FD) and check and replicate assays. The CRM, quartz sand blanks and Duplicate samples make up that portion of the QA/QC program which provides ongoing monitoring of the geochemical laboratories. The check assay and replicate assay samples are submitted at longer time intervals (less frequently) and provide a secondary control on the accuracy of the geochemical data.

Sampling protocols suggest the following submission rates:

- For the CRM, five random numbers are generated between 1 and 100;
- For the FD samples, two random numbers are generated between 1 and 100;
- For the PD samples, two random numbers are generated between 1 and 100; and

- In contrast, the blanks are inserted at points within the sample stream where, based on the geology, the geologist believes that there is a high likelihood of significant mineralization, and therefore potential for contamination.

GCM employed a database administrator of the QA/QC program at Marmato, during this period. SRK held discussions with the database administrator during the site visit to review how the data was captured.

SRK has been supplied with a complete QA/QC assay database for the Project in two separate excel files which summarize the submissions in 2018 and 2019 (up to borehole MT-IU-031). Within the 2018/2019 exploration period when the majority of the drilling has been completed by GCM a total of 510 certified standards, 412 blanks and 624 duplicates, representing approximately 15% of total routine sample submissions for the GCM drilling programs at Marmato to SGS. Additionally GCM a total of 73 certified standards, 58 blanks and 57 duplicates, representing approximately 13 % of total sample routine submissions for the GCM drilling programs at Marmato to ALS. In addition to the duplicates a series of reassays and check assays have been completed using alternative laboratories (SGS vs ALS). In 2018 a total of 188 assays along with associated QA/QC were re-assayed from SGS submissions, and 42 reassays from ALS submissions. In 2019 a total of 33 re-assays have been selected to date from SGS submissions. The pulp and reject check programs represent a total of 475 and 476 submissions respectively. SRK considers the level of QA/QC submissions to be of acceptable levels for the current stage of the Project.

A summary of the breakdown per sample type and laboratory are shown in Table 11-1 (2018) and Table 11-2 (2019).

Table 11-1: Summary Of QA/QC Sample Submissions During 2018 Submissions To SGS And ALS Laboratories

Marmato Exploration							
Original Shipments							
Sent Shipments			**Received Analysis**		**QAQC**		
Laboratory	Shipments	Samples	Shipments	Samples	Standard	Blanks	Duplicates
SGS	29	4530	29	4530	224	178	275
ALS	10	909	10	909	46	36	36
Re-Analysis Shipments							
Sent Shipments			**Received Analysis**		**QAQC**		
Laboratory	Shipments	Samples	Shipments	Samples	Standard	Blanks	Duplicates
SGS	6	203	6	203	12	0	1
ALS	3	46	3	46	3	0	1

Source: GCM, 2018

Table 11-2: Summary Of QA/QC Sample Submissions During 2019 Submissions To SGS And ALS Laboratories (Up to MT-IU-031)

Marmato Exploration							
Original Shipments							
Sent Shipments			**Received Analysis**		**QAQC**		
Laboratory	Shipments	Samples	Shipments	Samples	Standard	Blanks	Duplicates
SGS	48	5848	48	5848	286	234	349
ALS	7	526	7	526	27	22	21
Re-Analysis Shipments							
Sent Shipments			**Received Analysis**		**QAQC**		
Laboratory	Shipments	Samples	Shipments	Samples	Standard	Blanks	Duplicates
SGS	3	36	3	36	3	0	0
ALS	2	21	2	21	1	1	2

Source: GCM, 2019

11.4.1 Standards

In the 2018/19 program, 510 Certified Reference Material (Standards) were submitted during routine summissions to SGS and 73 with ALS submissions.

A summary of the submissions is shown in Table 11-3. In the submissions to SGS, SRK concluded the majority of standards had a greater number of overestimations than underestimations. The discrepancies noted are likely to be due to occasional laboratory issues, however this has not resulted in a material bias overall.

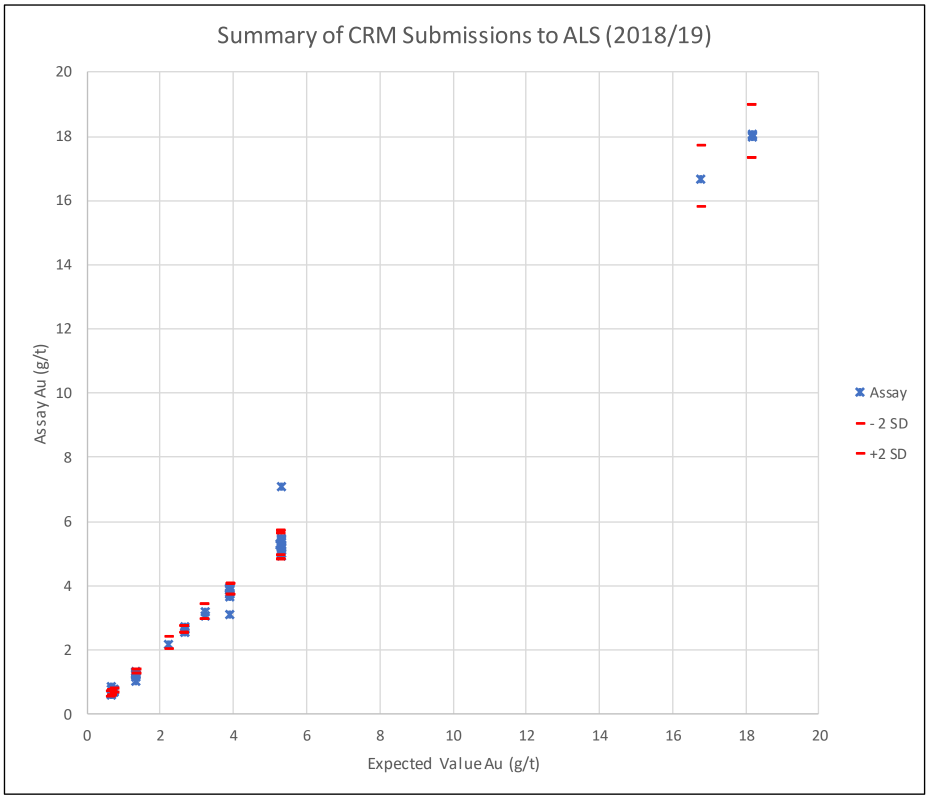
SRK has reviewed the CRM results and is satisfied that they demonstrate a high degree of accuracy at the assaying laboratory and hence give sufficient confidence in the assays for these to be used to derive a Mineral Resource estimate. GCM has utilized CRM from Geostats Pty Ltd., Rocklabs, and OREAS. In 2018/19, 22 CRM's were inserted into the sample stream; of these, four (10.5%) were identified by GCM as being erroneous (i.e., values exceeded three standard deviations from the recommended value).

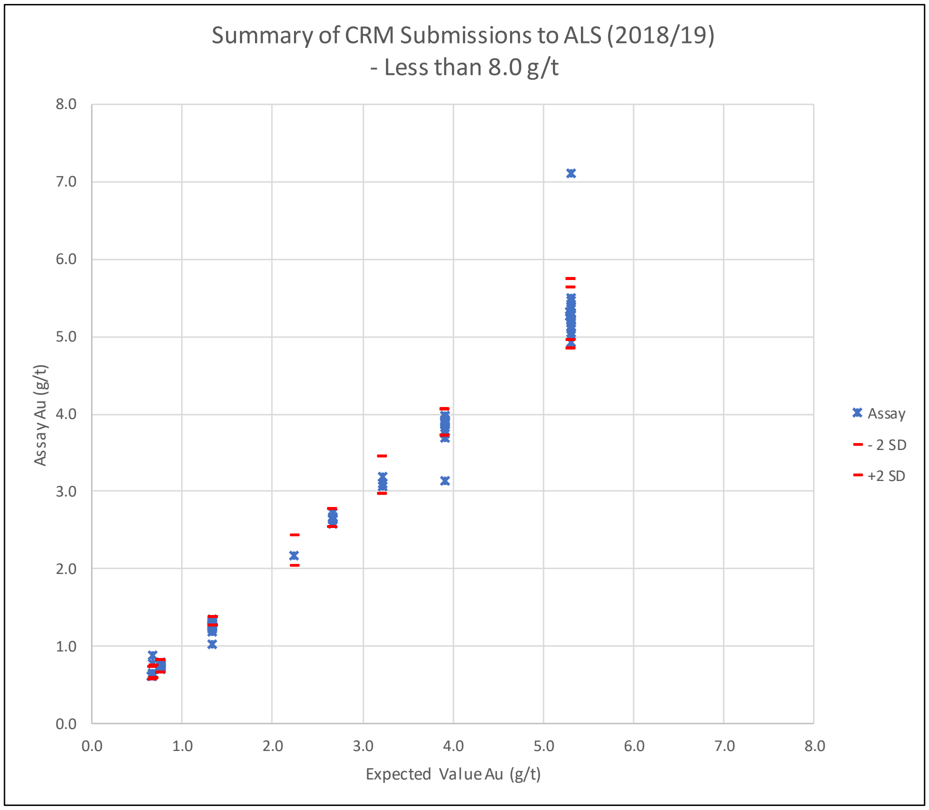
Table 11-3: Summary of CRM's Submitted During Routine Assay Submissions

STM_NAME	StdValue	SD1Low	SD1High	SD2Low	SD2High	SD3Low	SD3High
G310-6	0.65	0.61	0.69	0.57	0.73	0.53	0.77
G312-4	5.3	5.08	5.52	4.86	5.74	4.64	5.96
G313-1	1	0.95	1.05	0.9	1.1	0.85	1.15
G313-2	2.04	1.97	2.11	1.9	2.18	1.83	2.25
G314-1	0.75	0.71	0.79	0.67	0.83	0.63	0.87
G314-5	5.29	5.12	5.46	4.95	5.63	4.78	5.8
G315-2	0.98	0.94	1.02	0.9	1.06	0.86	1.1
G914-6	3.21	3.09	3.33	2.97	3.45	2.85	3.57
G914-9	16.77	16.29	17.25	15.81	17.73	15.33	18.21
G915-5	17.95	17.09	18.81	16.23	19.67	15.37	20.53
G915-6	0.67	0.63	0.71	0.59	0.75	0.55	0.79
OREAS-15Pc	1.61	1.571	1.649	1.532	1.688	1.493	1.727
OREAS-60P	2.6	2.56	2.64	2.52	2.68	2.48	2.72
OREAS-62Pa	9.64	9.56	9.72	9.48	9.8	9.4	9.88
OREAS-67A	2.238	2.142	2.334	2.046	2.43	1.95	2.526
SH35	1.323	1.279	1.367	1.235	1.411	1.191	1.455
SH82	1.333	1.306	1.36	1.279	1.387	1.252	1.414
SJ80	2.656	2.599	2.713	2.542	2.77	2.485	2.827
SK94	3.899	3.815	3.983	3.731	4.067	3.647	4.151
SN91	8.679	8.485	8.873	8.291	9.067	8.097	9.261
SP73	18.17	17.75	18.59	17.33	19.01	16.91	19.43
SQ88	39.723	38.776	40.67	37.829	41.617	36.882	42.564

Source: SRK, 2019

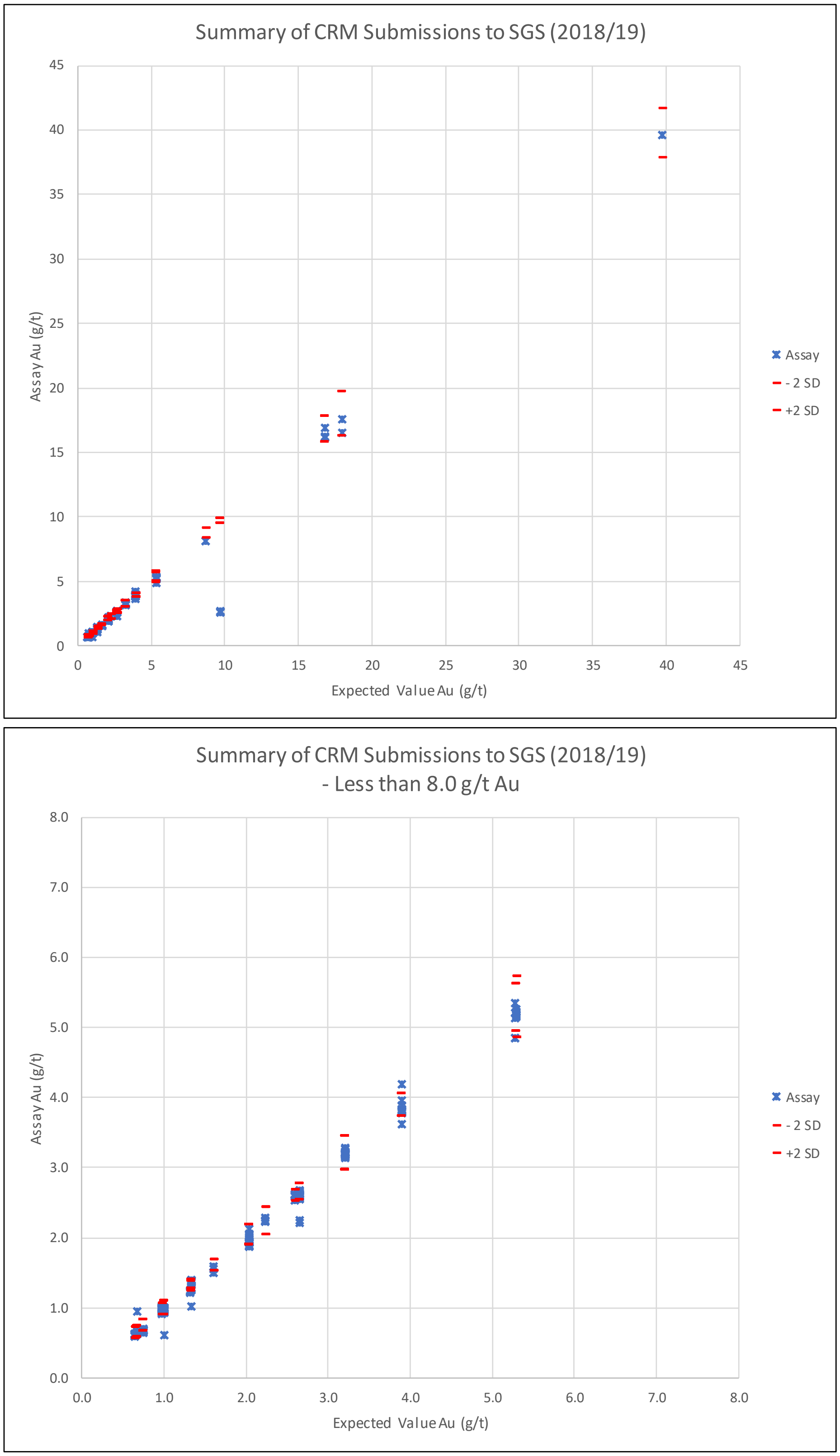
SRK has reviewed the results with the majority of the assays reporting within the desired two standard deviation limits which have been summarized in Figure 11-3 and Figure 11-4 below. SRK has analyzed each CRM in both summary and timeline form (example shown in Figure 11-5). The charts indicate that in general the laboratories have performed within the 2 standard deviation warning lines for each of the different CRM's, with no evidence of any high or low bias.





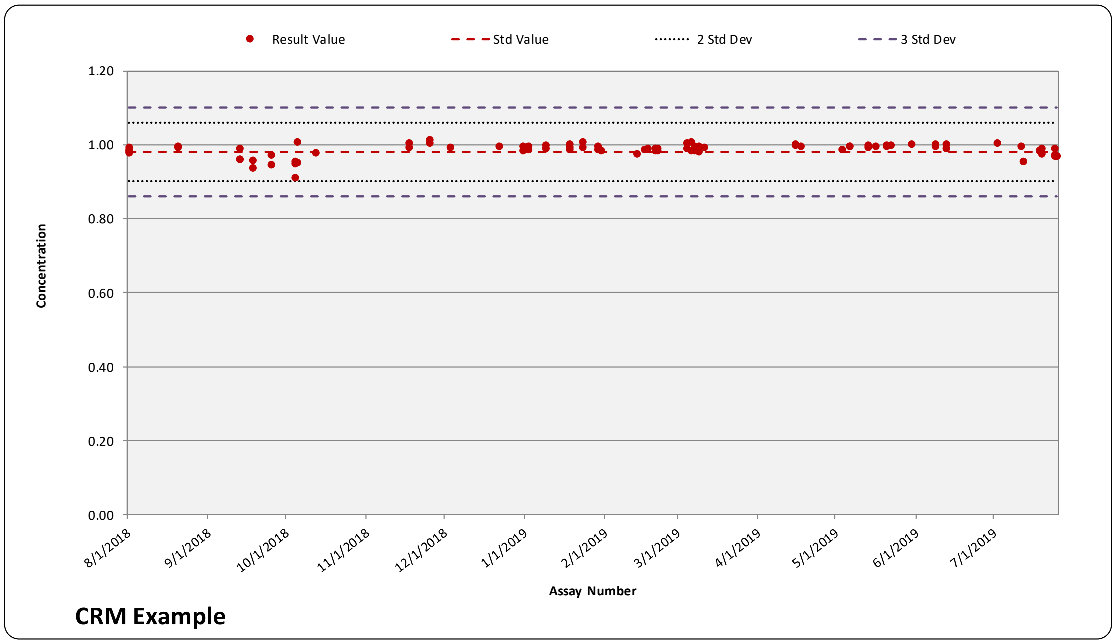
Source: SRK, 2019

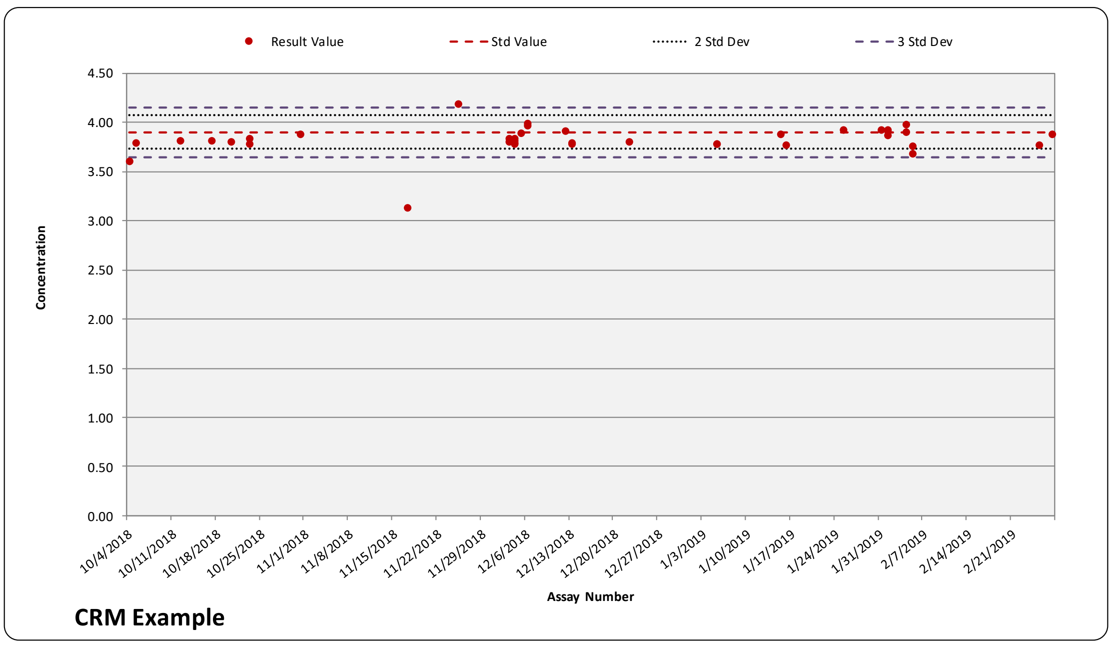
Figure 11-3: Summary Of CRM Submissions To ALS In 2018/2019 Program, Showing All Submissions (Left), And CRM's Below 8.0 G/T Au (Right)



Source: SRK, 2019

Figure 11-4: Summary of CRM Submissions To SGS In 2018/2019 Program, Showing All Submissions (Left), And CRM's Below 8.0 G/T Au (Right)



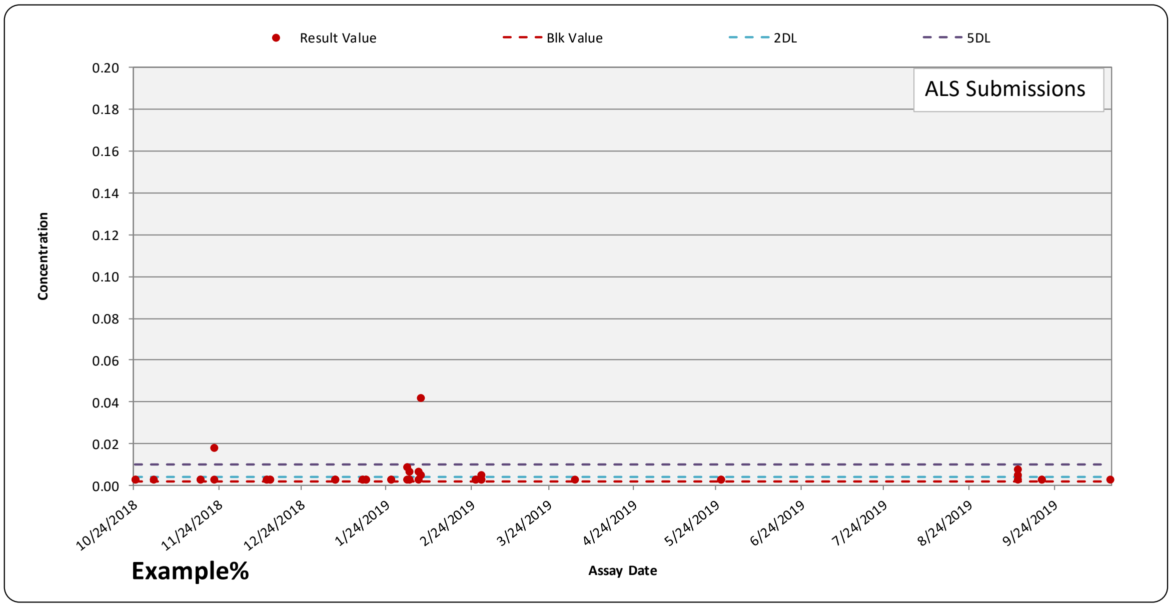


Source: SRK, 2019

Figure 11-5: Example of Timeline Review Of CRM G315-2 and SK94 Submissions

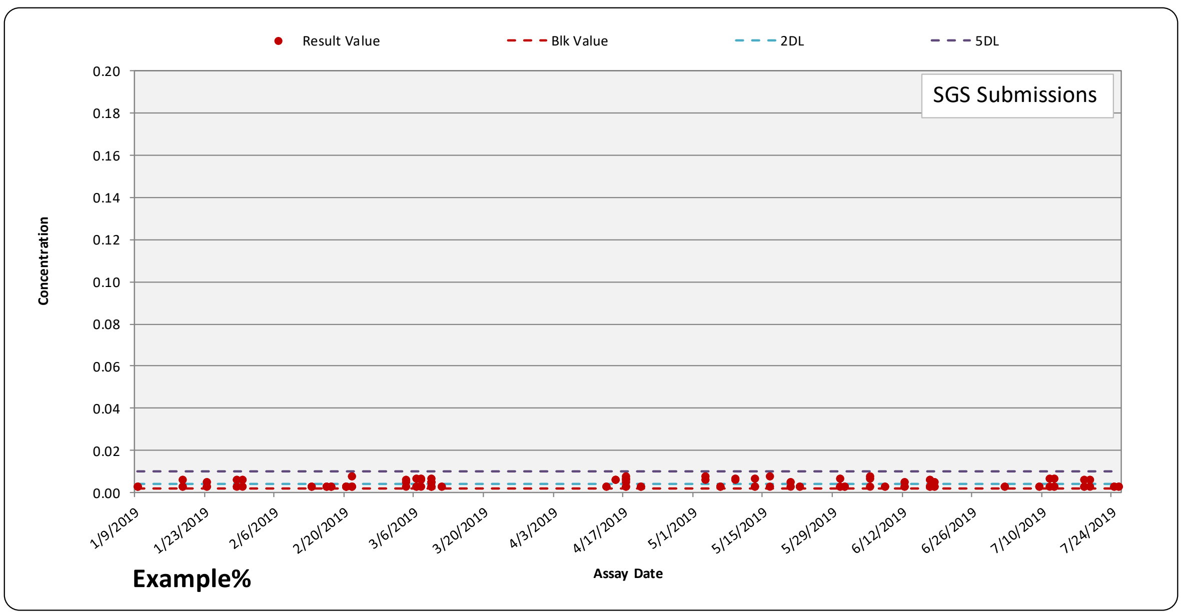
11.4.2 Blanks

Blanks are inserted at points within the sample stream where, based on the geology, the geologist believes that there is a high likelihood of significant or high-grade mineralization, and therefore potential for contamination. Coarse and Fine blank samples are submitted (412 in total) to both SGS and ALS (Figure 11-6 to Figure 11-9) the results have been reviewed to check for any potential evidence of contamination. SRK comments that no evidence has been noted with limited samples reporting above a value of 0.02 g/t Au.



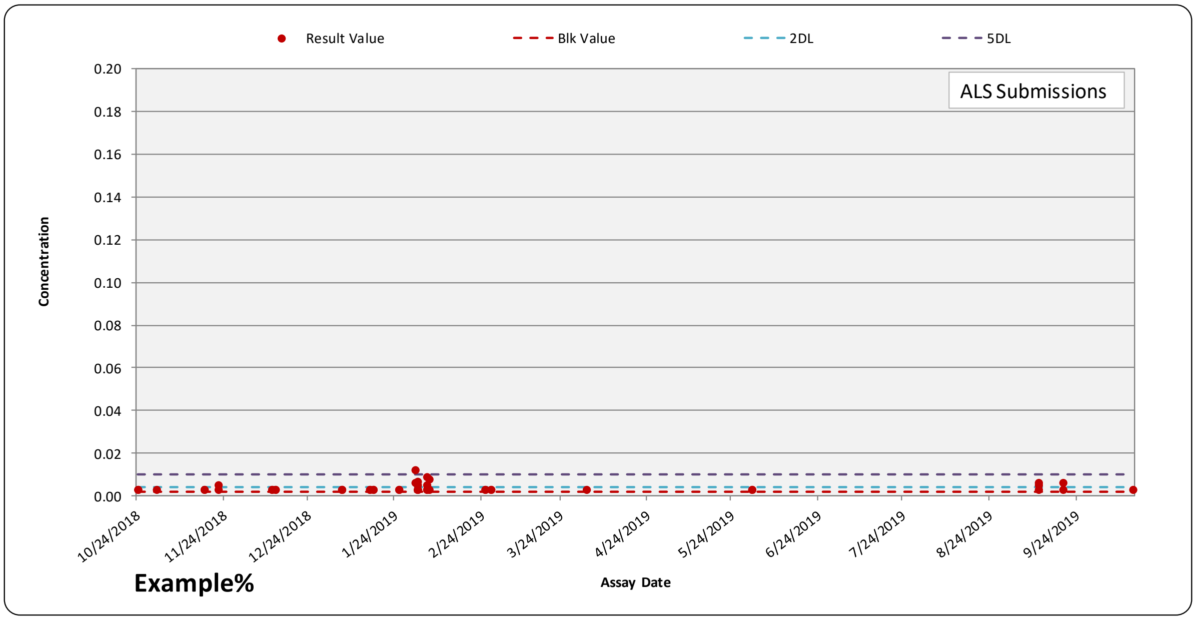
Source: SRK, 2019

Figure 11-6: ALS Coarse Blank Submissions



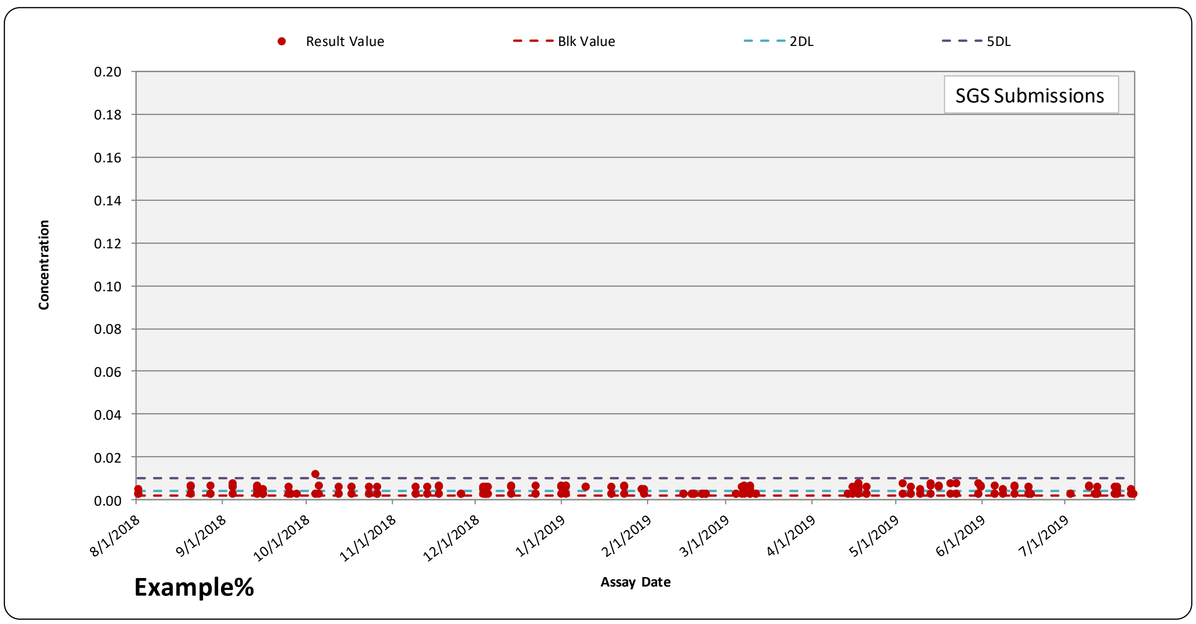
Source: SRK, 2019

Figure 11-7: SGS Coarse Blank Submissions



Source: SRK, 2019

Figure 11-8: ALS Fine Blank Submissions



Source: SRK, 2019

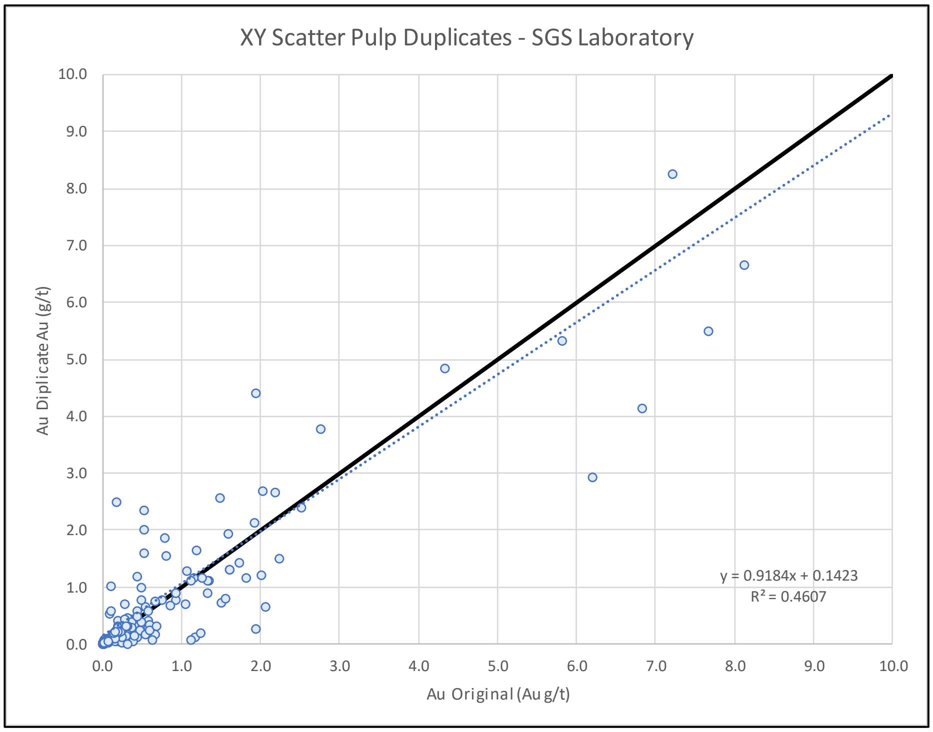
Figure 11-9: SGS Fine Blank Submissions

The Company has submitted three different types of duplicates during the routine sample submissions. The three different types have been defined as Field Duplicates, Coarse Duplicates and Pulp Duplicates. The field duplicates have only been submitted with the exploration drilling submitted to SGS with no duplicates in the mine drilling programs to ALS. A total of 223 field duplicates have been analyzed. The difference in the mean grades from the two data populations is 4% higher in the duplicate dataset, which returned 1.04 g/t and 1.06 g/t Au respectively. A summary of the results is shown in Table 11-4 and Figure 11-10.

Table 11-4: Summary Statistics for Field Duplicates to SGS laboratory

	Original Au (g/t)	Duplicate Au (g/t)
Mean	1.04	1.06
Standard Deviation	1.66	2.23
Sample Variance	2.76	2.23
Range	10.84	21.46
Minimum	0.01	0.01
Maximum	10.85	21.47
Count	223	223

Source: SRK, 2019



Source: SRK, 2019

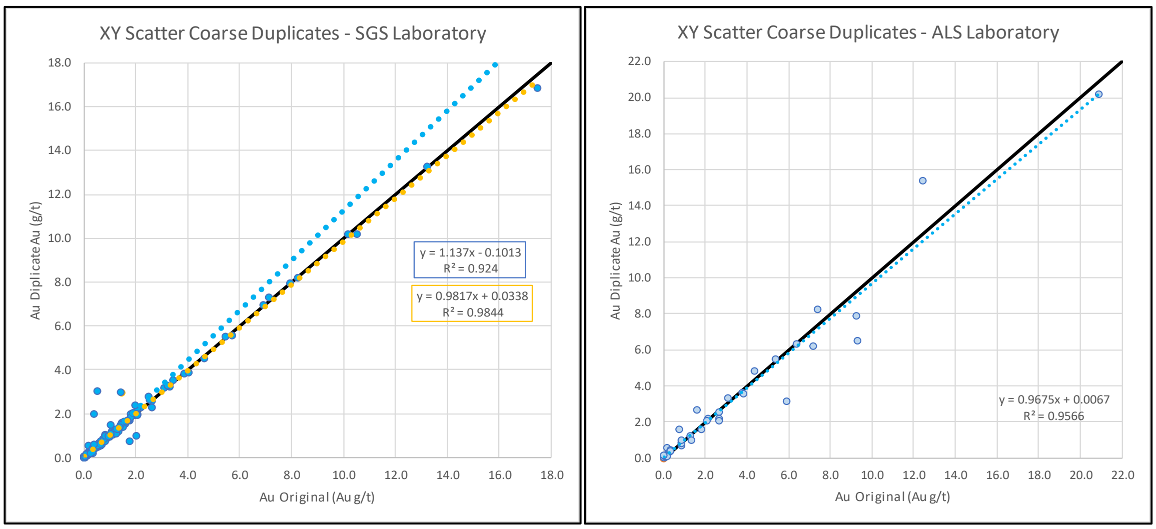
Figure 11-10: Summary of Field Duplicate Inserted with SGS Submissions

The coarse duplicates have been submitted in both exploration and mine drilling programs submitted to SGS and ALS. A total of 208 coarse duplicates have been analyzed at SGS and 42 at ALS. The difference in the mean grades from the two data populations is 5.5% higher in the duplicate dataset, which returned 1.24 g/t and 1.31 g/t Au respectively at SGS. This is skewed by a single high-grade sample, which once removed reduces the difference to 1.1 % in the mean grades. The difference in the mean grades from the two data populations is 3.0% lower in the duplicate dataset, which returned 2.97 g/t and 2.88 g/t Au respectively at ALS. A summary of the results is shown in Table 11-5 and Figure 11-11. The correlation coefficient in each case is greater than R2> 0.95, which indicates a strong correlation between the original and duplicate assays.

Table 11-5: Summary Statistics for Coarse Duplicates to SGS and ALS Submissions (Au g/t)

	SGS Submissions		ALS Submissions	
	Original Au (g/t)	Duplicate Au (g/t)	Original Au (g/t)	Duplicate Au (g/t)
Mean	1.24	1.31	2.97	2.88
Standard Deviation	2.42	2.87	4.16	4.11
Sample Variance	5.87	8.21	17.27	16.90
Range	17.48	28.68	20.86	20.14
Minimum	0.00	0.00	0.04	0.06
Maximum	17.48	28.68	20.90	20.20
Sum	258.36	272.69	124.75	120.98
Count	208	208	42	42

Source: SRK, 2019



Source: SRK, 2019

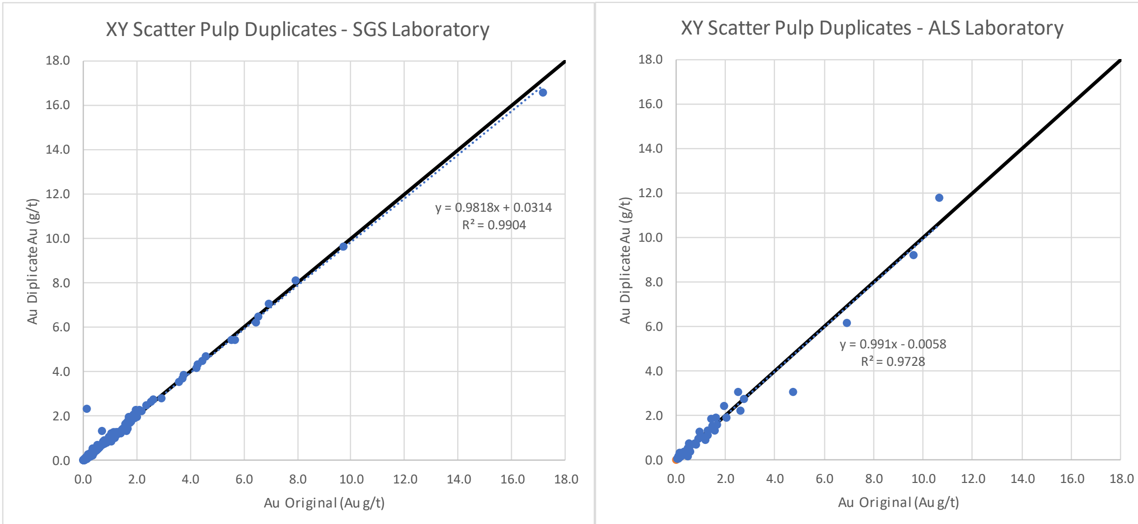
Figure 11-11: Summary of Coarse Duplicate Submissions to SGS (left) and ALS (right)

The pulp duplicates have been submitted in both exploration and mine drilling programs submitted to SGS and ALS. A total of 209 pulp duplicates have been analyzed at SGS and 46 at ALS. The difference in the mean grades from the two data populations is 1.2% higher in the duplicate dataset, which returned 1.03 g/t and 1.04 g/t Au respectively at SGS. The difference in the mean grades from the two data populations is 1.3% lower in the duplicate dataset, which returned 1.44 g/t and 1.42 g/t Au respectively at ALS. A summary of the results is shown in Table 11-6 and Figure 11-12.

Table 11-6: Summary Statistics for Coarse Duplicates to SGS and ALS Submissions (Au g/t)

	SGS Submissions		ALS Submissions	
	Original Au (g/t)	Duplicate Au (g/t)	Original Au (g/t)	Duplicate Au (g/t)
Mean	1.03	1.04	1.44	1.42
Standard Deviation	1.83	1.80	2.27	2.28
Sample Variance	3.33	3.24	5.17	5.22
Range	17.14	16.57	10.61	11.76
Minimum	0.01	0.01	0.04	0.04
Maximum	17.15	16.58	10.65	11.80
Sum	214.35	217.00	66.24	65.38
Count	209	209	46	46

Source: SRK, 2019



Source: SRK, 2019

Figure 11-12: Summary of Pulp Duplicate Submissions to SGS (left) and ALS (right)

11.4.3 Reassays

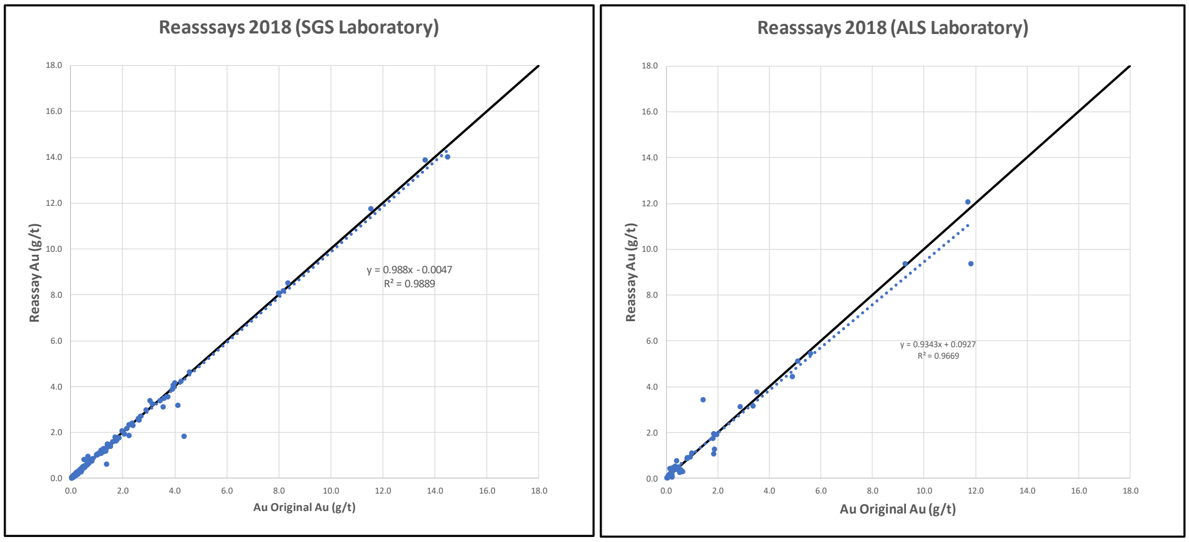
The Company has completed two reassay programs on the 2018 and 2019 submissions respectively. These represent resubmission of samples to a secondary laboratory (SGS to ALS and vice-versa). In 2018 a total of 188 sample pulp were reanalyzed from SGS samples which returned similar values across all grade ranges. A total of 42 ALS submissions were reassayed during the same period which return similar results.

The difference in the mean grades from the two data populations is 1.6 % lower in the reassay dataset, which returned 1.09 g/t and 1.08 g/t Au respectively in the original and reassays. Both results return very similar high-grades (205.79 g/t Au versus 202.42 g/t Au), and the correlation coefficient was R^2>0.98, which is considered an indication of no bias during this period at SGS. In the ALS reassays the mean grades returned 1.85 g/t and 1.82 g/t Au which is a reduction of 1.5 %, which SRK does not consider to be material during this time period. A summary of the results are shown in Table 11-7 and Figure 11-13.

Table 11-7: Summary Statistics for 2018 Reassays Program to SGS vs ALS Submissions (Au g/t)

	SGS Submissions (Primary)		ALS Submissions (Primary)	
	Original Au (g/t)	Duplicate Au (g/t)	Original Au (g/t)	Duplicate Au (g/t)
Mean	1.09	1.08	1.85	1.82
Standard Deviation	2.09	2.07	2.94	2.80
Sample Variance	4.36	4.30	8.67	7.82
Range	14.48	14.01	11.78	12.03
Minimum	0.01	0.01	0.02	0.02
Maximum	14.49	14.02	11.80	12.05
Sum	205.79	202.42	77.50	76.30
Count	188	188	42	42

Source: SRK, 2019



Source: SRK, 2019

Figure 11-13: Summary of 2019 Reassay (secondary laboratory) Submissions from SGS (left) and ALS (right)

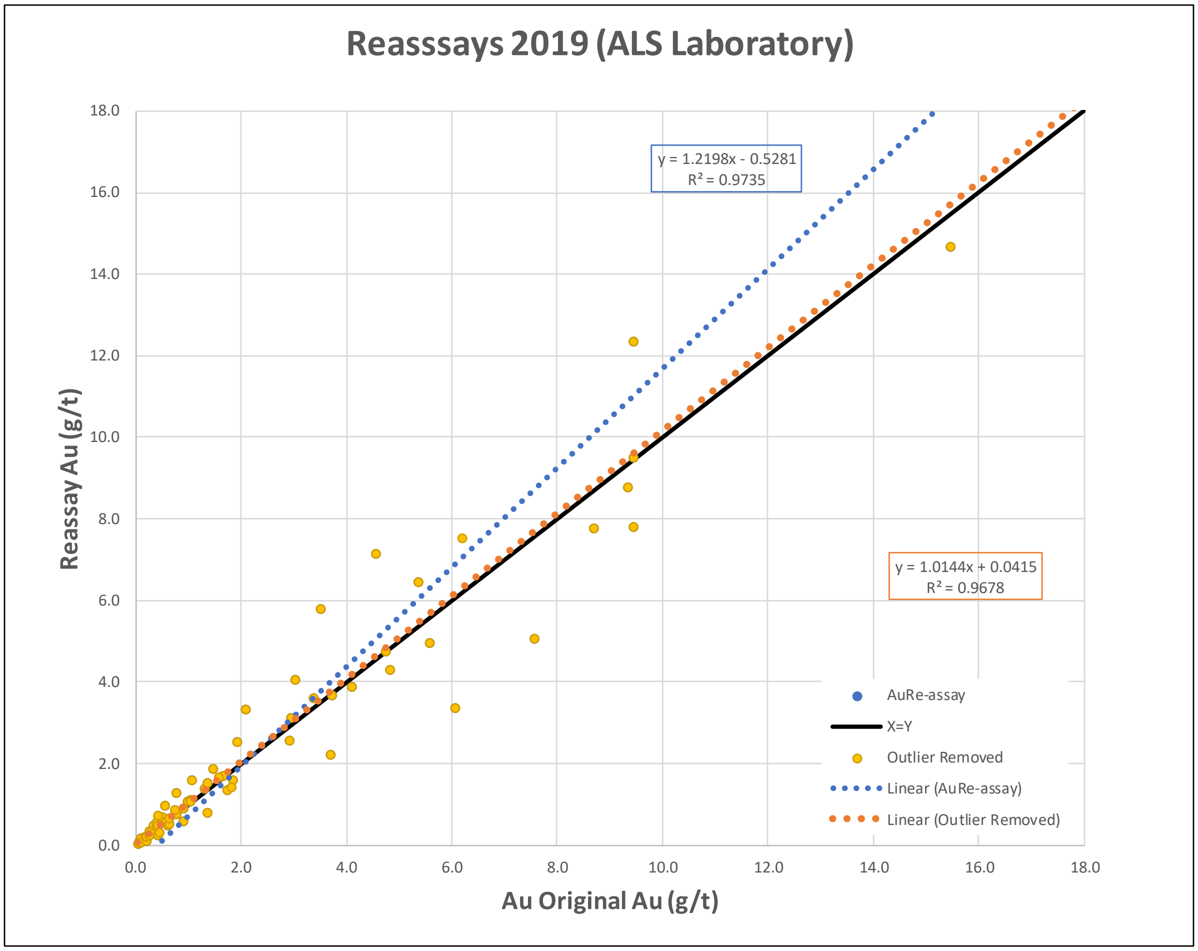
In 2019 a total of 33 sample pulp were reanalyzed from SGS samples which returned similar values across all grade ranges. A total of 72 ALS submissions were reassayed during the same period which returned similar results.

The difference in the mean grades from the two data populations is 1.6% lower in the reassay dataset, which returned 1.24 g/t and 1.25 g/t Au respectively in the original and reassays. While the mean grades show similar results the correlation coefficient was poor, which could be a result of the low sample population but should continue to be monitored during the remaining 2019 drilling programs. In the ALS reassays the mean grades returned 3.94 g/t and 4.31 g/t Au which is an increase of 9.2%. The results are influenced by a single high grade assays which impacts both the correlation and the comparative statistics. Review of the assays below values of 16.0 g/t Au, show a significant improvement in the comparison. After completing this analysis SRK does not consider it to be material during this time period. A summary of the results are shown in Table 11-8 and Figure 11-14.

Table 11-8: Summary Statistics for 2019 Reassays Program to SGS vs ALS Submissions (Au g/t)

	SGS Submissions (Primary)		ALS Submissions (Primary)	
	Original Au (g/t)	Duplicate Au (g/t)	Original Au (g/t)	Duplicate Au (g/t)
Mean	1.24	1.25	3.94	4.31
Standard Deviation	2.15	2.11	8.53	10.54
Sample Variance	4.63	4.47	72.70	111.01
Range	11.96	11.46	59.57	79.48
Minimum	0.04	0.07	0.03	0.02
Maximum	12.00	11.52	59.60	79.50
Sum	40.97	41.16	283.89	309.96
Count	33	33	72	72

Source: SRK, 2019



Source: SRK, 2019

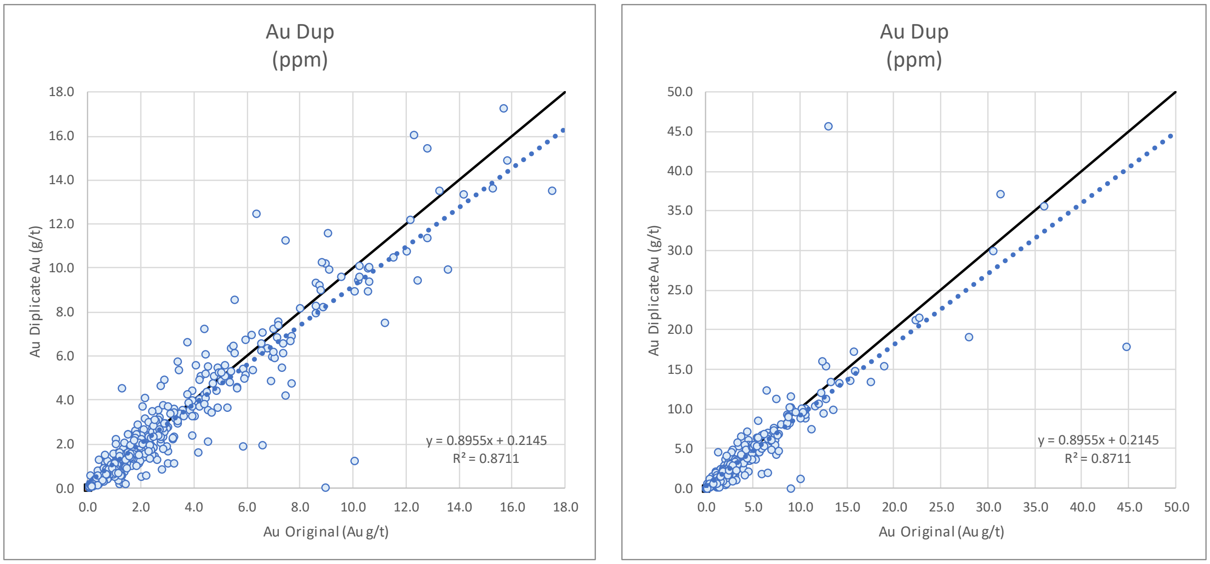
Figure 11-14: Comparison of 2019 Reassays from ALS (primary) Submissions, Showing All Data and Values Below 16.0 g/t Au

11.4.4 Check Analysis

In addition to the reassay program GCM also complete check analysis on pulps and reject material on a quarterly basis. A total of 475 sample pulp were reanalyzed from SGS samples which returned similar values across all grade ranges. A total of 476 rejects submissions were reassayed during the same period which returned similar results.

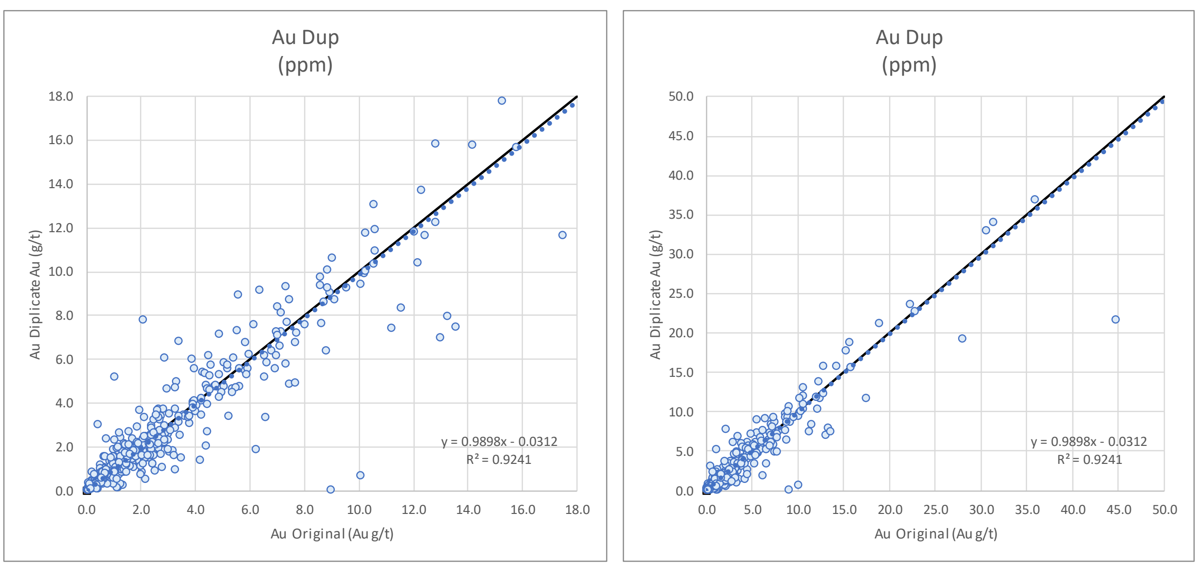
The difference in the mean grades from check pulps is 4.4 % lower in the check dataset, which returned 3.55 g/t and 3.39 g/t Au respectively in the original and check assays. Both results return very similar high-grades (62.97 g/t Au versus 59.60 g/t Au), and the correlation coefficient was R^2>0.87, which is considered an indication of no significant bias in the pulps. After completing this analysis SRK does not consider it to be material during this time period. A summary of the results are shown in Figure 11-15.

In comparison the difference in the mean grades from rejects is 1.9% lower in the check dataset, which returned 3.54 g/t and 3.48 g/t Au respectively in the original and check assays. SRK notes that the correlation coefficient is improved within the reject check analysis with R^2>0.92, A summary of the results are shown in Figure 11-16.



Source: SRK, 2019

Figure 11-15: Summary of Check Assays Completed on Pulp Material (Quarterly Checks), Showing Values Less Than 18.0 g/t (Left) and Full Dataset (Right)



Source: SRK, 2019

Figure 11-16: Summary of Check Assays Completed on Reject Material (Quarterly Checks), Showing Values Less Than 18.0 G/T (Left) and Full Dataset (Right)

11.5 Opinion on Adequacy

It is the opinion of SRK that the frequency of QA/QC sample inserted in the 2018 and 2019 campaign is at an acceptable rate as stipulated in the Company's internal guidelines (approximately 11%).

In general, it is the opinion of SRK that the results of the QA/QC analysis display a reasonably good correlation to the original assays and are acceptable for use in defining compliant Mineral Resource Estimates.

In the opinion of SRK, the sampling preparation, security and analytical procedures used by GCM are consistent with generally accepted industry best practices and are therefore adequate.

12 Data Verification

12.1 Procedures

12.1.1 Verifications by GCM

GCM has completed a number of verification sampling programs during the history of the Marmato Project. The work completed has ensured sample integrity and allowed SRK to have confidence to use the combined historical and GCM data as supplied by the Company. The work completed by GCM can be sub-divided into the validation and verification of the on-going exploration drilling programs, and the validation of underground channel sampling from the operating mine.

GCM employs a Database & GIS Manager who is responsible for tracking the samples through the laboratory. The Sample Order Form is given to the Database Manager. A Microsoft® Excel spreadsheet is used to track Company reference number, lab order number, date of delivery to lab, date of receipt of assays by email, date of receipt of certificate and date of receipt of invoice.

The Database & GIS Manager is responsible for receiving the assay results and importing these into the database. This is the only person with authority to do this in order to maintain integrity and quality control of the database.

On receipt of each batch of assays for the exploration drilling, the QA/QC samples are checked to accept or reject the batch. If there is a problem the Chief Geologist is notified and he requests that the laboratory identify and solve the problem, if possible, or carry out re-analyis, as necessary. If re-assay is required, either the whole batch or the sample tray between the good QC samples on either side is re-analyzed. Microsoft® Excel or Access spreadsheets and graphs are used to check QC results and update these with each batch so that the whole program is monitored progressively.

The laboratory also carries out its own internal QA/QC samples and the results for these are requested and monitored on an ongoing basis by GCM.

On-going validation included a detailed survey of historical collar positions using a DGPS which highlighted a number of minor discrepancies (typically less than 10 m). Based on the new survey data, the database has been updated accordingly and the interpretations adjusted to the new drillhole positions.

The on-going validation of the underground channel sampling database has been a considerable task, which has required capture of the sampling information from the mines operating long-sections into a 3D database. The program continued between 2017 and 2019, which has resulted in an increase of approximately 100% in the size of the channel sample database. To complete the task, GCM has completed surveys of the existing development to ensure accuracy of the placement of sampling on the main levels. GCM geologists have then created collar, survey and assay database for each sample relative to the strike of the deposit using the assumption that sampling has been completed perpendicular to the vein, as per the mines sampling procedure.

12.1.2 Verification by SRK

In accordance with NI 43-101 guidelines, Mr. Parsons of SRK most recently visited the Marmato Project on June 11, 2019. The main purpose of the site visit was to:

- Witness the extent of the exploration work completed to date;
- Complete an underground site inspection to understand the changes in the geological settings and possible exposure of the Marmato Deeps style mineralization;
- Inspect core logging and sample storage facilities;
- Discuss geological interpretation and inspect drill core;
- Assess logistical aspects and other constraints relating to the exploration property;
- Review data for the assay database; and
- Hold discussions with personnel involved in the current and historical exploration activities.

SRK did not complete an independent visit to the SGS Laboratory facility during the recent site visit, but visited the facility previously during the November 2011 site visit also completed by Ben Parsons.

SRK has been working with the GCM geological team between 2017 and through 2019, when data has been captured from the mine to generate a detailed geological model. For the most recent iteration of the database, in addition to the site inspection, SRK has completed a series of technical meetings with GCM geologists to review the on-going capture of the underground channel sampling program and integration into the database. SRK reviewed the capture and geo-referencing of the underground development with existing geological maps for each of the mining levels.

SRK has undertaken basic validation for all tabulated data in both Leapfrog and Datamine. Additionally, in order to independently verify the information incorporated within the latest drill program, SRK has:

- Completed a review of selected drill core for selected holes, to confirm both geological and assay values stored in the database show a reasonable representation of the Project;
- Verified the digital database against the original issued assay certificates;
- Visited underground workings to check the continuity of vein and veinlet mineralization at depth, including a site inspection to levels 19 through 21 to understand changes in the mineralization styles;
- Verified the quality of geological and sampling information and developed an interpretation of gold grade distributions appropriate to use in the resource model; and
- Reviewed the QA/QC database for the recent drilling and channel sampling programs.

12.2 Limitations

SRK has reviewed the data acquired for the Project and held a number of technical meetings at the Company office in Medellín to review the progress on the data validation. The efforts should remain ongoing and a lack of definition in portions of the 3D survey of the mines has limited the ability to accurately place all the samples in their “true” location. SRK notes that the information for the raise sampling shows the most significant variations from SRK geological interpretation using mapping between levels.

SRK has highlighted to GCM upon the validation phase, there still remains a large number of data points which contain significant mineralization that require constraining which lie outside of the revised

2019 vein interpretations. SRK noted a number of areas where, based on short channel samples, the geological model would likely result in overstating the tonnage if left unconstrained. Additionally, where these occur, the grade in any subsequent estimate will overstate grade locally with possible vein or veinlet material being incorrectly projected into the lower grade porphyry style domains.

The current structure of the database and naming convention for the underground channel sampling results in some limitations on generating an automated process to update the geological model. Isolated sampling of veins without surrounding samples can led to overstating the tonnage when using Leapfrog® and therefore caution has been required to review intersections on a case by case basis. Additionally, in a number of cases long cross-cut drift sampling has been logged as individual Hole_ID's, so restrictions based on length cannot be applied.

Regarding the significant rise in the amount of channel sampling, some of which has not been captured in the veins or splay model, it is the opinion of SRK that there is potential for over-estimation. SRK has noted this risk and therefore applied filters to the database to minimize these risks in the porphyry domain estimates. SRK considers this approach to be conservative, and it has resulted in a reduction in the contained metal within the domains if estimated with no filter; however, these estimates provide a more reasonable base case for classification. Further work is recommended between SRK and GCM to improve the database structure and geological modelling of this domain, prior to any consideration for use in a mining study.

SRK completed a series of test models, which removed the influence of any of these samples from the database, using filters on logging codes and channel length, which is considered a more conservative approach by discounting the influence of short channel samples. The resultant interpretation contained approximately half the volume of the optimistic scenario for the porphyry mineralization. Due to the uncertainty in the impact of these domains, SRK has therefore excluded this material from the current mining assessment. If a solution can be found in the short term to improve the confidence in the geological continuity, this may represent upside potential. SRK recommends follow-up work from GCM which includes mapping and verification of the presence of the porphyry-style mineralization and additional sampling (drilling if required), prior to inclusion in the mine plan.

12.3 Opinion on Data Adequacy

Based on the validation work completed by SRK, the database has been accepted as provided by GCM's Resource Geologist. SRK is satisfied with the quality of assays returned from the laboratory for the latest drilling program and that there is no evidence of bias within the current database which would materially impact on the estimate.

While there are areas for potential improvement, SRK is of the opinion that the exploration and assay data is sufficiently reliable to support evaluation and classification of Mineral Resources in accordance with generally accepted CIM Estimation of Mineral Resources and Reserves: Definitions and Guidelines (CIM, 2014).

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

A metallurgical program was conducted by SGS Lakefield on test composites from the MDZ and the results of this program are fully documented in SGS's report, "The Recovery of Gold from Marmato Deposit Samples" dated April 16, 2019. The metallurgical program included comminution testwork, mineralogical studies and an evaluation of several different flowsheet options including:

- Whole-ore cyanidation;
- Gravity concentration followed by cyanidation of the gravity tailing; and
- Gravity concentration followed by gold flotation from the gravity tailing and cyanidation of the flotation concentrate.

The test program also included solid-liquid separation and cyanide detoxification studies.

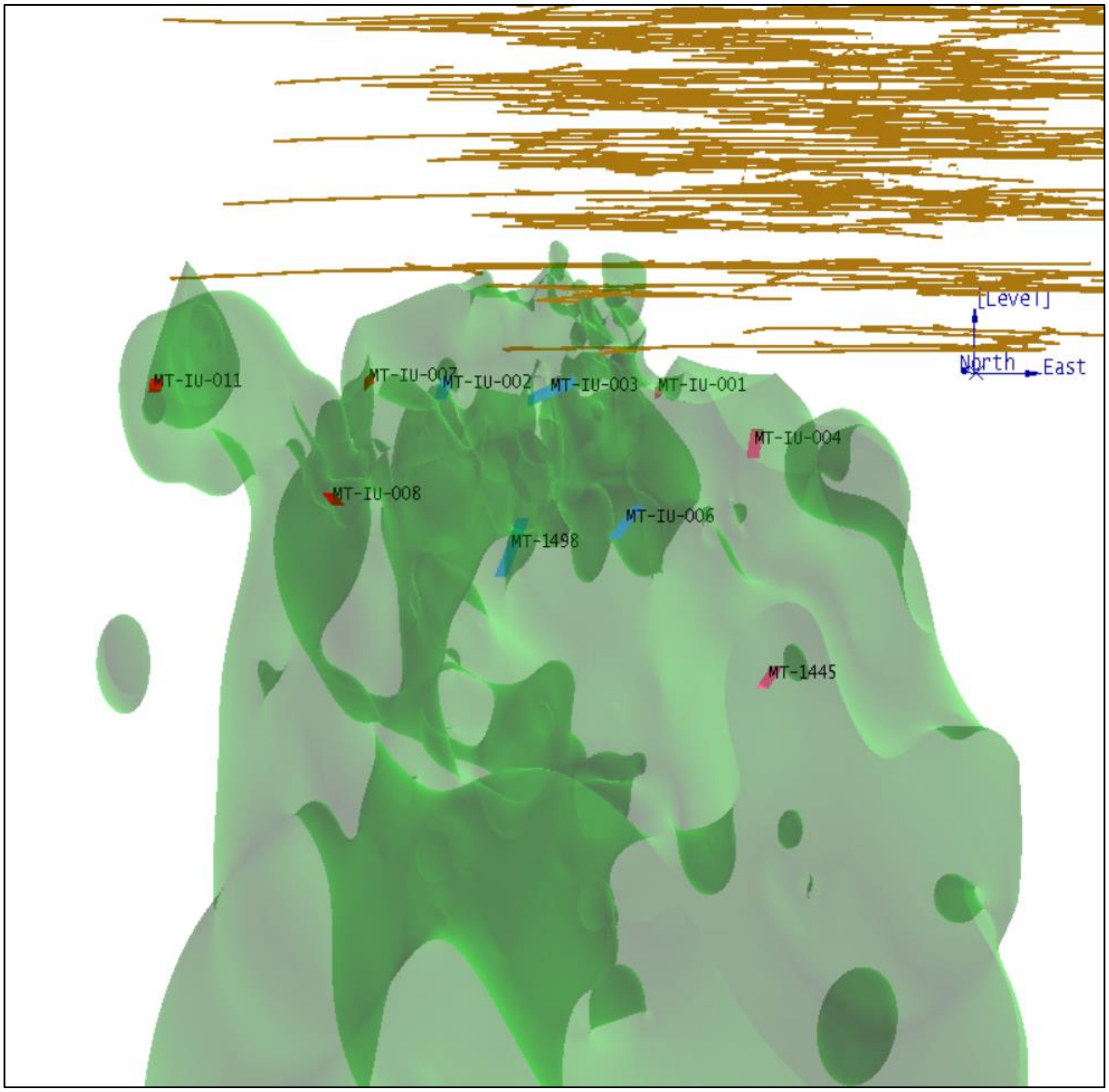
13.2 Metallurgical Sample Characterization

The test program was conducted on test samples prepared from drill core from the East, West and Central MDZ and a Master MDZ composite was formulated on a weighted basis from the East, West and Central MDZ samples. In addition, a composite representing the current Marmato material was also tested. The drill holes and intervals used to formulate the test composites are shown in Table 13-1 and the location of each drillhole is shown in Figure 13-1. Head analyses for each of the test composites are shown in Table 13-2.

Table 13-1: Drillholes and Intervals for MDZ Metallurgical Composites

Location	Drill Hole	From (m)	To (m)
East Zone	MT-IU-001	138.3	148.0
	MT-IU-004	152.4	182.2
	MT-1445	562.0	589.0
Central Zone	MT-IU-002	146.0	179.3
	MT-IU-003	209.4	256.5
	MT-IU-006	252.5	292.8
	MT-1498	334.0	386.0
West Zone	MT-IU-007	159.4	173.7
	MT-IU-008	251.0	266.6
	MT-IU-011	129.3	144.6

Source: SGS Lakefield, 2019



Source: SRK, 2019

Figure 13-1: Drillhole Locations

Table 13-2: Head Analyses for MDZ and Marmato Test Composites

Element	MDZ West	MDZ Center	MDZ East	MDZ Master	Marmato Comp
Au (S.M.) g/t	1.54	2.69	2.65	2.32	5.48
Au (Calc.) g/t	1.30	2.61	1.80	2.36	4.83
Ag g/t	0.90	3.9	6.7	4.2	19.6
S %	1.22	2.04	2.20	1.95	10.5
Te g/t	<4	<4	<4	<4	<4
Hg g/t	0.3	<0.3	<0.3	<0.3	<0.3

Source: SGS, 2019

13.3 Mineralogy

A mineralogical evaluation was conducted on a single sample from the MDZ Center Zone by Terra Mineralogical Services Inc. The results of this evaluation are fully documented in the report, “Determination of Gold Deportment in one Master Composite Sample from the Top of the Deep Marmato Deposit, Colombia” by Giovanni Di Prisco dated March 2, 2019. Key findings included the following:

- Native gold was by far the predominant gold carrier;
- The majority (>99%) of the gold particles occurred in locations that would be readily accessible by leaching solutions;
- The gold grains were predominately associated with silicate gangue minerals. Gold particles were not often in direct contact with sulfides, yet very commonly pyrrhotite, chalcopyrite, and bismuth minerals were found in close vicinity to the gold mineralization; and
- The average grain size of the gold particle identified could be best defined as very fine grained (<6 µm), yet a small amount of coarse gold particles also existed.

13.4 Comminution Testwork

Comminution testwork included SMC (SAG mill comminution), BWi (Bond ball mill work index) and Ai (Abrasion index) tests, the test results are shown in Table 13-3. The SMC tests were conducted on the East, West and Center MDZ and the reported Axb values ranged from 28 to 31 and averaged 29, indicating that the material is very hard with respect to SAG mill impact grinding. The BWi tests were conducted on all test composites using a 150 mesh (105 µm) closing screen, the MDZ composites ranged from 19.0 to 20.7 kWh/t and averaged 19.8 kWh/t. Indicating that the MDZ material is very hard with respect to ball mill grinding. It is also noted that the MDZ material is much harder than the current Marmato material which was reported to have a BWi of 15.7 kWh/t. The abrasion tests indicate that the MDZ material is very abrasive with abrasion indices ranging from 0.626 to 0.731 and averaging 0.678. The MDZ material is much more abrasive than the current Marmato material, which has a reported Ai of 0.199. Relatively high consumption of wear materials can be expected when processing MDZ material.

Table 13-3: Comminution Test Results on MDZ and Marmato Test Samples

Sample	Relative Density	JK Parameters			BWI (kWh/t)	AI (g)
		A x b	t_a	SCSE		
Marmato Material	-	-	-	-	15.7	0.199
MDZ Comp	-	-	-	-	20.7	0.652
Zone Center	2.65	31.0	0.30	11.0	19.0	0.704
Zone East	2.64	28.0	0.27	11.6	20.3	0.626
Zone West	2.69	28.0	0.27	11.7	19.0	0.731
Average	**2.66**	**29.0**	**0.28**	**11.4**	**19.0**	**0.582**

Source: SGS, 2019

13.5 Metallurgical Testwork

13.5.1 Whole-Ore Cyanidation

Two whole-ore cyanidation tests were completed on the MDZ composite. The tests were conducted at a grind size of 80% passing (P_{80}) 60 µm with a maintained cyanide concentration of 1 gram per liter (g/L) sodium cyanide (NaCN) and evaluated the impact of pre-aeration and dissolved oxygen concentration on gold extraction and leach kinetics. The results of these tests are summarized in Table 13-4. These tests showed that without preaeration and air injection to maintain the dissolved oxygen concentration during leaching at 5 to 8 milligrams per liter (mg/L) that 98% of the gold could be extracted after 72 hours of leaching with sodium cyanide consumption reported at 1.83 kilograms per tonne (kg/t). However, with inclusion of preaeration for two hours and oxygen injection sufficient to maintain the dissolved oxygen concentration during leaching at 20 mg/L, sodium cyanide consumption was reduced to 0.66 kg/t and leach kinetics were significantly increased with gold leaching complete after 24 to 48 hours.

Table 13-4: Whole-Ore Cyanidation Test Results on MDZ Test Composite

CN Test No.	Feed Size P_{80}, µm	Aeration Conditions		Reagent Cons. kg/t of CN Feed		Au Extraction (%)				Au Residue (g/t)			Au Head (g/t)	
		Pre-air	Leach	NaCN	CaO	8 h	24 h	48 h	72 h	A	B	Avg.	Calc.	Direct
5	56	n/a	Air, ~5-8 ppm	1.83	1.52	40.0	70.4	88.7	98.3	0.04	0.05	0.05	2.65	2.32
9	61	2 h O_2	O_2, ~20 ppm	0.66	1.29	93.5	95.6	~99	98.2	0.04	0.04	0.04	2.26	

Source: SGS, 2019

13.5.2 Gravity Concentration

Gravity concentration tests were conducted on the MDZ, West Zone, Center Zone, East Zone and Marmato composites at a grind sizes ranging from P_{80} 70 to 223 µm with a Knelson MD-3 centrifugal gravity concentrator followed by upgrading on a Mozley table. The results of these tests are shown in Table 13-5. This testwork demonstrated that the MDZ material (including West, Center and East Zone composites) is highly amenable to gravity concentration with gold recoveries ranging from 50.6 to 69.0% and silver recoveries ranging from 14.6 to 24.7% into gravity concentrates containing 0.05 to 0.16 weight percent (wt%) and 1,575 to 2,494 g/t Au and 182 to 1,432 g/t Ag. The Marmato test composite was also very responsive to gravity concentration with Au recoveries ranging from 56.9 to 57.9% and Ag recoveries of 11.6% into gravity concentrates containing 0.13 to 0.17 wt% and 1,586 to 2,192 g/t Au and 1,278 to 1,668 g/t Ag.

Table 13-5: Summary of Gravity Concentration Testwork on MDZ and Marmato Composites [(1)]

Test No.	Composite	Grind Size	Assay Head		Calc. Head		Gravity Concentrate			Distribution (%)	
		P_{80} µm	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Mass (%)	Au (g/t	Ag (g/t)	Au	Ag
G-1	MDZ	223	2.32	4.2	2.31	3.9	0.07	1,815	1,006	55.1	17.9
G-3	MDZ	223	2.32	4.2	2.27	4.5	0.05	2,494	1,432	50.6	14.6
G-9	MDZ	112	2.32	4.2	2.51	4.0	0.10	1,575	921	63.7	23.6
G-6	West Zone	88	1.54	0.9	1.30	1.2	0.16	531	182	66.1	24.7
G-7	Center Zone	94	2.69	3.9	2.61	4.3	0.12	1,498	752	69.0	21.0
G-8	East Zone	99	2.65	6.7	1.80	8.1	0.11	831	1,149	51.7	15.9
G-4	Marmato	70	5.48	19.6	4.67	18.8	0.17	1,586	1,278	57.9	11.6
G4R	Marmato	78	5.48	19.6	4.98	18.5	0.13	2,192	1,668	56.9	11.6

Note: [1] Marmato test G-2 not shown due to high variance between assay and calculated head
Source: SGS Lakefield, 2019

13.5.3 Cyanidation of Gravity Tailing

MDZ Master Composite

Cyanidation tests were conducted on the gravity tailings from the MDZ composite. The leach conditions are shown in Table 13-6 and the test results are summarized in Table 13-7. Tests were conducted over a range of grind sizes and cyanide concentrations, both with and without preaeration and oxygen injection. These tests demonstrated that overall gold extractions (gravity concentration + gravity tailing cyanidation) of about 97 to 98% could be achieved. A grind size of about P_{80} 100 µm appeared optimum with a cyanide concentration of 0.5 g/L NaCN. Cyanide consumption under these conditions was very low at about 0.15 kg/t NaCN. It should be noted that the tests conducted at an initial cyanide concentration of 0.5 g/L (CN-22 an CN-23) allowed the initial cyanide concentration to attenuate throughout the test with no additional cyanide added. The results were very similar to the other tests and indicated that maintaining the cyanide concentration at 0.5 g/L throughout the leach cycle was not required for the MDZ gravity tailing. Tests CN-12, 13, and 14 were designed to evaluate grind size over the range from P_{80} 138 to 76 µm. The residue gold grades for these tests were 0.08 g/t, 0.06 g/t, and 0.06 g/t indicating an optimum grind size of about P_{80} 100 µm. The leach kinetics for all tests are shown in Figure 13-2 and indicate a leach retention time of about 30 hours is sufficient to maximize gold extraction. Preaeration appears to be beneficial in reducing cyanide consumption, but the need for lead nitrate requires further evaluation. Additionally, the need for oxygen injection requires further evaluation.

Table 13-6: MDZ Gravity Tailing Leach Conditions

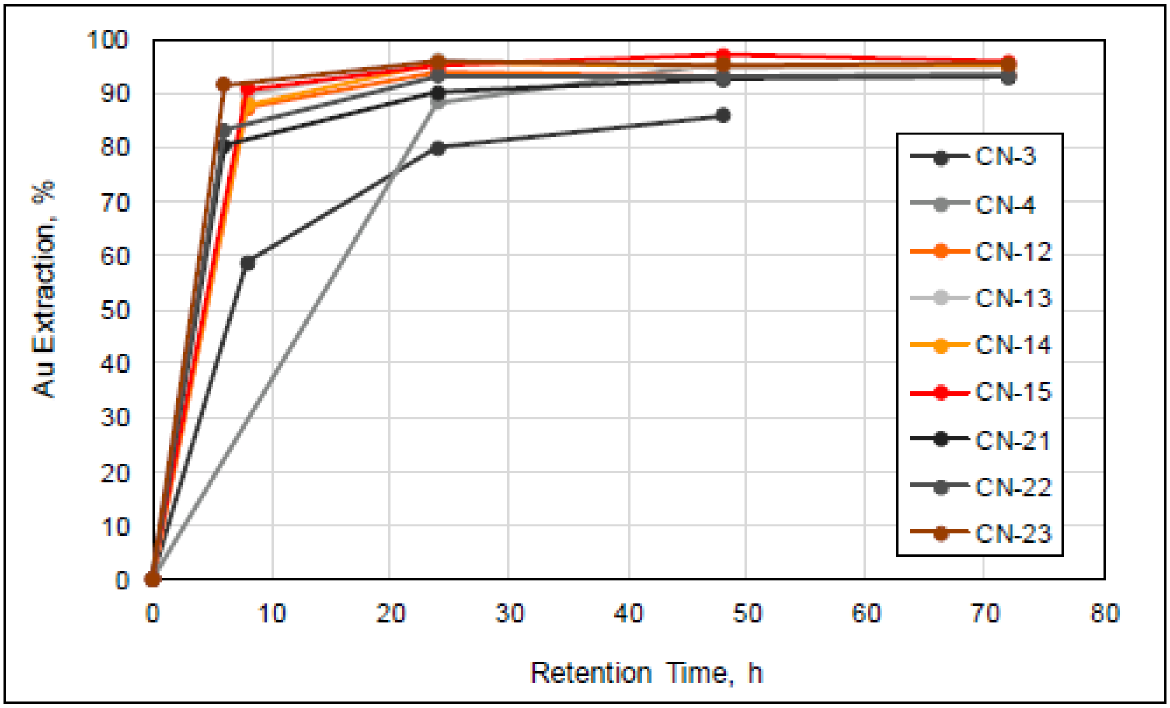
CN Test No.	Feed	Feed Size P_{80}, µm	Aeration Conditions		Lead Nitrate 100 g/t	Cyanide Concentration g/L NaCN	Reagent Addition kg/t of CN Feed		Reagent Cons. kg/t of CN Feed	
			Pre-air	Leach			NaCN	CaO	NaCN	CaO
3	G-1	223	n/a	Air, ~5-8 ppm	N	1.0	1.65	1.09	0.83	1.09
4	G-1	70	n/a	Air, ~5-8 ppm	N	1.0	2.27	1.09	1.55	1.09
12	G-3	138	4 h, O_2	O_2, ~20 ppm	N	1.0	1.23	1.22	0.24	1.18
13	G-3	96	4 h, O_2	O_2, ~20 ppm	N	1.0	1.24	1.27	0.22	1.23
14	G-3	76	4 h, O_2	O_2, ~20 ppm	N	1.0	1.24	1.32	0.25	1.30
15	G-3	80	2 h, O_2	O_2, ~20 ppm	Y	1.0	1.24	1.20	0.28	1.17
21	G-3	99	4 h, O_2	O_2, ~20 ppm	N	0.75	0.75	1.10	0.19	1.09
22	G-3	98	4 h, O_2	O_2, ~20 ppm	N	0.50	0.50	1.11	0.16	1.08
23	G-3	97	4 h, O_2	O_2, ~20 ppm	Y	0.50	0.50	1.13	0.14	1.11

Source: SGS Lakefield, 2019

Table 13-7: Gravity Concentration + Gravity Tailing Cyanidation Test Results

CN Test No.	Feed	Au Extraction/Recovery, %						Au Residue, g/t			Au Calc. Head g/t
		CN (Unit)				Grav	Grav +CN				
		8 h	24 h	48 h	72 h			A	B	Avg.	
3	G-1	58.6	79.9	85.8		55.1	93.6	0.12	0.18	0.15	1.06
4	G-1		88.2	95.6		55.1	98.0	0.07	0.06	0.07	1.48
12	G-3	87.3	93.8	93.2	93.3	50.6	96.7	0.08	0.08	0.08	1.19
13	G-3	89.8	96.1	94.7	95.1	50.6	97.6	0.06	0.05	0.06	1.13
14	G-3	87.8	95.6	95.2	95.0	50.6	97.5	0.05	0.06	0.06	1.10
15	G-3	90.7	95.1	97.0	95.9	50.6	98.0	0.04	0.05	0.05	1.08
21	G-3	80.3	90.1	92.6	93.1	50.6	96.6	0.08	0.07	0.08	1.08
22	G-3	83.1	93.3	92.9	93.4	50.6	96.7	0.07	0.07	0.07	1.06
23	G-3	91.5	95.9	95.2	95.4	50.6	97.7	0.05	0.06	0.06	1.21

Source SGS Lakefield, 2019



Source: SGS Lakefield, 2019

Figure 13-2: Gold Extraction Versus Retention Time (MDZ Gravity Tailings)

Variability Composites

The individual East, Center and West Zone composites and the Marmato composite were subjected to gravity concentration followed by cyanidation of the gravity tailings using the following test conditions:

- Grind size: ~ P_{80} of 100 µm
- Preaeration: 4 hours with oxygen
- Dissolved O_2: 20 mg/L
- NaCN: 1.0 g/L (maintained)
- Retention Time: 48 hours
- Slurry Density: 50% solids

The results of these tests are summarized in Table 13-8, which show that overall gold recoveries for the West, Center and East Zone composites were very similar to the results obtained from the MDZ composite and ranged from 96.7 to 97.9% with cyanide consumption ranging from 0.19 to 0.34 kg/t. Overall gold recovery from the Marmato composite was about 92% with cyanide consumption at about 0.50 kg/t.

Table 13-8: Summary of Gravity Concentration + Gravity Tailing Cyanidation (Variability Composites)

Gravity Test	Cyanidation Test	Composite	Grind Size	Au Distribution (%)	
			P80 µm	Gravity	Grav. + Cyan.
G-6	CN-18	West Zone	88	66.1	97.3
G-7	CN-19	Center Zone	94	69.0	97.9
G-8	CN-20	East Zone	99	51.7	96.7
G-4	CN-16	Marmato	70	57.9	92.0
G4R	CN-16R	Marmato	78	56.9	91.8

Source: SGS Lakefield, 2019

13.5.4 Flotation from Gravity Tailing

Rougher flotation tests were conducted on gravity tailings from the MDZ and Marmato composites using flotation conditions provided by Gran Colombia. All tests were conducted at natural pH with 20 minutes of retention time and used potassium amyl xanthate (PAX) and MX5160 as the collectors, copper sulfate as a sulfide mineral activator and Dowfroth 250 as the frother. The results of selected tests are shown in Table 13-9. Overall gold recoveries (gravity concentration + rougher flotation) of 96% to 97% were reported for the MDZ composite and 97.4% for the Marmato composite. Rougher flotation concentrate grades produced from the MDZ composites ranged from 10 to about 13 g/t Au. The rougher flotation concentrate produced from the Marmato composite contained 43.6 g/t Au. Although generally high overall gold recoveries were reported, it should be noted that the gold grade of the final flotation tailing produced from the MDZ composite was significantly higher than the cyanidation leach residues (0.09 g/t Au versus 0.06 g/t Au).

Table 13-9: Summary of Rougher Flotation Tests on Gravity Tailings from MDZ and Marmato Composites

Composite	Test	Grind P80 µm	Au Recovery (%)			Au Grade (g/t)	
			Grav	Flot (unit)	Flot + Grav	Flot Conc	Flot Tail
MDZ	G1/F2	74	55.1	93.4	97.0	9.57	0.08
MDZ	G1/F4	63	55.1	91.9	96.4	13.1	0.09
MDZ	G1/F5	63	55.1	90.8	95.9	12.1	0.10
Average		**67**	**55.1**	**92.0**	**96.4**	**11.6**	**0.09**
Marmato	G-2/F3	210	11.5	97.1	97.4	43.6	0.39

Source: SGS Lakefield, 2019

13.5.5 Cyanidation of Flotation Concentrates

Cyanidation tests were conducted on the rougher flotation concentrates that had been reground to about 22 µm. Cyanidation tests were conducted at 1 g/L NaCN for 48 hours and the results are summarized in Table 13-10. These tests demonstrate that about 98% of the gold contained in the flotation concentrates could be extracted by cyanidation. It is important to note that the 98% gold extraction from the flotation concentrate implies an overall gold recovery of about 95% to 96% from a gravity + flotation + cyanidation flowsheet. This is about 2% lower gold recovery than by the gravity + gravity tailing cyanidation flowsheet.

Table 13-10: Summary of Flotation Concentrate Cyanidation Test Results

CN Test No.	Sample	Feed	Au Extraction, % CN (Unit)			Au Residue (g/t)			Au Calc. Head (g/t)
			8 h	24 h	48 h	A	B	Avg.	
1	MDZ	G-1/F-1	95.4	97.1	98.1	0.22	0.20	0.21	11.1
2	MDZ	G-1/F-2	93.3	95.4	98.2	0.17	0.17	0.17	9.57
10	MDZ	G-1/F-4	93.0	95.1	97.2	0.38	0.36	0.37	13.1
11	MDZ	G-1/F-5	94.2	95.9	98.1	0.24	0.23	0.24	12.1
7	Marmato	G-2/F-3	56.6	92.4	98.0	0.87	0.84	0.86	43.6

Source: SGS, 2019

13.6 Cyanide Detoxification

The cyanidation leach residue produced from the MDZ composite under optimized leach conditions was subjected to cyanide detoxification testing using the industry-standard SO_2/Air process to reduce the weak acid dissociable cyanide (CN_{WAD}) to less than 10 mg/L. The main parameters adjusted during the testwork were sodium metabisulphite and copper addition rates. The results of the detoxification testwork are shown in Table 13-11. The initial leach residue contained 150 mg/L CN_T, which was subsequently reduced to 8.95 CN_{WAD} (Test 1-7) with the addition of 8.05 g SO_2/g CN_{WAD} and 0.22 g Cu/g CN_{WAD}. This testwork established that the following operating conditions will achieve a discharge CN_{WAD} concentration of <10 mg/L.

- Slurry density: 50% solids (w/w)
- SO_2 addition: 8 g SO_2 /g CN_{WAD}
- Cu addition: 0.22 g Cu /g CN_{WAD}
- pH: 8.5 (with lime added as needed (~0.5 kg/t)
- Retention time: 90 minutes

It should be noted that aging the detoxified sample for one to five days further attenuated the CN_{WAD} to approximately 2.5 mg/L or less. Additional cyanide detoxification testwork should be completed in the next phase of study to further optimize process requirements.

Table 13-11: Summary of Cyanide Detoxification Testwork on MDZ Composite Leach Residue

Test	Test Duration	Reten. Time	Product (Solution Phase)							Reagent Addition								
				CN_T	CN_{WAD} by			Cu	Fe	g/g CN WAD			g/L Feed Pulp			kg/t Solids		
	Min	Min	pH	mg/L	Ana. Lab mg/L	Picric Acid mg/L	Aged Picric Acid mg/L	mg/L	mg/L	SO_2 Equiv.	Lime	Cu (1)	SO_2 Equiv.	Lime	Cu (1)	SO_2 Equiv.	Lime	Cu (1)
Feed (CN-24)		…	10.6	150	151	…	…	26.6	2.56	…	…	…	…	…	…	…	…	…
Batch Test																		
CND 1	270	270	8.5	…	…	1.12	…	…	…	11.1	9.71	0.070	1.23	1.08	0.007	1.67	1.46	0.011
Continuous Tests																		
1-1	94	60	8.5	…	…	55.6	…	…	…	6.98	0.54	0.000	0.78	0.061	0.000	1.05	0.081	0.000
1-2	90	55	8.7	5.7	<0.1	14.7	1.14*	0.70	1.7	**6.12**	0.11	**0.070**	0.68	0.013	0.007	0.92	0.02	0.011
1-3	90	60	8.5	…	…	50.2	…			**7.95**	4.84	0.070	0.88	0.55	0.007	1.20	0.73	0.011
1-4	60	55	8.5	…	…	30.5	…	…	…	7.31	3.60	**0.130**	0.81	0.41	0.015	1.10	0.54	0.020
1-5	115	59	8.5	2.5	<0.1	9.88	2.51**	4.3	1.2	7.57	2.19	**0.220**	0.88	0.26	0.025	1.14	0.33	0.033
1-6	120	59	8.5	…	…	18.3	…	…	…	7.37	2.38	**0.350**	0.86	0.28	0.041	1.11	0.36	0.053
1-7	100	**89**	8.5	…	…	8.95	…	…	…	**8.05**	3.62	0.220	0.94	0.43	0.026	1.21	0.55	0.033
1-8	170	**91**	8.5	<0.1	<0.1	6.45	…	14.7	<0.1	**10.7**	4.66	0.240	1.19	0.53	0.026	1.61	0.70	0.036

… No sample submitted for assays
(1) Cu added using $CuSO_4$, $5H_2O$, SO_2 added using sodium metabisulphite
* CND1-2 aged five-day sample
** CND1-5 aged one day sample
Bold red values indicate a key parameter that has changed
Source: SGS Lakefield, 2019

13.7 Solid-Liquid Separation

Thickening and rheological studies were conducted on a cyanidation leach residue at a P_{80} 105 µm grind size that had adjusted to pH 8.5 with lime to simulate the detoxified slurry pH.

13.7.1 Flocculant Screening

Flocculant screening tests identified BASF Magnafloc 10, which is a very high molecular weight, slightly anionic polyacrylamide flocculant, as a suitable flocculant for this application at an application rate of 40 g/t. Both static and dynamic thickening testwork were conducted with this flocculant.

13.7.2 Static Thickening

Preliminary static settling tests were performed in two-liter graduated cylinders which were fixed with rotating picket-style rakes. Static settling test results were used to determine the starting conditions for subsequent dynamic thickening tests. The selected conditions based on these tests are summarized in Table 13-12, and indicated a thickener settling area of 0.11 square meter per tonne per day (m^2/[t/day]) with an underflow density of 62% solids and an overflow containing 61 mg/L of total suspended solids (TSS).

Table 13-12: Static Thickener Test Conditions

Sample ID	Flocculant Dose (g/t)	Feed %w/w	U/F %w/w	Unit Area m^2/(t/day)	ISR m^3/m^2/day	Supernatant Clarity	TSS mg/L
MDZ Comp	40	8.0	62	0.11	833	Hazy	61

Source: SGS Lakefield, 2019

13.7.3 Dynamic Thickening

Dynamic thickening testwork was initiated with a 50 g/t dosage of BASF Magnafloc 10 flocculant at a feedwell slurry density of 8%w/w solids. The dynamic thickening test responded very differently to the static thickening test under these conditions with a very turbid overflow with TSS measured at 450 mg/L. In order to improve the overflow clarity, BASF Magnafloc 1687 coagulant was applied to the diluted thickener feed prior to flocculant dosing. A series of additional tests established a dosage of Magnafloc 1687 at 15 g/t followed by a dosage of 25 g/t of Magnafloc 10 as optimal. The result of dynamic thickener tests conducted over a range of unit settling areas (m^2/[t/d]) are summarized in Table 13-13.

Table 13-13: Summary of Dynamic Thickener Test Results

1687 Dosage (g/t)	10 Dosage (g/t)	Unit Area m^2/(t/d)	Solids Loading (t/m^2/h)	Net Rise Rate (m^3/m^2/d)	Underflow %w/w solids	Overflow TSS (mg/L)	Residence Time (h)	U/F Yield Stress (Pa)
15	25	0.13	0.32	84.2	64.1	54	1.36	60
15	25	0.11	0.38	99.5	62.9	57	1.15	49
15	25	0.09	0.46	121.7	61.7	43	0.94	52
15	25	0.07	0.60	156.4	59.5	58	0.73	38
Underflow extended for 30 minutes:					65.1			93

Note: Bed height was maintained around 160 mm
Source: SGS Lakefield, 2019

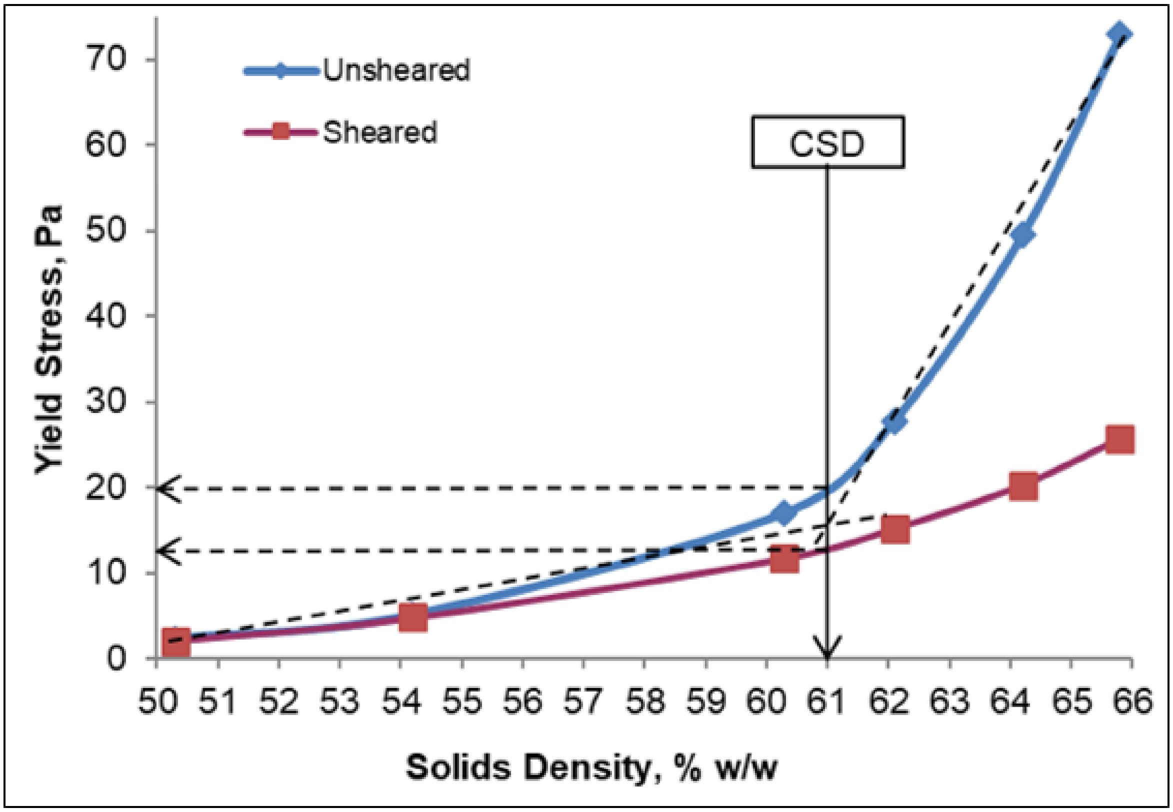
13.7.4 Rheology on Thickener Underflow

The results of rheology testwork on the thickener underflow are summarized in Table 13-14. A Critical Solids Density (CSD) of approximately 61% solids was established, which corresponds to approximately 20 pascals (Pa) on the unsheared yield test and 13 Pa on the sheared yield test. As shown in Figure 13-3, CSD is the solids density at which a small increase of the solids density causes a significant decrease of the flowability of the slurry. The CSD value is also predictive of the maximum underflow solids density achievable in a commercial thickener.

Table 13-14: Results of Rheology Testwork on MDZ Thickener Underflow Sample

Test Code	Solids %w/w	Unsheared Sample			Sheared Sample			Observations
		γ	$\tau_y B$	η_P	γ	$\tau_y B$	η_P	
		Range, 1/s	Pa	mPa.s	Range, 1/s	Pa	mPa.s	
CSD= ~61% solids, corresponding to ~20 Pa unsheared and 13 Pa sheared yield stress.								
T1	65.8	Plug Flow	73	--	200-400	26	50	Thixotropic
T2	64.2	200-400	50	3.9	200-400	20	36	Thixotropic
T3	62.1	200-400	28	15	200-400	15	24	Thixotropic
T4	60.3	200-400	17	16	200-400	12	17	Thixotropic
T5	58.3	200-400	12	15	200-400	8.9	15	Minor Thixotropic, minor settling
T6	54.2	200-400	5.1	13	200-400	4.8	13	Minor settling
T7	50.3	200-400	2.3	10	200-400	2.1	15	Moderate settling, dilatant after 425 1/s

Notes: The MDZ Comp underflow samples contained 15 g/t BASF Magnafloc 1687 coagulant and 25 g/t BASF Magnafloc 10 flocculant
The values are based on data produced by the unsheared and sheared slurry sample.
Variable shearing was produced in the 0 to 600 s^{-1} range, increasing and decreasing (up and down curves).
Constant shearing was produced by subjecting the slurry sample to a constant rotation at 300 1/s for 180 seconds.
Bingham Plastic parameters: yield stress ($\tau_y B$) and plastic viscosity (η_P) values, for the specified γ range.
γ – Shear rate range at which the rheological parameters were calculated.
Source: SGS Lakefield, 2019



Source: SGS, 2019

Figure 13-3: Yield Stress Versus Thickener Underflow Slurry Density

13.8 Gold and Silver Recovery Estimate

As shown in Table 13-15, SRK has estimated achievable gold and silver recoveries from the MDZ for each of the process flowsheet options tested based on the reported results from selected tests conducted on the MDZ composite. SRK recommends discounting laboratory-reported gold extractions/recoveries by 2% and silver extractions/recoveries by 5% to account for inherent plant inefficiencies. As such, the following gold and silver recoveries are projected for each of the flowsheet options based on the testwork conducted to-date:

Flowsheet	Gold Recovery	Silver Recovery
• Whole-ore cyanidation	96%	38%
• Gravity + cyanidation	95%	41%
• Gravity + flotation + cyanidation	94%	44%

It should be noted that the whole-ore cyanidation testwork was conducted at a finer grind, and with more aggressive leach conditions than were used in the gravity + cyanidation testwork. Overall gold recoveries from either the whole-ore cyanidation or the gravity + cyanidation flowsheets operated under similar conditions can be expected to be similar. It should also be noted that no cyanidation tests were conducted to assess gold and silver extractions from the gravity concentrates. Generally, gold extractions from gravity concentrates in plant practice are very high (>98%) and it is assumed that the 2% adjustment to overall gold recovery adequately accounts for gold losses that may occur during intensive leaching of the gravity concentrates. Gold and silver cyanidation extraction from gravity concentrates should be evaluated during the next phase of study.

Table 13-15: Estimated Gold and Silver Recoveries for Flowsheet Options

Flowsheet	Test	Gravity Recovery (%)		Unit Flotation Recovery (%)		Unit Cyan. Extraction (%)		Overall Recovery (%)		Adjusted Overall Recovery (%)	
		Au	**Ag**	**Au**	**Ag**	**Au**	**Ag**	**Au**	**Ag**	**Au**	**Ag**
Whole-Ore Cyanidation	CN-5					98.3	39.8	98.3	39.8	96	35
Whole-Ore Cyanidation	CN-6					98.2	45.2	98.2	45.2	96	40
Average								**98.3**	**42.5**	**96**	**38**
Gravity + Cyanidation	G-3/CN-21	50.6	14.6			93.1	34.6	96.6	44.1	95	39
Gravity + Cyanidation	G-3/CN-22	50.6	14.6			93.4	37.9	96.7	47.0	95	42
Average								**96.7**	**45.6**	**95**	**41**
Gravity + Flot + Cyan	G1/F2	55.1	17.9	93.4	86.8	98.2	44.8	96.3	49.8	94	45
Gravity + Flot + Cyan	G1/F4	55.1	17.9	91.9	83.4	97.2	42.3	95.2	46.9	93	42
Gravity + Flot + Cyan	G1/F5	55.1	17.9	90.8	83.9	98.1	46.8	95.1	50.1	93	45
Average								**95.5**	**48.9**	**94**	**44**
Gold Adjustment Factor	2										
Silver Adjustment Factor	5										

Source: SGS Lakefield and SRK, 2019

13.9 Significant Factors

The following significant metallurgical and mineral processing factors have been identified:

- Metallurgical testwork was conducted on a test composite representing the average material from the MDZ as well as variability composites from East, Central and West Zones of the MDZ deposit;
- Native gold was by far the predominant gold carrier and the majority (>99%) of the gold particles occurred within mineral structures that would be readily accessible by leaching solutions. Gold particles were not often in direct contact with sulfides, yet very commonly pyrrhotite, chalcopyrite, and bismuth minerals were found in close vicinity to the gold mineralization;
- The SAG mill comminution (SMC) tests were conducted on the East, West and Central MDZ and the reported Axb values ranged from 28 to 31 and averaged 29, indicating that the material is very hard with respect to SAG mill impact grinding. Bond ball mill work indices conducted on the MDZ composites ranged from 19.0 to 20.7 kWh/t indicating that the MDZ material is very hard with respect to ball mill grinding. The MDZ material is much harder than the current Marmato material which was reported to have a BWi of 15.7 kWh/t;
- Metallurgical testwork was conducted to evaluate three different flowsheet options that included:
  - Whole-ore cyanidation;
  - Gravity concentration followed by cyanidation of the gravity tailing; and
  - Gravity concentration followed by flotation and cyanidation of the flotation concentrate.
- The MDZ material responded very well to each of the process flowsheet options tested with gold recoveries estimated at 94 to 96% and silver recoveries estimated at 38 to 44%;
- Cyanidation leach residues can be detoxified to < 10 mg/L CN_{WAD} using the industry-standard SO_2/Air process; and
- Additional testwork should be conducted during the next phase of study to fully define the process design criteria and operating parameters.

14 Mineral Resource Estimate

The Mineral Resource Statement presented herein represents the latest mineral resource evaluation prepared for the Marmato Project reported in accordance with the standard adopted for the reporting of Mineral Resources of the CIM guidelines, and with NI 43-101 disclosure standards.

SRK has been supplied with electronic databases covering the sampling at the Project, all of which have been validated by the Company. The databases comprise of a combination of historical and recent diamond core and underground channel samples. In total, there are 1,165 diamond drillholes for a combined length of 240,855 m; plus 13,489 individual underground channel samples, inclusive of current mine sampling contained in the databases.

The resource estimation was completed by Ben Parsons, MSc, MAusIMM (CP), Membership Number 222568, an appropriate “independent qualified person” as this term is defined in National Instrument 43-101. The Effective Date of the resource statement is July 31, 2019.

The database used to estimate the Marmato Project mineral resources was audited by SRK. SRK is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for gold and silver mineralization and that the assay data are sufficiently reliable to support mineral resource estimation.

Leapfrog® (version 4.5.1) was used to generate the geological and mineralization models used to define the 2017 Marmato model. Datamine™ Studio RM (version 1.5.62.0) was used to construct the geological solids, prepare assay data for geostatistical analysis, construct the block model, estimate metal grades and tabulate mineral resources. Snowden Supervisor software (version 8.7.1) was used for the statistical/geostatistical analysis and variography.

The estimation methodology involved the following procedures:

- Database compilation and verification;
- Construction of wireframe models for the boundaries of the veins;
- Construction of wireframe models for the boundaries of the main other domains including, fault network, mineralized porphyry, low-grade porphyry, deeps/feeder structures;
- Definition of resource domains;
- Data conditioning (compositing and capping) for statistical analysis, geostatistical analysis, and variography;
- Block grade interpolation;
- Resource classification and validation;
- Assessment of “reasonable prospects for economic extraction” and selection of appropriate reporting CoGs; and
- Preparation of the Mineral Resource Statement.

14.1 Drillhole Database

SRK was supplied with ASCII files (.csv) containing the latest drilling and sampling information, in the form of collar, survey, lithology and assay files. The database provided included all sampling from the combined drilling database and channel sampling programs. A summary of the database used in the final estimate is detailed in Table 14-1.

SRK reviewed the database and imported into both Leapfrog® and Datamine™ RM which have validation steps included as part of the importing routine. These include reviewing absent collar or survey information, plus overlapping sampling intervals, conflicting data logging and absent data in intervals.

SRK is satisfied with the quality of the database for use in the construction of the geological block model and associated Mineral Resource Estimate

Table 14-1: Summary of Geological Database Information Available by Sample Type and Company

Summary of Channel Sampling Database by Company						
		Count	**Minimum Length (m)**	**Maximum Length (m)**	**Average Length (m)**	**Sum Length (m)**
Company	CGD	165	0.0	14.0	1.0	168
	CMdC	918	0.0	58.2	3.0	2,764
	CNQ	39	1.3	154.8	22.3	871
	MAdO	308	1.3	102.2	9.6	2,962
	MNL	24,824	0.0	122.2	1.4	35,115
Channel Subtotal		**26,254**	**0.0**	**154.8**	**1.6**	**41,881**
Summary of Drilling Sampling Database by Company						
		Count	**Minimum Length (m)**	**Maximum Length (m)**	**Average Length (m)**	**Sum Length (m)**
Company	CGD	20	50.9	559.6	296.9	5,937
	CGD-GCL	75	16.8	527.4	149.2	11,187
	CMdC	205	1.2	587.3	226.9	46,519
	CNQ	47	39.2	600.2	316.5	14,874
	CNQ-MNL	25	14.0	180.0	72.1	1,803
	CNQ-PDG	6	70.6	175.0	116.0	696
	GCM	35	127.2	705.5	459.3	16,076
	MAdO	342	12.8	1012.1	356.1	121,790
	MNL	562	4.0	400.4	84.5	47,509
Drillhole Subtotal		**1,317**	**1.2**	**1012.1**	**202.3**	**266,390**
Grand Total		**27,571**	**0.0**	**1012.1**	**11.2**	**308,271**

Source: SRK, 2019

SRK notes, in the tables presented, three additional holes (MT-IU-032, MT-IU-033, MT-IU-034), were included but did not have complete datasets and therefore were excluded from the final PEA model, and will be included in the next update.

14.2 Geologic Model

14.2.1 Topographic Wireframes

GCM commissioned a detailed topographic map with 2 m contour intervals derived from Ikonos satellite imagery which was received in early 2007. The new topographic map provides a detailed base map for improved accuracy when plotting the results of the exploration programs, as well as a high-resolution satellite image. The topography was converted to a solid model in Vulcan™ to limit the grade estimate to the surface. This model has been supplied to SRK by the Company.

14.2.2 Lithological Wireframes

As part of the updated Mineral Resource, GCM and SRK initially focused on the creation of a more complete geological model (i.e., one encompassing the major geological features inclusive of the current veins being mined). The main geological units and entities modelled for the resource were:

- Major Fault Network;
- Meta Schist;
- Porphyry (P1 – P4);
- Vein; and
- MDZ.

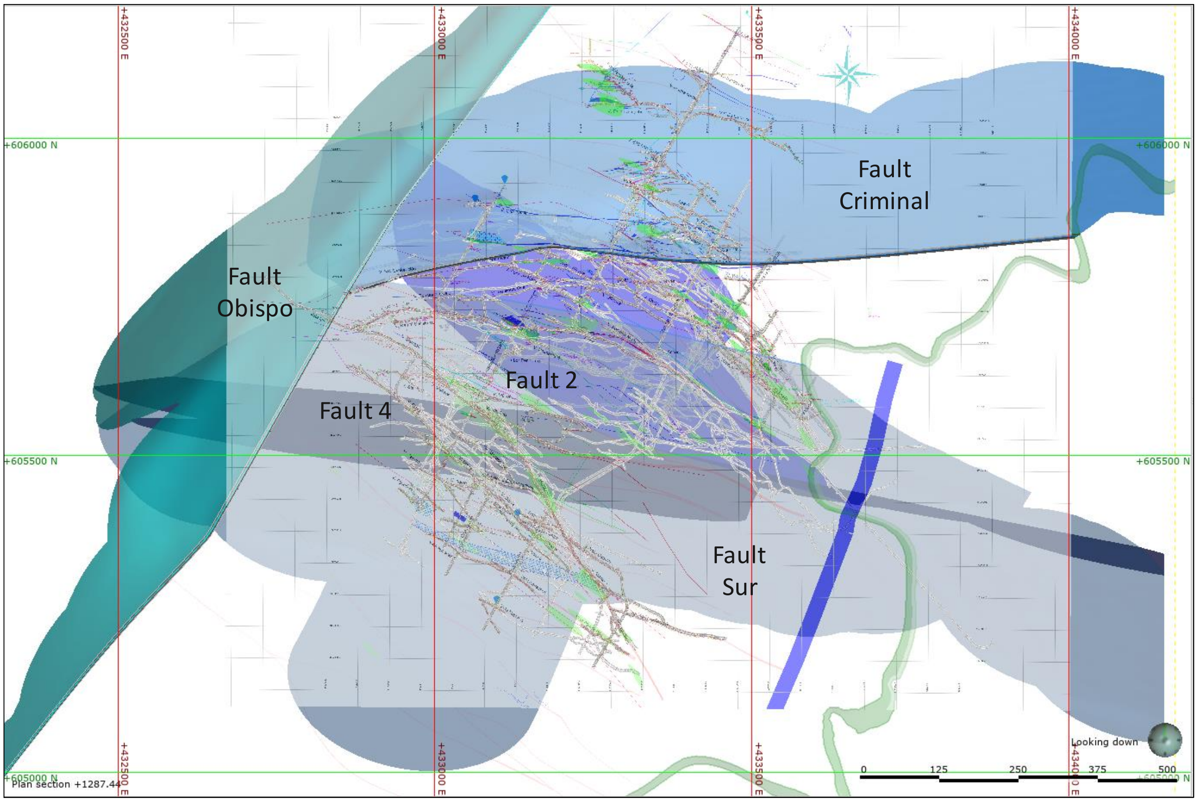
Drilling during 2011/2012, along with check logging of the drill core by the Company, SRK has confirmed the presence of a high-grade feeder zone at the core of the main mineralization (MDZ), which SRK initially modelled based on interpretations provided by the GCM geologists.

14.2.3 Fault Network

GCM geological staff undertook a structural interpretation for the deposit using logged faults and breccia in drill core, underground mapping and surface traces from the digital topography. A series of fault wireframes were provided to SRK, which were generally localized around areas of structural data. SRK has extruded these wireframes to surface using regional mapping and underground development as a guide, and subsequently clipped these to the topography to create a fault network which has been used to review the vein interpretations during the geological modelling (Figure 14-1). Where possible, SRK has attempted to tie the fault interpretations into the latest geological logging.

The final fault network consists of five faults used in the geological model are named:

- Fault Obispo;
- Fault Criminal;
- Fault Sur;
- Fault 2; and
- Fault 4.



Source: SRK, 2019

Figure 14-1: Summary of Fault Network Plan as Used in the Marmato Geological Model

14.2.4 Vein Models

The vein models have been updated using a combination of the drillhole information, channel sampling, geological mapping and the initial depletion shapes provided by GCM. The process has been a collaboration between SRK and GCM to ensure accuracy to the geological conditions mapped underground are considered. To complete the process GCM geologists supplied SRK with an updated underground channel database based on the procedures discussed in Section 11.2 of this report. Additionally, GCM created an initial stope model based on the average dip and strike taken from the underground long-sections produced by the mine.

SRK imported all the available information into Leapfrog® to aid in the generation of the vein model. The vein model was created via a staged process. SRK has made a number of modifications to the geological modelling process in the 2019 update. The changes included combining the vein and halos domains used during the 2017 process. The decision to combine these domains was based on the mining methods typically taking both and therefore a single domain will account for some of the edge dilution, although not all due to minimum mining widths. Also, the incremental gain from the model was not deemed to add value, and therefore a single pass combining both mineralization domains was preferred.

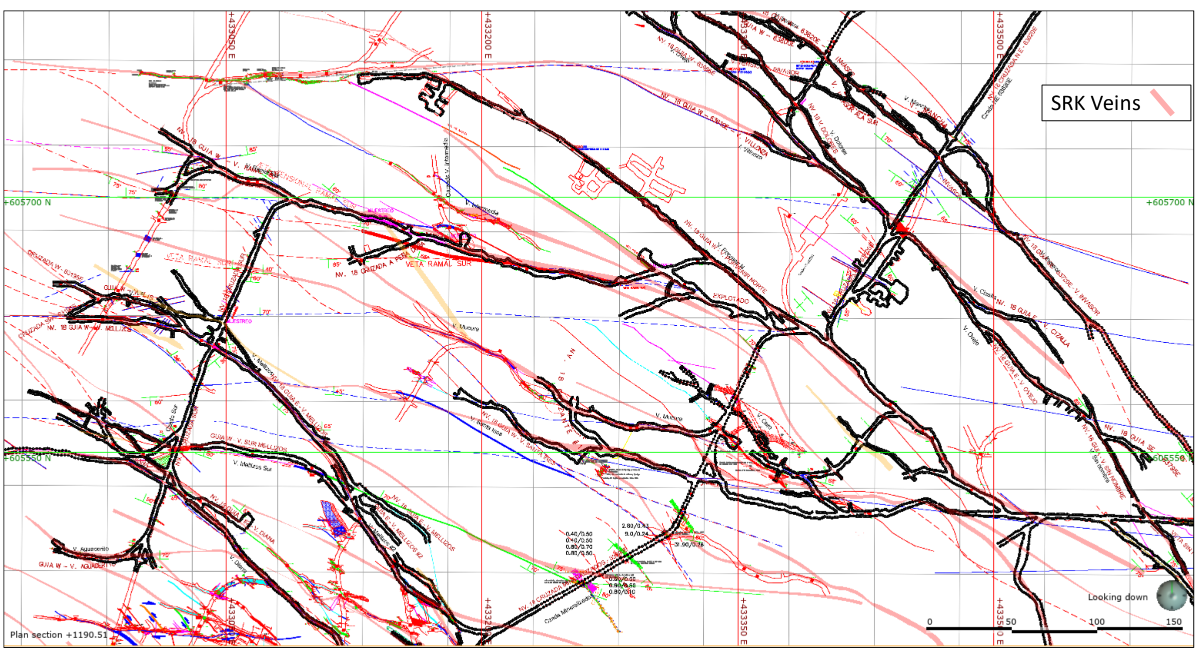
The position of the vein within the channel sampling was based on a combination of lithology logging, gold grade, geological mapping and (base of) stope wireframe data, with interpretation for the vein in less well-informed areas guided by visually evident step changes in the gold grade and characteristics of the vein (i.e., position, thickness and grade continuity) shown in adjacent samples.

The additional channel (VEN) samples have been sub-divided per GCM 'VEIN' code in to three types:

- Vein samples that influence the wireframe. These were deemed by SRK to be visually spatially positioned correctly with respect to geological mapping, mining development/stope data and surrounding drillhole assays and logging and have the following 'Veins_F' code: 'V_XXX';
- Vein samples that do not influence the wireframe, but have a code corresponding to the relevant structure for use in statistics and grade estimation. These are visually spatially offset with respect to other geological data (which if incorporated would result in 'pull points' to the vein wireframes and potentially overstate the tonnage) and have the following 'Veins_F' code: 'V_XXX_STM'; and
- Vein samples that were provided by GCM with a structure code, but are (based on visual review) more likely to be located in discontinuous splay veins in the HW or FW of the modelled vein. These are left unassigned in the background 'Veins_F' coding, and have been considered for the SPLAY domain.

The process was initially completed on the areas with strong geological control in the mine and then expanded into the upper and lower levels of the deposit which are predominantly supported by drilling information only.

SRK has validated the vein model using a series of level plans from the current mining operation for Levels 16 – 20, and reviewed by GCM geologists for approval (Figure 14-2).

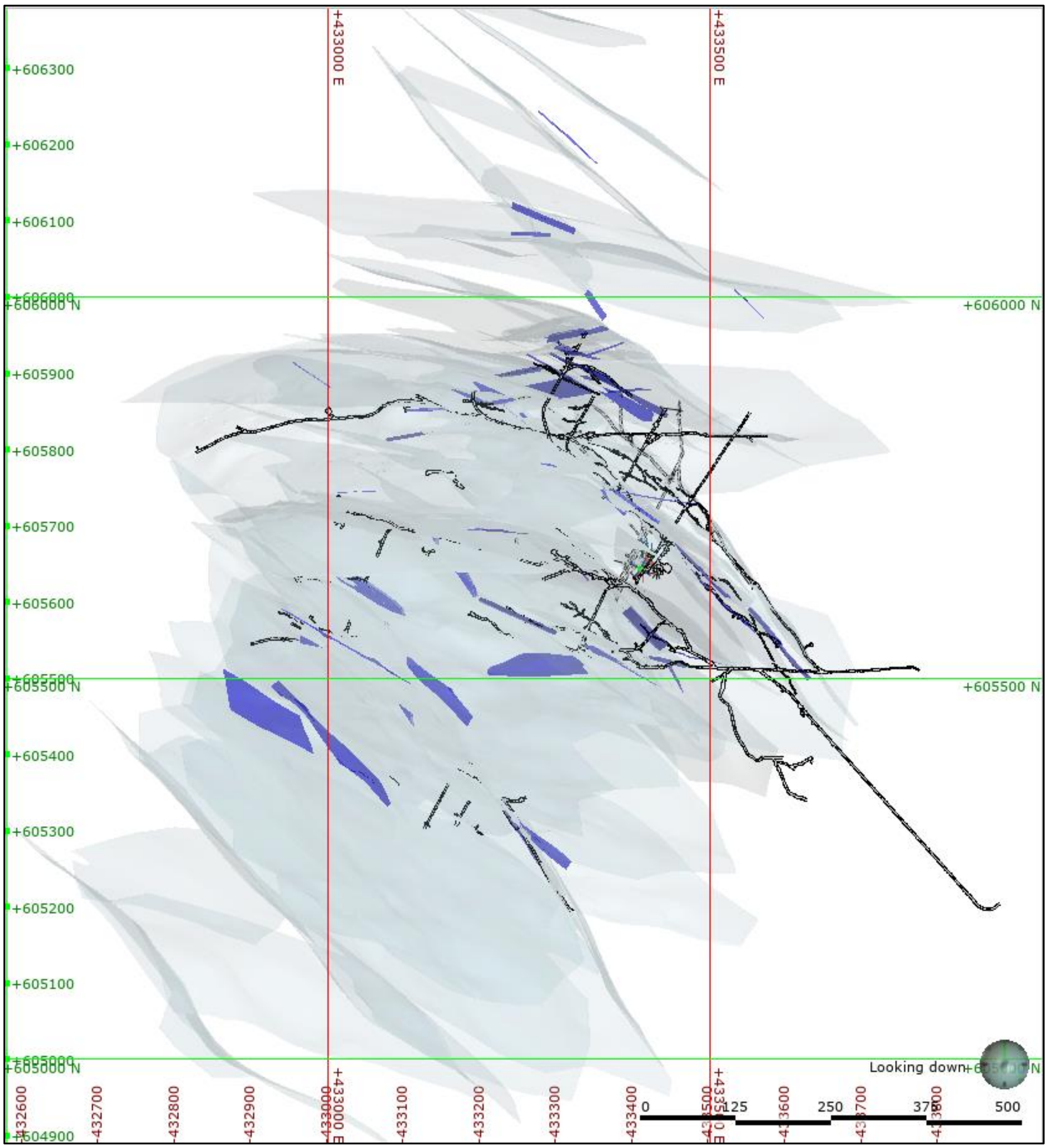


Source: SRK, 2019

Figure 14-2: Level Plan Showing SRK Validation Versus Mining Geological Mapping

14.2.5 Splay Veins

As discussed in Section 12, there are a number of intersections left after the definition of the veins which still have the logging code for "VEN". SRK has reviewed these samples along with the geological mapping to identify a number of small splays of the main structures. SRK identified a total of 52 structures which show some degree of geological continuity to be able to define wireframes. SRK considers the splays to have lower geological confidence to the main veins and further sampling will be required to confirm potential prior to mining. The wireframe has been created using the vein tools in Leapfrog® which the boundaries to limit the model defined using polylines by SRK to crop the veins as appropriate to the main structures (Figure 14-3).



Source: SRK, 2019

Figure 14-3: Plan Showing Location of Splays (Blue) vs Veins Interpretation (Transparent)

14.2.6 Porphyry Model

The modelling of the porphyry mineralization within the pyrite zone has also undertaken a number of changes from the previous SRK model. The key changes have been to sub-divide the domain into a number of major fault blocks, which have been modelled independently. An Indicator shell has been created in using a 0.5 g/t Au CoG, and an iso-value of 0.45 using 2 m composites. Due to the issue of the increase in the channel sampling database, with many short channel samples located outside of the vein models, a restriction has been placed on the samples used during the process. The restriction has removed any samples logged with a vein or splay code, plus any samples which have a geological logging code of “VEN, VNA, VNS or VOI”, or have a channel sampling length of less than 3 m (typical maximum width of mining). SRK has worked under the assumption that the porphyry mineralization will have some relationship to the orientation of the epithermal vein systems which cross-cut the porphyry, and that any veinlets would likely be structurally controlled. To apply this condition, SRK has used a structural trend within Leapfrog when generating the Indicator Grade Models. In order to generate the structural trends, SRK identified key veins and orientations so not to overload the process with the complete library of structural features. Structural trends are then based on the orientations of these veins and structural orientations and are applied to new interpolants to force anisotropy along these trends.

The resultant wireframes have shown a significant reduction in volume. SRK considers the revised model more representative and a reasonable interpretation of the geology.

14.2.7 MDZ

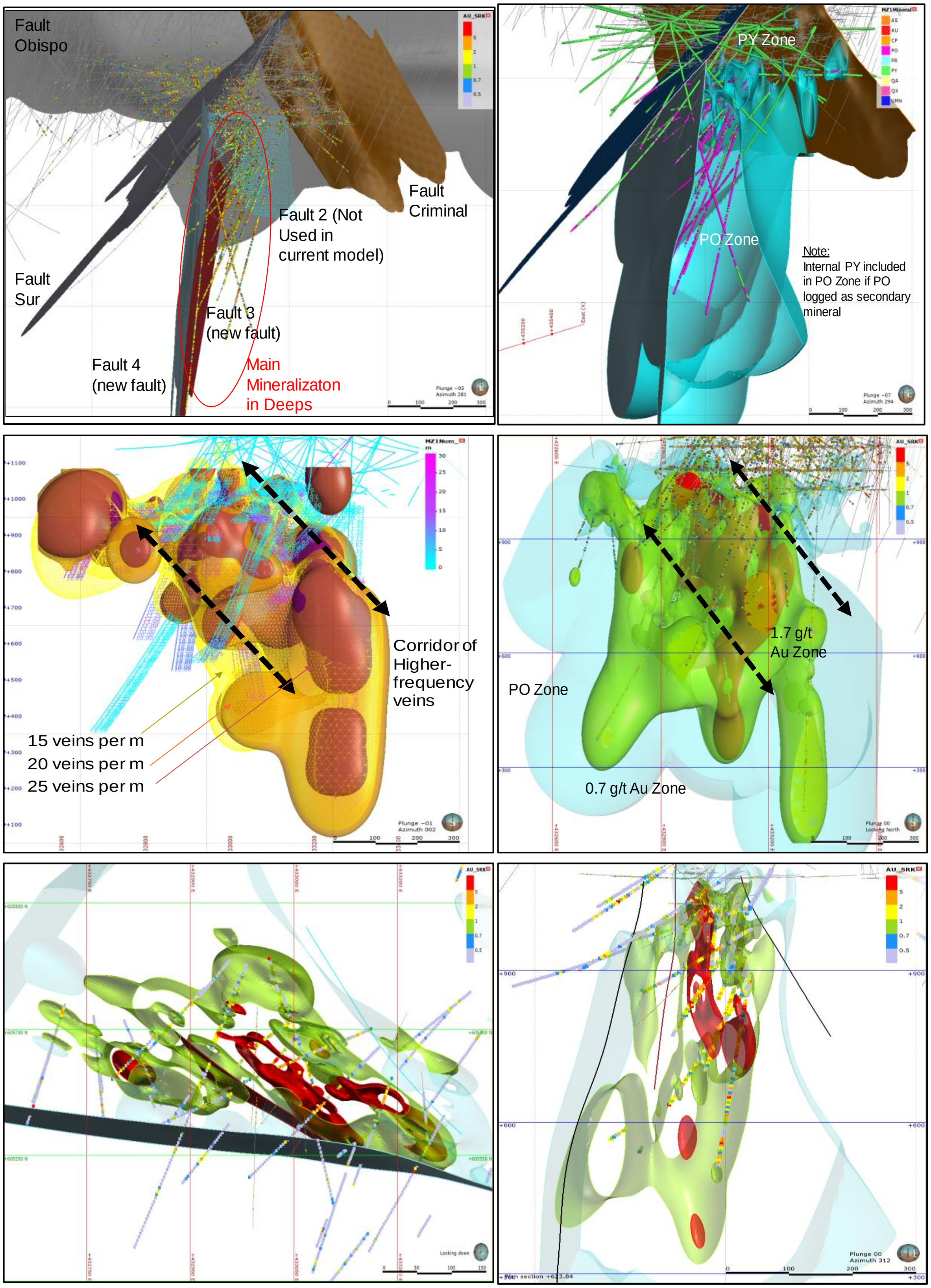
In the 2017 model, the MDZ was modelled using a broad outline and an approximate cut-off of 0.7 g/t Au, but limited geological knowledge restricted the ability produce any sub-domains. In the 2019 models, it is known that the high-grades are typically associated with areas of increase veinlet density as highlighted by Stillitoe (2019). SRK initially attempted to model the geological continuity of the areas logged with higher frequency of veinlets, but due to inconsistency in the logging of the historical holes abandoned this approach for the current model. SRK has provided GCM with a list of holes which will require relogging to enable further investigation prior to the PFS update. A comparison of the vein frequency and indicator grade shells showed a strong correlation in the areas of new drilling between core with a frequency of greater than 20 veinlets per m and a grade shell of 1.7 g/t Au. SRK has therefore used grade as a pseudo-geological control to produce a higher-grade core within the MDZ. A summary of the process is shown in Figure 14-4.

To create the Indicator model the following assumptions have been used:

- Only drillholes have been used to define the domain to remove potential errors or overstating tonnage related to isolated short channel sampling;
- All holes have been composited to 5.0 m, with samples lengths less than 0.5 m at the end of holes appended to the previous sample;
- CoG of 0.7 and 1.7 g/t were used to define the outer limit of the mineralization, which represents 13.1% of the database being assigned and indicator value of 1, with all other values assigned an indicator of 0;
- The general trend for the search ellipse has been orientated to a Dip 85º, to a Dip Azimuth of 195º, with the ratio set for the ellipse of 5 by 3 by 1.5 in the maximum, intermediate and minimum directions; and

- A spherical model has been used using a sill of 0.2 and a nugget of 0.03, using a base range of 150 m for the interpolant.

An ISO (probability) value of 0.40, with a grid resolution of 5 m has been used to define the wireframes. Upon review SRK was concerned of potential blow-outs within the 0.7 g/t indicator models at the edge of the model where limited data exists. SRK has utilized the use of control lines and minimum distances to restrict the high-grades from artificially increasing the tonnage. These areas are also reviewed during the classification to limit the chances of over estimation by applying a limit to the Inferred boundary.



Source: SRK, 2019

Figure 14-4: Summary of the Process Used to Define the MDZ Model in Leapfrog

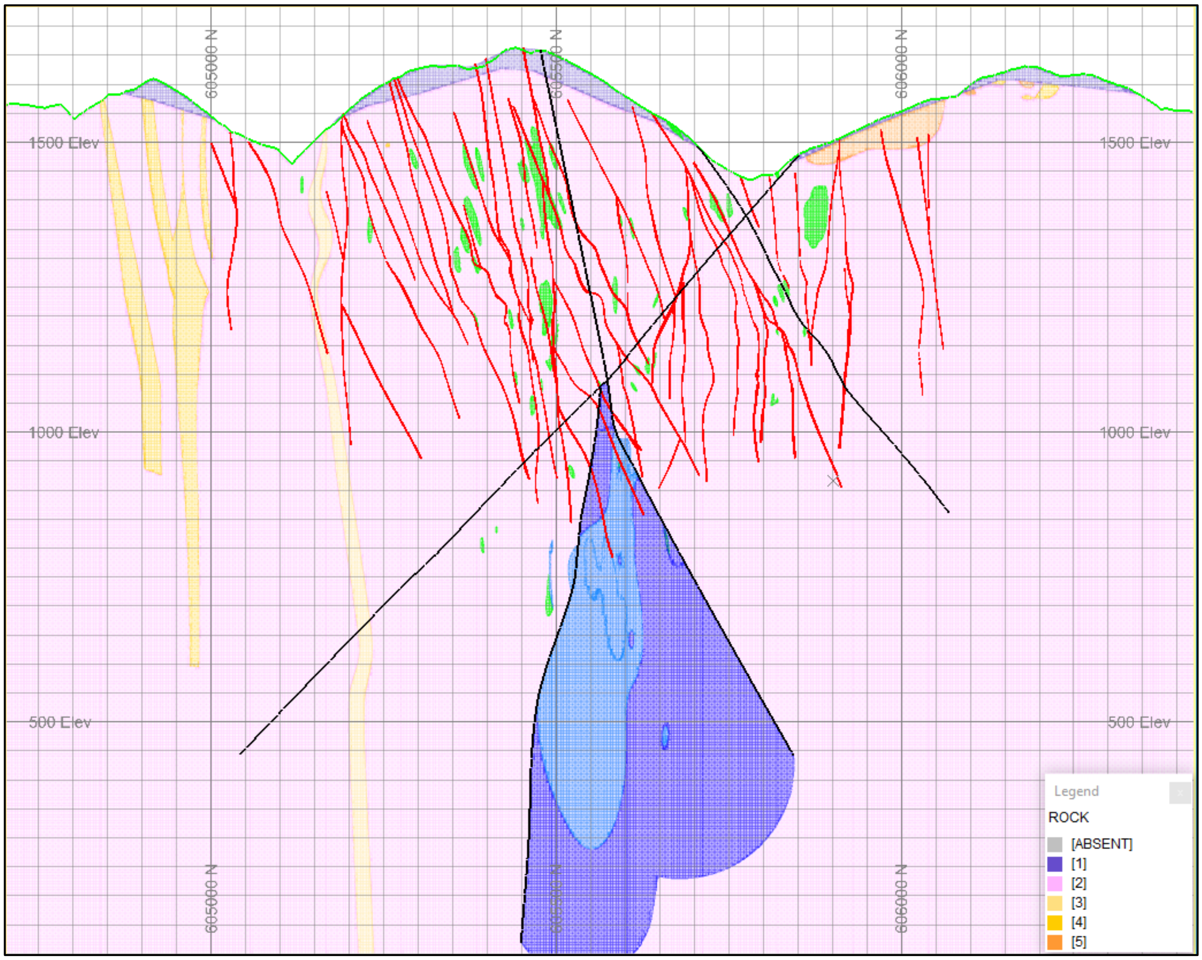
14.3 Domains

All geological surfaces were cut to the topography, and the final geological model has been reviewed by the Company for approval and has been deemed acceptable by SRK for use in determining the Mineral Resource Estimate. Using the wireframes, SRK has coded the drilling and block model information into five domains which are stored in the block model under the field "GROUP" and "KZONE" to distinguish between mineralization style and individual mineralized structure. A list of the domains used is shown in Table 14-2 and in cross section in Figure 14-5.

Table 14-2: Summary of Domain Coding Used in the 2017 Mineral Resource Estimate

Group	No Subdomains	Wireframe	Domain	Description
1000	75	Group1000.dxf	Vein	High grade sulfide veins
2000	0	Group2000.dxf	Halo	Not used in the current model and combined with veins in areas of mining to reflect current mining procedures
3000	59	Group3000.dxf	Splays	Splays of main structure within limited continuity
4000	5	Group4000.dxf-	Grade Shell	Mineralized porphyry material (contained within veinlet), characterized by a mixed population of higher grade above an elevation of 850 m, low grade and barren material, marks the default unit for all material, split by fault domain
5000	3	Group5000.dxf	Deeps	High grade core or feeder zone to the main mineralization. Located at depth within the porphyry system with limited veinlet mineralization, split into low, medium (0.7 g/t) and high (1.7 g/t).

Source: SRK, 2019



Source: SRK, 2019

Figure 14-5: Cross-Section Showing SRK Domained Model

14.4 Assay Capping and Compositing

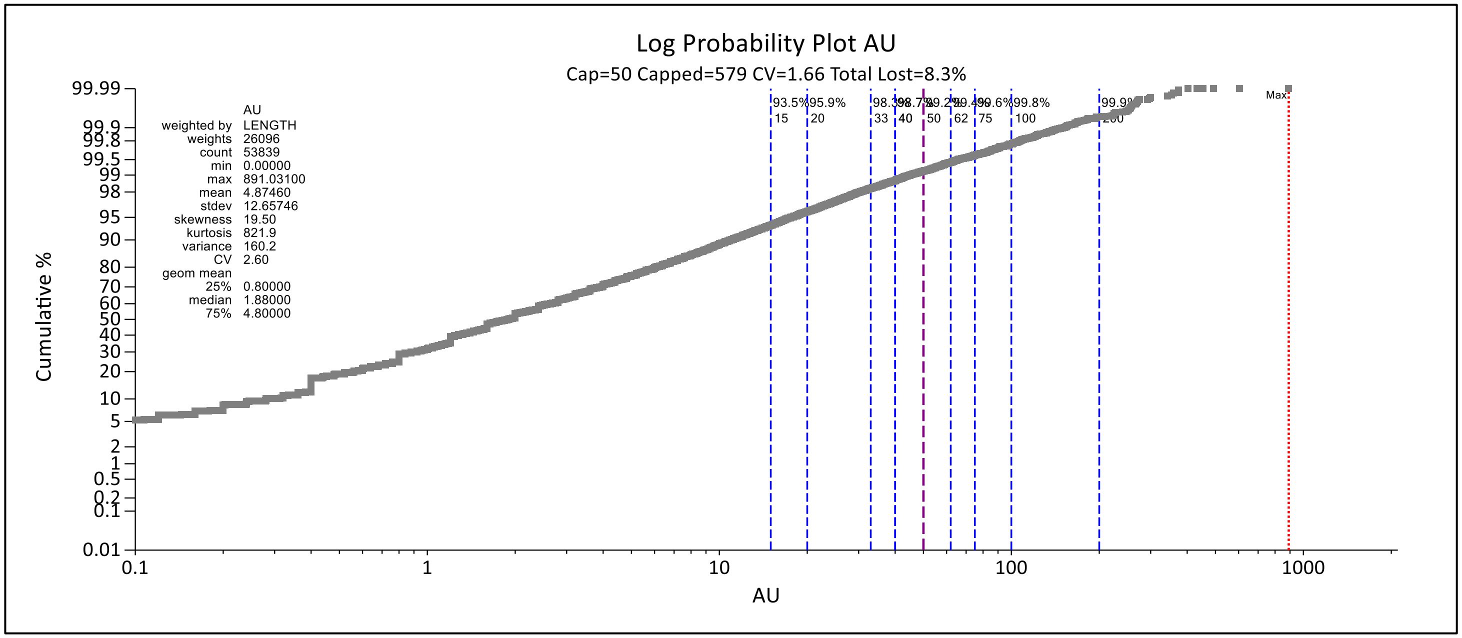
14.4.1 Outliers

High grade capping is typically undertaken where data is no longer considered to be part of the main population. Useful discussions on the need for, and application of capping of high grades are found in Leuangthong and Nowak (2015). Capping is an appropriate technique for dealing with high-grade outlier values, given that appropriate analysis is undertaken to validate the results of the implementation of capping. The following procedure is recommended for treating outliers during resource estimation:

- Determine data validity. Are the data free of sampling, handling, measurement and transfer errors;
- Review geology logs for samples with high-grade assays. Capping may not be necessary for assays where the logs clearly explain the presence of high-grade;
- Capping should not be considered for deleterious substances that have negative impacts on project economics;

- Decide if capping should be considered before or after compositing;
- Keep capping to a necessary minimum. If high-grade assays unduly affect overall grade average, cap them; and
- Restrict influence of very high-grade assays during the estimation process if required.

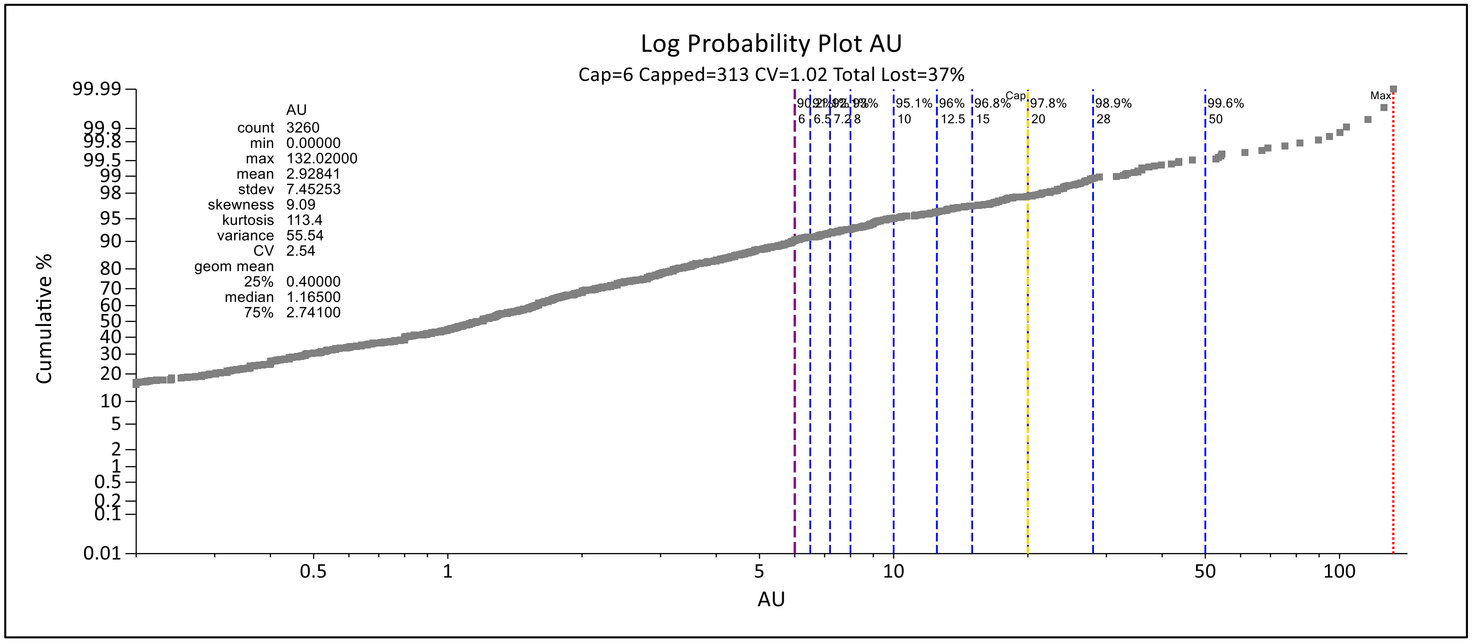
Upon review of the domained samples, SRK elected to apply the capping post compositing for the current estimate. To define the appropriate capping levels, SRK completed analysis of the grade distributions using log probability plots, raw and log histograms Figure 14-6 to Figure 14-10 (with the selected caps highlighted in yellow) to distinguish the grades at which samples have significant impacts on the local estimation and whose effect is considered extreme. Full details of the capping analysis and comparison historgrams are shown in Appendix B.



Column	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Variance	CV
AU							53839	0	891	4.88	160.2	2.6
AU	200	36	99.90%	0.10%	1.40%	15%	53839	0	200	4.81	113.9	2.22
AU	100	169	99.80%	0.30%	3.70%	25%	53839	0	100	4.70	84.02	1.95
AU	75	284	99.60%	0.50%	5.30%	30%	53839	0	75	4.62	71.28	1.83
AU	62	402	99.40%	0.70%	6.60%	33%	53839	0	62	4.56	63.4	1.75
AU	50	579	99.20%	1.10%	8.30%	36%	53839	0	50	4.47	54.87	1.66
AU	40	861	98.70%	1.60%	10%	40%	53839	0	40	4.37	46.71	1.56
AU	40	861	98.70%	1.60%	10%	40%	53839	0	40	4.37	46.71	1.56
AU	33	1188	98.30%	2.20%	12%	43%	53839	0	33	4.27	40.09	1.48
AU	20	2739	95.90%	5.10%	20%	51%	53839	0	20	3.92	25.14	1.28
AU	15	4236	93.50%	7.90%	25%	55%	53839	0	15	3.66	18.19	1.16
AU	AU > 50						579	50.1	891	98.13	5907	0.78
AU	AU <= 50						53260	0	50	4.09	37.67	1.50

Source: SRK, 2019

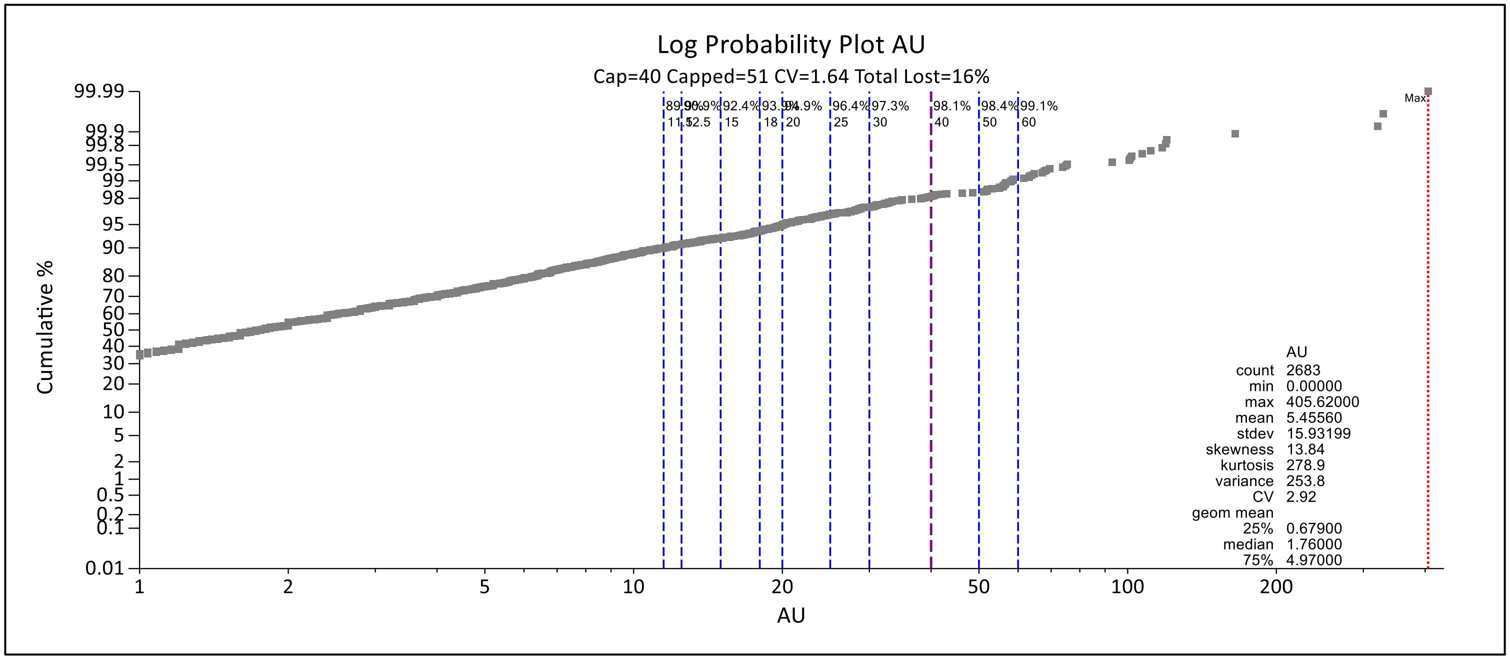
Figure 14-6: Capping Analysis Veins Domain Au (g/t) – (KZONE<9000)



Column	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Variance	CV
AU							3260	0	132	2.93	55.54	2.54
AU	50	16	99.6%	0.5%	5.6%	20%	3260	0	50	2.77	31.49	2.03
AU	28	36	98.9%	1.1%	11.0%	32%	3260	0	28	2.61	20.51	1.74
AU	20	74	97.8%	2.3%	16.0%	39%	3260	0	20	2.47	14.90	1.56
AU	15	105	96.8%	3.2%	20.0%	45%	3260	0	15	2.34	10.87	1.41
AU	12.5	130	96.0%	4.0%	23.0%	48%	3260	0	12.5	2.25	8.85	1.32
AU	10	159	95.1%	4.9%	27.0%	52%	3260	0	10	2.14	6.84	1.22
AU	8	229	93.0%	7.0%	31.0%	56%	3260	0	8	2.02	5.23	1.13
AU	7.2	256	92.1%	7.9%	33.0%	57%	3260	0	7.2	1.96	4.57	1.09
AU	6.5	288	91.1%	8.8%	35.0%	59%	3260	0	6.5	1.90	3.99	1.05
AU	6	313	90.2%	9.6%	37.0%	60%	3260	0	6	1.86	3.59	1.02
AU	AU > 20						74	20.16	132	40.06	702.70	0.66
AU	AU <= 20						3186	0	19.9	2.07	7.94	1.36

Source: SRK, 2019

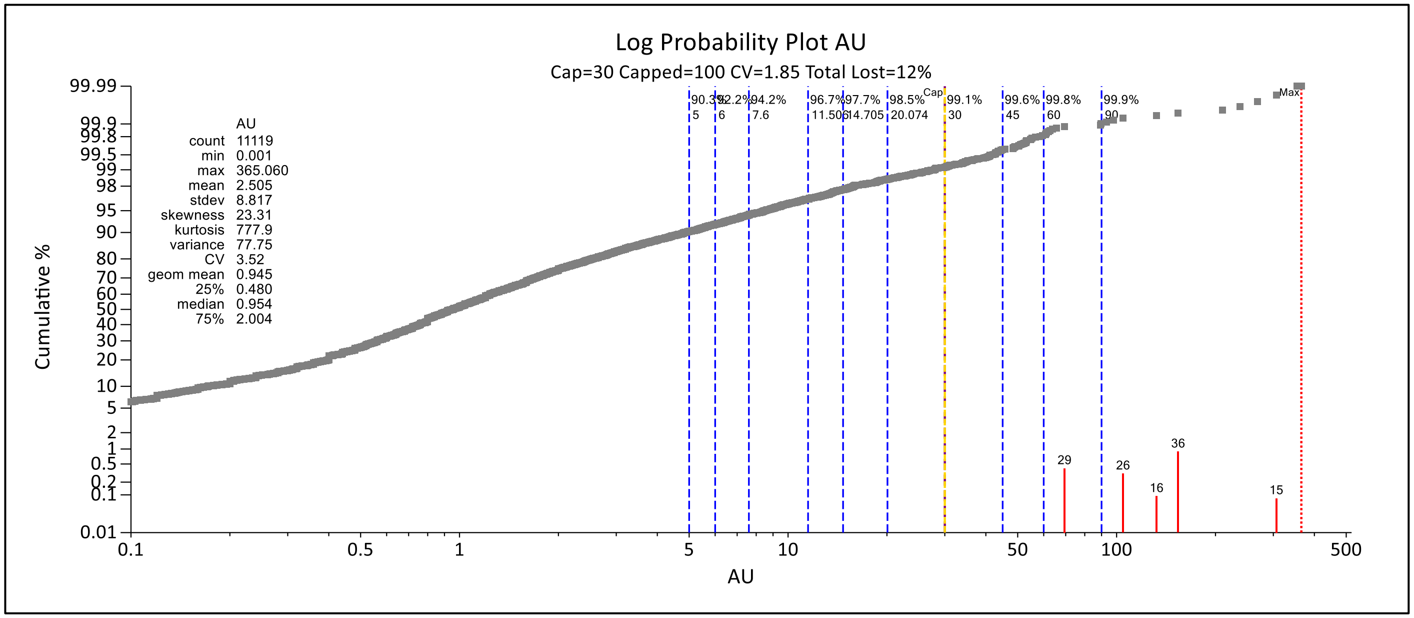
Figure 14-7: Capping Analysis Veins Domain Au (g/t) – (KZONE>9000)



Column	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AU							2683	0	405.6	5.46	14637	253.8	2.92
AU	60	25	99.10%	0.90%	10%	36%	2683	0	60	4.89	13131	82.8	1.86
AU	50	43	98.40%	1.60%	13%	40%	2683	0	50	4.77	12788	70.09	1.76
AU	40	51	98.10%	1.90%	16%	44%	2683	0	40	4.60	12336	56.54	1.64
AU	30	75	97.30%	2.80%	20%	49%	2683	0	30	4.37	11734	42.97	1.5
AU	25	96	96.40%	3.60%	23%	52%	2683	0	25	4.21	11305	35.59	1.42
AU	20	136	94.90%	5.10%	27%	55%	2683	0	20	4.00	10739	27.88	1.32
AU	18	163	93.90%	6.10%	29%	56%	2683	0	18	3.89	10437	24.49	1.27
AU	15	205	92.40%	7.60%	33%	59%	2683	0	15	3.68	9878	19.22	1.19
AU	12.5	242	90.90%	9.00%	36%	62%	2683	0	12.5	3.48	9323	15.02	1.12
AU	11.5	270	89.90%	10.10%	38%	63%	2683	0	11.5	3.38	9066	13.39	1.08
AU	AU > 40						51	40.09	405.6	85.13	4341	5286	0.85
AU	AU <= 40						2632	0	39.72	3.91	10296	32.87	1.47

Source: SRK, 2019

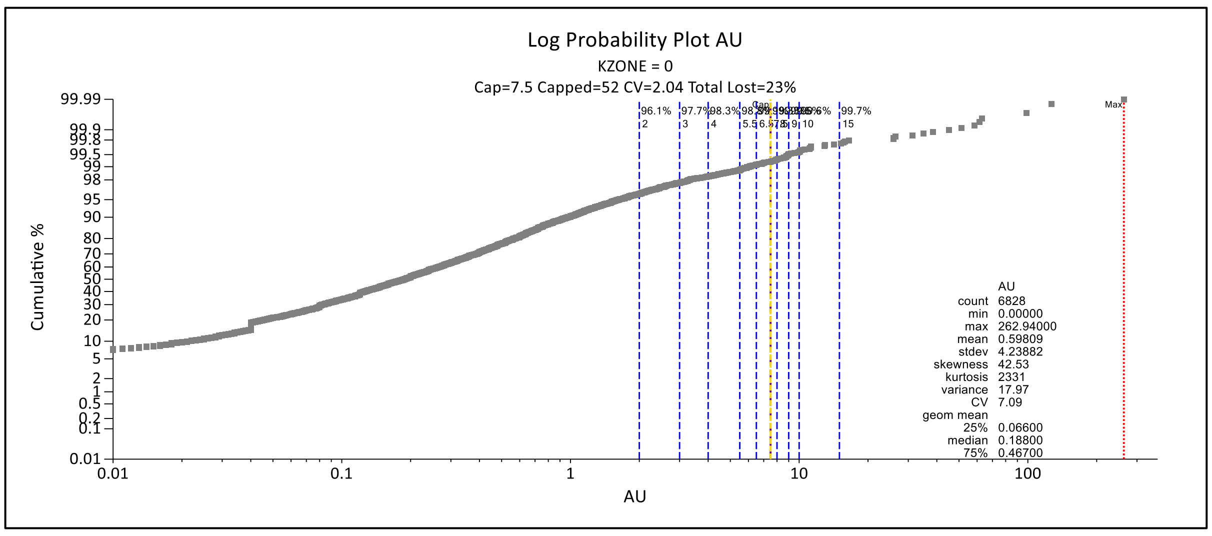
Figure 14-8: Capping Analysis Splays Domain Au (g/t) – (Group = 3000)

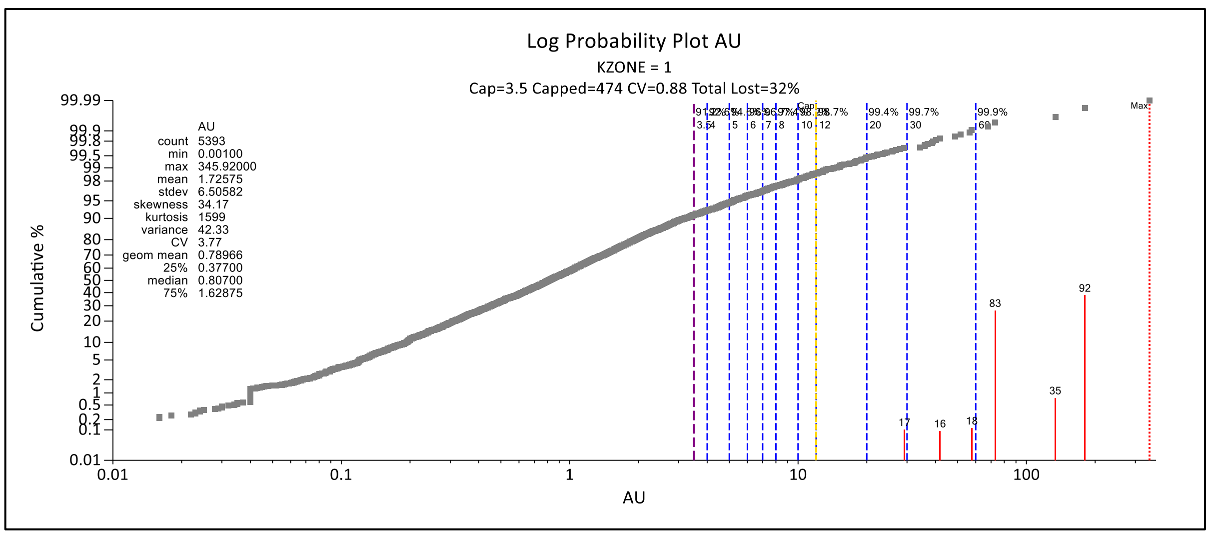


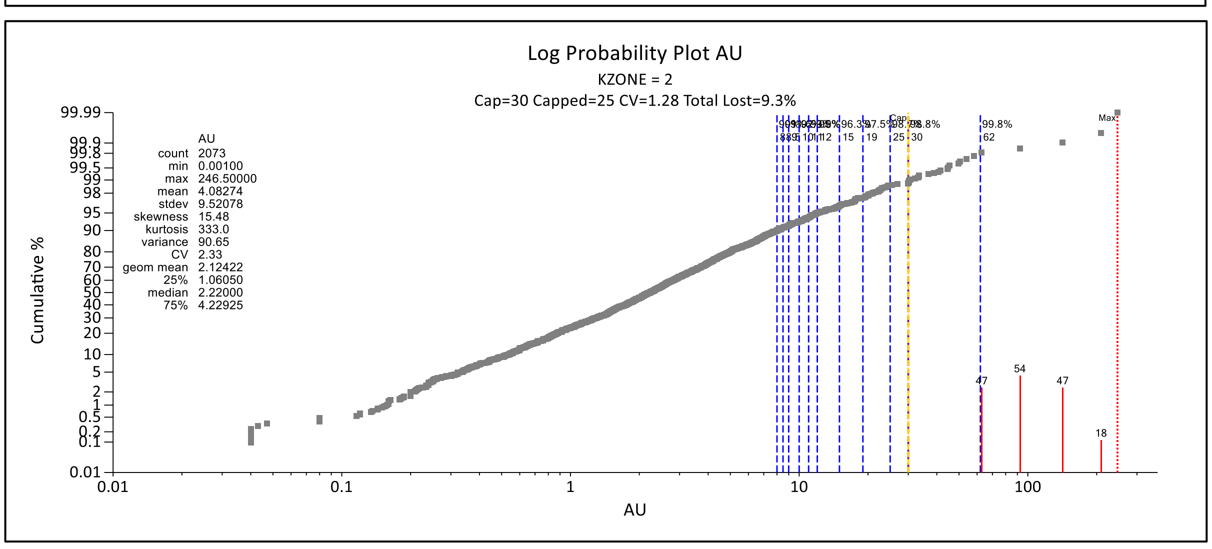
Column	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Variance	CV
AU							11119	0.001	365.1	2.51	77.75	3.52
AU	90.0	11	99.90%	0.10%	4.80%	32%	11119	0.001	90	2.39	32.82	2.40
AU	60.0	21	99.80%	0.20%	6.30%	37%	11119	0.001	60	2.35	27.45	2.23
AU	45.0	43	99.60%	0.40%	8%	41%	11119	0.001	45	2.31	23.26	2.09
AU	30.0	100	99.10%	0.90%	12%	47%	11119	0.001	30	2.21	16.77	1.85
AU	20.1	167	98.50%	1.50%	16%	54%	11119	0.001	20.07	2.10	11.57	1.62
AU	14.7	256	97.70%	2.30%	20%	58%	11119	0.001	14.71	2.00	8.57	1.47
AU	11.5	367	96.70%	3.30%	24%	62%	11119	0.001	11.51	1.91	6.61	1.35
AU	7.6	639	94.20%	5.70%	31%	67%	11119	0.001	7.6	1.74	4.05	1.16
AU	6.0	865	92.20%	7.80%	35%	70%	11119	0.001	6	1.63	2.96	1.06
AU	5.0	1076	90.30%	9.70%	38%	72%	11119	0.001	5	1.54	2.27	0.98
AU	AU > 30						100	30.08	365.1	62.74	3937.00	1.00
AU	AU <= 30						11019	0.001	29.8	1.96	9.85	1.60

Source: SRK, 2019

Figure 14-9: Capping Analysis Porphyry Domain Au (g/t) – (Group = 4000)







Column	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Variance	CV
AU	KZONE 0 = Raw						6828	0	262.9	0.60	17.97	7.09
AU	15	18	99.70%	0.30%	18%	65%	6828	0	15	0.49	1.47	2.48
AU	10	29	99.60%	0.40%	21%	69%	6828	0	10	0.47	1.11	2.22
AU	9	34	99.50%	0.50%	22%	70%	6828	0	9	0.47	1.02	2.16
AU	8	45	99.30%	0.70%	23%	71%	6828	0	8	0.46	0.93	2.08
AU	7.5	52	99.30%	0.80%	23%	71%	6828	0	7.5	0.46	0.87	2.04
AU	6.5	62	99.10%	0.90%	25%	73%	6828	0	6.5	0.45	0.77	1.94
AU	5.5	82	98.70%	1.20%	26%	74%	6828	0	5.5	0.44	0.65	1.83
AU	4	115	98.30%	1.70%	30%	77%	6828	0	4	0.42	0.47	1.63
AU	3	159	97.70%	2.30%	33%	79%	6828	0	3	0.40	0.35	1.48
AU	2	267	96.10%	3.90%	38%	82%	6828	0	2	0.37	0.23	1.29
AU	KZONE = 0 - AU > 7.5						52	7.52	262.9	25.71	1691	1.6
AU	KZONE = 0 - AU <= 7.5						6776	0	7.44	0.41	0.5	1.74
AU	KZONE 1 = Raw						5393	0.001	345.9	1.73	42.33	3.77
AU	60	5	99.90%	0.10%	5.40%	42%	5393	0.001	60	1.63	12.58	2.17
AU	30	17	99.70%	0.30%	8.70%	52%	5393	0.001	30	1.58	7.99	1.79
AU	20	30	99.40%	0.60%	11%	57%	5393	0.001	20	1.53	6.07	1.61
AU	12	74	98.70%	1.40%	15%	63%	5393	0.001	12	1.46	4.07	1.38
AU	10	100	98.20%	1.90%	17%	65%	5393	0.001	10	1.43	3.47	1.3
AU	8	138	97.40%	2.60%	20%	68%	5393	0.001	8	1.39	2.8	1.21
AU	7	176	96.70%	3.30%	21%	69%	5393	0.001	7	1.36	2.45	1.15
AU	6	216	96%	4%	23%	71%	5393	0.001	6	1.32	2.07	1.09
AU	5	292	94.60%	5.40%	26%	73%	5393	0.001	5	1.28	1.68	1.02
AU	4	398	92.60%	7.40%	30%	75%	5393	0.001	4	1.21	1.27	0.93
AU	KZONE = 1 - AU > 3.5						474	3.508	345.9	9.82	404.3	2.05
AU	KZONE = 1 - AU <= 3.5						4919	0.001	3.5	0.95	0.59	0.81
AU	KZONE 2 = Raw						2073	0.001	246.5	4.08	90.65	2.33
AU	62	5	99.80%	0.20%	5.20%	34%	2073	0.001	62	3.87	35.45	1.54
AU	30	25	98.80%	1.20%	9.30%	45%	2073	0.001	30	3.70	22.61	1.28
AU	25	28	98.70%	1.40%	11%	48%	2073	0.001	25	3.64	19.57	1.22
AU	19	52	97.50%	2.50%	14%	52%	2073	0.001	19	3.53	15.55	1.12
AU	15	76	96.30%	3.70%	17%	56%	2073	0.001	15	3.41	12.3	1.03
AU	12	104	95%	5%	20%	59%	2073	0.001	12	3.27	9.66	0.95
AU	11	126	93.90%	6.10%	21%	61%	2073	0.001	11	3.22	8.75	0.92
AU	10	148	92.90%	7.10%	23%	62%	2073	0.001	10	3.15	7.79	0.89
AU	9	174	91.60%	8.40%	25%	64%	2073	0.001	9	3.08	6.82	0.85
AU	8	207	90%	10%	27%	65%	2073	0.001	8	2.99	5.82	0.81
AU	KZONE = 2 - AU > 30						25	30.09	246.5	61.64	3110	0.9
AU	KZONE = 2 - AU <= 30						2048	0.001	29.88	3.38	14.33	1.12

Source: SRK, 2019

Figure 14-10: Capping Analysis MDZ Sub-domains Domain Au (g/t) – (Group = 5000)

SRK reviewed and updated the capping/composite strategy at Marmato as part of the 2019 Mineral Resource update. In 2019, the selected capping limits are as follows:

- Veins: A 50 g/t Au cap in the major veins with large numbers of samples, dropping to 20 g/t Au in veins with lower sampling density. A standard cap for all veins of 150 g/t Ag has been selected;
- Splay: A 40 g/t Au and 80 g/t silver (Ag) cap for all splays has been used;
- Porphyry: A 15 g/t Au cap has been used for all material, but the silver capping has been split based on the assumption of higher silver grades within the Echandia area of 120 g/t and 550 g/t Ag respectively; and
- MDZ: A sliding cap related to the varying grade shells used to model at depth has been applied with a cap ranging from 7.5 g/t Au, to 12.0 g/t Au and 30 g/t Ag cap, for the low, medium and higher grade components.

Overall, these levels remain consistent with the capping levels used previously, with application of the capping restrictions supported by improvements in the definition of the grade populations for the veins from the channel sampling.

In general, SRK aims to limit the impact of the capping to less than 5% change in the mean value, however in some cases with extreme outliers, the change in the mean exceeds 5%. The highly positively skewed nature of the gold distributions and the very high values seen in the population result in the significant changes in the mean values. A comparison of the raw versus capped values is shown in Table 14-3.

Table 14-3: Comparison of Raw vs. Capped Composite Statistics

Element	GROUP	KZONE	Number Values	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	Variance	Standard Deviation	Coefficient of Variation	Difference (%)
Au Raw	1000	All	54,179	0.001	891.03	4.67	154.74	12.44	2.67	
Au Raw	3000	All	2,716	0.001	405.62	4.81	222.03	14.90	3.10	
Au Raw	4000	All	7,508	0.001	306.41	1.46	19.13	4.37	3.00	
Au Raw	5000	0	6,827	0.001	262.94	0.31	3.20	1.79	5.76	
Au Raw	5000	1	5,393	0.001	345.92	1.42	21.66	4.65	3.28	
Au Raw	5000	2	2,073	0.001	246.50	3.69	58.33	7.64	2.07	
Ag Raw	1000	All	53,892	0.001	1,995.00	25.09	1,242.74	35.25	1.41	
Ag Raw	3000	All	2,583	0.001	432.00	21.48	765.70	27.67	1.29	
Ag Raw	4000	All	7,444	0.001	5,613.00	14.30	8,559.22	92.52	6.47	
Ag Raw	5000	0	6,828	0.001	316.08	1.66	33.78	5.81	3.49	
Ag Raw	5000	1	5,391	0.001	258.92	2.57	36.65	6.05	2.36	
Ag Raw	5000	2	2,073	0.001	162.06	5.00	81.91	9.05	1.81	
Au Comp	1000	all	21,893	0.001	50.00	4.54	35.89	5.99	1.32	-2.8%
Au Comp	3000	all	1,093	0.001	40.00	4.40	33.98	5.83	1.33	-8.6%
Au Comp	4000	all	5,733	0.001	15.00	1.44	3.48	1.86	1.30	-1.2%
Au Comp	5000	0	5,714	0.003	7.50	0.32	0.45	0.67	2.11	2.8%
Au Comp	5000	1	3,683	0.003	12.00	1.46	2.75	1.66	1.14	3.0%
Au Comp	5000	2	1,215	0.005	30.00	3.52	12.59	3.55	1.01	-4.7%
Ag Comp	1000	all	21,893	0.001	150.00	24.62	535.67	23.14	0.94	-1.9%
Ag Comp	3000	all	1,093	0.001	80.00	20.78	311.72	17.66	0.85	-3.2%
Ag Comp	4000	all	5,693	0	550.00	12.71	1,433.50	37.86	2.98	-11.1%
Ag Comp	5000	0	5,714	0.005	30.00	1.66	11.19	3.34	2.02	-0.3%
Ag Comp	5000	1	3,683	0.001	30.00	2.66	11.32	3.36	1.27	3.4%
Ag Comp	5000	2	1,215	0.005	30.00	4.69	29.06	5.39	1.15	-6.3%

Source: SRK, 2019

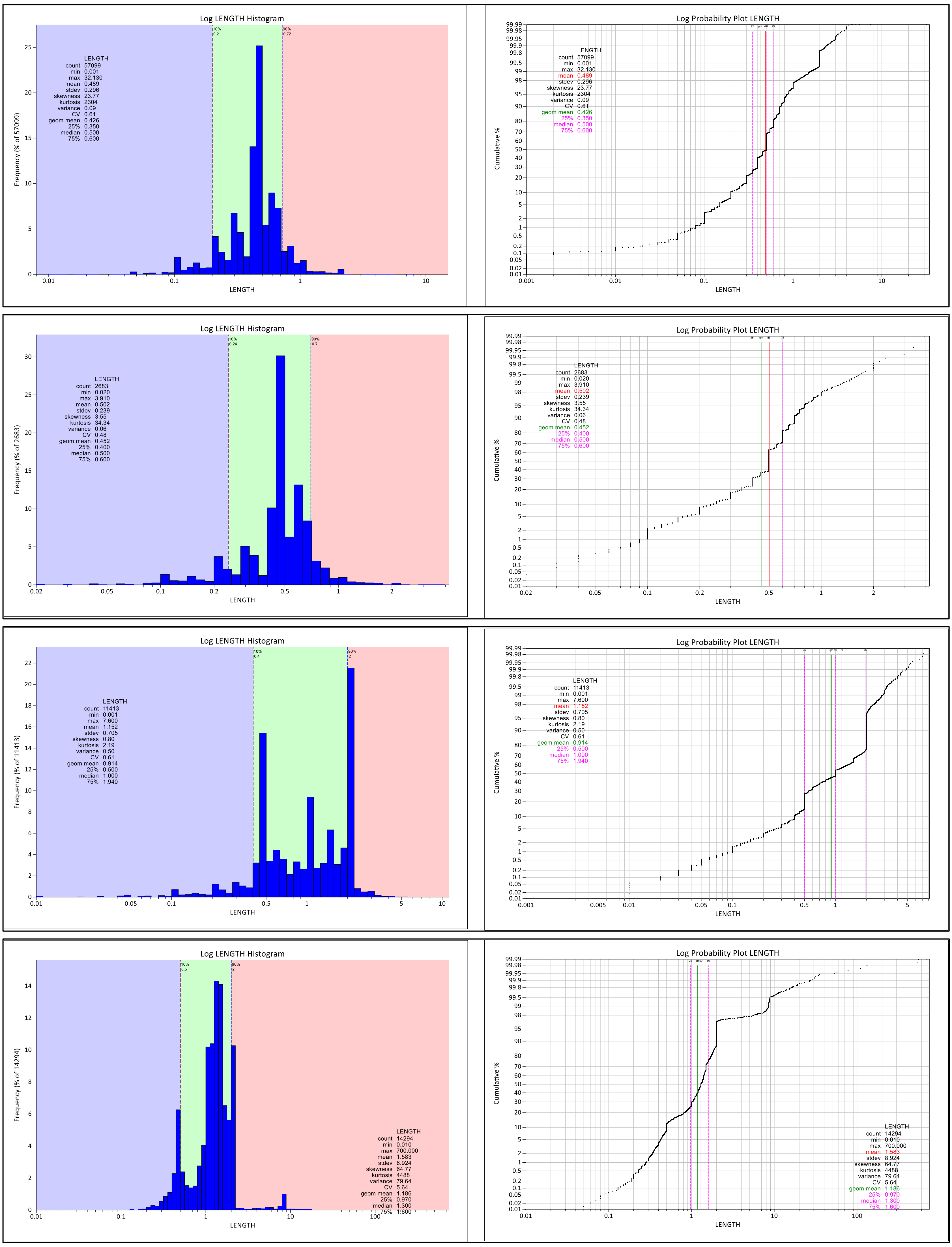
14.4.2 Compositing

Prior to the undertaking of grade interpolation, samples need to be composited to equal lengths for constant sample volume, honoring sample support theories.

SRK has undertaken a sample composite analysis for gold in order to determine the optimal sample composite length for grade interpolation. This investigated both changes in composite length and minimum composite lengths for inclusion. The analysis compared the resultant mean grade against the length weighted raw sample mean grades, and the percentage of samples excluded when applying the minimum composite length. In addition to the analysis completed, SRK has reviewed the histograms of the raw sampling lengths within the various domains (Figure 14-11). During the review SRK noted the following:

- A review of the sample lengths indicated that the mean sample length is approximately 0.5 m veins, but 45% of the samples are between 0.5 to 1.0 m, with a further 5% between 1.0 to 2.0 m.
- The average length of the raw sampling in the porphyry and deep mineralization is 1.0 m, with the majority of the samples ranging between 1.0 to 2.0 m.

Given the narrow nature of the veins, it is SRK's view that increasing the sample lengths to 2.0 m is preferred so only a single composite will exist across in the vein in narrow areas. SRK has elected to also use the 2.0 m composite lengths in porphyry and deep zones.



Source: SRK, 2019

Figure 14-11: Histogram of Sample Lengths Within the Various Domains

14.5 Density

Density measurements are made routinely by GCM geologists during core logging and sample preparation. Each geologist tries to make one density measurement daily and to complete the calculation the following procedure has been used:

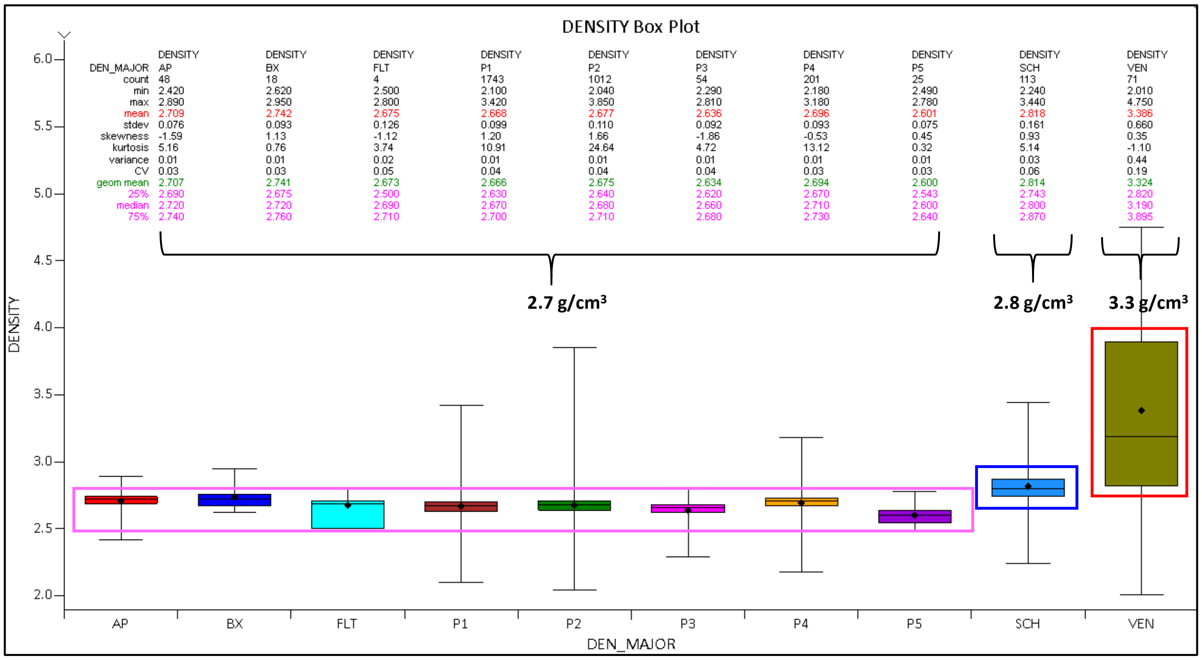
- A piece of unbroken core is selected;
- A 14 to 15 cm long piece of core from the interval of interest is cut;
- As the core is cut, the geologist must ensure that the cut is perpendicular to the core axis and does not result in the loss of any material along the cut line;
- The length of the core is measured and the diameter of the core is determined with a digital caliper at 3 to 4 cm intervals and the average diameter is calculated; and
- The core is weighed on a digital balance and the density is calculated as follows:
    - Pi * core diameter * core length = core volume; and
    - Core weight / core volume = density.

SRK completed a statistical review of the density measurements in the database provided up to drillhole MT-IU-031. The database included a total of 3,289 samples with results ranging from 2.01 to 4.75 g/cm^3. The majority of the samples have been taken within the various phases of porphyry mineralization which have a range of 2.04 to 3.85 g/cm^3. The highest measured density based on the drillcore is taken from the vein material which returned an average of 3.39 g/cm^3, but also contained the highest variability. A summary of the measured density per major rocktype is shown in Table 14-4 and Figure 14-13.

Table 14-4: Summary of Density Values

Domain	Count	Min	Max	Mean	Total	Variance	StDev	CV
	3289	2.01	4.75	2.69	8857	0.03	0.177	0.07
AP	48	2.42	2.89	2.71	130	0.01	0.076	0.03
BX	18	2.62	2.95	2.74	49.36	0.01	0.093	0.03
FLT	4	2.5	2.8	2.68	10.7	0.02	0.126	0.05
P1	1743	2.1	3.42	2.67	4650	0.01	0.099	0.04
P2	1012	2.04	3.85	2.68	2709	0.01	0.11	0.04
P3	54	2.29	2.81	2.64	142.3	0.01	0.092	0.04
P4	201	2.18	3.18	2.70	541.9	0.01	0.093	0.03
P5	25	2.49	2.78	2.60	65.03	0.01	0.075	0.03
SCH	113	2.24	3.44	2.82	318.4	0.03	0.161	0.06
VEN	71	2.01	4.75	3.39	240.4	0.44	0.66	0.19

Source: SRK 2019

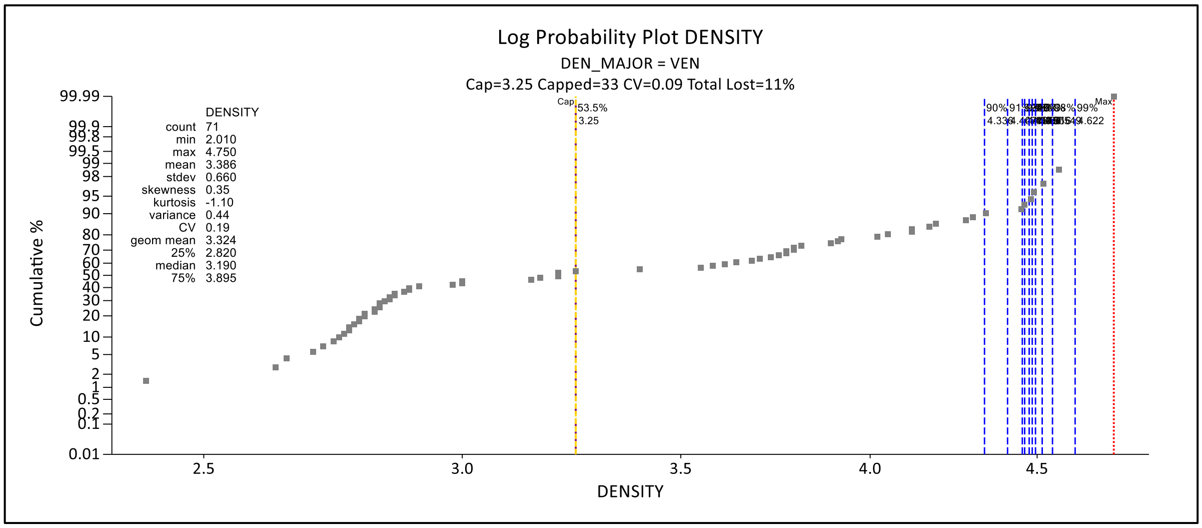


Source: SRK, 2019

Figure 14-12: Summary of Review by Rocktype of Density Values at Marmato

In the 2017 Mineral Resource, the density has been grouped into three main units, which are comprised of the porphyry, schist and vein rocktypes. SRK has compared these values to the 2017 model and discussed with the current mining operations team. The porphyry domain has been kept at the same value as the 2017 block model which used an average density of 2.7 g/cm^3. SRK still considers this to be reasonable but comments that it could fluctuate between 2.65 to 2.70 g/cm^3, and should therefore be reviewed within the Marmato Deeps Zone based on the results of the current on-going infill drilling program being completed by the Company for preparation for a future PFS. If a lower density is applied, it would impact the overall tonnage less than 2%.

The mine reported that it currently uses a lower density for the vein material based on the feed at the plant closer to 2.7 to 2.9 g/cm^3 range and that there could be risk of overstating the tonnage using a 3.4 g/cm^3 as shown from the statistical analysis. The basis for the lower density is likely due to favorable sampling of higher sulphides in the core which do not reflect the mined stope widths. Therefore, to provide a more relatistic assessment of the density, SRK completed a statistical review of the vein samples using histograms and log-probability plots which still indicates a density of over 3.0 g/cm^3 is reasonable based even on extreme cappinig of values greater than 3.25 g/cm^3.



Cap	Capped	Percentile	Capped%	Count	Min	Max	Mean
				71	2.01	4.75	3.386
4.622	1	99%	1.40%	71	2.01	4.622	3.384
4.549	2	98%	2.80%	71	2.01	4.549	3.383
4.516	3	97%	4.20%	71	2.01	4.516	3.382
4.495	3	96%	4.20%	71	2.01	4.495	3.381
4.485	4	95%	5.60%	71	2.01	4.485	3.38
4.475	5	94%	7.00%	71	2.01	4.475	3.38
4.461	5	93%	7.00%	71	2.01	4.461	3.379
4.453	6	92%	8.50%	71	2.01	4.453	3.378
4.407	7	91%	9.90%	71	2.01	4.407	3.374
4.336	8	90%	11.30%	71	2.01	4.336	3.366
3.25	33	53.50%	46.50%	71	2.01	3.25	3.029
DEN_MAJOR = VEN - DENSITY > 3.25				33	3.4	4.75	4.018
DEN_MAJOR = VEN - DENSITY <= 3.25				38	2.01	3.25	2.837

Source: SRK, 2019

Figure 14-13: Log Probability Plot of Density Measurements Logged as Vein

To reflect density values more consistent with the mining, SRK has updated the statisitical analysis to using a sub-set of the density database and the veins wireframes generated from the geological model using a halo of 0.5 m either side of the veins and rerun the analysis. The results of the analysis returned a mean density of 2.97 g/cm^3 (rounded to 2.95 g/cm^3). In discussion with the GCM geological team it was felt this was a more reasonable represention of the density for the vein domain. In summary, the final density values used in the 2019 Mineral Resources are presented in Table 14-5 which also compares them to the density values used in the 2017 block model.

Table 14-5: Summary of Density Values Used in 2019 Mineral Resource

Rock Type	No. Samples	2017	2019
		Density (gcm3)	Density (gcm3)
Vein	71	3.5	2.95
Porphyry	2,857	2.7	2.7
Schist	97	2.8	2.8

Source: SRK, 2019

14.6 Variogram Analysis and Modeling

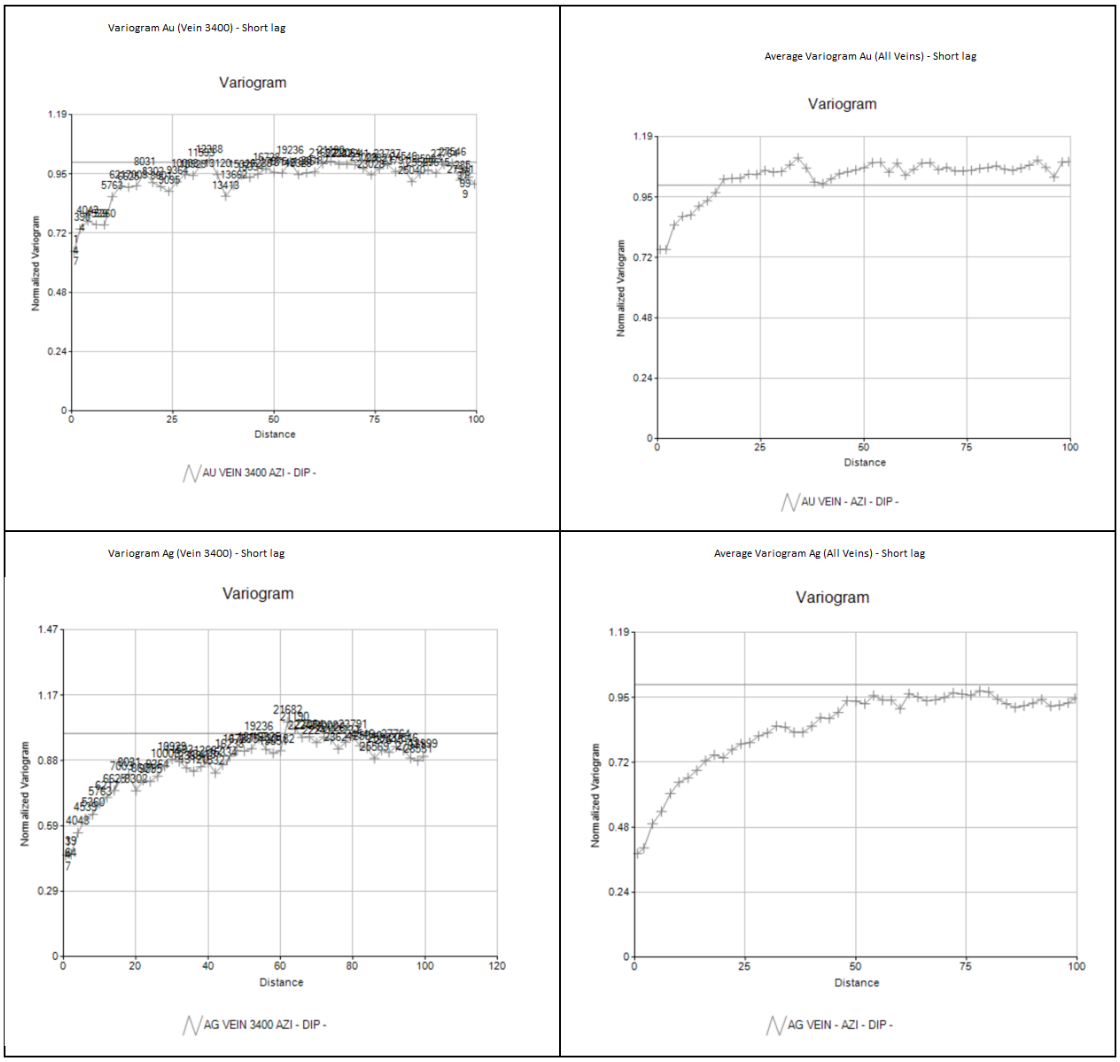
The composite drillhole database was imported into Snowden Supervisor software for the geostatistical analysis. Semi-variograms have been completed for both gold and silver values. The resultant experimental semi-variogram models produced were poor in terms of definition to fit a statistical model. In order to define variograms of sufficient clarity to be modelled, the data has been calculated using pairwise relative variograms, which removes the influence of some of the variability.

Following the pairwise transformation, the next stage was to define the nugget effect from down-hole variogram and then to model the longer variogram ranges from longer lag directional variograms in the three principle directions, downdip (N0), along-strike (D90) and perpendicular to the bedding plane (N90). An example of the semi-variograms is shown in Figure 14-14. SRK noted poor continuity in the initial analysis of the directional semi-variograms so has elected to use omni-directional models for the current estimate (Table 14-6). Further infill drilling is required to improve the distance between samples at depth to improve the variography.

Table 14-6: Summary of Semi-Variogram Parameters Used in the 2019 Estimation Process

Domain	**Vein**		**Splay**		**Porphyry**		**Deeps**		
Element	Au	Ag	Au	Ag	Au	Ag	Au_0.7	Au_1.7	Ag
Ref Number	1001	1002	3001	3002	4001	4002	5001	5002	5003
Nugget	0.35	0.25	0.35	0.25	0.99	0.5	0.48	0.24	0.36
Range 1 (m)	3.6	9.4	1.6	2.3	7	19	15	15	15
Sill 1	0.277	0.341	0.181	0.178	0.09	0.53	0.47	0.5	0.26
Range 2 (m)	22	56.1	15.4	11.7	35	73	80	65	60
Sill 2	0.295	0.32	0.091	0.153	0.26	0.14	0.26	0.26	0.42
Range 3 (m)	139.5	384.5	55.2	54.8	70				
Sill 3	0.078	0.089	0.378	0.419	0.14				

Source: SRK, 2019



Source: SRK, 2019

Figure 14-14: Example of Semi-Variogram Analysis for Veins Domain (Group 1000)

14.7 Block Model

14.7.1 Protoype Definition

SRK has produced a parent block model with block dimensions of 5 by 5 by 5 m (X,Y,Z), as a function of the sample spacing within the veins, Previous test work undertaken for block variance suggested a 10 by 10 by 10 m block size was appropriate, but the smaller block size was preferred with the increase in the information within the vein domains, and reflect potential for selectivity for mining. SRK acknowledges a larger block size could be more appropriate, but the decision was taken to use a uniform block size across the deposit in conjunction with the mining team. SRK is recommending a more detailed study be completed during a future prefeasibility study to test the sensitivity of this assumption. Sub-blocking has been allowed to a resolution 0.5 m along strike, 0.5 m across strike and 1 m in the vertical direction to provide an appropriate geometric representation.

Given the orientation of the orebody striking to the northwest, the decision was made to rotate the database (for block model grade interpolation) from UTM coordinates through 55 degrees into a north-south local grid orientation, to enable an improved representation of grade continuity along strike. To rotate the interpretation the “CDTRAN” Datamine™ command has been utilized.

The details of the block model origin, rotation and local dimensions are shown in Table 14-7.

Table 14-7: Summary of Block Model Parameters used for Geological Model

Dimension	Origin (UTM)	Origin (Local)	Block Size	Number of Blocks	Rotation	Min Sub-blocking (m)
X	433,250	0	5	420	-	0.5
Y	604,250	0	5	460	-	0.5
Z	100	100	5	400	-55	1

Source: SRK, 2019

14.7.2 Model Codes

Using the wireframes created and described in Section 14.2 several codes have been developed to describe each of the major geological properties of the rock types. Table 14-8 summarizes geological fields created within the geological model and the codes used.

Table 14-8: Summary of Block Model Fields and Description

Field Name	Description
SVOL	Search Volume reference (range from 1 to 3)
KV	Kriging Variance
SLOPE	Slope of regression
NS	Number of samples used to estimate the block
AU	Final Gold Estimate using for Reporting
AG	Final Silver Estimate using for Reporting
AUOK	Gold Estimate using OK
AGOK	Silver Estimate using OK
AUIDW	Gold Estimate using IDW (Power 2)
AGIDW	Silver Estimate using IDW (Power 2)
AUNN	Gold NN Methodology
AGNN	Silver NN Methodology
CLASS	Classification
GROUP	Mineralized structures grouped by domain
VEIN	Vein coding for individual mineralized structure GROUP1000 coding
SPLAY	Vein coding for individual mineralized structure GROUP3000 coding
DENSITY	Density of the rock
DEPLETE	Mined out areas
ROCK	Coding for Major Rock type

Source: SRK, 2019

14.8 Estimation Methodology

A Kriging Neighborhood Analysis (KNA) exercise has been completed for gold, in order to optimize the parameters used in the estimation and kriging calculations. Initial grade estimation was undertaken in Datamine™. To complete the exercise, a number of scenarios were tested using various estimation and kriging parameters. Different input parameters have been changed and the differences in the slope of regression, kriging variances, and block estimates recorded.

To complete the analysis, SRK ran different estimates for Au, changing the following parameters:

- Search ellipse sizes;
- Minimum and maximum number of samples; and

- Orientation of search ellipse.

In order to assess the best grade estimate, the following data fields were analyzed in most detail: kriging variance; number of samples; and proportion of blocks estimated in each search volume. Additional fields monitored included the resultant grade in comparison with the sample data. To test the optimum search volume to be used, SRK has selected a first pass minimum and maximum number of samples and adjusted the expansion factor of the semi-variogram range used per estimate per zone.

The optimum parameters selected allowed an appropriate proportion of block estimates in the initial search volumes, whilst achieving a reduction in variance and a relative increase in slope of regression (in SVOL 1 and 2) without excessive smoothing.

Based on the outcome of the validation process, SRK has selected to use either OK algorithm, or ID2 estimates to compile the final grade estimates. Typically zones with larger sample populations are supported by OK, while zones with less data are supported by ID2, which has been used for the primary interpolation within anisotropic elliptical search ranges using suitable parameters. The search distances used for the interpolations were based upon an expansion factor of the semi-variogram ranges. Individual searches were specified for each data field.

In completing a detailed review of the vein and the veinlet style mineralization, SRK concluded that given the presence of two principal strike and dip directions a bias could be introduced if a single search orientation was selected per zone. To ensure the block model reflects the nature of the vein mineralization as accurately as possible, SRK therefore utilized the wireframe interpretation to aid in determining the search orientations used during the kriging equations on a block by block basis. This has been done using the dynamic anisotropy function in Datamine™.

In addition to varying the number of samples, second and third radius factored search volumes have been used for all estimation domains. The first search represents an optimized search distance (selected from a kriging sensitivity analysis), ensuring (in general) that block estimates use a minimum of two drillholes, whilst the second and third search volumes use expansion factors that produce more smoothed block estimates, appropriate to the limit of geological continuity.

The third expansion volume was sufficient to ensure that all appropriate blocks (in areas with reasonable geological confidence) were assigned grade values. These blocks were generally classified with lower confidence.

SRK tested the estimation sensitivity of the gold grade distribution and mean grade and tonnage of the MDZ using a progressively steepening (dip) of the search ellipse. In general, the global mean grade and tonnage of MDZ shows only limited sensitivity to variation in dip of the search ellipse, whilst the visual gold grade distribution appears to most accurately honor the dip continuity of the sample grades at a dip of 80°.

The final kriging parameters selected for gold and silver are presented in Table 14-9. A discretization grid of 3 by 3 by 3 has been used within each parent block during the estimation. The discretization grid ensures that single blocks near the edge of each estimation zone are assigned a grade that is characteristic of the modelled domain and not just those values at the block midpoint.

Table 14-9: Summary of Final Ordinary Kriging Parameters for Gold at Marmato

Group	Rotation Axis						Search Range			Number Samples				Octants
	Angle 1	Axis	Angle 2	Axis	Angle 3	Axis	Along Strike	Down Dip	Across Strike	Min	Max	Max Per Hole	Max Per Octant	
	First Range													
1000	Dynamic						50	50	50	5	20	na	4	Yes
3000	Dynamic						50	50	50	5	20	na	4	Yes
4000	15	Z	-80	X	0	Y	35	35	15	6	12	4	na	No
5000	30	Z	85	X	0	Y	75	75	15	4	12	3	na	No
	Second Range													
1000	Dynamic						250	250	250	5	10	na	4	Yes
3000	Dynamic						250	250	250	5	10	na	4	No
4000	15	Z	-80	X	0	Y	70	70	40	4	12	3	na	No
5000	30	Z	-80	X	0	Y	150	150	30	4	8	3	na	No
	Third Range													
1000	Dynamic						250	250	250	1	8	na	4	Yes
3000	Dynamic						250	250	250	1	8	na	4	Yes
4000	15	Z	-80	X	0	Y	140	140	80	1	8	3		No
5000	30	Z	-80	X	0	Y	300	300	60	1	8	3		No

Source: SRK: 2019

14.9 Model Validation

SRK has undertaken a thorough validation of the resultant interpolated model in order to confirm the estimation parameters, to check that the model represents the input data on both local and global scales and to check that the estimate is not biased. SRK has undertaken this using a number of different validation techniques:

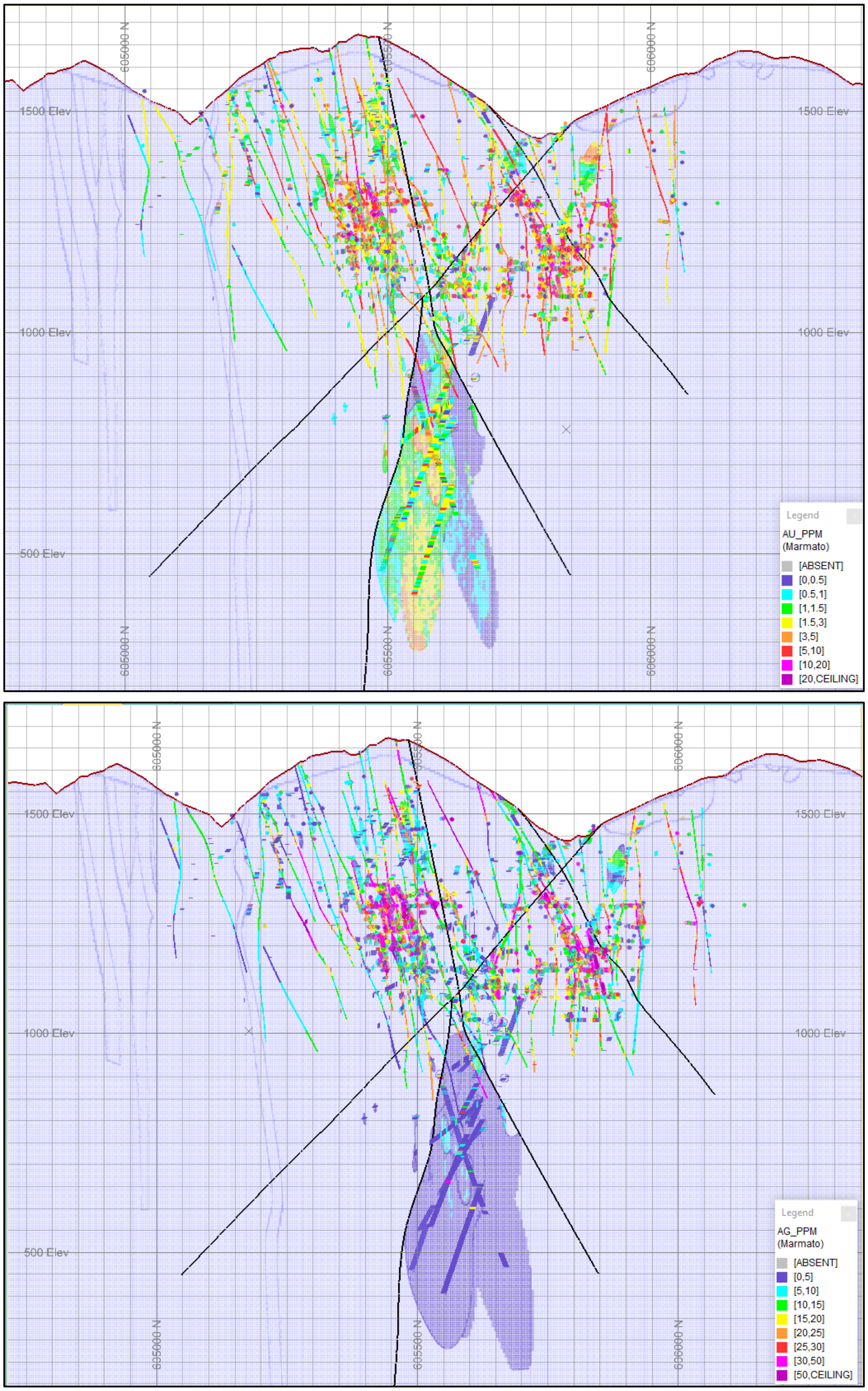
- Visual inspection of block grades in comparison with drill hole data;
- Inspection of block grades in plan and section and comparison with drillhole grades;
- Statistical validation of declustered means versus block estimates;
- Comparison of estimates using different estimation methods (NN, IDW, OK); and
- Swath plots of the mean block and sample grades.

The geology model, geostatistical analysis, variography, selection of resource estimation parameters, and construction of the block model work were completed by SRK. The current drilling information is sufficiently reliable to interpret with confidence the boundaries of the various veins, and the assaying data is sufficiently reliable to support Mineral Resource estimation.

14.9.1 Visual Comparison

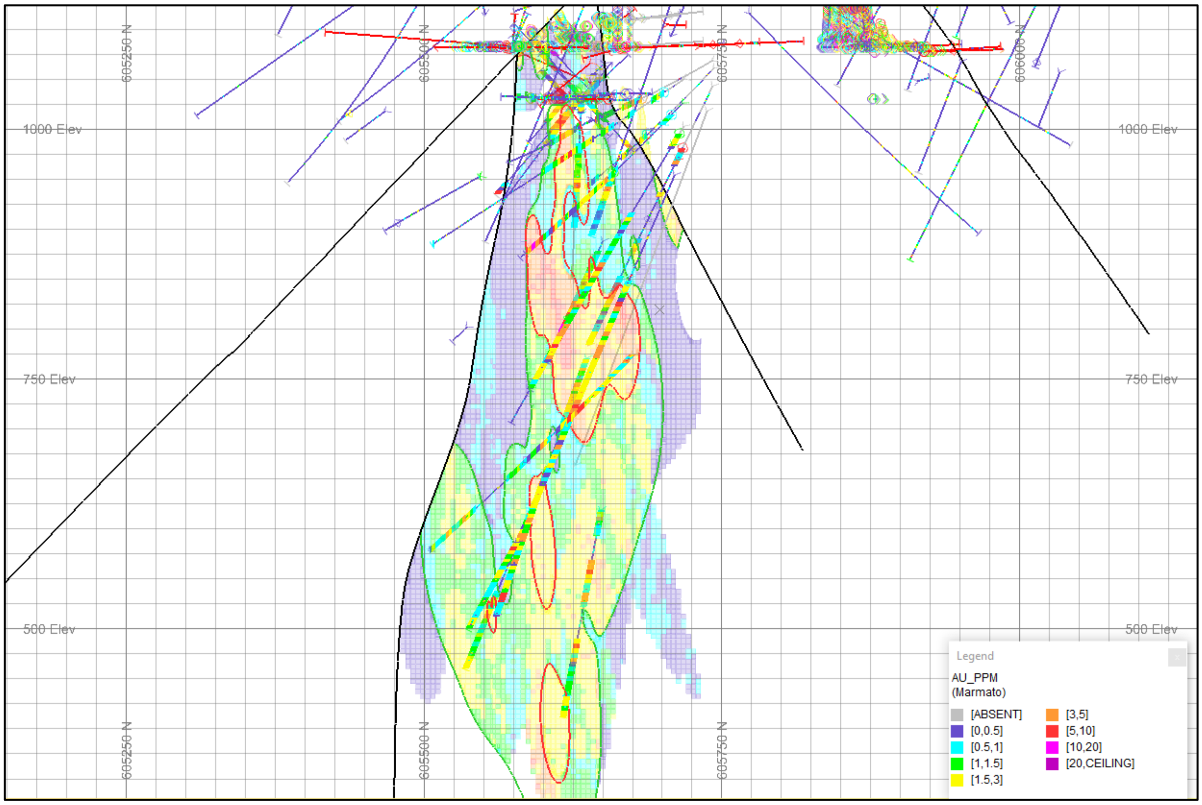
Visual validation provides a local validation of the interpolated block model on a local block scale, using visual assessments and validation plots of sample grades versus estimated block grades. A thorough visual inspection of cross-sections, long-sections and bench/level plans, comparing the sample grades with the block grades has been undertaken, which demonstrates good comparison between local block estimates and nearby samples, without excessive smoothing in the block model.

Figure 14-15 and Figure 14-16 show examples of the visual validation checks and highlights the overall block grades corresponding with raw samples grades.



Source: SRK, 2019

Figure 14-15: Example of Visual Validation of Grade Distribution for Gold (Top) and Silver (Bottom)



Source: SRK, 2019

Figure 14-16: Comparison of Au Grade Within Deeps Domain Between Boreholes and Estimates

14.9.2 Comparative Statistics

SRK has completed a statistical validation of the block estimates (NN, OK and ID2) versus the mean of the composite samples per zone. In general, the results indicate a reasonable comparison (Table 14-10) between the sample mean grades (declustered) and the block estimates.

SRK notes that the comparison between the mean grades and the raw samples often exceeds the desired levels of error, but this is often a function of the clustering within the dataset from either channel sampling, or the fan drilling. SRK has therefore focused on using a comparison to the declustered mean grades, which have been determined within Snowden Supervisor, by testing multiple declustering block sizes ranging between 0 to 50 m, and selecting points where the mean stabilizes. In the case of the higher grade domains such as the veins or the high-grades MDZ, this typically represents a reduction in the average grades, while the lower grade domains have typically been under sampled and therefore there is a slight increase in the average grades within these zones.

For the comparison SRK has compared the grouped statistics for the veins, splays, and porphyry material, but has broken out the MDZ into the three main sub-domains. The zones show satisfactory correlations between the composite and block estimates, with the highest errors noted within the splays. This is minor domain and reducing limiting the study to the higher confidence blocks estimated within the first estimation pass reduces the average differences from ± 25% to less than 15%. SRK notes these differences which have subsequently been accounted for during the classification process.

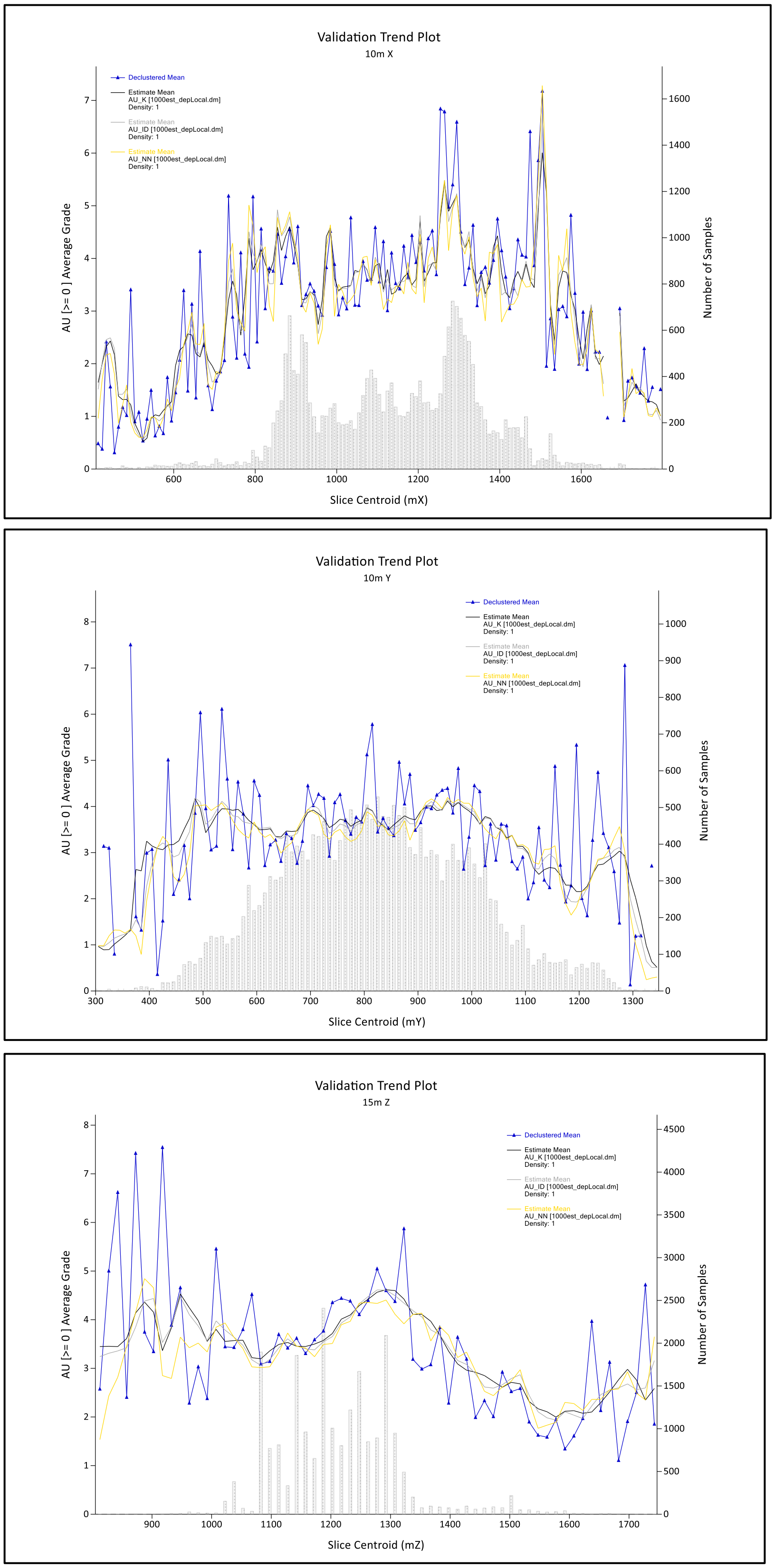
Table 14-10: Comparison of Raw, Declustered Composites vs. OK, ID2 and NN Statistics (1)

Statistic	Points	Mean	Standard Deviation	Variance	Coefficient of Variation	% Difference (raw)	% Difference (decluster)
Group 1000							
Composites	21,893	4.82	6.51	42.42	1.35		
Declustered	21,893	3.59	5.57	31.04	1.55		
OK	3,556,735	3.63	2.91	8.48	0.8	-24.78	1.16
ID2	3,556,735	3.61	3.41	11.65	0.95	-25.19	0.59
NN	3,556,735	3.53	5.37	28.83	1.52	-26.79	-1.55
Group 3000							
Composites	1,092.00	4.75	6.50	42.26	1.37		
Declustered	1,092.00	4.47	6.20	38.48	1.39		
OK	138212	3.42	3.21	10.30	0.94	-27.90	-23.38
ID2*	138212	3.44	3.41	11.61	0.99	-27.59	-23.05
NN	138212	3.32	6.09	37.13	1.84	-30.04	-25.65
Group 4000							
Composites	7,682	1.77	2.45	6.00	1.38		
Declustered	7,682	1.67	2.30	5.28	1.38		
OK	724,536	1.58	1.50	2.24	0.95	-10.98	-5.45
ID2	724,536	1.58	1.53	2.35	0.97	-10.82	-5.29
NN	724,536	1.71	3.33	11.10	1.95	-3.45	2.54
Group 5000 - KZONE=0 (<0.7 g/t)							
Composites	6237	0.29	0.66	0.43	2.24		
Declustered	6237	0.33	0.65	0.42	1.97		
OK	453408	0.39	0.28	0.08	0.73	32.22	18.47
ID2	453408	0.39	0.29	0.08	0.75	32.05	18.31
NN	453408	0.40	0.65	0.43	1.65	35.03	20.98
Group 5000 - KZONE=1 (0.7 - 1.7 g/t)							
Composites	4100	1.32	1.65	2.72	1.25		
Declustered	4100	1.35	1.65	2.73	1.22		
OK	566499	1.45	0.59	0.35	0.41	9.93	7.17
ID2	566499	1.47	0.61	0.37	0.41	11.48	8.68
NN	566499	1.39	1.57	2.48	1.14	5.40	2.75
Group 5000 - KZONE=0 (>1.7 g/t)							
Composites	1245	3.52	3.67	13.44	1.04		
Declustered	1245	3.33	3.54	12.50	1.06		
OK	158057	3.25	1.16	1.35	0.36	-7.58	-2.22
ID2	158057	3.37	1.23	1.51	0.37	-4.29	1.25
NN	158057	3.13	3.00	8.98	0.96	-11.01	-5.86

Source: SRK, 2019

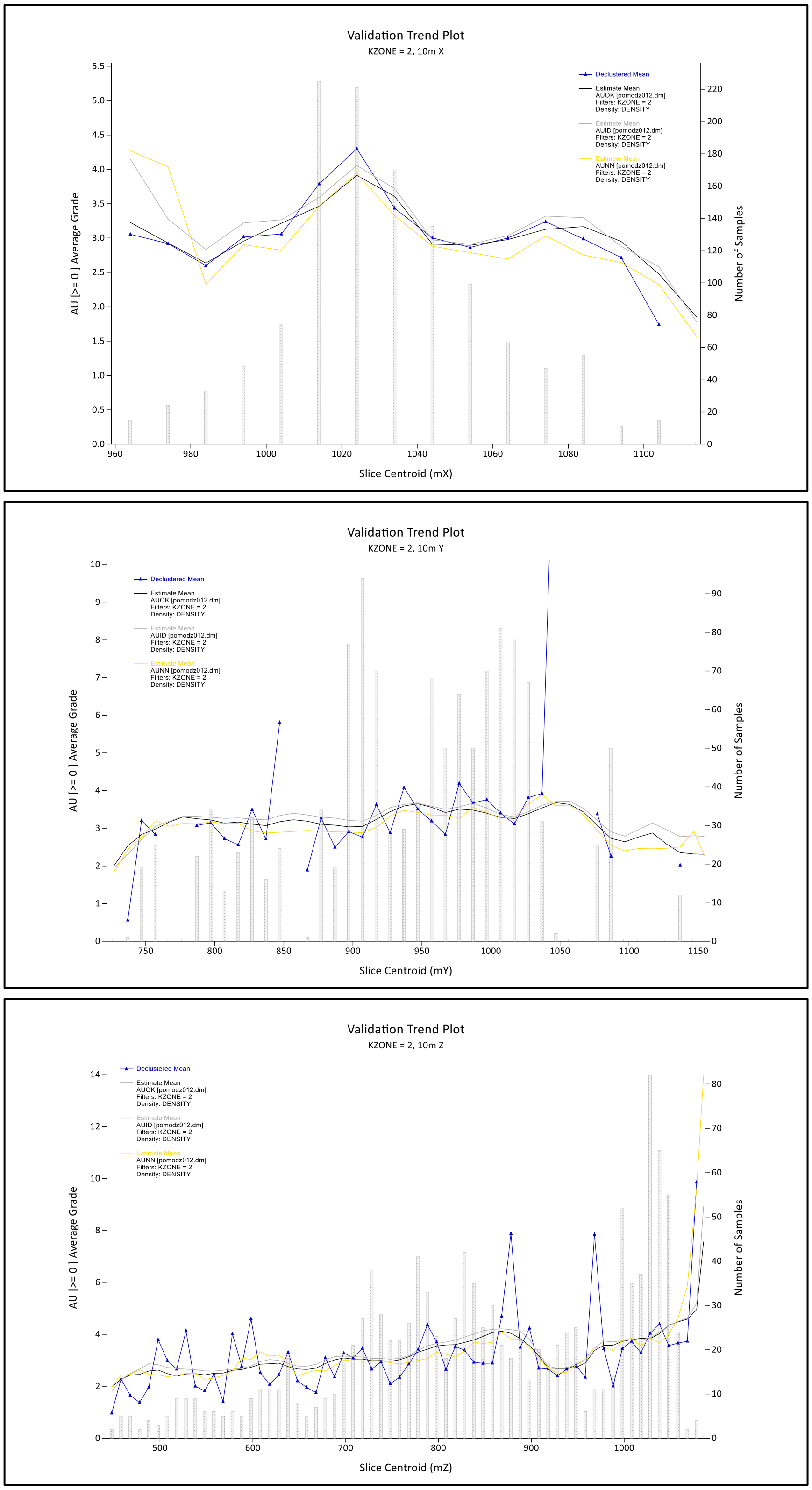
14.9.3 Swath Plots

As part of the validation process, the input composite samples were compared to the block model grades within a series of coordinates. The results of this were then displayed on graphs to check for visual discrepancies between grades. Figure 14-17 and Figure 14-18, show the results for the gold grades for the Marmato vein domain and MDZ high-grade domains respectively, based on all three principal directions. The graph shows the block model grades (black line), ID2 (grey), NN (yellow) and the declustered composite grades (blue line). A copy of the swath validation study per zone is contained in Appendix C.



Source: SRK, 2019

Figure 14-17: Example of Swath Analysis Used During Validation, Showing Vein Au (g/t)



Source: SRK, 2019

Figure 14-18: Example of Swath Analysis Used During Validation, Showing MDZ (High-Grade) Au (g/t)

14.10 Resource Classification

Block model quantities and grade estimates for the Project were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014).

Mineral Resource classification is typically a subjective concept. Industry best practices suggest that classification should consider the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim to integrate both concepts to delineate regular areas at similar resource classification.

Data quality, drillhole spacing and the interpreted continuity of grades controlled by the veins have allowed SRK to classify portions of the veins in the Measured, Indicated and Inferred Mineral Resource categories.

SRK's classification system differs from the previous estimate which used very broad classification as it was based on the assumption of mining adjustments based on increased knowledge of the deposit from on-going mine planning support.

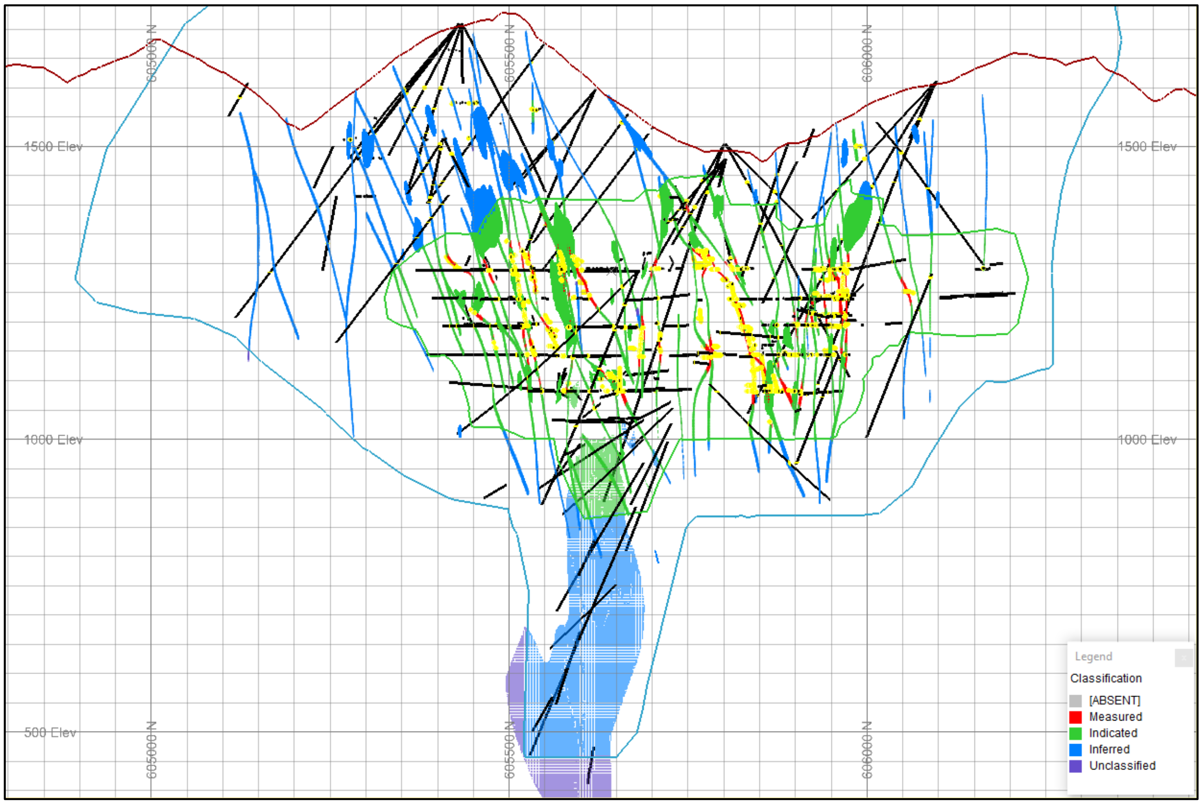
Measured: Measured Resources are limited to vein material within the current levels being mined by the Company and estimated within the first search volume which required a minimum of five composites and a maximum of 20 composites. These areas are considered to have strong geological knowledge as they have been traced both down-dip and along strike via mapping, plus underground channel samplings provide sufficient data populations to define internal grade variability.

Indicated: For the 2017 Mineral Resource estimate, SRK has delineated Indicated Mineral Resources at Marmato primarily between Level 16 to 21 currently in operation at Mineros Nacionales. Indicated Mineral Resources have been given at the following approximate data spacing, as a function of the confidence in the grade estimates and modelled variogram ranges. SRK has expanded the limits of the Indicated to also cover areas within Echandia where:

- Spacing of 50 m by 50 m (XY) existed from the nearest drillhole;
- Multiple holes were enabled to be used during the estimation process; and
- Support from both diamond drilling and channel sampling was present.

Inferred: In general, Inferred Mineral Resources have been limited to within areas of reasonable grade estimate quality and sufficient geological confidence, and are extended no further than 150 m from peripheral drilling on the basis of modelled variogram ranges.

A summary of the classification at Marmato is shown in Figure 14-19.



Source: SRK, 2019

Figure 14-19: Cross Section Showing Classification Systems at Marmato

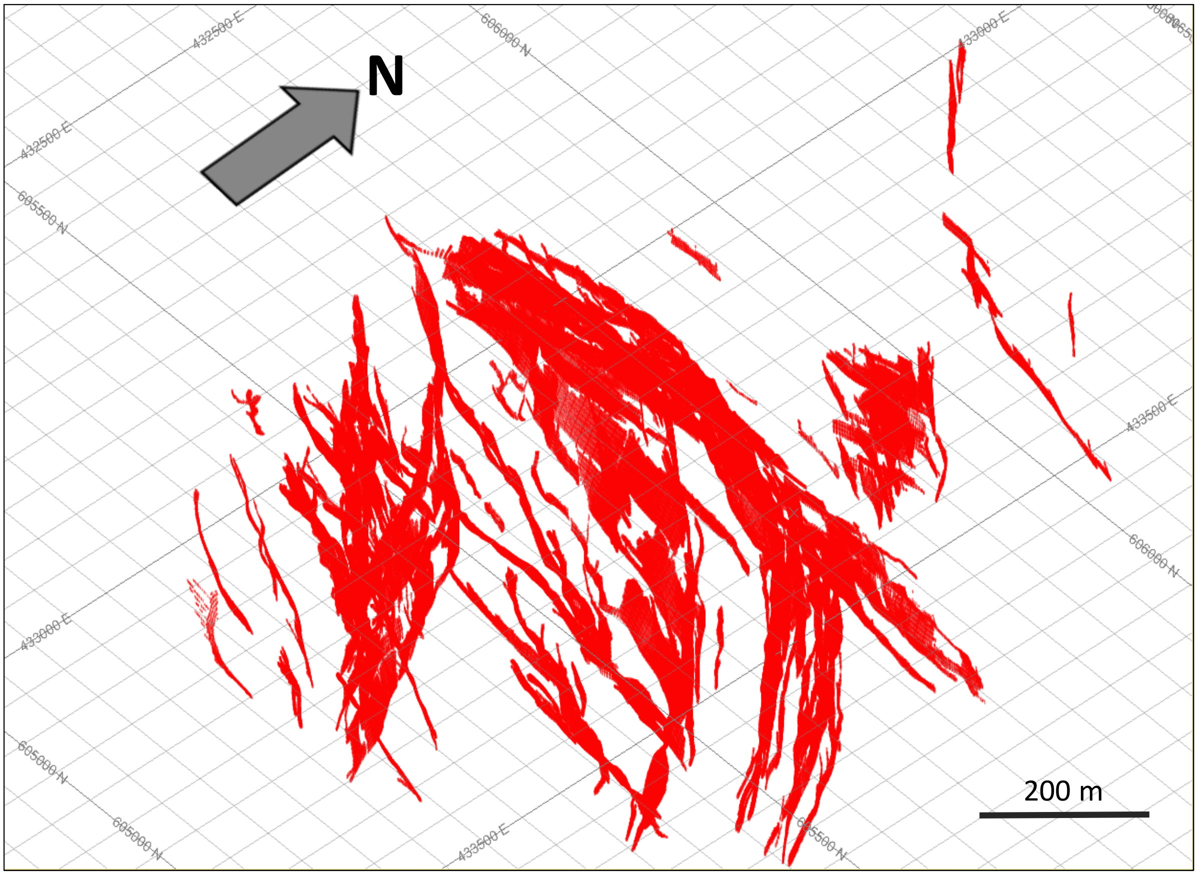
14.11 Depletion

To define the Mineral Resource SRK has created a block model to represent the depletion for the veins. In order to complete this task SRK has used a combination of AutoCAD™ polylines provided by GCM and generated Vulcan™ (.00t) files by projecting the strings in perpendicular to the strike. These wireframes have subsequently been used to copy out the assigned vein. The process is manual and labor intensive and requires each vein to be individually assessed.

This process may result in some errors of over or under depletion at the edges but given the size of the deposits is not considered to be material. Once the block model has been established the model has been combined with the final geological model to code all the blocks for depletion.

It is recommended that GCM work towards generating a 3D digital layout of the mine which has the depletion assigned to each of the main structures, but this will take time to initially set-up and was beyond the scope of the current Project.

An example of the resultant model used is shown in Figure 14-20.



Source: SRK, 2019

Figure 14-20: 3D View Example of Depletion Model Created by SRK for Marmato

14.12 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) defines a Mineral Resource as:

"(A) concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge".

The "reasonable prospects for eventual economic extraction" requirement generally imply that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate CoG taking into account extraction scenarios and processing recoveries. In order to meet this requirement, SRK considers that portions of the vein system to be amenable for underground mining.

To determine the potential for economic extraction SRK has used the following key assumptions for the costing but notes that the deposit has variable mining costs depending on the mining types resulting in a range of CoGs (Table 14-11). A metallurgical recovery of 95.0% Au has been assumed based on the current performance of the operating plant.

Table 14-11: Summary of Cut-Off Grade Assumptions at Marmato Based on Assumed Costs (Averaged for All Mining Styles)

	Units	Vein Mining 2017 Averaged CoG	Deeps Option (Long Hole) Averaged CoG
Assumptions			
Gold Price	US$/oz	$1,500	$1,500
Gold Price	US$/g	$48.23	$48.23
Au Recovery	%	95%	95%
Operating Costs			
Mining	US$/t mined	$48.00	$32.00
Processing	US$/t ore	$14.97	$14.35
Royalties	US$/t ore	$9.20	$9.20
G&A and Other	US$/t ore	$14.0	$5.00
Other	US$/t ore	$0.00	$0.00
Subtotal	**US$/t**	**$86.17**	**$60.55**
CoG - Head Grade	g/t	1.9	1.3

Source: SRK, 2019

SRK has defined the proportions of Mineral Resource to have potential for economic extraction for the Mineral Resource based on two separate CoGs, relating to the different mining methods involved. The initial cut-off is based on the mining of the veins using the current mining processes and assumed costs, with a second method (longhole) defined for mining the MDZ and potentially areas of wider porphyry mineralization in the upper levels.

The estimation domains have therefore been grouped for the Mineral Resource Statement into veins, porphyry and MDZ mineralization. The veins account for the veins, halos and splay material and have used a 1.9 g/t Au cut-off, while all other domains (grade-shell, deeps, porphyry) have used a lower cut-off of 1.3 g/t to account for the larger bulk mining methods involved.

The Mineral Resource Statement for the Project is shown in Table 14-12.

Table 14-12: SRK Mineral Resource Statement for the Marmato Project, Dated July 31, 2019

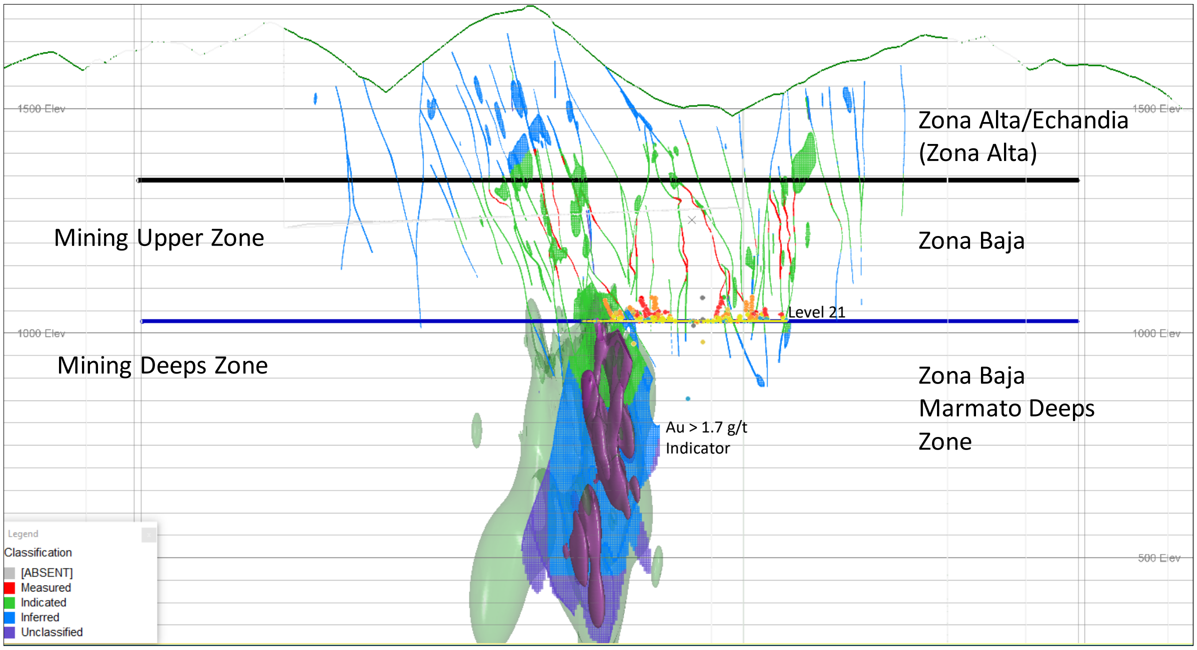
Category	Quantity	Grade		Metal	
		Au	Ag	Au	Ag
	Mt	gpt	gpt	000'oz	000'oz
Underground Vein**					
Measured	2.7	5.0	25.5	433	2,190
Indicated	10.8	4.5	20.5	1,579	7,133
Measured and Indicated	13.5	4.6	21.5	2,013	9,323
Inferred	8.6	4.2	19.3	1,154	5,330
Underground Porphyry***					
Measured					
Indicated	3.7	2.9	26.4	350	3,149
Measured and Indicated	3.7	2.9	26.4	350	3,149
Inferred	3.0	4.3	48.0	420	4,680
Underground MDZ***					
Measured					
Indicated	6.4	2.6	4.7	537	978
Measured and Indicated	6.4	2.6	4.7	537	978
Inferred	41.2	2.1	2.7	2,812	3,609
Underground Combined					
Measured	2.7	5.0	25.5	433	2,190
Indicated	21.0	3.7	16.7	2,466	11,260
Measured and Indicated	23.6	3.8	17.7	2,899	13,450
Inferred	52.9	2.6	8.0	4,387	13,619

* Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate.
** Vein and Porphyry mineral resources are reported at a CoG of 1.9 g/t. CoGs based on a price of US$1,500 per ounce of gold, suitable benchmarked technical and economic parameters and gold recoveries of 95 percent for underground resources, without considering revenues from other metal.
*** MDZ mineral resources are reported at a CoG of 1.3 g/t. CoGs based on a price of US$1,500 per ounce of gold, suitable benchmarked technical and economic parameters and gold recoveries of 95 percent for underground resources, without considering revenues from other metal in the pit shell.
Source: SRK, 2019

14.13 Mineral Resource Sensitivity

The proposed mining plan focuses on splitting the above Mineral Resources into three styles of mineralization within two distinct areas. The three styles of mineralization are based on the key geological types defined in the Mineral Resources of Veins, Porphyry and MDZ. The mining areas are split into Zona Alta and Zona Baja based on the presence of the existing mining infrastructure, with Zona Alta representing material above the existing mines, and Zona Baja as all material, including the Echandia Licences, below an elevation of 1,340 masl.

A summary of the breakdown for the Mineral Resources as used in the current study is shown in Figure 14-21, and with the tonnages and grades presented in Table 14-13.



Source: SRK, 2019

Figure 14-21: Summary Breakdown of Mining Areas for Study

Table 14-13: SRK Mineral Resource Statement for the Marmato Project, Dated July 31, 2019*, Breakdown by Mining Areas

Licence Grouping			Zona Alta			Zona Baja				Total
Type			Veins	Porphyry	*Total*	Veins	Porphyry	Deeps	*Total*	
Measured	Tonnes	(kt)	0.6		*0.6*	2.1			*2.1*	**2.7**
	Grade	Au (g/t)	5.6		*5.6*	4.9			*0.1*	**5.0**
		Ag (g/t)	33.2		*5.6*	23.2			*4.9*	**25.5**
	Au Metal	Au (koz)	109		*109*	325			*325*	**433**
		Ag (koz)	648		*648*	1,543			*1,543*	**2,190**
Indicated	Tonnes	(kt)	3.6	2.1	*5.7*	7.2	1.6	6.4	*15.2*	**21.0**
	Grade	Au (g/t)	4.6	3.1	*4.1*	4.5	2.7	2.6	*3.5*	**3.7**
		Ag (g/t)	25.3	38.9	*4.1*	18.1	10.1	4.7	*3.5*	**16.7**
	Au Metal	Au (koz)	542	210	*752*	1,037	140	537	*1,714*	**2,466**
		Ag (koz)	2,966	2,622	*5,588*	4,167	527	978	*5,672*	**11,260**
Measured & Indicated	Tonnes	(kt)	4.2	2.1	*6.3*	9.2	1.6	6.4	*17.3*	**23.6**
	Grade	Au (g/t)	4.8	3.1	*4.2*	4.6	2.7	2.6	*3.7*	**3.8**
		Ag (g/t)	4.8	3.1	*4.2*	4.6	2.7	2.6	*3.7*	**17.7**
	Au Metal	Au (koz)	650	210	*860*	1,362	140	537	*2,039*	**2,899**
		Ag (koz)	3,614	2,622	*6,236*	5,709	527	978	*7,214*	**13,450**
Inferred	Tonnes	(kt)	5.3	2.7	*7.9*	3.3	0.3	41.2	*44.9*	**52.9**
	Grade	Au (g/t)	4.1	4.5	*4.2*	4.4	3.1	2.1	*2.3*	**2.6**
		Ag (g/t)	4.1	4.5	*4.2*	4.4	3.1	2.1	*2.3*	**8.0**
	Au Metal	Au (koz)	688	386	*1,074*	466	34	2,812	*3,312*	**4,387**
		Ag (koz)	3,753	4,573	*8,326*	1,577	107	3,609	*5,293*	**13,619**

Source: SRK, 2019

The mineral resource given above is sensitive to the selection of the reporting CoG. To illustrate the sensitivity the block model quantities and grade estimates

The reader is cautioned that the figures presented in the tables should not be misconstrued with a Mineral Resource Statement. Figure 14-22 and Figure 14-23 are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade. The following tables (Table 14-14 to Table 14-19) have been split by the mineralization style and classification (Measured and Indicated have been combined for ease of reporting).

Table 14-14: Grade Tonnage Curve Measured and Indicated - Vein Domains (Group 1000 to 3000)

Cut-Off (Au g/t)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
0.00	19,077	3.6	18.9	2,235	11,565
0.50	18,701	3.7	19.1	2,232	11,495
1.00	17,472	3.9	19.8	2,201	11,126
1.20	16,704	4.0	20.2	2,173	10,831
1.30	16,249	4.1	20.3	2,155	10,624
1.50	15,342	4.3	20.7	2,114	10,227
1.70	14,433	4.5	21.1	2,068	9,792
1.80	13,966	4.5	21.3	2,041	9,553
1.90	13,483	4.6	21.5	2,013	9,323
2.00	13,021	4.7	21.7	1,984	9,092
2.20	12,095	4.9	22.1	1,921	8,592
2.50	10,894	5.2	22.7	1,830	7,935
2.70	10,094	5.4	23.0	1,763	7,477
3.00	9,058	5.7	23.6	1,669	6,866
3.50	7,407	6.3	24.6	1,497	5,865
4.00	6,094	6.8	25.4	1,339	4,986
4.50	5,028	7.4	26.2	1,193	4,240
5.00	4,203	7.9	26.9	1,068	3,629

Source: SRK, 2019

Table 14-15: Grade Tonnage Curve Measured and Indicated - Porphyry Domain (Group 4000)

Cut-Off (Au g/t)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
0.00	21,077	1.4	11.6	976	7,884
0.50	20,735	1.5	11.8	972	7,846
1.00	13,830	1.8	13.8	791	6,130
1.20	10,433	2.0	15.6	671	5,230
1.30	8,980	2.1	16.7	613	4,835
1.50	6,614	2.4	19.6	507	4,160
1.70	4,894	2.7	23.0	418	3,625
1.80	4,299	2.8	24.7	385	3,409
1.90	3,711	2.9	26.4	350	3,149
2.00	3,258	3.1	27.7	322	2,903
2.20	2,547	3.3	30.9	274	2,529
2.50	1,714	3.8	35.0	211	1,931
2.70	1,362	4.1	38.0	182	1,665
3.00	1,063	4.5	40.7	154	1,392
3.50	691	5.2	46.7	116	1,039
4.00	473	5.9	46.2	89	704
4.50	307	6.8	45.1	67	446
5.00	192	8.0	49.3	49	305

Source: SRK, 2019

Table 14-16: Grade Tonnage Curve Measured and Indicated - MDZ Domain (Group 5000)

Cut-Off (Au g/t)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
0.00	23,407	1.1	2.9	795	2,150
0.50	12,673	1.8	3.9	716	1,588
1.00	8,637	2.2	4.4	618	1,217
1.20	7,057	2.5	4.6	562	1,047
1.30	6,434	2.6	4.7	537	978
1.50	5,410	2.8	4.9	491	858
1.70	4,651	3.0	5.1	452	762
1.80	4,308	3.1	5.2	433	716
1.90	3,984	3.2	5.2	413	668
2.00	3,732	3.3	5.3	397	631
2.20	3,254	3.5	5.4	365	560
2.50	2,684	3.7	5.4	322	469
2.70	2,314	3.9	5.4	292	403
3.00	1,817	4.2	5.4	246	317
3.50	1,189	4.7	5.4	181	208
4.00	806	5.2	5.5	135	142
4.50	531	5.7	5.6	97	95
5.00	354	6.2	5.4	70	62

Source: SRK, 2019

Table 14-17: Grade Tonnage Curve Inferred - Vein Domains (Group 1000 - 3000)

Cut-Off (Au g/t)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
0.00	16,680	2.7	16.0	1,448	8,567
0.50	15,777	2.8	16.6	1,439	8,431
1.00	13,541	3.2	18.0	1,384	7,825
1.20	12,440	3.4	18.5	1,345	7,388
1.30	11,926	3.5	18.7	1,324	7,175
1.50	10,669	3.7	19.1	1,268	6,565
1.70	9,633	3.9	19.2	1,214	5,940
1.80	9,119	4.0	19.3	1,186	5,645
1.90	8,591	4.2	19.3	1,154	5,330
2.00	8,149	4.3	19.4	1,126	5,090
2.20	7,247	4.6	19.5	1,066	4,535
2.50	6,065	5.0	20.3	977	3,958
2.70	5,467	5.3	20.7	927	3,634
3.00	4,732	5.7	21.5	860	3,275
3.50	3,852	6.2	22.4	768	2,770
4.00	3,064	6.8	23.2	673	2,286
4.50	2,473	7.5	24.0	593	1,907
5.00	2,039	8.0	24.9	527	1,630

Source: SRK, 2019

Table 14-18: Grade Tonnage Curve Inferred- Porphyry Domain (Group 4000)

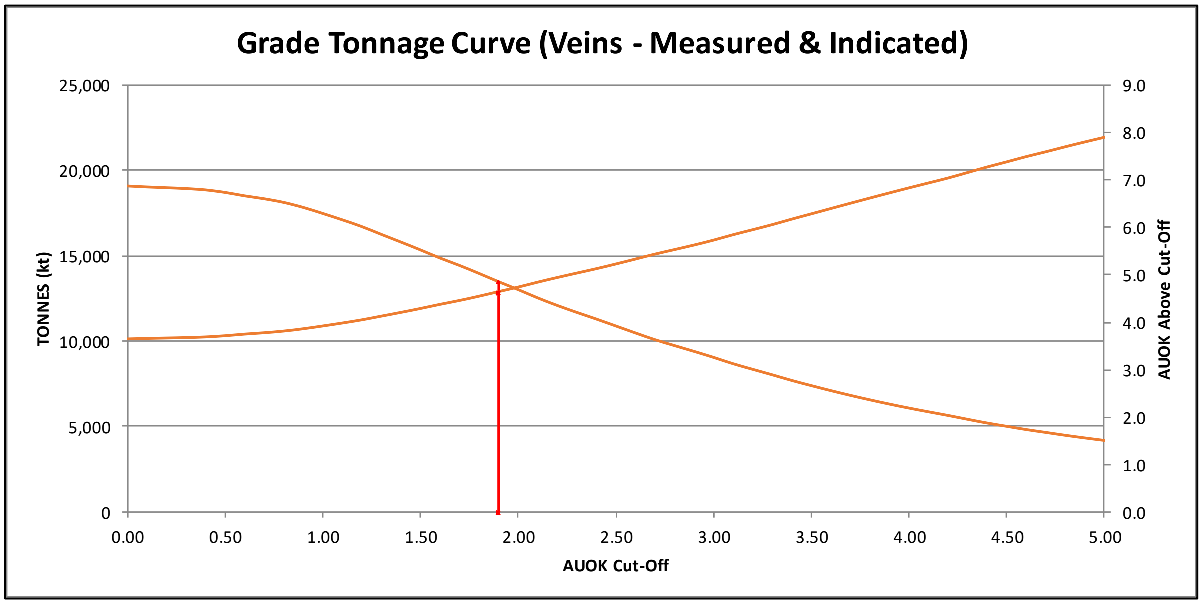
Cut-Off (Au g/t)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
0.00	14,791	1.8	19.0	847	9,034
0.50	14,539	1.8	19.3	844	9,004
1.00	10,111	2.3	24.5	732	7,969
1.20	7,880	2.6	29.2	654	7,400
1.30	6,981	2.8	32.0	617	7,174
1.50	5,304	3.2	39.2	542	6,681
1.70	3,906	3.8	40.9	471	5,135
1.80	3,438	4.0	44.3	445	4,893
1.90	3,030	4.3	48.0	420	4,680
2.00	2,694	4.6	49.9	399	4,325
2.20	2,172	5.2	47.1	364	3,290
2.50	1,725	6.0	44.0	331	2,441
2.70	1,572	6.3	46.0	318	2,325
3.00	1,410	6.7	49.3	303	2,234
3.50	1,139	7.5	56.1	275	2,055
4.00	936	8.3	64.8	250	1,950
4.50	773	9.2	74.4	228	1,850
5.00	674	9.8	82.2	213	1,780

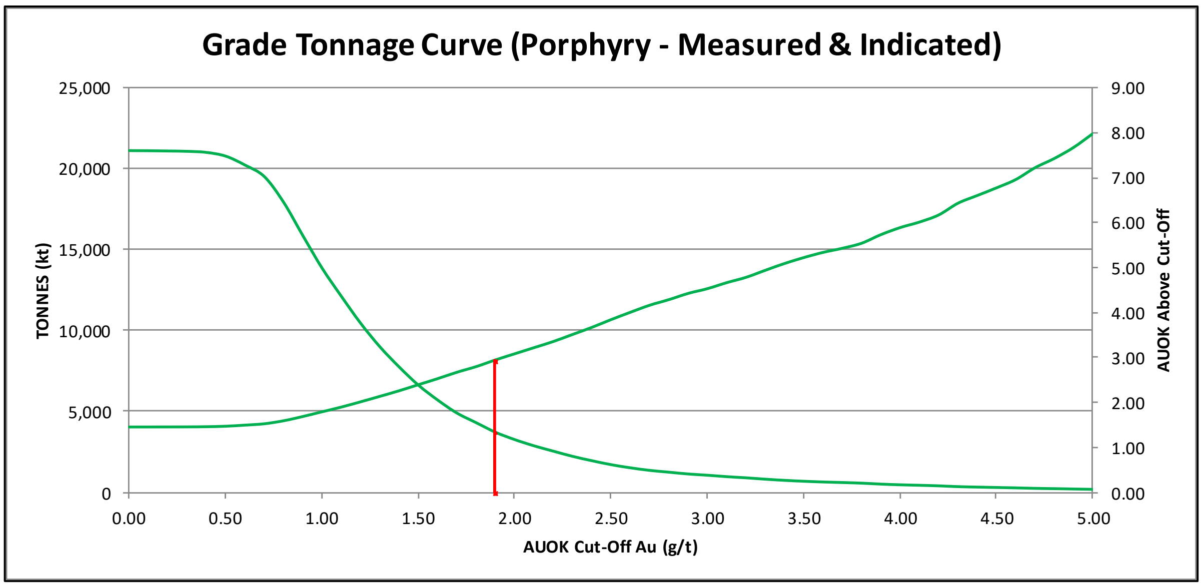
Source: SRK, 2019

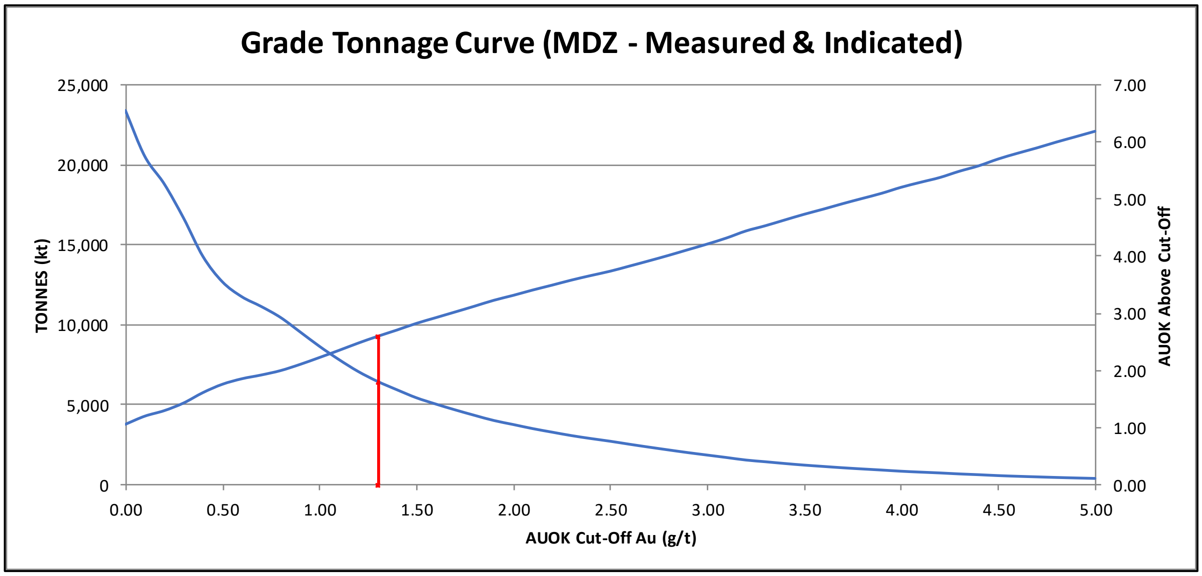
Table 14-19: Grade Tonnage Curve Inferred - MDZ Domain (Group 5000)

Cut-Off (Au g/t)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
0.00	122,708	1.1	2.0	4,418	7,931
0.50	84,035	1.5	2.3	4,067	6,169
1.00	59,142	1.8	2.5	3,473	4,765
1.20	46,957	2.0	2.6	3,042	3,981
1.30	41,237	2.1	2.7	2,812	3,609
1.50	31,542	2.3	2.9	2,378	2,979
1.70	24,753	2.5	3.2	2,029	2,515
1.80	22,005	2.6	3.3	1,875	2,313
1.90	19,555	2.8	3.4	1,729	2,123
2.00	17,350	2.9	3.5	1,591	1,945
2.20	13,656	3.1	3.7	1,343	1,627
2.50	9,878	3.3	4.0	1,058	1,272
2.70	7,772	3.5	4.2	882	1,049
3.00	5,289	3.9	4.6	656	778
3.50	2,684	4.5	4.9	386	425
4.00	1,528	5.0	5.4	248	267
4.50	960	5.5	5.9	171	183
5.00	598	6.0	6.4	116	124

Source: SRK, 2019

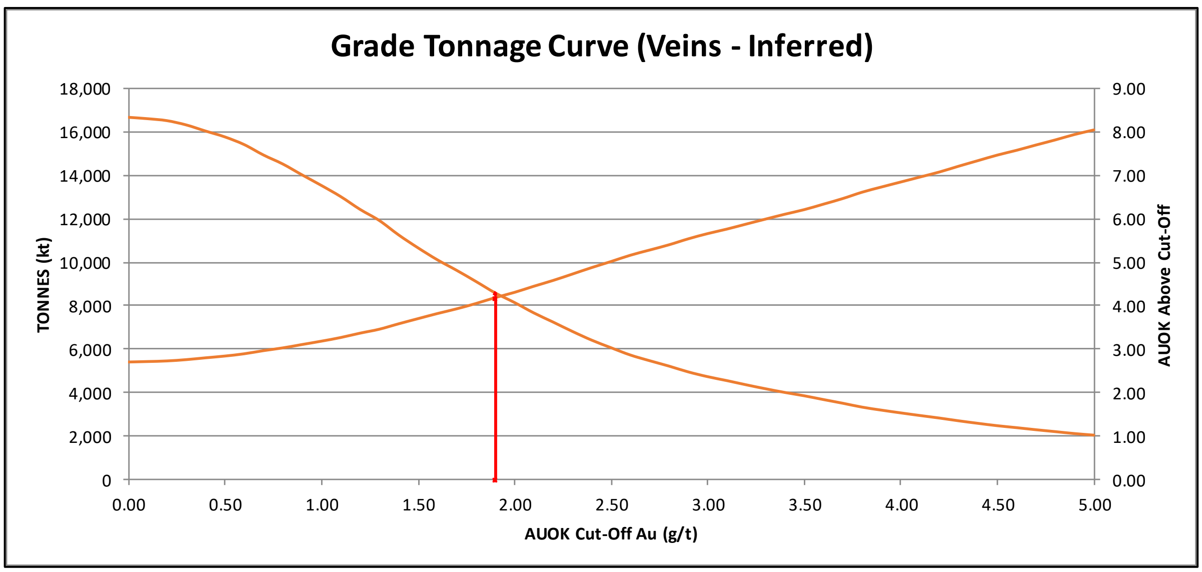

Grade Tonnage Curve (Veins - Measured & Indicated)
TONNES (kt)
AUOK Above Cut-Off
AUOK Cut-Off
25,000
20,000
15,000
10,000
5,000
0
9.0
8.0
7.0
6.0
5.0
4.0
3.0
2.0
1.0
0.0
0.00
0.50
1.00
1.50
2.00
2.50
3.00
3.50
4.00
4.50
5.00

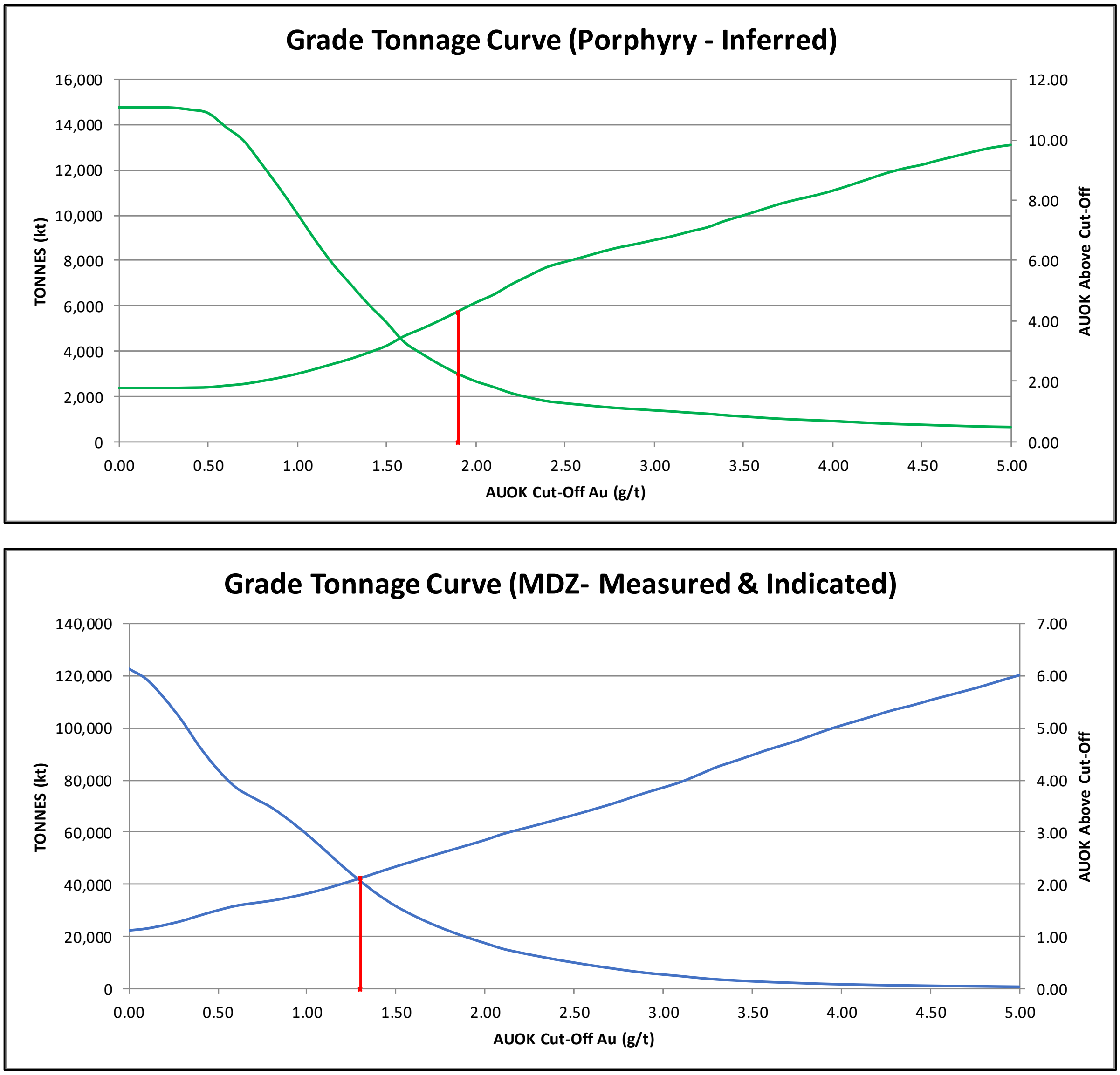

Grade Tonnage Curve (Porphyry - Measured & Indicated)
TONNES (kt)
AUOK Above Cut-Off
AUOK Cut-Off Au (g/t)
25,000
20,000
15,000
10,000
5,000
0
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
0.00
0.50
1.00
1.50
2.00
2.50
3.00
3.50
4.00
4.50
5.00



Source: SRK, 2019

Figure 14-22: Grade Tonnage Curves Showing Sensitivity to Changes in Cut-Off for Measured and Indicated Mineralized Material





Source: SRK, 2019

Figure 14-23: Grade Tonnage Curves Showing Sensitivity to Changes in Cut-Off for Inferred Mineralized aterial

14.14 Comparison to Previous Estimates

A comparison to the previous (2017) Mineral Resource Estimate for the Marmato Project demonstrates a limited overall change in the global estimates for the Project but some significant adjustment within the sub-categories of the Mineral Resource statement.

SRK urges caution that it will be important that these changes are clearly explained in any future press release to avoid potential issues of investor confidence.

The main changes in the Mineral Resource Statement can be defined as follows:

- Reduction in the Mineral Resources contained within the Porphyry domain within the current operating mine, which have reduced from 27.0 Mt at a grade of 2.1 g/t (1,861 koz) to 3.7 Mt at a grade of 2.9 g/t Au (350 koz), within the Indicated category, and 13.5 Mt at a grade of 1.8 g/t (786 koz) to 3.0 Mt at 4.3 g/t Au (420 koz), within the Inferred.
- A significant portion of the reduction has been due to an increase in the CoG used to define the mineral resource from 1.2 g/t to 1.9 g/t Au, which is more consistent with the current operating mining cut-off. This reduced the 2017 Mineral Resource from 27.0 Mt at a grade of 2.1 g/t (1,861 koz) to 12.4 Mt at a grade of 2.9 g/t Au (1,158 koz), within the Indicated category, and 13.5 Mt at a grade of 1.8 g/t (786 koz) to 4.2 Mt at 2.5 g/t Au (340 koz), within the Inferred.
- Using a direct comparison of the cut-offs the reduction in the contained metal is in the order of 800 koz from the Indicated portion and an increase of 80 koz in the Inferred category
- SRK attributes these differences to the following:
    - Reassignment of a portion of the porphyry pyrite domains to the MDZ based on new geological information; and
    - Application of a more restrictive sample selection criteria's during the geological modelling and estimation stage to only use information from drilling in grade estimation. This has eliminated potential upside of known mineralization within channel sampling taken from cross-cuts, but the database structure does not enable filters between short grade control channel sampling and more detailed exploration channel sampling of cross cuts and development. If SRK and GCM can define a suitable methodology during the coming months there remains potential to add additional material to this domain for the PFS. This may also provide additional feed to the current plant and therefore should be considered a priority for future studies.
- There has been an increase in the Indicated portion of the MDZ from 0.9 Mt at 2.0 g/t (60 koz), to 6.4 Mt at 2.6 g/t (537 koz) on the basis of infill drilling and better understanding of the geological controls;
- There has been an increase in the Inferred portion of the MDZ from 29.3 Mt at 2.3 g/t (2,142 koz), to 41.2 Mt at 2.1 g/t (2,812 koz);
- SRK highlights there was an increase in the CoG from 1.2 g/t to 1.3 g/t between the 2017 and 2019 Mineral Resource statements, which accounts for some of the increase in the grade;
- The overall increase within the MDZ is due to some of the reclassification of the porphyry material but primarily can be accounted for by the additional drilling completed to date.
- There is limited change within the veins models between 2017 and 2019, with the only change being a 10% reduction in the contained metal within the Inferred category, which may be a result of combining the Halo and vein mineralization domains during the geological modelling stage.

To provide GCM with a clear summary of the impact of these changes, SRK has completed a reconciliation between the 2017 and 2019 Mineral Resource statements. The 2017 Mineral Resource statement had a combined **41.2 Mt at a grade of 2.9 g/t Au for 3.90 Moz**, which has been reduced to **23.6 Mt at 3.8 g/t Au for 2.90 Moz,** when reviewing the Indicated portion of the Mineral Resource for all material. This represents a reduction of approximately 20% in the contained metal. SRK attributes the majority of this change due to the increase in the cut-off of the porphyry domain. When compared at the same CoGs the comparison of the Indicated Mineral Resources is **26.5 Mt at a grade of 3.7 g/t for 3.20 Moz Au**, which has reduced to **23.6 Mt at 3.8 g/t for 2.90 Moz Au** which is a reduction of approximately 9% in contained metal. Within the Inferred category there has been an increase in the Mineral Resources from **40.0 Mt at 2.8 g/t for 3.65 Moz Au** to **52.9 Mt at 2.6 g/t for 4.38 Moz Au,** which is an increase of 16%.

A direct comparison of the Mineral Resource statements using the 2019 CoG is shown in Table 14-20.

Table 14-20: Mineral Resource Comparison of 2017 vs. 2019 Roll Forward Numbers for Marmato[1]

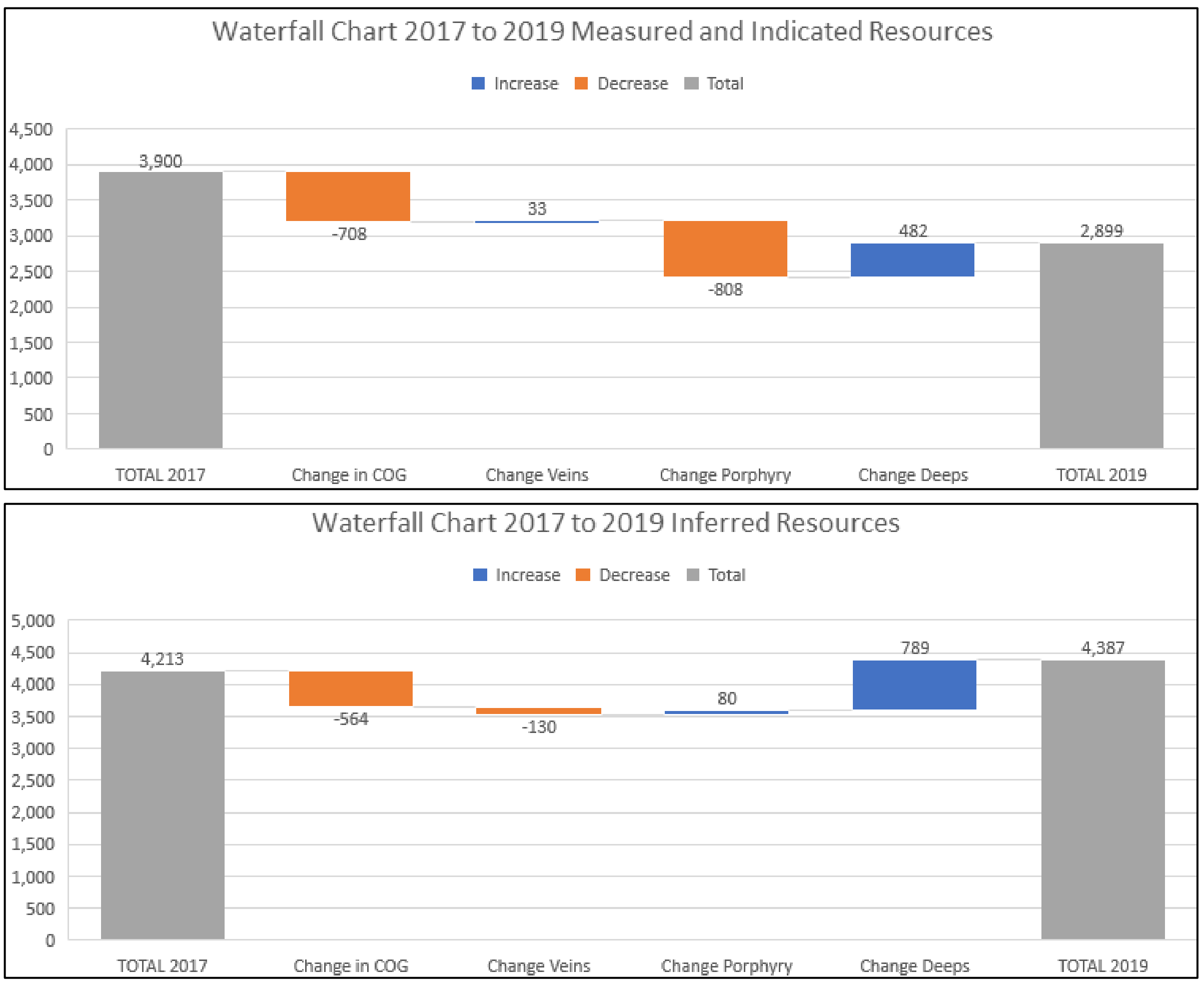
Category	Quantity	Grade		Metal		Quantity	Grade		Metal		Difference	
		Au	Ag	Au	Ag		Au	Ag	Au	Ag	Au	Ag
	Mt	gpt	gpt	000'oz	000'oz	Mt	gpt	gpt	000'oz	000'oz	000'oz	000'oz
Underground Vein(2)												
Measured	2.6	4.7	21.3	396	1,774	2.7	5.0	25.5	433	2,190	37	416
Indicated	10.7	4.6	22.3	1,583	7,663	10.8	4.5	20.5	1,579	7,133	-4	-530
Measured and Indicated	13.3	4.6	22.1	1,979	9,437	13.5	4.6	21.5	2,013	9,323	33	-114
Inferred	9.5	4.2	18.9	1,285	5,765	8.6	4.2	19.3	1,154	5,330	-130	-435
Underground Porphyry(3)												
Measured											0	0
Indicated	12.4	2.9	19.3	1,158	7,690	3.7	2.9	26.4	350	3,149	-808	-4,542
Measured and Indicated	12.4	2.9	19.3	1,158	7,690	3.7	2.9	26.4	350	3,149	-808	-4,542
Inferred	4.3	2.5	23.8	340	3,258	3.0	4.3	48.0	420	4,680	80	1,422
Underground Deeps(4)												
Measured											0	0
Indicated	0.8	2.2	8.3	55	211	6.4	2.6	4.7	537	978	482	768
Measured and Indicated	0.8	2.2	8.3	55	211	6.4	2.6	4.7	537	978	482	768
Inferred	26.3	2.4	2.9	2,024	2,410	41.2	2.1	2.7	2,812	3,609	789	1,199
Underground Combined												
Measured	2.6	4.7	21.3	396	1,774	2.7	5.0	25.5	433	2,190	37	416
Indicated	23.9	3.6	20.2	2,796	15,564	21.0	3.7	16.7	2,466	11,260	-330	-4,304
Measured and Indicated	26.5	3.7	20.3	3,192	17,338	23.6	3.8	17.7	2,899	13,450	-293	-3,888
Inferred	40.0	2.8	8.9	3,649	11,433	52.9	2.6	8.0	4,387	13,619	738	2,186

(1) The 2017 and 2019 mineral resources have been reported at a CoGs of 1.9 and 1.3 g/t. CoGs based on a price of US$1500 per ounce of gold, without considering revenues from other metal within a limiting pit shell.
(2) Vein mineral resources have been reported at a CoG of 1.9 g/t.
(3) Porphyry mineral resources are reported at a CoG of 1.9 g/t. The 2017 Mineral Resources statement has been restated at the higher cut-off to account for the significant change due to the increase in CoGs and allow direct comparison and gold recoveries of 90 percent for underground resources, without considering revenues from other metal.
(4) Deeps mineral resources are reported at a CoG of 1.3 g/t. CoGs based on a price of US$1,500 per ounce of gold, suitable benchmarked technical and economic parameters and gold recoveries of 95 percent for underground resources, without considering revenues from other metals.

SRK completed a high-level reconciliation of the 2017 to 2019 Mineral Resource Statements to show the key drivers in the differences. The 2019 Mineral Resource represents a number of changes in the defined Mineral Resource due to the following key factors:

- Increased CoGs used to report the porphyry domain from 1.2 g/t to 1.9 g/t Au, which is in line with the current CoGs used for the veins within the same areas. There is potential this could be reduced in the future with more detailed mining studies, but these areas have been excluded from the current study due to uncertainty in the database;
- Additional Deeps Mineralization (MDZ) has been defined in the 2019 model as a result of infill drilling. This has been the focus of the current study and represents the potential future of the Project.

SRK highlights that the current MDZ mineralization represents a notable change in the style of mineralization and considerations for mining methods at the Project. The largest gains in the Mineral Resources have been a result of the current drilling on the MDZ (Figure 14-24).



Source: SRK, 2019

Figure 14-24: Waterfall Chart Showing Changes from 2017 to 2018 Mineral Resources with Impact of Key Changes

14.15 Relevant Factors

SRK is not aware of any Environmental, permitting, legal, title, taxation marketing or other factors that could affect resources, however SRK considers that there may be some degree of sensitivity for the potential to extract mineralization based on the various mining domains.

At Marmato there are currently a number of small scale operations working in the upper portion of the deposit (Zona Alta). While the mineralization within these areas current falls within the concessions owned by Gran Colombia, due to social issues the GCM mining operation currently does not extract material from within these areas.

15 Mineral Reserve Estimate

No Mineral Reserves have been estimated for the Project.

16 Mining Methods

The Project has been in operation in various forms since the mid-1500s. Mineras Nacionales (MN) was awarded the contract for the concessions in 1989. The Project was originally developed as a 300 tpd underground project in 1997 and has expanded through the years to the existing 1,200 tpd capacity operation. Table 16-1 shows the production from 2015 to June 2019.

Table 16-1: 2015 to 2019* Production

Year	Unit	2015	2016	2017	2018	2019*
Tonnes of Mineral Processed	t	303,279	341,309	365,117	338,909	181,800
Recovered Gold	g	745,044	729,352	782,661	772,438	387,897
	oz	23,954	23,449	25,163	24,834	12,471

*First six months of 2019
Source: GCM, 2019

The mine is currently developed to the 1,025 m elevation. A transition is occurring from narrow vein mineralization to large porphyry mineralized areas (gold associated with pyrrhotite veinlets). For this PEA, there are three different mining methods, separated into three distinct zones.

- The first zone is the mineralized material between 1,025 m elevation to 1,350 m elevation, referred to as the Veins. This is the current mine and will be mined using the current conventional cut and fill stope method.
- The second zone is the wider porphyry material between 1,025 m elevation and 1,070 m elevation, referred to as Level 21. This material is on Level 21 with some development accesses already in place. A modified longhole stoping method will be used in this area. The stope size is 10 m wide by 15 m high with varying length of up to 26 m. 5 m dip pillars are left due to the use of unconsolidated hydraulic fill.
- The third zone is the porphyry material below 1,025 m elevation, referred to as MDZ. There is a sill pillar left in-situ between the MDZ and Level 21. The MDZ material is mined using a longhole stoping method with stope sizes that are 10 m wide by 25 m high, varying lengths of up to 20 m. The MDZ area is currently not developed.

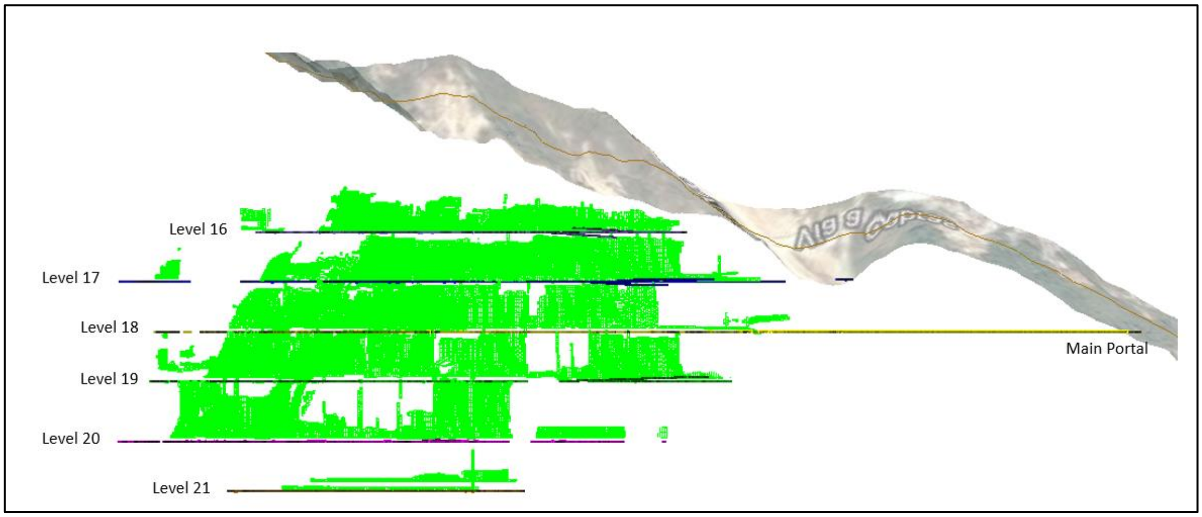
The first two zones are considered the Upper Mine, and the material is processed in the existing processing facility. The third zone is considered the MDZ and the material is envisioned to be sent to a new processing facility. Separate mine plans are presented for the Upper Mine area and MDZ area.

16.1 Current Mining Methods

GCM currently mines the vein material using a conventional cut and fill (CAF) stope mining technique that supplies approximately 950 tpd of material to a 1,200 tpd capacity processing facility. The CAF panels are typically 35 m long by 50 m high, with varying thicknesses depending on the vein. The panels are accessed from 2.2 m by 2.2 m haulage levels on the top and bottom. Raises are developed along the vein to break the panel into discreet mining stopes as well as to provide ventilation. Sub-levels are then driven horizontally along strike. Once the sublevel is opened, vertical holes are drilled up at a length of approximately 1.7 m to 2.3 m over the width of the vein. After blasting, the mineralized material is mucked using either slushers, bobcats or mini-scoops and loaded into trains and hauled out. Once mucking is complete, concrete walls are built on either end of the stope and the stope is filled with hydraulic fill. When the fill is sufficiently drained, the next slice of mineralized material can be mined.

16.1.1 Mine Layout

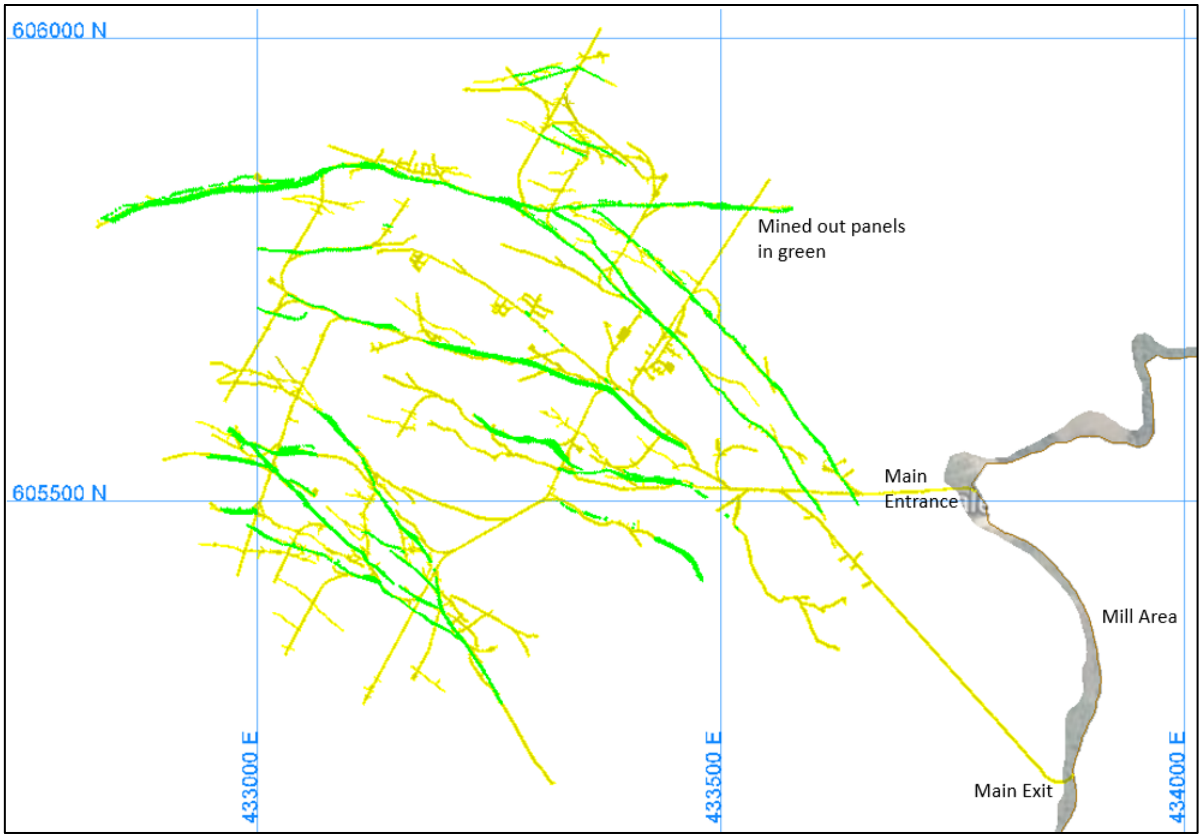
The current Zona Baja mining extends approximately 300 m vertically and approximately 900 m along the vein structure. The mine has been developed with level accesses proceeding horizontally from the main portal at the surface to horizontal cross cuts that provide access to the veins. There are currently six production levels, the highest being Level 16 and the lowest level being Level 21 (Figure 16-1). Each level is spaced 50 m apart vertically, with the exception of Level 20, which is 60 m from Level 19.



Source: SRK, 2019

Figure 16-1: Marmato Zona Baja Cross Section Looking NE with Active Levels

Level 18 is the main haulage level and the primary access for the mine and is shown in Figure 16-2. A track drift provides the main haulage for all material. The trains exit via the south portal, unload at the mill area and enter the mine via the north portal. Personnel and material enter via the north portal only. All levels can be reached via ladder-ways. Level 16 and Level 17 also have adit accesses from surface, mainly for ventilation. A rail decline from Level 18 to Level 19 provides the ability to move material and supplies between levels. Levels below 18 are accessed by an apique (inclined shaft) hoist and skip system that allows transport of material. There are other apiques that transport supplies to the lower levels.



Source: SRK, 2019

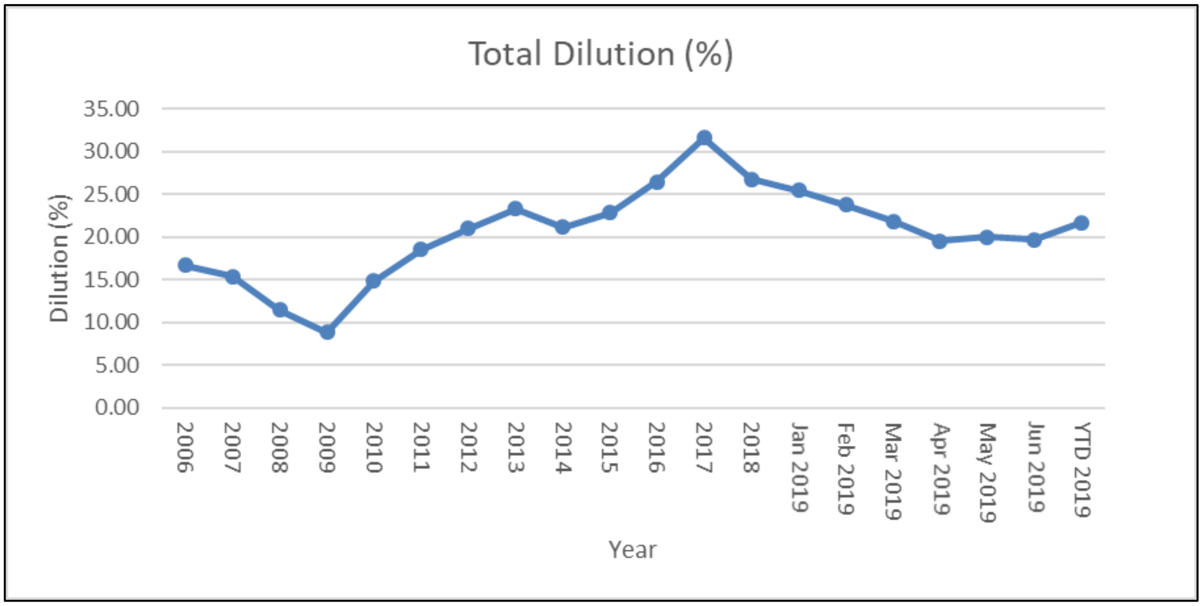
Figure 16-2: Marmato Level 18 with Main Haulage

16.1.2 Reconciliation

Reconciliation information was not provided at this time and is not carried out on a regular basis. SRK recommends that production information is reconciled to the mine plan on a regular basis to ensure the mine plan is predicting appropriate grades. Within a known mining area, the tonnes/grades mined should be compared to the tonnes/grades in the block model. If there are continuous discrepancies between the mined material and the predicted mine plan, modifications to the mine plan process should be made to more accurately predict future mining.

16.1.3 Dilution

Dilution data is provided by GCM and summarized in Figure 16-3. The total dilution has increased since 2006 to a high of 31.6% in 2017. GCM has introduced better drilling practices and training to control dilution. Since 2017, the total dilution has decreased with 2019 June year-to-date at 21.7%.



Source: GCM, 2019

Figure 16-3: Total Dilution

16.2 Geotechnical

SRK evaluated expected ground conditions and provided design parameters based on the available geotechnical data. This information included core logging, laboratory testing and field observations from a site visit completed in July 2019. This section summarizes the main findings.

16.2.1 Engineering Geology

GCM has conducted a preliminary field geotechnical characterization program that included field data collection and laboratory testing. SRK conducted a site visit to the Marmato mine operation in July 2019 to visually observe ground conditions and reviewed data collection procedures and protocols. In addition, SRK conducted traverse mapping on the 19 and 20 levels to validate the rock mass quality and strength parameters as determined by GCM.

At the time of this report, the geotechnical field investigation consisted of 35 drillholes (~8,800 m). Data from these holes have been used for rock mass characterization. The holes were positioned to examine rock mass fabric in and around the mineralized zone at different depths and orientations necessary to collect data on discontinuities. Holes were drilled at varying orientations into the hangingwall, footwall, and mineralized rock. The field investigation included geotechnical core logging and core sample collection for laboratory strength testing.

The laboratory testing program included 70 unconfined compression strength (UCS) tests, 36 triaxial compression strength (TCS) tests, and 14 direct shear strength (DSS) tests of rock joints. A set of 70 static and dynamic elastic moduli measurements, and 13 Brazilian tensile strength (BTS) tests were conducted. The laboratory tests were used to develop discontinuity shear strength parameters and estimates of the static and dynamic elastic properties.

During the July site visit, SRK observed that GCM has followed industry standards and the International Rock Mechanic Society (ISRM) suggested methods and the ASTM standards. In SRK's opinion, the data collected and the quantity of laboratory tests are consistent with the industry standards for a PEA level design.

Based on the GCM data collected and the laboratory tests provided to SRK, a statistical assessment was conducted to determine variations in rock mass strength parameters of the MDZ area of the Marmato Project. SRK has grouped all lithological units into one geotechnical domain for this PEA level study due to the lack of rock unit specific data. Table 16-2 summarizes the rock mass quality in terms of Rock Mass Rating (RMR_{89}) (Bieniawski, 1989), Barton's Q' system and Rock Quality Designation (RQD). SRK understands that there might be differences in rock mass quality between the different lithologies, and this should be evaluated for the next level of study.

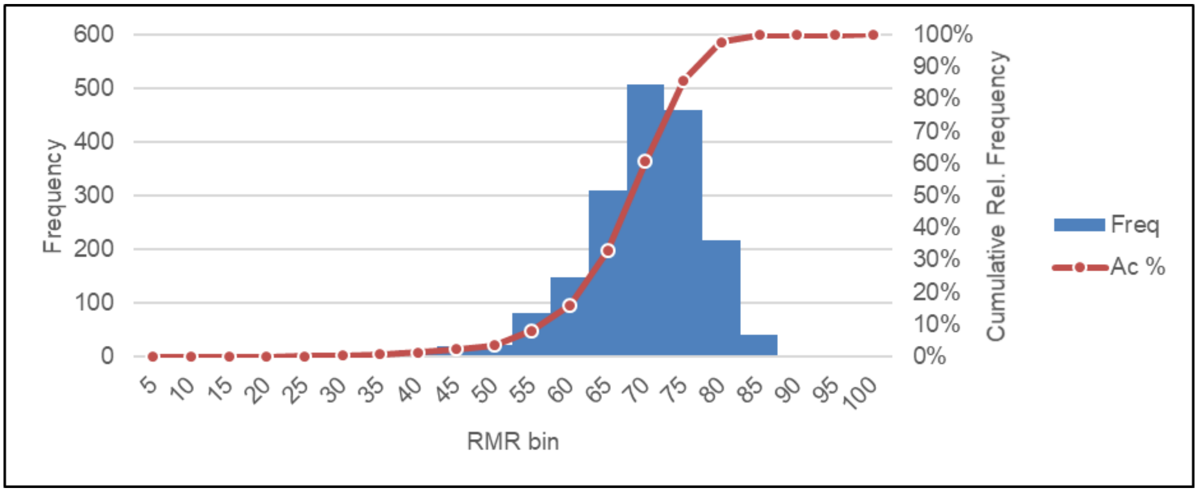
Preliminary indications are that the rock mass characteristics are similar between the upper vein mining areas with the lower MDZ mining areas and thus, for this PEA, SRK treated them as similar. These two mining areas plan to utilize different mining methods because of mineralization shape, however, the same geotechnical design methodologies apply given the similar nature of the rock mass.

Figure 16-4 shows the statistical distribution of RMR_{89} calculated values. SRK understands that more drillholes are in progress to support a PFS level design, and this additional data may introduce some changes to the rock mass distribution. It is reasonable to accept the present statistical distribution for the PEA level design as a representative indicator of the rock mass quality.

Table 16-2: Rock Mass Quality

Parameter	Wt. Avg	Std. dev	CV	25th Percentile	33rd Percentile	50th Percentile
RMR	66	10	0.15	63	65	68
Q'	4.67	6.07	1.30	1.56	1.95	2.72
RQD Rating	17.5	4.82	0.28	17	17	20
Spacing Rating	10.91	3.20	0.29	10	10	10
Strength Index Rating	8.95	3.21	0.36	7.00	7	7
Joint Condition Rating	12.85	3.10	0.24	11.00	12	13

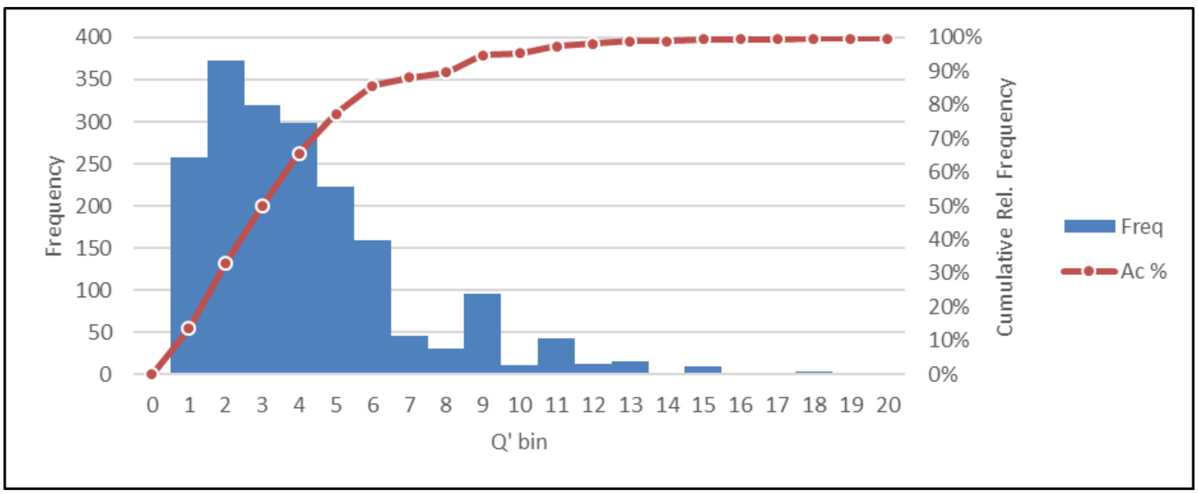
Source: SRK, 2019



Source: SRK, 2019

Figure 16-4: Rock Mass Rating Histogram for the MDZ

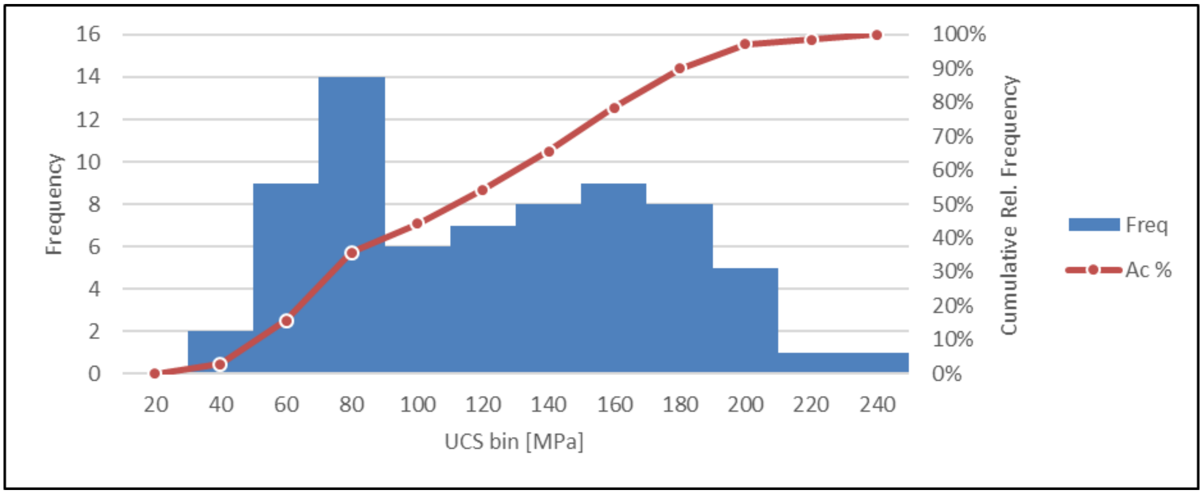
SRK has estimated the Barton's Q' index (Barton, 1974) for each run, which was used for the design of stopes, pillars and ground support. Figure 16-5 shows the statistical distribution of the Q' index. The average value for Q' corresponds to 4.2, while the mode value was 2.7. SRK considers that a Q' value of 2.7 is more adequate to represent the rock mass quality for the PEA design.



Source: SRK, 2019

Figure 16-5: Barton's Q' Histogram for the MDZ

Figure 16-6 shows a histogram of UCS values obtained from laboratory testing completed by GCM. SRK observes two different statistic populations in the intact rock strength testing results (i.e., below and above 100 MPa). SRK estimates that most of the observable rock is in the R5 range ISRM strength index (100-200 MPa) based on field observations and in-situ hammer tests completed during the site visit. This would correspond to the second population. The average UCS is 151 MPa and the 25th percentile strength is 128 MPa. SRK considers that a 25th percentile value is adequate for a PEA stability assessment, which has been used for the analyses. SRK recommends increasing the number of laboratory tests for a PFS level study.



Source: SRK, 2019

Figure 16-6: Uniaxial Compressive Strength Histogram

16.2.2 Vein Zone

A rock mass characterization program was completed by Knight Piésold in March 2012 (KP, 2012) for the previous open pit conceptual study. Data gathered from this program was from coreholes that adequately covered the upper vein area of the mine. SRK has used this data for the PEA level design of the upper vein areas.

The Knight Piésold drilling program included drilling 15 coreholes from the surface for a total of about 6,300 m of core. The geotechnical activities included geotechnical core logging, sample collection and laboratory testing. Core logging included recording RQD and RMR_{89}.

The Knight Piésold program also included a laboratory testing program consisting of 35 unconfined compression strength (UCS) tests, 30 triaxial confined compression strength (TCS) tests, and 33 Brazilian tensile strength (BTS) test measurements that were performed by the Rock Mechanics Laboratory of the Robert M. Buchan Department of Mining, Queen's University at Kingston. SRK has reviewed the raw database, and is of the opinion that the laboratory tests and characterization logging have been conducted in accordance with industry standards and results are appropriate for PEA geotechnical design for the upper vein region.

Knight Piésold reported an average RQD of 85, which suggests that the intact rock can be considered massive/blocky. The RMR_{89} ranges between 60 to 70, with an average of 65 and standard deviation of about 3. This suggests that rock quality can be expected to be Good Quality (Class II). The RMR_{89} ranges of 60 to 70 represents the 30% and 70% percentile of the data.

The vein zone area has been mainly mined using conventional cut and fill mining method, and in some limited areas using open stoping mining methods. Ground conditions are routinely assessed to address ground support needs. Typical ground conditions include:

- Fair Quality – Rock type III - (RMR_{89}: 40-60);
- Poor Quality – Rock type IV (RMR_{89}: 20-40); and
- Very Poor Quality – Rock type V (RMR_{89}: <20).

Knight Piésold observed that most of the reduced rock quality is the consequence of joint weathering, especially in fault zones that includes veins and their cross jointing.

At the time of the geotechnical site visit, SRK did not have access to all the mine areas in this zone and could not verify the ground conditions. The stope optimization shown in Section 16.3.3 proposed by SRK are based on limited data. However, in wide vein areas it is assumed for this PEA level study that the same analyses are applicable.

SRK has identified an opportunity for optimizing the stope design at a PFS level, once more geotechnical data becomes available. SRK understands that GCM is currently completing a geotechnical investigation and database. SRK recommends for PFS level design that a numerical stability model be conducted for verifying the proposed optimized design and a review of the interaction between the Veins and MDZ mining be carried out. SRK also recommends carrying out stress measurements to improve the confidence in the numerical model results.

16.2.3 MDZ Stope Stability

For the MDZ area, a stope stability assessment was conducted according to the Matthews' empirical method. The stability graph method (Mathews et al., 1981; Potvin, 1988; Potvin and Milne, 1992 and Nickson, 1992) is an empirical relationship that has been developed for open stope design based on the depth of mining, rock mass quality and stope span. The stability graph is a plot of stope Hydraulic Radius (HR) versus Modified Stability Number (N'). Stopes plotted on the graph are classified as stable, unstable, stable with support or caved. A stable stope will exhibit little or no wall deterioration during its mining cycle, while an unstable stope will exhibit limited wall deterioration (30% of face area) and a caved stope will exhibit unacceptable failure (Hutchinson and Diederichs, 1996). The stability number, N', is defined as:

$$N' = Q' x A x B x C$$

Where:

- Q' = the modified Q rock mass classification;
- A = the rock stress factor (value between 0.1 to 1.0);
- B = the joint orientation adjustment factor (value between 0.2 to 1.0); and
- C = the gravity adjustment factor (values between 0 and 10).

The Q' is a modified version of the Q rock mass classification formula that has the stress reduction factor (SRF) equal to one, which represents a moderately clamped but not overstressed rock mass and the joint water reduction (Jw) factor equal to one, representing a dry excavation for underground stopes. Therefore, the Q' represents the inherent characteristic of the rock mass (block size and joint properties). The rock mass rating that has been calculated for the MDZ has been based on the RMR rock mass classification system. The groundwater factor in the RMR calculation assumed dry conditions (i.e., depressurized)

The HR used in the stability graph is defined as the stope area/stope perimeter. The HR is calculated for the stope face (width = strike) and the end walls (width = hangingwall to footwall) and is defined by the following equation:

$$HR = \text{stope width x stope height}/2(\text{stope width + stope height})$$

The rock stress factor (A) replaces the SRF factor calculated in Q. The A factor is the ratio of intact rock strength (UCS) to induced stress (stress acting parallel to the exposed stope wall or roof). The UCS values are based on the laboratory tests discussed before.

The vertical stress is estimated to be the weight of the overlying rock (0.027 MN/m^3) which is defined as follows:

$\sigma_V = 0.027\ MN/m^3 \times Z$ (vertical depth in m).

The horizontal stress values (σ_{H1} and σ_{H2}) have been estimated to be equal and a ratio of the vertical stress using the following relationship:

$\sigma_{H1} = 1.0 \times \sigma_V$ and
$\sigma_{H2} = 1.0 \times \sigma_V$.

Due to the range of vertical depth of mining, three stress ranges have been analyzed assuming depths of 200 m, 400 m and 600 m. The deepest is the approximate depth to the lowest level in the mine.

Based on Matthews Method, Table 16-3, summarizes the Mathews approach applied to the MDZ area.

Table 16-3: Stope Design Parameters Marmato Deep

Area	Depth (m)	Q' Index	Factor			Stability Number N'	H (m)	W (m)	L (m)
			A	B	C				
Back	200	2.74	0.74	0.20	2.00	0.80	25	10	20
Sidewall		2.74	0.74	0.20	8.00	3.20			
Hangingwall		2.74	0.74	0.20	8.00	3.20			
Footwall		2.74	0.74	0.20	8.00	3.20			
Back	400	2.74	0.30	0.20	2.00	0.3			
Sidewall		2.74	0.30	0.20	8.00	1.3			
Hangingwall		2.74	0.30	0.20	8.00	1.3			
Footwall		2.74	0.30	0.20	8.00	1.3			
Back	600	2.74	0.15	0.20	2.00	0.2			
Sidewall		2.74	0.15	0.20	8.00	0.7			
Hangingwall		2.74	0.15	0.20	8.00	0.7			
Footwall		2.74	0.15	0.20	8.00	0.7			

A = the rock stress factor (value between 0.1 to 1.0);
B = the joint orientation adjustment factor (value between 0.2 to 1.0); and
C = the gravity adjustment factor (values between 0 and 10).
Source: SRK, 2019

Dilution into the stopes has been estimated using an empirical design chart for ELOS based on work completed by Pakalnis (1986). For the stope parameters specified above, the ELOS chart estimates about 0.35 m of sloughing. Assuming dilution in the near vertical stopes comes from both the hangingwall and footwall rock.

SRK considers that an empirical method is acceptable for a PEA level study. However, for further stages of the Project, SRK recommends that a numerical model is built to reevaluate stope stability for the different portions of the mine. This approach would account for the variable stress conditions through the mining sequence, and the effect of backfill on stability. The proposed stope design is valid for a PEA level only and cannot be applied for construction.

16.2.4 MDZ Sill Pillar Design

SRK considered two different empirical methodologies to determine adequate sill pillar dimensions, which are detailed below.

Scaled Span Method (1990)

The characterization data has been reviewed for the rock mass above the mine. The top of mining is planned to begin at about 400 m below ground. The sill pillar stability assessment was conducted using the empirical Scaled Span Method. The concept for a scaled span for defining crown pillar stability was proposed by Golder in a CANMET report (Golder, 1990). The concept was formulated based on back-analysis of historic crown pillar failure and review of precedence experience.

For any given geometry and rock quality, the crown pillar stability is estimated empirically by plotting rock mass quality, Q, against a scaled pillar span index, C_S, according to the following equation:

$$C_S = S\left\{\frac{\gamma}{t(1+S_R)(1-0.4*\cos(dip))}\right\}^{0.5}$$

Where:

S = sill pillar span;
γ = density of rock mass (t/m^3);
t = thickness of crown pillar;
S_R = span ratio = S/L (crown pillar span / crown pillar length); and
Dip = dip of the orebody or foliation in degrees.

The effects of groundwater and clamping stresses are included in the determination of rock mass quality (Carter, 1992).

With the rock quality, Q, known, an estimate of the maximum stable span can be made by reference to the critical span, S_C, as follows:

$$S_c = 3.3 * Q^{0.43} * \sinh Q^{0.0016}$$

The sill pillar can be considered stable when $C_s > S_c$, assuming the crown pillar has not been artificially reinforced by bolting, cabling or tight backfilling of stopes. The ratio of $S_c/C_s = F_c$ can be considered equivalent to a pillar stability Factor of Safety (FOS).

For the MDZ area the assumed values include the following:

- Sill Pillar span, S = 10 m;

- Density, g = 2.7 ton/m^3;
- Sill Pillar length, L =20;
- Dip structural, Dip =30;
- Barton Q = 2.7;
- Span Ratio, Sr = 0.5;
- Critical Span, Sc = 5.3; and
- Span Index, Cs = 3.5.

According to this empirical method, the required sill pillar thickness is in the 10 to 13 m range, assuming short-term mine exposure and that the stopes will be tightly backfilled.

Lunder and Pakalnis Empirical Method (1997)

SRK completed an additional assessment for sill pillar thickness based on the work completed by Lunder and Pakalnis (1997), who compiled an extensive database of hard-rock pillar failures, including 178 case histories. While many of these pillars were rib or sill pillars from steeply dipping ore bodies, most of them were in the Canadian Shield. The authors developed an empirical formula based on these case studies. The proposed 'Confinement Formula" for estimating hard rock pillar strength was defined as follows:

$$\sigma_{ps} = 0.44 * UCS * (0.68 + 0.52 * kappa)$$

where: UCS = the unconfined compressive strength of intact pillar material (MPa),

kappa = the mine pillar fiction term, which is defined through the following equation:

$$kappa = \tan\left[\cos^{-1}\frac{1 - c_{pav}}{1 + c_{pav}}\right]$$

The average pillar confinement (c_{pav}) is defined as the ratio of the average minor to the average major principal stress at the mid-height of a pillar. The average pillar confinement can be estimated with the following expression:

$$c_{pav} = 0.46 * \left[log\left(\frac{W_p}{h} + 0.75\right)\right]^{\frac{1.4}{\frac{W_p}{h}}}$$

where: W_p = Pillar width

h = Pillar height

The parameters used by SRK for the calculation of the sill pillar thickness are listed below:

- Sill pillar depth = 500 m;
- Sill Pillar span = 10 m;
- K ratio = 1.0;
- Density = 2.7 ton/m^3; and
- UCS = 128 MPa.

With this methodology, the required sill pillar thickness was estimated in the 9.5 to 10.5 m range, provided the stopes will be tightly backfilled. Considering that the two approaches resulted in similar values, SRK determined that a sill pillar with a thickness of 10 m is adequate with the available information.

SRK recommends that additional data is collected to confirm the input parameters used in the analyses and validate the PEA recommended pillar dimensions. Additionally, a numerical model should be built for future studies to properly assess the effect of the mining sequence and stress redistribution on pillar stability.

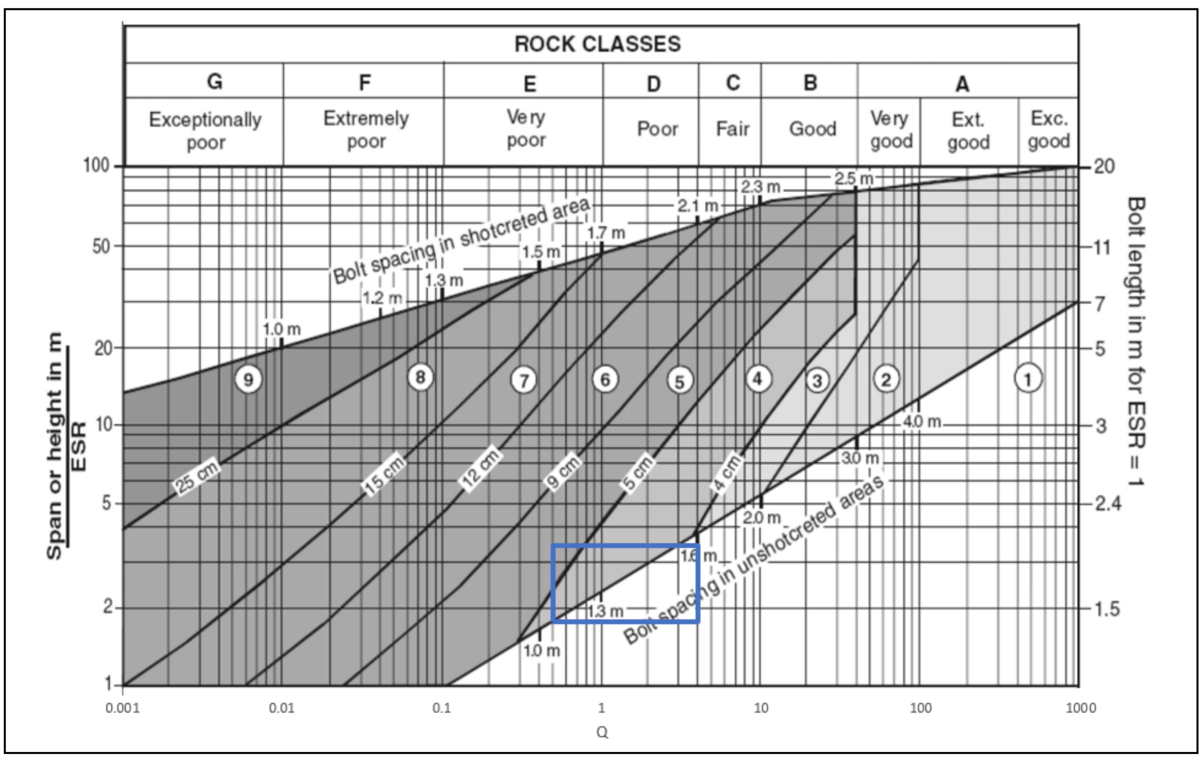
16.2.5 Ground Support

Ground support requirements were estimated using empirical support charts developed by Barton (1974). The method relates the rock mass quality (*Q*) to the equivalent dimension of the excavation (*De*). *De* is the ratio of the excavation width (*D*) to the excavation support ratio (*ESR*) index. The *ESR* value required depends on the use for t the excavation. Values ranges from 0.8 rating for underground public storage facilities to 3 to 5 rating for temporary mining excavations.

The equivalent dimension is determined from the formula:

$$D_e = \frac{D}{ESR}$$

Table 16-4 shows the ground support requirements as a function of Barton's Q' index and the equivalent dimension. Estimated support categories for the various development types are shown in Figure 16-7.



Source: SRK, 2019 Adopted from Barton

Figure 16-7: Estimated Support Categories Based on Q Index

Table 16-4: Summary of Support Requirements by Excavation Type

Excavation Type	ESR	De [m]	Support Category
Ramp	1.0	3.5	Q<0.8: Category 5 0.8<Q<3: Category 4 Q>3: Category 1
Access tunnels	1.3	2.7	Q<0.6: Category 5 0.6<Q<2: Category 4 Q>2: Category 1
Permanent mine openings, drifts	1.6	2.2	Q<0.8: Category 4 Q>0.8: Category 1
Permanent mine openings, drifts	2.0	1.8	Q<0.5: Category 4 Q>0.5: Category 1

Note: The excavation dimensions were assumed to be 3.5 by 3.5 meters.
Source: SRK, 2019

The support categories shown in the previous tables are summarized below:

- Support category 1: Unsupported;
- Support category 4: Systematic bolting, unreinforced shotcrete 4 to 10 cm; and
- Support category 5: Fiber reinforced shotcrete and bolting, 5 to 9 cm.

16.3 Upper Mine Mining

The following sections outline mining in the Veins and Level 21 area. The material is between elevations of 1,025 m and 1,350 m.

16.3.1 Cut-off Grade Calculations

SRK was provided with the cost structure for GCM for their current operations. This information was summarized to calculate an AuEq CoG for veins material as shown in Table 16-5. AuEq is calculated using the following formula:

$$Aueq = \frac{\frac{tonnage * Ag\ grade * Ag\ recovery * Ag\ price}{Au\ price} + tonnage * Au\ grade * Au\ recovery}{tonnage * Au\ recovery}$$

Table 16-5: Cut-Off Grade Parameters for Veins Material

Parameter	Amount	Unit
Mining Cost	48.11	US$/t
Process cost	14.97	US$/t
G&A	14.00	US$/t
Royalties	8.10	US$/t
Total Cost*	85.18	US$/t
Gold Price	1,300.00	US$/oz
Silver Price	17.00	US$/oz
Gold Recovery	87.00	%
Silver Recovery	40.00	%
AuEq CoG	2.34	g/t

* Cost is based on GCM 2018 costs
Source: SRK, 2019

From these vein material costs, SRK developed costs and cut-off information for the Level 21 mining method as shown in Table 16-6.

Table 16-6: Cut-Off Grade Parameters for Level 21 Material

Parameter	Amount	Unit
Mining Cost	42.00	US$/t
Process cost	13.08	US$/t
G&A	14.00	US$/t
Royalties	8.10	US$/t
Total Cost	77.18	US$/t
Gold Price	1,300.00	US$/oz
Silver Price	17.00	US$/oz
Gold Recovery	90.00	%
Silver Recovery	40.00	%
AuEq CoG	2.05	g/t

Source: SRK, 2019

16.3.2 Hydrogeology and Mine Dewatering

The mine area is located in the hydrogeological regional area of Magdalena Cauca. The region is comprised of igneous and metamorphic rocks with limited groundwater storage capacity and hydraulic conductivity (IDEAM, 2013). Specifically, the Marmato mine area is connected to an emplacement of dacite and andesite porphyry stocks of the Late Miocene age and is hosted by a sheeted pyrite veinlet system associated with intermediate argillic and propylitic alteration in the porphyry intrusions. The upper portion of the Marmato district (Zona Alta) has been historically mined by artisanal mines. The lower portion of the Marmato district (Zona Baja) is located below 1,207 m amsl to 1,298 m amsl (Wilson, 2010).

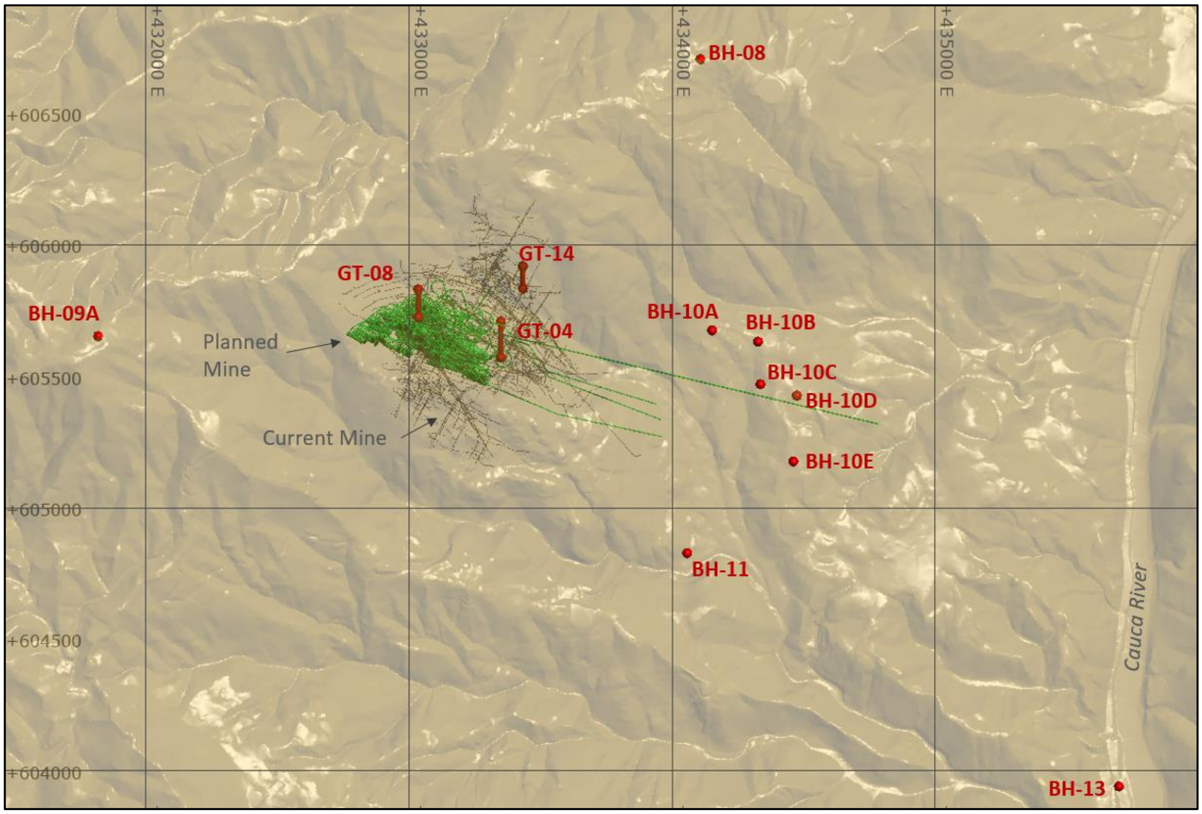
Hydrogeological Field Data

A hydrogeological field investigation was completed by Knight Piésold from October 2011 to March 2012 (Knight Piésold, 2012). The investigation included hydraulic tests performed in the porphyry intrusion units and the construction of three piezometers completed from the underground mine and 11 piezometers completed from the ground surface (Table 16-7 and Figure 16-8).

Table 16-7: 2012 Campaign Piezometers and Boreholes

Borehole ID	ID 2	Easting (m)	Northing (m)	Elevation (m amsl)	Total Depth (m)	Dip (Degrees)	Diameter	Water level (m amsl)	Date
BH-06 [1]	MG-026	432,213	607,734	1885	100.1	90	HQ	1,855.6	3/21/2012
BH-07 [1]	MG-027	435,651	607,603	722	76	90	HQ	691.4	3/20/2012
BH-08	MG-021	434,107	606,706	1457	72.5	90	HQ	1,431.2	3/21/2012
BH-09A	MG-024	431,818	605,653	2083	301.9	90	HQ/NQ	2,021.8	3/25/2012
BH-10A	-	434,151	605,675	1201	30.5	90	HQ	dry	2012
BH-10B	-	434,326	605,632	1157	30.5	90	HQ	dry	2012
BH-10C	-	434,335	605,470	1120	29.5	90	HQ	dry	2012
BH-10D	-	434,473	605,427	1112	30.5	90	HQ	dry	2012
BH-10E	MG-016	434,460	605,176	1124	93	90	HQ	1,028.5	3/20/2012
BH-11	MG-022	434,056	604,828	1050	45.4	90	HQ	1,043.1	3/20/2012
BH-13	MG-028	435,698	603,941	726	100.3	90	HQ/NQ	660.8	3/24/2012
BH-14 [1]	MG-023	434,098	603,327	1102	74.1	90	HQ/NQ	1,044.6	3/20/2012
GT-04	MG-015	433,348	605,572	1186.9	301.5	63	HQ	1,099.6	3/16/2012
GT-08	MG-012	433,034	605,725	1192	250.6	65	HQ	1,092.0	3/16/2012
GT-14	MG-014	433,432	605,834	1188.4	200.5	65	HQ	1,089.8	3/16/2012
KP-TSF-11-03 [1]	MG-029	437,779	602,194	935	Unknown	90	Unknown	889.3	3/18/2012
KP-TSF-11-05 [1]	MG-030	440,343	601,816	1037	Unknown	90	Unknown	1,004.7	3/18/2012

Source: Knight Piésold, 2012
(1) Piezometer located out of area of interest)



Source: Knight Piésold, 2012

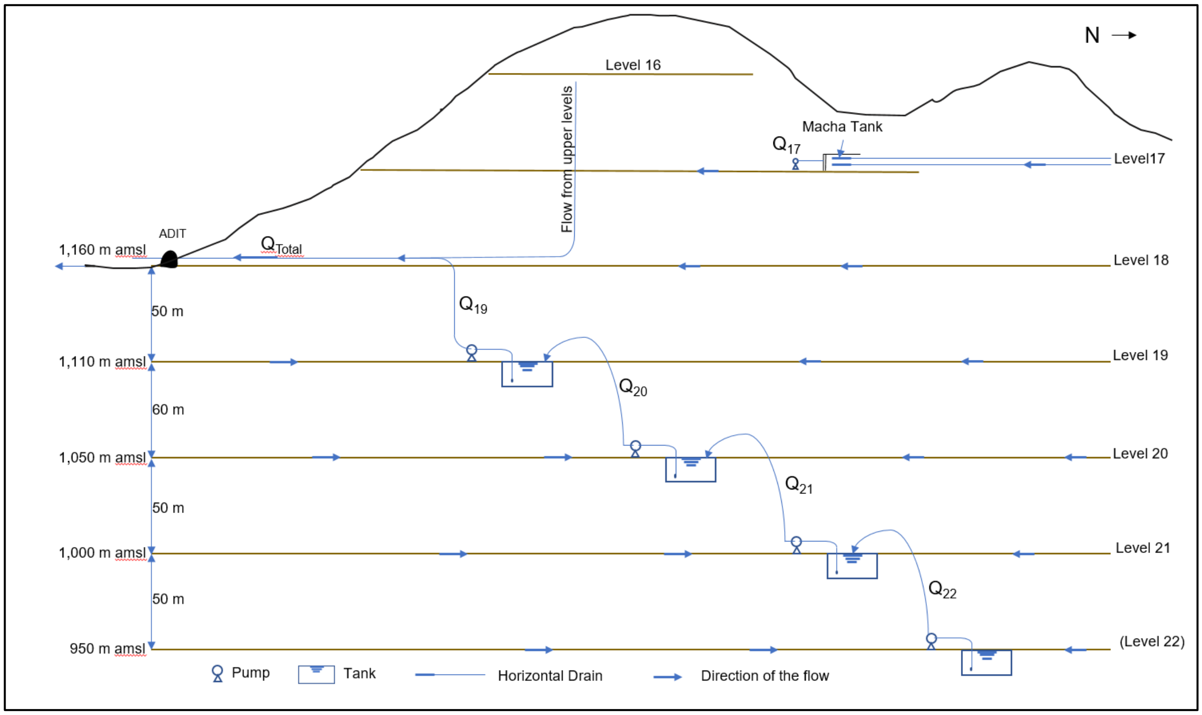
Figure 16-8: Location of Piezometers from 2012 Campaign

Water levels recorded until 2012 show elevations from to 661 to 2,022 masl. Ground surface piezometers show water levels following the topography within 100 m depth. The underground piezometers present water levels from 1,051 to 1,119 masl with depths varying from 7 to 94 m.

In the underground mine, a total of 105 Lugeon tests were conducted in five geotechnical borehole (angled) drilled into andesite/dacite porphyry and basalt (Knight Piésold, 2012). The geometric mean of the hydraulic conductivity for these intrusive units is 1.4 x 10^{-2} m/d, being slightly higher in the dykes and slightly lower in the basalt. 41 Lugeon tests were performed in five vertical boreholes surrounding the mine area additionally, most of which were conducted at less than 100 m depth within the same porphyry units. The results of the tests showed a geometric mean of 6.4 x 10^{-2} m/d. Nine Le Franc tests were conducted in the same vertical boreholes within the saprolite, and the results showed a hydraulic conductivity of 5.9 x10^{-2} m/d as geometric mean. It is important to note than several of the hydraulic tests were completed in unsaturated conditions and into the shallow part of the intrusive units or into the development mine zone area; therefore, the hydraulic conductivity values may be overestimated and not representative of the groundwater flow conditions toward the underground mine, values around 0.01 m/d could be considered more representative. SRK has not completed comprehensive review of the hydraulic tests conducted during the 2012 campaign to validate the field procedures and analysis. It should be completed as part of PFS.

Mine Dewatering

The mine has a series of pumps and tanks from Level 22 to Level 19, where the water is pumped to the processing plant to be used as makeup water. Each Level collects the water produced in its developments in addition to infiltration coming from Levels above. The water is briefly stored in a tank and pumped to the next Level above. Water from Level 16 and Level 17 is collected by gravity, discharges to Level 18 and through to the process plant. Figure 16-9 shows a simplified scheme of the current dewatering system.

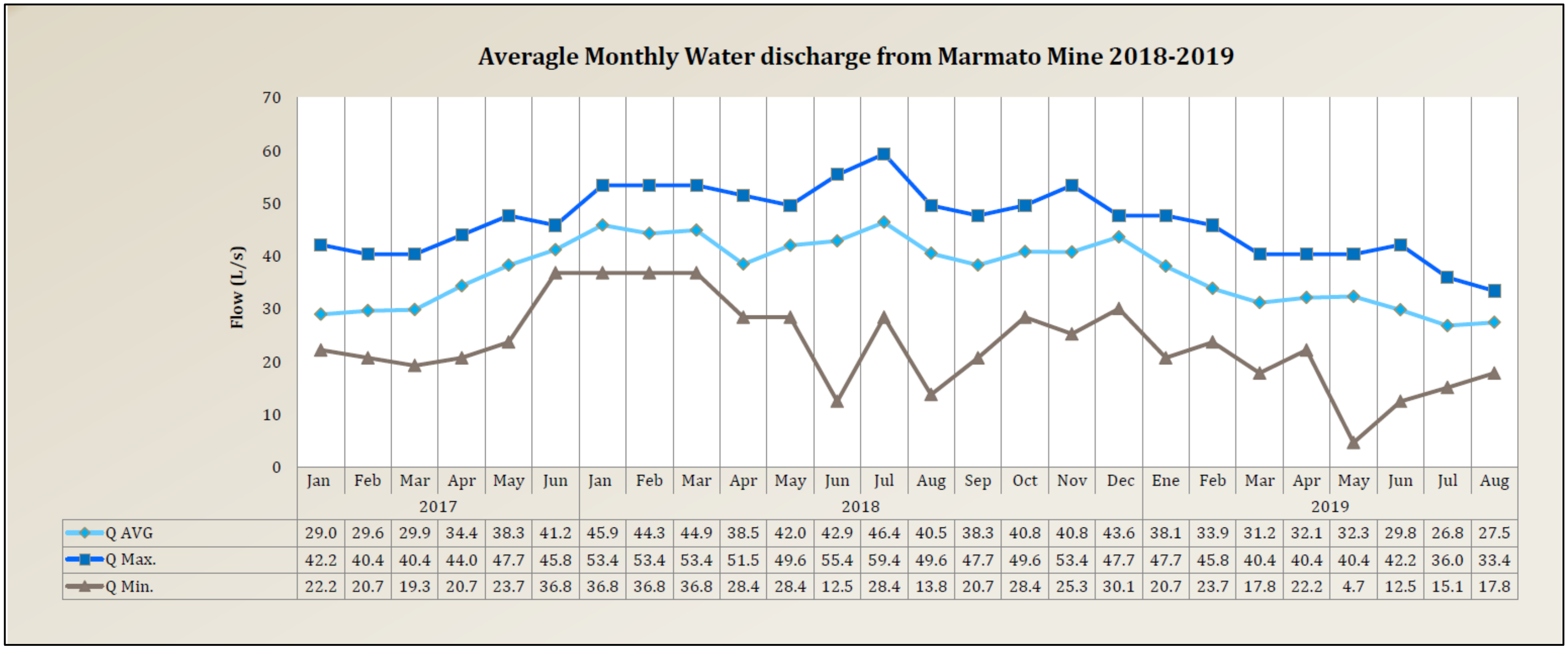


Source: Grand Colombia Mine, 2019

Figure 16-9: Scheme of Current Dewatering System

Preliminary flow measurements were conducted at mine adits and are presented in the environmental baseline report prepared by GCM (date unknown). The measurement values vary from a total of 6.3 to 15.9 L/s in summer and winter. However, there is no information about which mine levels were in operation during the flow measurements.

A measurement record of total mine water discharge is available from January 2017. The measured monthly average of total dewatering in Marmato mine is 37 L/s, varying from 26.8 to 46.4 L/s. Strong seasonal trends were not observed; however, a decrease of approximately 16 L/s can be detected in the last 12 months. Figure 16-10 shows the total water discharge from the Marmato mine.



2017	Jan	Feb	Mar	Apr	May	Jun
Q AVG	29.0	29.6	29.9	34.4	38.3	41.2
Q Max.	42.2	40.4	40.4	44.0	47.7	45.8
Q Min.	22.2	20.7	19.3	20.7	23.7	36.8

2018	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Q AVG	45.9	44.3	44.9	38.5	42.0	42.9	46.4	40.5	38.3	40.8	40.8	43.6
Q Max.	53.4	53.4	53.4	51.5	49.6	55.4	59.4	49.6	47.7	49.6	53.4	47.7
Q Min.	36.8	36.8	36.8	28.4	28.4	12.5	28.4	13.8	20.7	28.4	25.3	30.1

2019	Ene	Feb	Mar	Apr	May	Jun	Jul	Aug
Q AVG	38.1	33.9	31.2	32.1	32.3	29.8	26.8	27.5
Q Max.	47.7	45.8	40.4	40.4	40.4	42.2	36.0	33.4
Q Min.	20.7	23.7	17.8	22.2	4.7	12.5	15.1	17.8

Source: Gran Colombia Mine, 2019
Data from July to December 2017 is not available.

Figure 16-10: Measured Mine Water Discharge

The dewatering flow is a combination of groundwater inflows and water content in the backfill material (60-65% of water). According to Marmato operational personnel, the contribution of the backfill material is 7 to 14 L/s, depending on the number of hydraulic backfill equipment in operation. Therefore, the average fresh groundwater inflow into the mine could vary from 23 to 30 L/s. A significant amount of groundwater flow comes from the north section of Level 17 (crossing the Criminal Fault) where horizontal boreholes contribute 7 to 8 L/s.

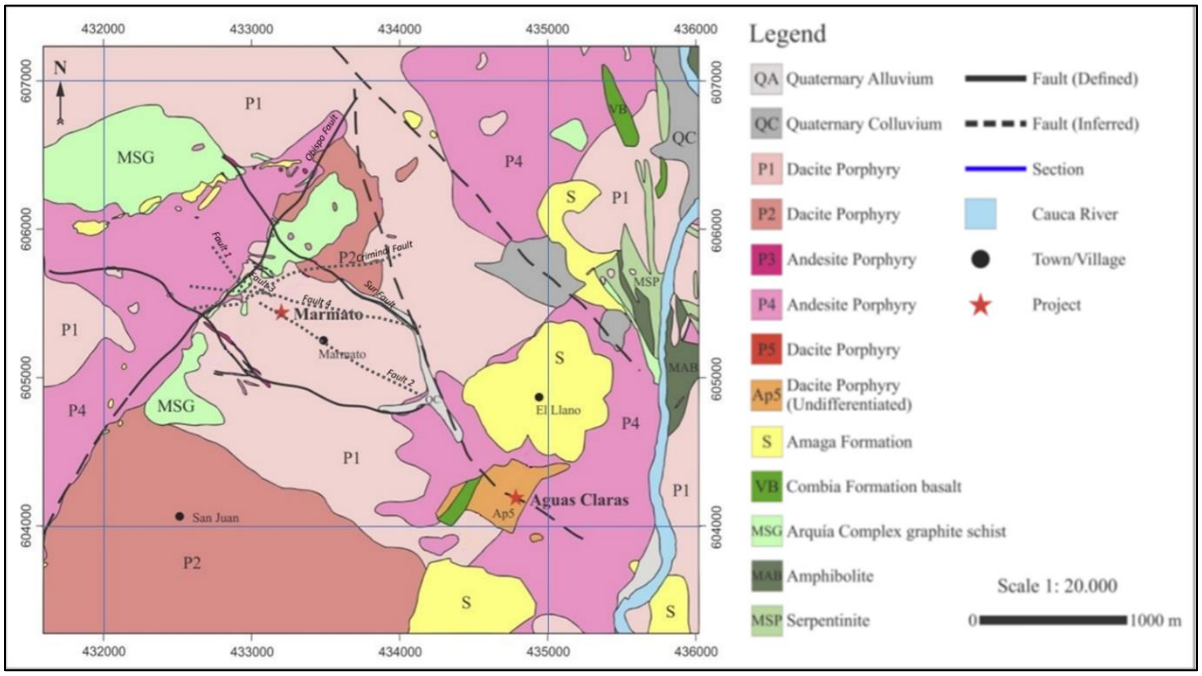
The existed dewatering system fits the current needs for the mine operations at Marmato mine.

Hydrogeological Conceptual Model

Saprolitic coverage and intrusive fracture rock are the two major hydrogeological units defined in the Marmato mine area: The saprolite is formed by clay material that has weathered on the top of intrusive rock units. It can reach over 10 m in some locations and is usually dry in the mine area. Subsurface flow can be observed as part of the recharge process from runoff and direct the precipitation. The intrusive fractured rock corresponds to dacite and andesite porphyry stocks and a sheeted pyrite veinlet system associated with intermediate argillic and propylitic alteration. As shown in Figure 16-7 the groundwater level is located approximately 100 m below the surface. However, the depressurization effect around the underground mine produces a more horizontal water table following the mine levels. This effect creates two water levels separated by the zone of depressurization as result of underground mine dewatering.

In a more regional scale, the groundwater flows west to east toward to the Cauca River, located at 668 m amsl, which represents the main discharge for the hydrogeological system. The water table follows surface topography. Given the high correlation between the topography and the groundwater levels, the hydrological and hydrogeological basins are the same. However, the effect of the mine dewatering/depressurization on the groundwater boundaries is currently unknown.

Groundwater flow is compartmentalized with structural blocks with limited hydraulic communication across fault boundaries due to fault gouge, weathering, or offset of geological units (Knight Piésold, 2012). This is evident in the Criminal Fault, which is located to the north of Cerro los Burros and runs toward Cauca River on Quebrada Los Pentanes. The fault represents a contact point between the Dacite Porphyry P1 in Cerro Lo Burros and Dacite Porphyry P2 and Graphitic Schist exposed to the north. The Criminal Fault is 15 m to 20 m wide, has a clayey alteration, and has a low water filtration flow. However, to the north of this contact, horizontal boreholes produce water flow from 7 to 8 L/s in Level 17. Also, water flow has been reported in the same location on Level 21. Figure 16-11 shows the geological units and faults described above.



Source: GCM, 2017

Figure 16-11: Local Geological Map

Since 1527 (492 years of mining), mining has affected the natural drainage and facilitated groundwater infiltration through the mine workings. Currently, there is no detailed recharge estimate into the Marmato mine area. However, preliminary estimates of infiltration from precipitation present values of 30% of annual precipitation (1,868.7 mm/y El Descanso Met. Station [Hatch, 2012]). A preliminary estimate of the recharge area for the current Marmato mine developments varies from 260 to 350 ha. Therefore, the recharge would represent 45 to 60 L/s of fresh groundwater inflow into the mine. However, measured values of mine water discharge indicate flowrates of 23 to 30 L/s for 2018 to 2019 mining conditions. Under this last consideration, the recharge in the mine area would vary from 15% to 20% of the annual precipitation.

16.3.3 Stope Optimization

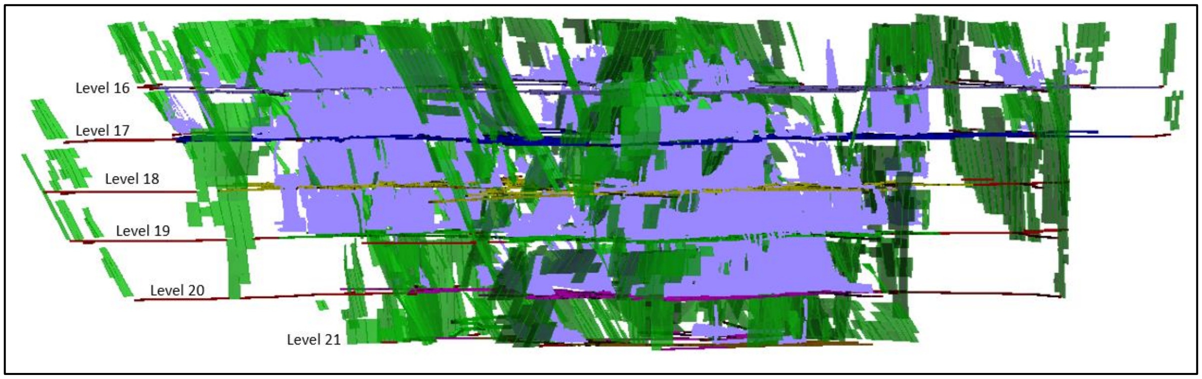
Stope optimization was completed on the veins material using Maptek Vulcan's implementation of Alford Mining Systems' Stope Optimization program and the resource block model used is discussed in Section 14 of this report.

<u>Vein Stope Optimization</u>

Vein area optimization parameters are as follows:

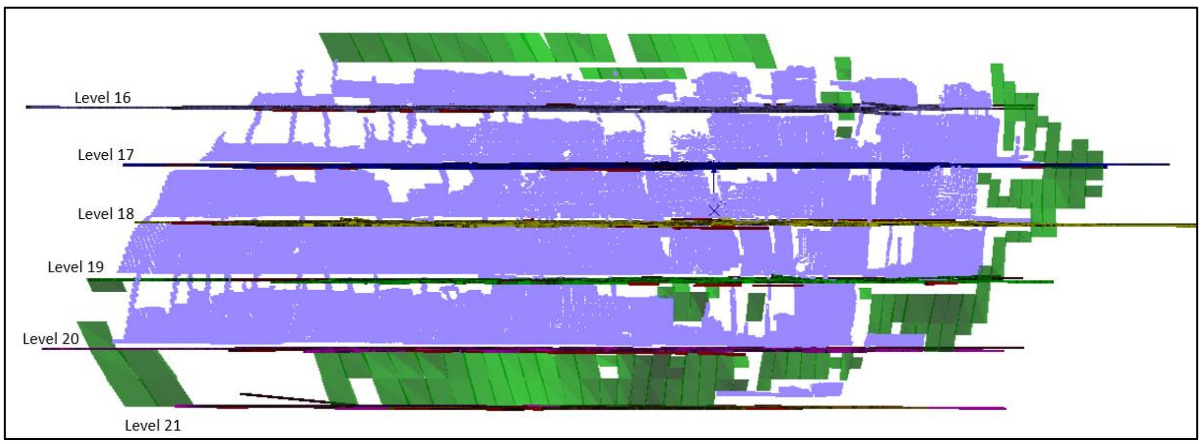
- CoG of 2.34 g/t AuEq;
- 1 m minimum mining width;
- 10 m and 25 m block heights;
- 17.5 m block length along strike;
- Angled stopes based on vein geology; and
- Elevation between 1,025 m and 1,350 m.

The optimization results for the 10 m and 25 m high blocks are assessed and combined to form 35m long by 50 m high stopes. Figure 16-12 and Figure 16-13 show the results of the optimization. Optimization was performed for various cut-offs ranging from 2.00 g/t AuEq to 2.60 g/t AuEq. There was little difference in the results between the various cut-offs.



Source: SRK, 2019

Figure 16-12: Stope Optimization Results in Green and Mined Out Area in Purple (Looking Northwest)



Source: SRK, 2019

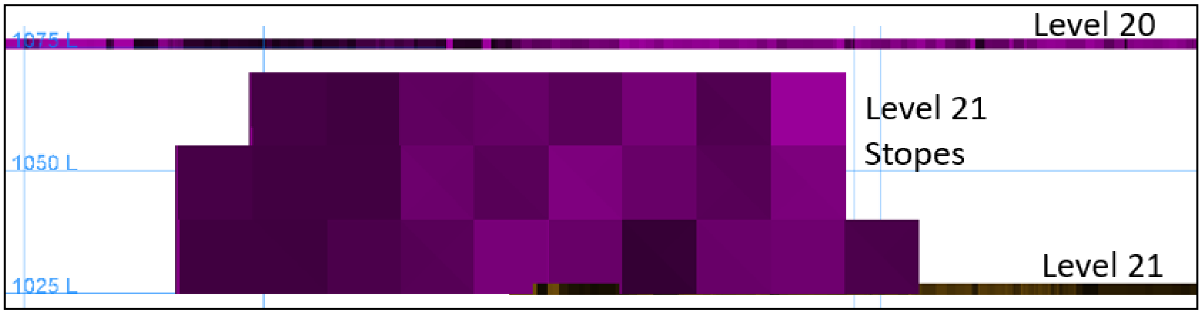
Figure 16-13: Example of Stope Optimization Results in Green and Mined Out Area in Purple for "Veta Exploracion NW" (Looking North)

Level 21 Stope Optimization

Level 21 area optimization parameters are as follows:

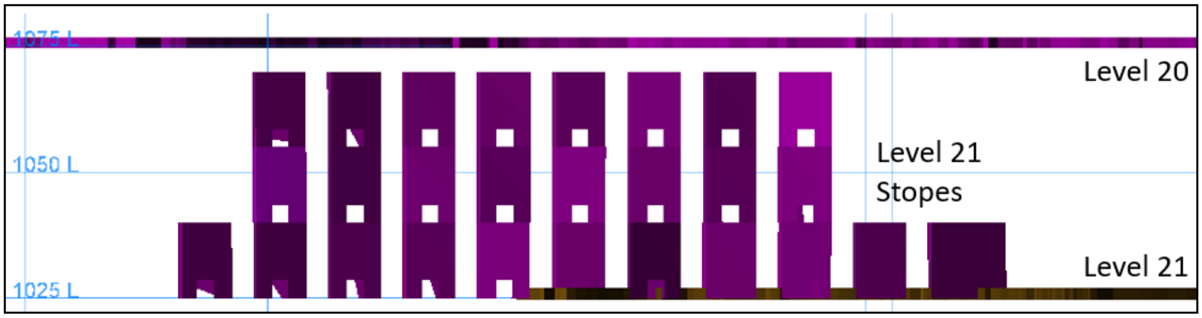
- Cut-off of 2.05 g/t AuEq;
- 25 m to 26 m mining length;
- 15 m block heights;
- 15 m block width;
- Angled stopes based on vein geology; and
- Elevation 1,025 m to 1,070 m.

Figure 16-14 shows the optimization results for Level 21. A sensitivity for different cut-offs was not completed for this study due to time constraint. Due to the use of unconsolidated hydraulic fill, 5 m dip pillars need to be left in-situ as shown in Figure 16-15. The stope size and pillar locations should be optimized for the next level of study.



Source: SRK, 2019

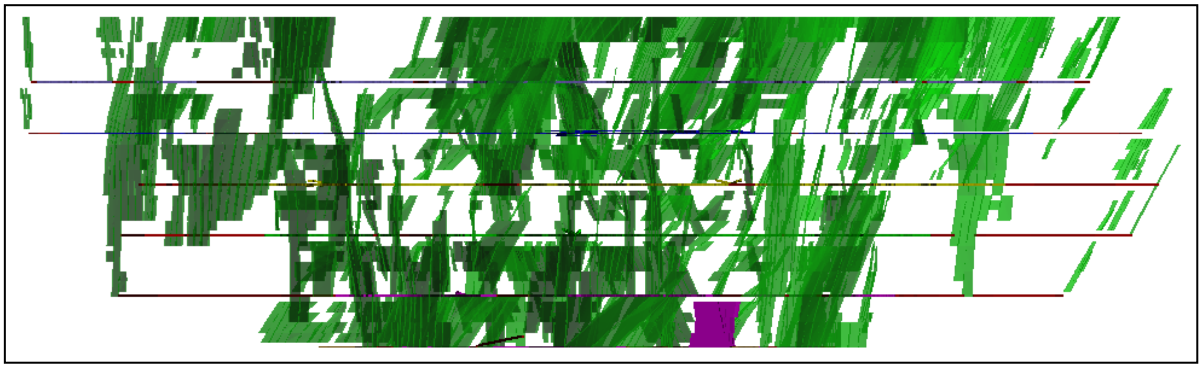
Figure 16-14: Stope Optimization Results



Source: SRK, 2019

Figure 16-15: Stope Optimization Results with Pillars Removed

Figure 16-16 shows the results of the vein optimizations and the Level 21 optimizations. The Level 21 stopes are located in a single area and make up approximately 5% of the total material tonnes.



Source: SRK, 2019

Figure 16-16: Comparison of Level 21 Stopes (magenta) and Vein Stopes (green)

16.3.4 Mine Design

Veins Stope Design

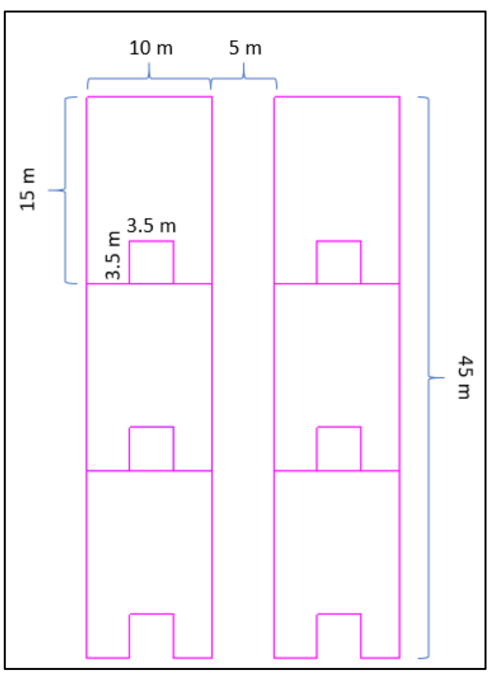
The stopes generated from the stope optimizer were amalgamated into 35 m long by 50 m high blocks. The minimum mining width is 1 m and the block width varies between 1 m and 3 m. This is consistent with the block size GCM is currently using to mine the veins. There are certain gaps in the blocks generated by the optimizer and these are not included as part of the mining plan. These gaps could potentially be mined as marginal material.

Veins Development Design

Due to the number of stopes from the optimizer, a simplified approach was taken for the development design. The development was calculated from the general optimizer stopes with the assumption that the access drifts are 2.2 m by 2.2 m. A diluted grade was calculated for this development material and tonnages/grades for development material are tracked separately from stope material in the production schedule. For areas where development is required to reach the stopes, 2.2 m by 2.2 m access drifts were designed following the veins where possible. Vertical raise development at the ends of the stopes is also calculated into the production rate for the stopes.

Level 21 Stope Design

The stopes generated from stope optimizer are 15 m wide by 15 m high and the length varies depending on the orebody to a maximum length of 26 m. A 5 m pillar is then removed to form 10 m wide by 15 m high stope (Figure 16-17). This area is envisioned to be mined from the bottom up in a primary/secondary stoping sequence. The stope will be drilled from a bottom access and mucked from the same access. Backfill is placed from the top access where available. For the top stopes, it is assumed that fill holes can be drilled up to the top of the stope or from adjacent accesses.

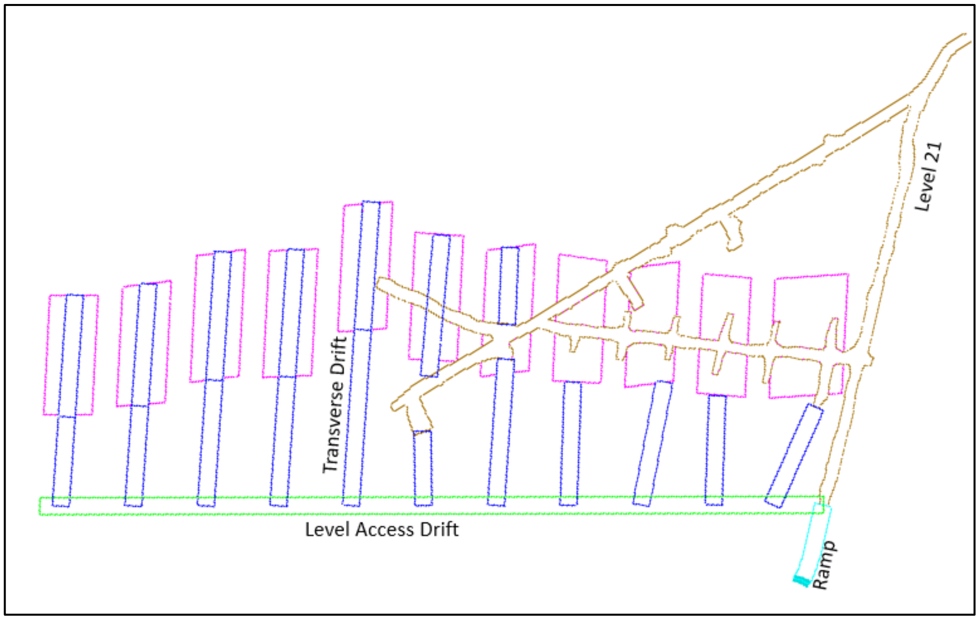


Source: SRK, 2019

Figure 16-17: Level 21 Stope Design Dimensions

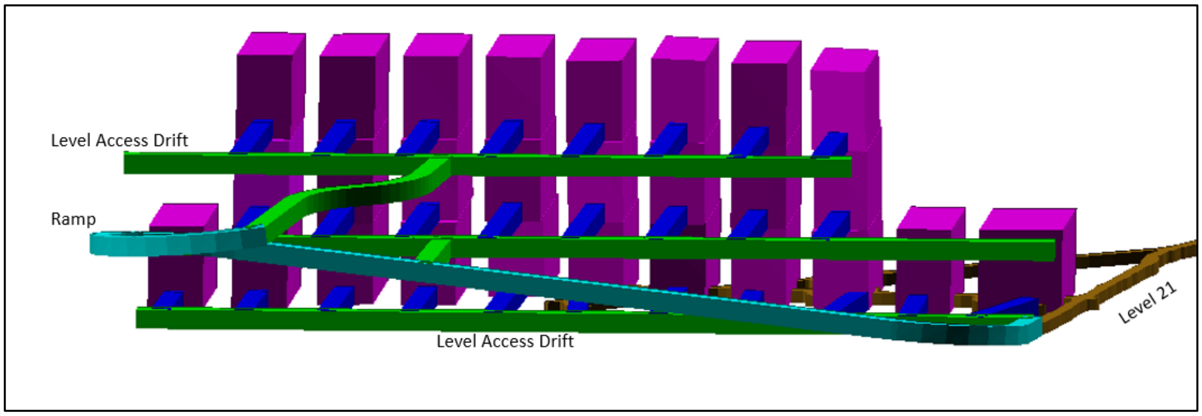
Level 21 Development Design

There are existing development accesses on Level 21 near planned stopes. A level access drift , 3.5 m by 3.5 m, is planned to be driven with transverse drifts perpendicular to it to access the stope for drilling and mucking as shown in Figure 16-18. The next lift/level of stopes is accessed through a 3.5 m by 3.5 m ramp, driven at a 13% grade as shown in Figure 16-19.



Source: SRK, 2019

Figure 16-18: Plan View of Level 21 Development Design



Source: SRK, 2019

Figure 16-19: Rotated View of Level 21 Development Design

16.3.5 Mine Plan Resource

The underground mine design for the Marmato veins and Level 21 blocks results in a mine plan resource of 5.5 Mt (diluted) with an average grade of 3.82 g/t Au and 15.39 g/t Ag (Table 16-8). This estimate is based on a 2.81 g/t AuEq cut-off, a 90% mining recovery and a 20% dilution for the veins. Level 21 blocks are based on a 2.05 g/t AuEq cut-off, a 90% mining recovery and a 15% dilution.

A 5% allowance is added to the development for overbreak.

Table 16-8: Upper Mine – Mine Plan Resource Classification – Vein and Level 21 Material(1)

Description	Tonnes (kt)	Au (g/t)	Ag (g/t)	Contained Au Oz (koz)	Contained Ag Oz (koz)
Measured	802	3.94	17.95	102	463
Indicated	4,308	3.84	14.64	532	2,028
Inferred	433	3.34	18.06	46	251
Total	**5,543**	**3.82**	**15.39**	**680**	**2,742**

Source: SRK, 2019

(2) Includes Measured, Indicated and Inferred material based on CoG of 2.81g/t AuEq for the veins, and 2.05 g/t AuEq for Level 21 material.

The grades reported here are reported from the mine design that is based on economic areas above a CoG. This grade is higher than the as-mined reported grades from Marmato (Table 16-9) due to factors such as Marmato mining outside of the planned areas, mining of marginal areas below the CoG, and uncontrolled dilution.

Table 16-9: GCM Reported Historical Grades

Year	GCM Grade (g/t Au)
2006	3.64
2007	3.32
2008	3.44
2009	3.51
2010	3.39
2011	3.18
2012	2.85
2013	2.90
2014	2.85
2015	2.79
2016	2.56
2017	2.46
2018	2.67

Source: GCM, 2019

The mine plan presented here assumes that GCM will focus mining only in planned areas and will minimize dilution going forward.

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that the PEA will be realized.

16.3.6 Production Schedule

The production schedule is based on the productivity rates shown in Table 16-10.

Table 16-10: Productivity Rates

Activity Type	Dimension	Rate (1)
Veins Stopes with Development		15 tpd
Veins Stopes without Development		19 tpd
Level 21 Stopes		500 tpd
Backfill (Veins and Level 21)		715 m3/d
Development	2.2 m x 2.2 m	1 m/d
Level 21 Level Access	3.5 m x 3.5 m	3.5 m/d
Level 21 Ramp	3.5 m x 3.5 m	3.5 m/d
Apique widening	2 m x 4 m	0.56 m/d

Source: SRK, 2019
(1) All rates are per face. Multiple areas/faces are mined together to generate the production schedule.

Stopes with development rate include the development of a 2.2 m by 2.2 m drift at the bottom of the stope, mining (raising, drilling, blasting and mucking) of the stope and backfilling. The rate for stopes without development includes mining the stope and backfill. Level 21 rates include mining of the stope. Backfill for Level 21 uses the same system as the veins and is filled at 715 m^3/d.

The LoM for the Upper Mine area is 15 years at 350,000 tpy for a total of 5.5 Mt of mineralized material. The average grade is 3.82 g/t Au and 15.39 g/t Ag for an average of 43 koz of Au per year.

Scheduling targeted 700 tpd (245,000 tpy) from the vein stopes from 2019 to 2021, then increasing to 1,000 tpd (350,00 tpy) for the rest of the mine life. Level 21 stopes are producing at 50 tpd (17,500 tpy), 300 tpd (105,000 tpy) and 450 tpd (157,500 tpy) for years 2019, 2020 and 2021 respectively. The production schedule is based on 350 days per year, with two eight hour shifts per day. Table 16-11 and Table 16-12 show the annual production schedule based on these assumptions. The annual schedule was completed using iGantt scheduling software. Figure 16-20 and Figure 16-21 show the production schedule colored by time period and AuEq grade.

Table 16-11: Marmato Upper Mine Total Production Schedule

Marmato	Unit	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	Totals
Mineralized Tonnes	kt	109	350	402	350	350	350	350	350	350	350	350	350	350	350	350	350	130	**5,543**
Waste Tonnes	kt	28	43	30	21	19	17	25	22	11	16	29	17	13	4	4	4	-	**301**
Au	g/t	3.94	3.77	3.53	3.64	3.63	3.64	3.75	3.92	4.08	4.08	3.89	3.58	3.82	4.04	4.03	3.95	3.54	**3.82**
Ag	g/t	14.32	12.16	10.95	14.41	15.28	15.06	14.81	15.29	15.57	16.42	17.68	16.33	17.04	18.08	17.28	16.14	13.57	**15.39**
Development Length	m	1,385	2,718	2,247	1,984	1,811	1,685	2,269	2,068	1,280	1,790	2,671	1,730	1,431	660	618	521	28	**26,895**
Backfill Volume	m^3	36,876	118,997	140,420	120,942	120,772	120,489	120,603	120,301	120,015	119,101	120,204	120,978	120,163	119,090	119,050	119,180	44,045	**1,901,227**

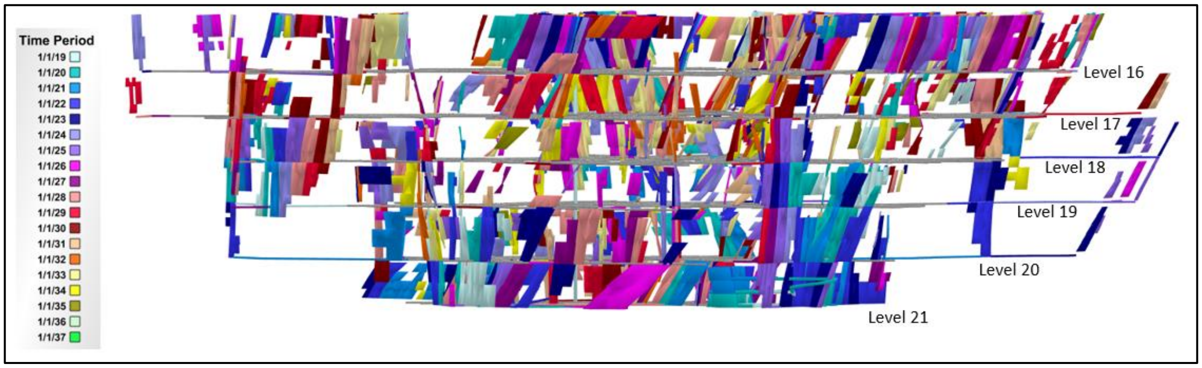
Source: SRK, 2019

Table 16-12: Marmato Upper Mine Production Schedule by Veins and Level 21

Marmato	Unit	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	Totals
Veins Mineralized Tonnes	kt	98	237	238	321	343	343	343	343	343	340	340	343	343	345	345	347	130	**5,143**
Veins Au	g/t	4.00	4.02	3.71	3.70	3.64	3.65	3.76	3.93	4.09	4.10	3.90	3.59	3.84	4.05	4.04	3.95	3.54	**3.86**
Veins Ag	g/t	15.17	15.59	14.33	15.00	15.37	15.12	14.91	15.32	15.60	16.46	17.70	16.32	17.09	18.15	17.34	16.17	13.59	**16.00**
Level 21 Mineralized Tonnes	kt	5	95	151	22	-	-	-	-	-	-	-	-	-	-	-	-	-	**272**
Level 21 Au	g/t	3.24	3.13	3.23	2.84	-	-	-	-	-	-	-	-	-	-	-	-	-	**3.17**
Level 21 Ag	g/t	4.96	4.43	5.78	6.55	-	-	-	-	-	-	-	-	-	-	-	-	-	**5.36**
Veins Development Mineralized Tonnes	kt	4	8	6	7	7	7	7	7	7	10	10	7	7	5	5	3	0.4	**108**
Veins Development Au	g/t	3.65	3.73	3.16	3.16	3.25	3.35	3.06	3.47	3.64	3.49	3.45	3.25	2.95	3.04	3.37	3.42	2.56	**3.34**
Veins Development Ag	g/t	9.72	11.43	10.99	11.58	10.9	11.98	10.06	13.94	14.01	15.01	16.99	16.46	14.41	13.33	13.58	13.44	8.09	**13.22**
Level 21 Development Mineralized Tonnes	kt	2	11	7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**20**
Level 21 Development Au	g/t	3.63	4.03	4.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**3.98**
Level 21 Development Ag	g/t	5.54	5.28	6.98	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**5.91**

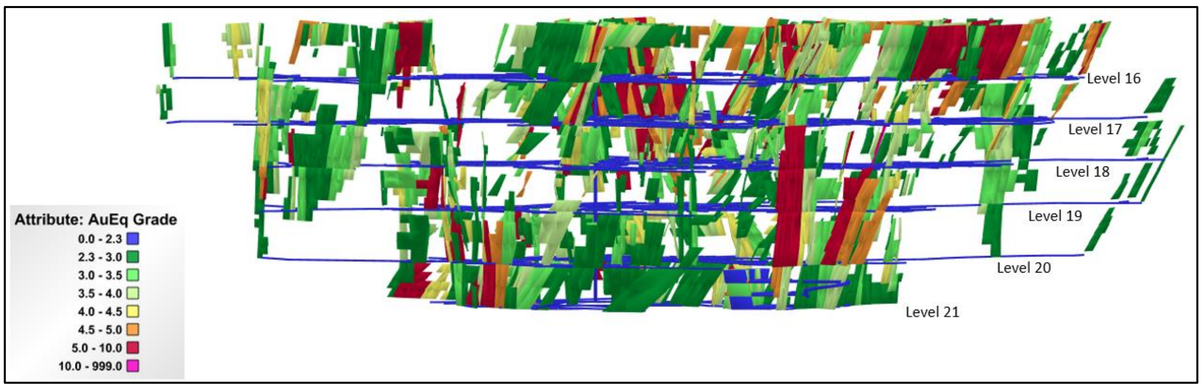
Source: SRK, 2019
Note: Numbers may not sum due to rounding.

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves.



Source: SRK, 2019

Figure 16-20: Production Schedule Colored by Time Period



Source: SRK, 2019

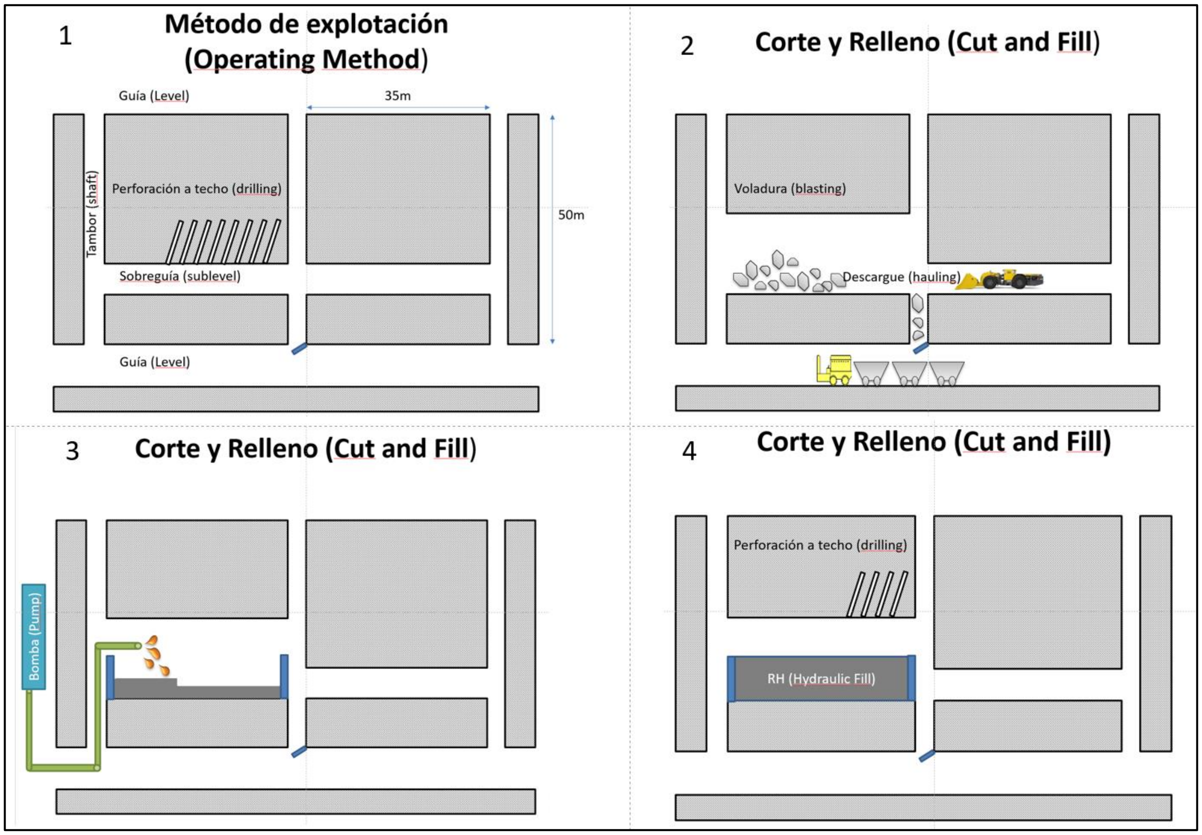
Figure 16-21: Production Schedule Colored by AuEq Grade

16.3.7 Mining Operations

Stoping

Cut and fill stopes are mined using stopers and jacklegs. Once the level access is driven, raises (tambores) are driven on either side of the stope. A sublevel is driven laterally and used as a drilling platform, where 1.7 m to 2.3 m slices are drilled up into the back. After blasting and bolting, the stope is mucked out either using slushers for higher grade stopes or bobcats/mini-scoops for lower grade blocks. Once the stope is mucked out, concrete barricades are built on either side of the stope and filled with unconsolidated hydraulic fill. Figure 16-22 shows the typical cut and fill mining sequence:

- Drilling and blasting;
- Mucking or slushing to a raise and removing the mineralized material from the raise and hauling by train along the production level;
- Sand backfill; and
- Repeat the cycle on top of the sandfill.



Source: GCM, 2019

Figure 16-22: Marmato Cut and Fill Mining Method

Level 21 blocks are 10 m by 15 m blocks and will be mined using a modified longhole stoping method in a general bottom up orientation. The block is drilled with upholes from the bottom access using a Muki long hole drilling rig. A slot will first be drilled and blasted, before the rest of the block is slashed into the slot. The material will be mucked from the bottom using a remote scoop. The blocks will be backfilled using unconsolidated hydraulic fill from the existing plant.

Development

Development is completed using jacklegs. Level accesses are 2.2 m by 2.2 m, sublevels vary depending on the width of the vein.

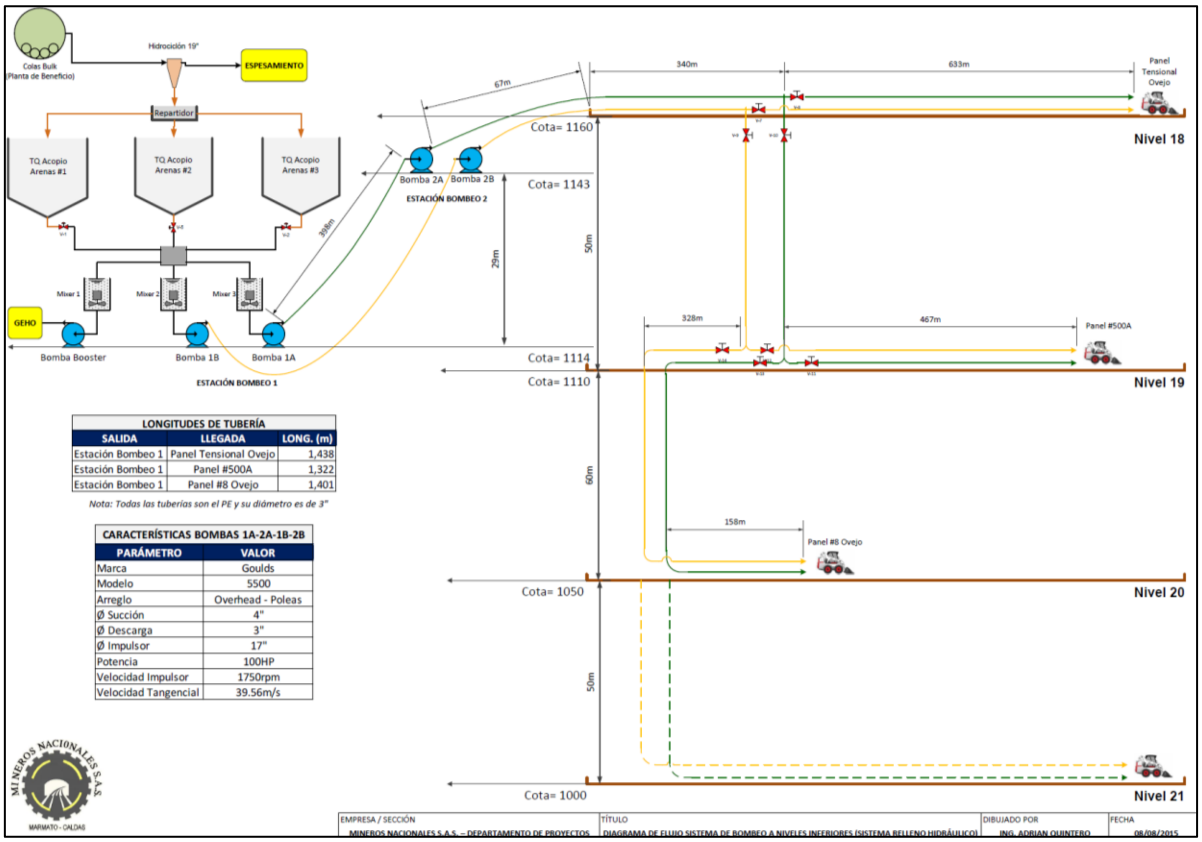
A 3.5 m by 3.5 m ramp is used to access the Level 21 blocks. The level access and transverse drifts are the same size as the ramp. Development for the Level 21 blocks is done using a jumbo.

Haulage

All materials are hauled using rail haulage on the level. Material from levels below Level 18 are hauled up using the apique hoist.

Backfilling

Backfilling is completed using unconsolidated hydraulic backfill from the plant. Currently, approximately 70% of the mill tailings are returned to the mine as backfill. There are four tailings pipelines going underground to different levels with each pipeline having a capacity of 290 m^3/d. The plant's backfill capacity is 715 m^3/d. One limitation of the backfill system is the lack of a surge tank, so there is limited catch up possibility should a delay occur. Waste rock generated underground that is not hauled out is used as backfill. The backfill system is shown in Figure 16-23. The backfill system also has a positive displacement pump that moves backfill materials to the levels above Level 18 that is not shown in Figure 16-23.



Source: GCM, 2019

Figure 16-23: Marmato Backfill System

16.3.8 Ventilation

Ventilation requirement, as provided by GCM, is 41 kCFM for personnel and 94 kCFM for diesel equipment, for a total of 135 kCFM. A ventilation model was not provided to SRK for review, however, the following information was provided by GCM, 139 kCFM of fresh air enters through the portals from Level 17 and Level 18 via 50 hp fans. 96 kCFM is exhausted from the portals on Level 16. There is a

43 kCFM difference in the intake and exhaust which GCM attributes to leakage in the system and the artisanal mining up above. For levels below 18, including Level 21, the fresh air is pulled down the apiques using 30 hp fans onto the level. Secondary ventilation is provided by 15 hp fans and vent tubing to the face. The flow in the level is mainly controlled by vent doors. In each level, the air flows west to an exhaust raise, where the air goes up to Level 16.

16.3.9 Mine Services

Pumping

The mine has an operating system of ditches, sumps, and small pumps that control water on the individual mine level and pump water to the main pump system.

The main pumping system used in the mine is a staged system of 10,000 to 15,000 liter sumps/tanks and pumps that move water from lowest levels of the mine at Level 21 up to the mine portal where the water is used in the mine processing plant. On Level 21, at the bottom of the currently developed mine, there is a storage tank with three Krebs pumps that pump through two-4 inch and one-6 inch pipelines up to Level 20. On Level 20, another tank and pump system with three pumps moves the water to Level 19 to a concrete lined sump with two Goulds 5500 slurry pumps that pump through 4 inch pipelines to the portal Level 18 to the process plant water tank. There is redundancy built into the system with extra pumps on Level 19 and additional locations to place pumps on Level 20. The pump system handles on average 37 L/s with a range from 26.8 L/s to 46.4 L/s.

Electrical Supply

The existing project electrical system includes an 8.1 MVA main project substation with six transformers that provide power to the mine and mill. The mine system power is provided at 33 kV through three transformers transform power for secondary transformers that feed the mine surface and underground facilities. The three mine related transformers and loads they feed are summarized as follows:

- Transformer 1 (2000 KVA) steps the 33kV power down to 13.2kV and feeds the three mine substations that in turn feed the compressors, pumps and offices/shops 440 VAC;
- Transformer 2 (2000 KVA) feeds the mine at 13.2kV through three separate mine transformers that in turn feed the various mine levels, hoists, pumps, and mine equipment. The equipment operates on 440 VAC; and
- Transformer 4 (1250 KVA) and 5 (630 KVA) feeds two compressors each at 440VAC.

The largest loads at the mine are the compressors, pumps, and hoists which currently account for an approximate total of 65% percent of the mine load.

Health and Safety

The mine has a mine phone system and emergency egress is provided through stairs in the apique declines and a series of ladders to the surface portal level. The mine has a health and safety response plan and miner safety training sessions for instruction on proper work procedures and safe work activities.

Manpower

Currently there are 1,158 personnel working at the site, this includes, underground staff, process plant staff and other support staff. The mining staff is approximately 67% of the total staffing. GCM projects an increase in manpower primarily in the mine underground operations over the next 5 years (Table 16-13).

Table 16-13: Manpower by Department

Department	2019	2020	2021	2022	2023
Underground Operations	777	868	890	910	910
Process Plant	34	34	34	34	34
Support	217	217	217	217	217
Administration	59	59	59	59	59
Others	71	71	71	71	75
Total	**1158**	**1249**	**1271**	**1291**	**1295**

Source: GCM, 2019

Equipment

The mine utilizes a large number jackleg drills and small electric and air operated equipment. There are also small diesel microscoops and bobcats. The mine is adding additional small drills and microscoops in the future.

Table 16-14 shows the current equipment list as provided by GCM.

Table 16-14: Marmato Equipment List

Equipment	Amount
Bobcat	27
Jacklegs/Stopers	313
Microscoops	6
Locomotives	27
Mine Pumps	18
Hydraulic Fill Pump	18
Winches	5
Slushers	91
Fans	83
Railcars	215
Compressors	9
Pumps	33
Transformers	12
Electric Grid	6
Total Equipment	**863**

Source: GCM, 2019

16.3.10 Recommendations

The following recommendation and opportunities have been identified:

- Using 3D modeling software and block model to generate mine design;
- Apply a 3D Gantt type scheduling software to generate a mine plan. This will more accurately show times of potential higher and lower grade in the mine plan;
- Reconciliation of mining plan and actual production data to improve the accuracy of the mine plan and dilution estimates;
- Develop a ventilation model for the current mine; and
- Optimizing shift schedule to improve overall efficiency.

16.4 MDZ Mining

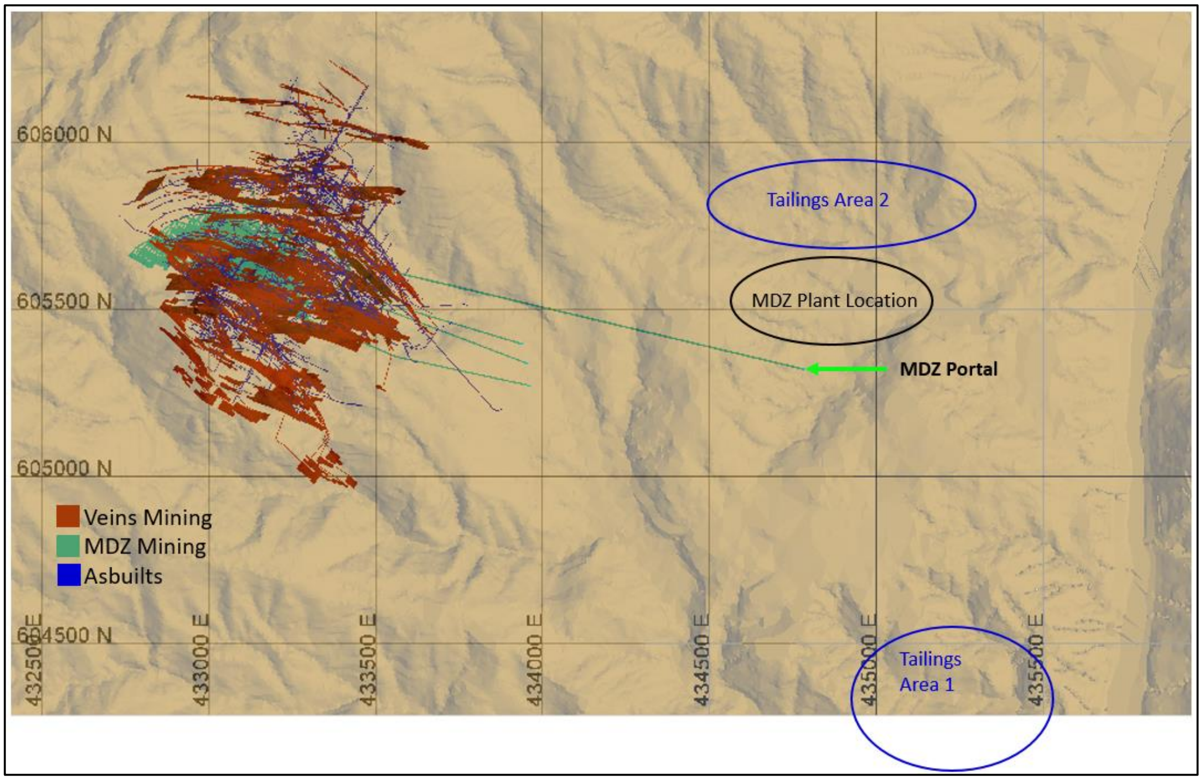
The MDZ area is currently in the exploration phase and has not been developed. Mineralization is located approximately 600 to 1,200 m below the surface (530 masl to 1,015 masl). Based on geomechanical information and mineralization geometry an underground longhole stoping method (LHS) is suitable for the deposit and vein mining can continue to occur above the MDZ area.

The stopes will be 10 m wide and stope length will vary based on mineralization grade. A spacing of 25 m between levels has been used. The deposit is mined in blocks where mining within a block occurs from bottom to top with the use of paste backfill. Sill pillars are left in situ between blocks. The backfill will have sufficient strength to allow for mining adjacent to filled stopes, thus eliminating the need for dip pillars. In the top mining block a higher grade core is extracted first, mined from bottom to top. Subsequently additional stopes are mined from the bottom of the block up, mining adjacent to (but not underneath) backfilled stopes.

The mine will be drift access, mineralization will be transported from stopes to the surface by underground trucks. Internal intake and exhaust raises will be developed using raisebore machines and air will flow into dedicated intake and exhaust ventilation drifts to surface.

The mine design process involved using stope optimization within Vulcan™ software to determine potentially mineable areas based on a CoG and minimum mining dimensions. Dilution and recovery were added to the designed tonnage to account for unplanned stope dilution and unrecoverable material within the stope.

A process facility of 4,000 tpd (1.4 Mtpy) is used. Access and infrastructure development underground was designed to support the mining method and sized based on mining equipment and production rate requirements. Surface infrastructure and tailings were designed to fit the mine plan resource. The general layout of the mine and mill is shown in Figure 16-24.



Source: SRK, 2019

Figure 16-24: MDZ General Layout

16.4.1 Cut-off Grade Calculations

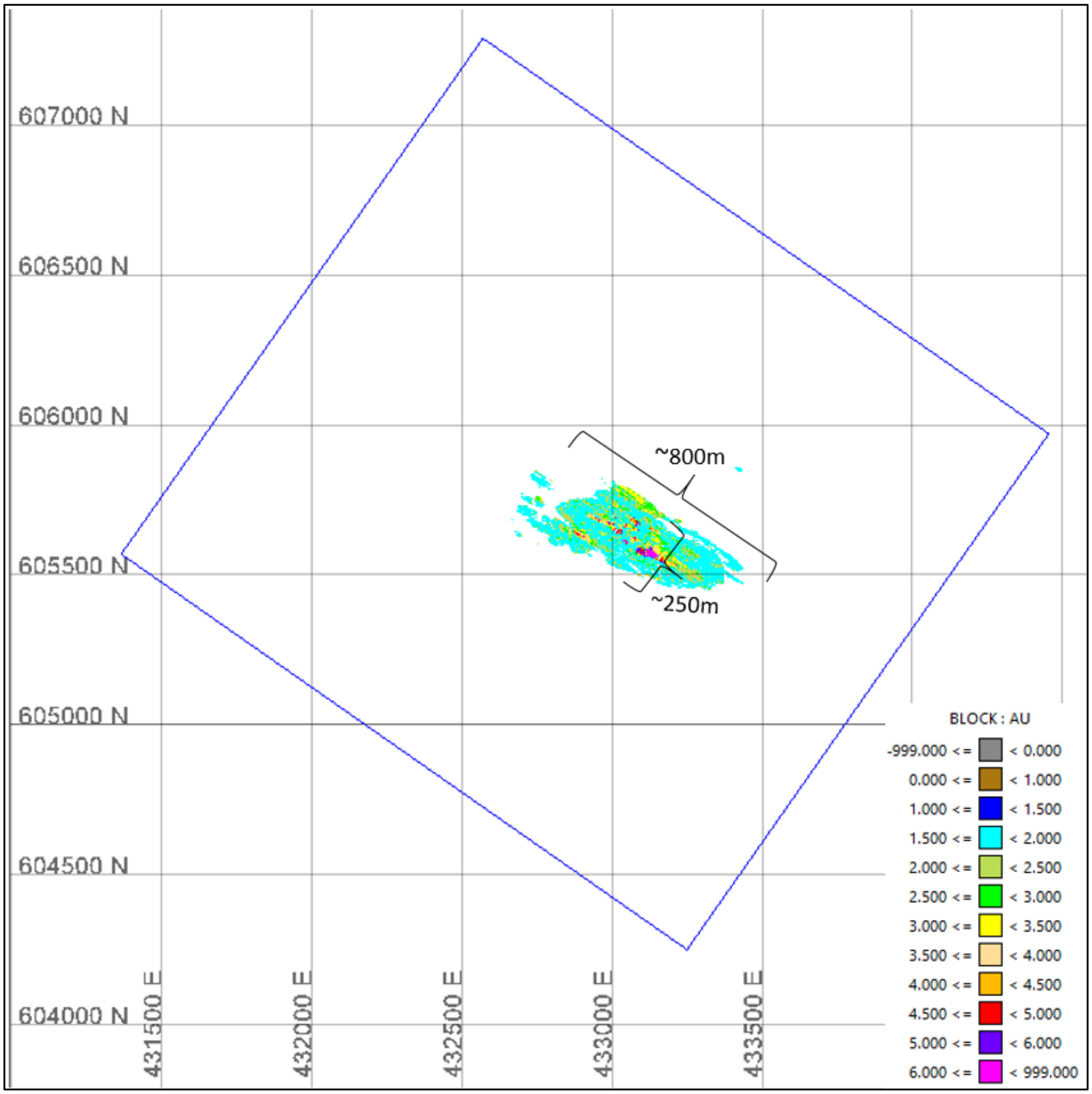
Current estimated project costs and the calculated Au CoG are shown in Table 16-15. An Au cut-off of 2 g/t was targeted for design work and stopes were extended to a 1.75 g/t cut-off where no additional development was required. For reporting within the design, a minimum cut-off of 1.5 g/t Au was used. A silver recovery of 40% is expected however was not used for the CoG calculation.

Table 16-15: Underground Cut-off Grade Calculation

Parameter	Amount	Unit
Mining cost (1)	30.00	US$/t
Process cost	17.75	US$/t
G&A, Other	6.50	US$/t
Royalty	9.2%	%
Total Cost	**$59.75**	**US$/t**
Gold price	1,300.00	US$/oz
Au Mill Recovery	95%	
CoG	1.50	g/t

Source: SRK
(1) Includes Backfill

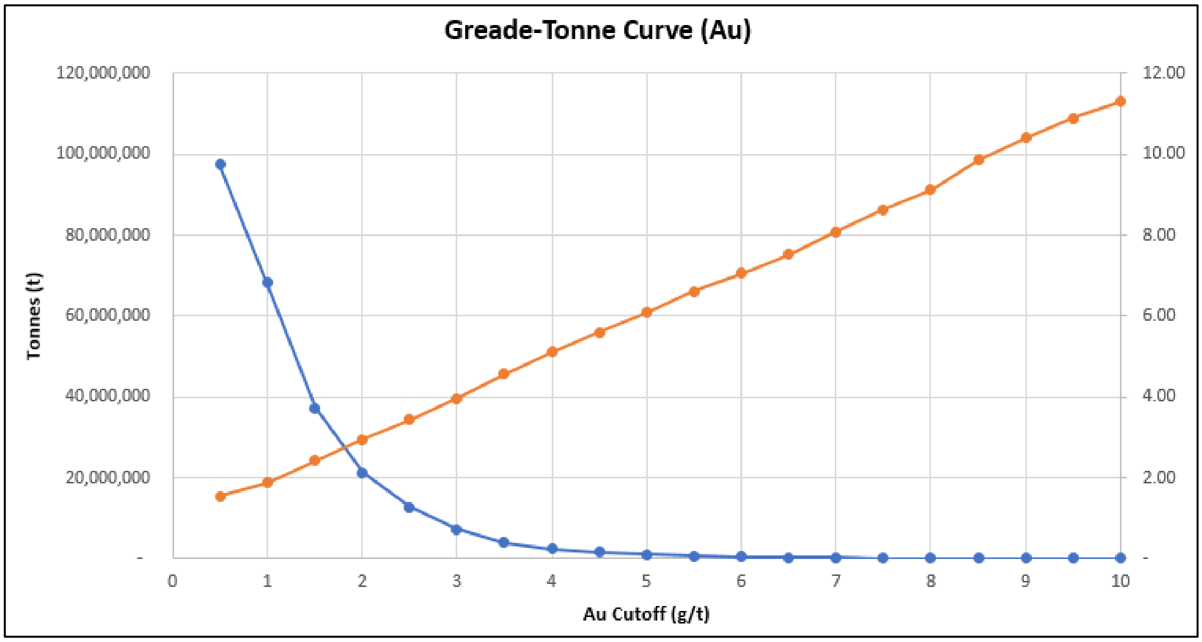
The basis for the mine plan resource work is the MDZ model described in Section 14. Figure 16-25 shows the block model orientation and mineralized blocks.



Source: SRK, 2019

Figure 16-25: MDZ Block Model and Mineralization Extents

Figure 16-26 shows a grade-tonnage curve for the MDZ area (below an elevation of 1,126 m). The underground Mineral Resources shown are classified as Indicated and Inferred.



Source: SRK, 2019

Figure 16-26: MDZ Grade/Tonne Curve Based on Au Cut-Off

16.4.2 Hydrogeology and Mine Dewatering

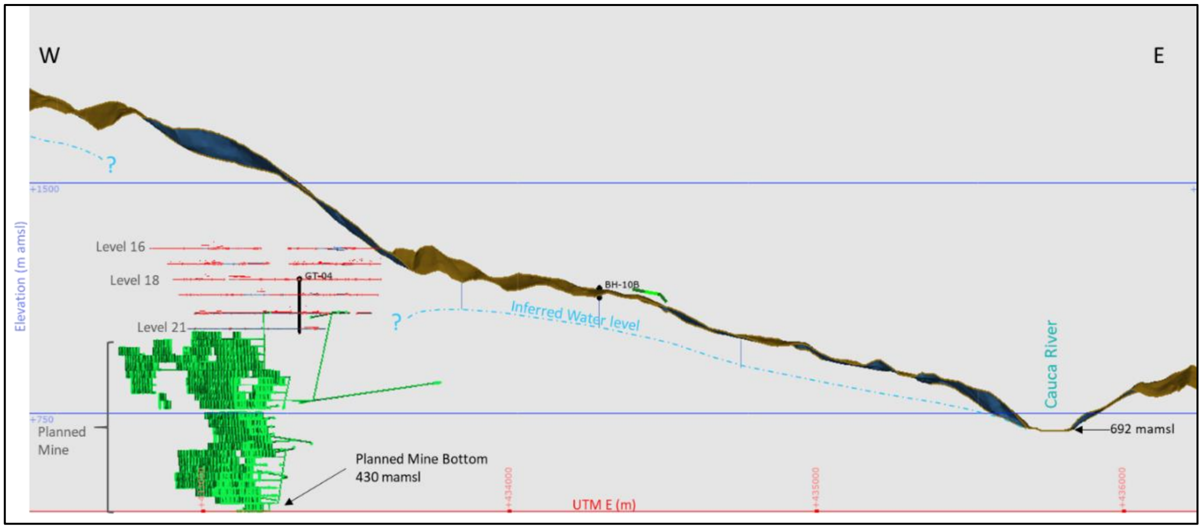
The planned mine will reach 765 m amsl in the first year of excavation. 10 levels will be mined above this elevation over the next seven years. The bottom of the mine will reach 430 m amsl at the end of the seventh year and will continue the mine upward for the next 11 years.

A preliminary analytical model was used to estimate the groundwater flow into the planned mine. A Theis solution method corrected for an unconfined aquifer and presence of bounded conditions (Q=0) was implemented and calibrated using the current water discharge condition. Assumptions used in the analytical model are as follows:

- Hydraulic conductivity: 0.01 m/d;
- Specific storage: 1 x 10^{-6} 1/m;
- Mine radius: 300 m;
- Aquifer thickness: 800 m;
- Aquifer radius (distance to no flow boundary): 2,000 km;
- Zero inflow contribution within the aquifer radius; and
- Recharge contribution to the inflows not simulated by analytical formula and added to total inlfow: 30 L/s.

Preliminary predicted inflows into the planned mine will be approximately 70 L/s in the first seven years of mining (765 m amsl and above) and up to 110 L/s in the last 11 years of mining (430 m amsl and above).

The bottom of the mine will be 262 m below and 2.5 km to the east of the Cauca River. There is a risk of water intrusion from the river through the river bed sediments (Figure 16-27). Structures with features similar to those detected to the north of the Criminal Fault could connect mine developments with the river. Further hydrogeological investigations of the area are required to evaluate potential sources of groundwater inflow into proposed underground mine.



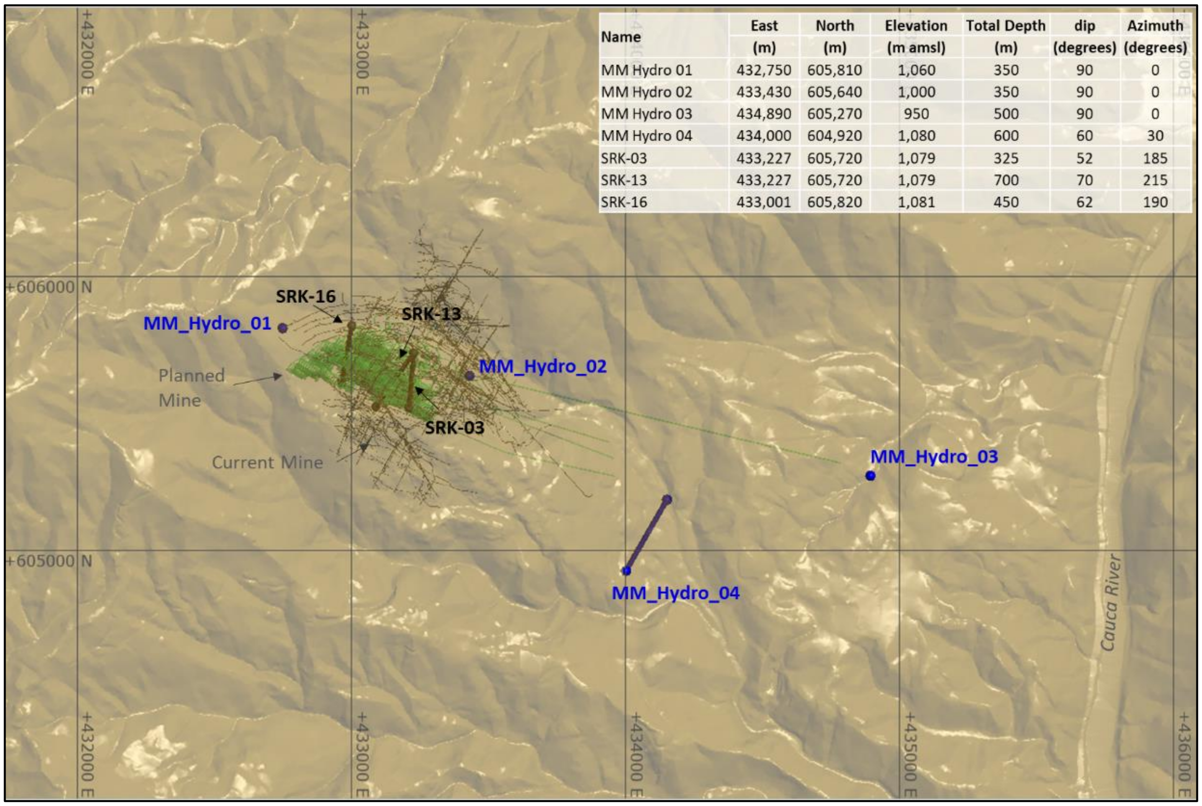
Source: SRK, 2019

Figure 16-27: Cross Section from Marmato Mine to Cauca River

Based on the current understanding of hydrogeological conditions, a minimum data set will be needed to meet a PFS-level hydrogeological analysis, SRK recommends the following work plan:

- Verify the field status of the 2012 piezometers presented in Table 16-7 and conduct a detailed QA/QC of the hydraulic tests performed in 2012. Based on the verification findings, a detailed work plan will be designed. The following preliminary tasks are strongly recommended:
  - Conduct short-term hydraulic tests (airlift test, slug test constant head test, etc.) in the available standpipe piezometers.
  - Conduct shutdown tests in the piezometers located La Macha Tank (Level 17).
  - Construct multiple VWP and/or standpipe piezometers upstream and low stream of planned underground mine (proposed Hydro 001 and 002).
  - Hydraulic testing of three NQ exploration/geotechnical holes from Level 21, (packer tests, flow/shut-in testing if flowing conditions or slug testing if water level is below underground developments) with a bottom target from 421 to 635 m amsl (boreholes SRK 3, SRK 13 and SRK 16).
  - Install piezometers/monitoring wells in three geotechnical holes (SRK 13 and SRK 16) for water level measurements (string of vibrating wire piezometers, VWPs, to define vertical hydraulic gradient or install pressure transducers in standpipes).
  - Install two deep-VWP targeting potential structures with connection to the Cauca River. (Hydro 3 Hydro 4).
  - Collecting groundwater samples in proposed piezometers.
- Continue monitor flow (total, from different levels). Surveying underground developments by collecting groundwater seepage rates, seepage-water strike elevations and water quality data.

Figure 16-28 shows the location of the proposed piezometers.

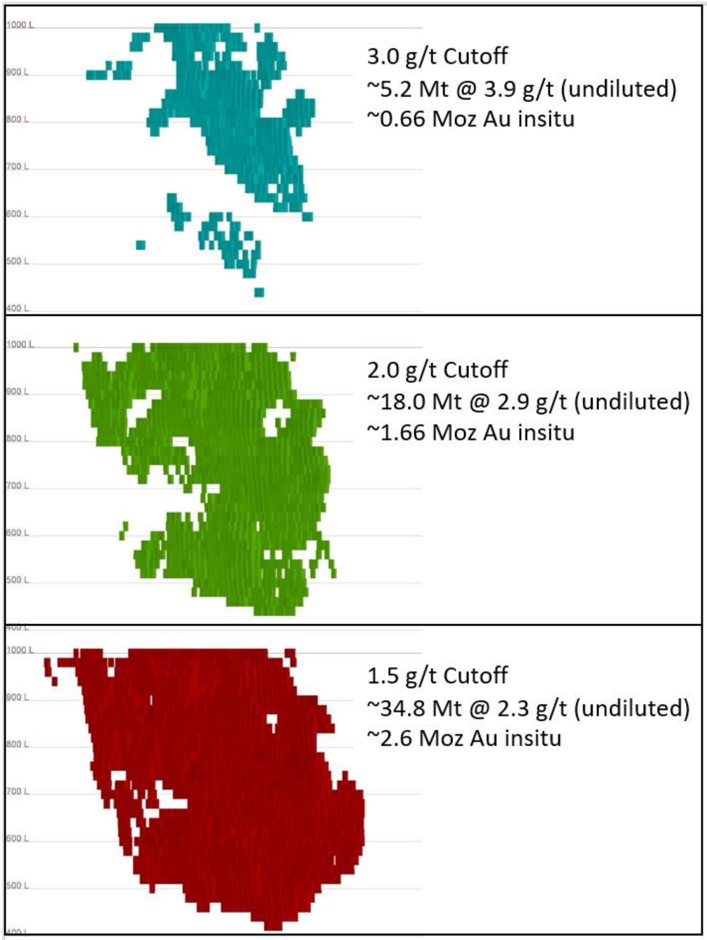


Source: SRK, 2019

Figure 16-28: Location of Proposed Piezometers

16.4.3 Stope Optimization

Stope optimization within Vulcan software was used to determine potentially economically minable material. Stope sizes used in the optimization ranges from 5 m wide and 10 m high to 15 m wide and 25 m high, with a minimum length of 5 m. Stope walls were vertical and wall dilution was not applied at the optimization stage. A 10 m width by 25 m height stope size was selected to maximize stope size yet keep dilution to a minimum. Optimizations were run using various CoG to identify higher grade mining areas and understand the sensitivity of the deposit to CoG. Results show large quantities of lower grade material where a small increase/decrease in CoG has a material impact on the material available for design. Figure 16-29 shows stope optimization results for cut-offs of 3.0 g/t, 2.0 g/t, and 1.5 g/t using a stope size of 10 m wide by 20 m high.



Source: SRK, 2019

Figure 16-29: Stope Optimization Results at Various Cut-offs (Looking Towards the Northeast)

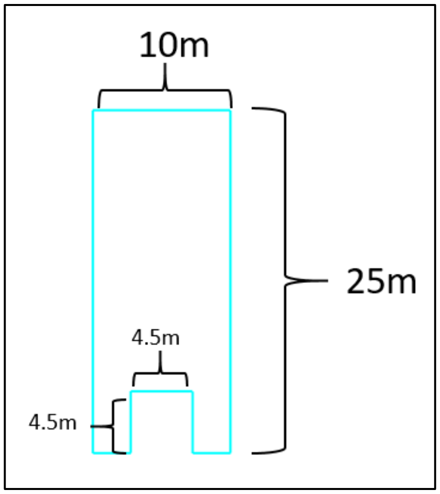
For mine planning work, a cut-off of 2 g/t was targeted for design work and stopes were extended to a 1.75 g/t cut-off where no additional development was required. For reporting within the design, a minimum cut-off of 1.5 g/t Au was used.

16.4.4 Mine Design

Stope optimization results were used as a basis for the underground mine design. The top of the MDZ mineralization is approximately 600 m below surface and extends to a depth of approximately 1,200 m below surface.

Stope Design

Stopes are 10 m wide and 25 m high with varying length. Each stope has a 4.5 m by 4.5 m access located at the bottom of the stope as shown in Figure 16-30. Top accesses are available on most levels to give access to stopes on the next level and to allow for backfilling. For upper most stopes in a block or where there is no mining above it is assumed a hole can be drilled from adjacent development into the stope for backfilling purposes. The stopes are drilled top down and rings are blasted from the end of a stope toward the access. The blasted material is remotely mucked from the stope access. A typical level is made up of approximately 40 stopes along strike.



Source: SRK, 2019

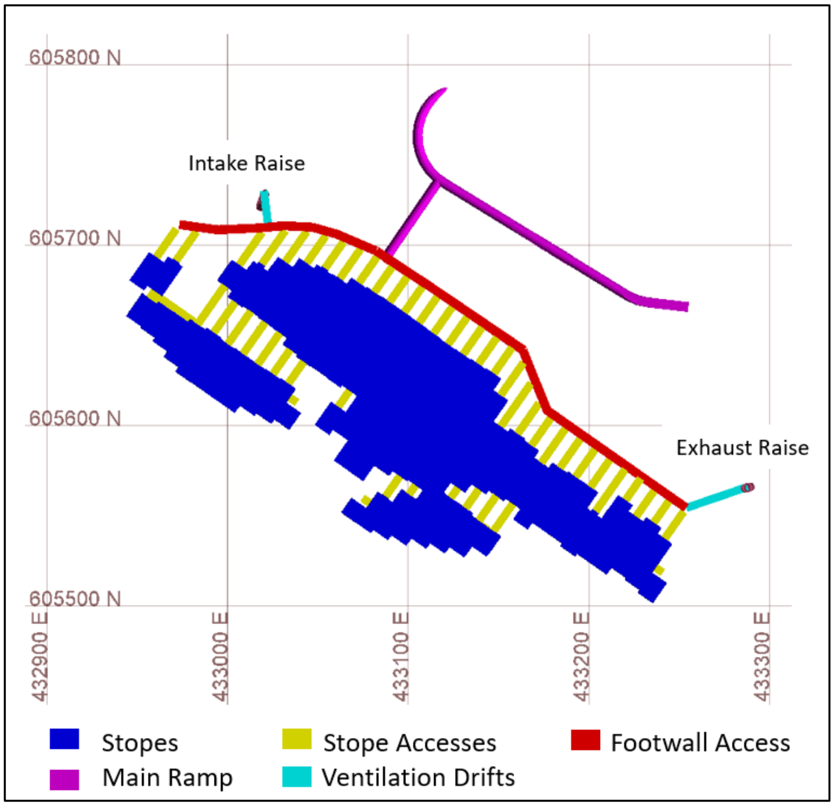
Figure 16-30: Stope Cross Section

A primary/secondary stoping sequence will be used, where on any given level, primary stopes must be separated by a secondary stope. Extraction of the secondary stope can only occur after the two immediately adjacent primary stopes have been mined, backfilled, and have had time to cure. Backfilling will be an integral part of the LHS mining cycle and a shorter cure time (7 to 14 days) is planned.

Development Design

The stope accesses are connected to a level access which is offset approximately 20 m away from the end of stopes. Each stope access typically connects to the level access except in cases where stopes are small and long development is required to reach the stope. In those instances, a connection from an adjacent stope is included in the design. This minimizes the amount of development, however it limits the sequencing order.

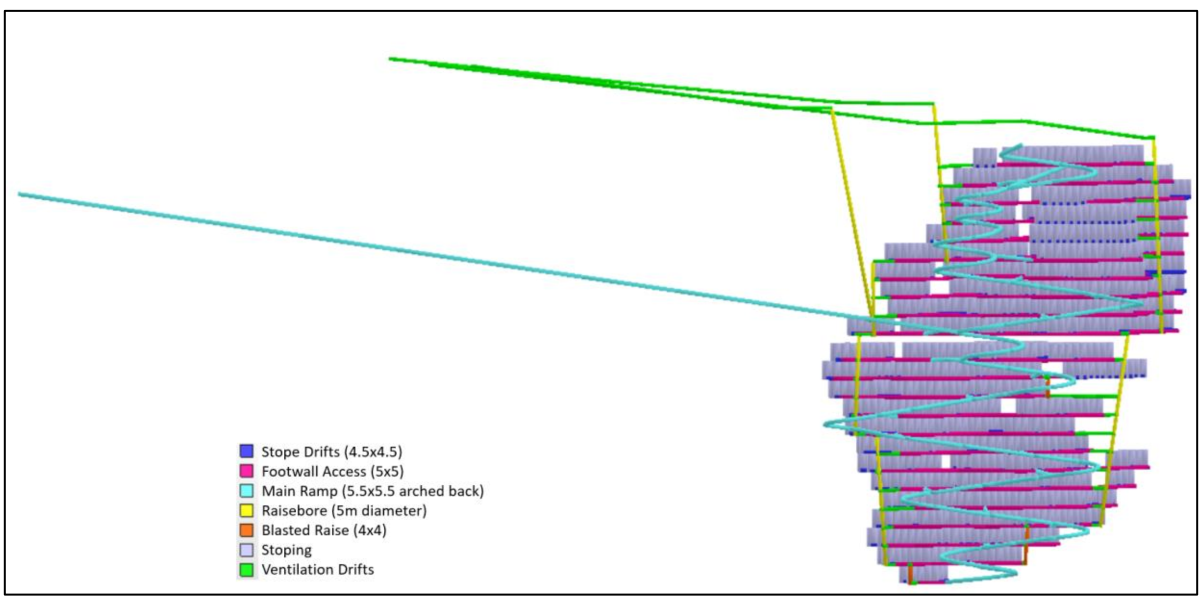
The level accesses connect to the main ramp which is offset at least 75 m from stoping into the footwall. On the northeast side of each level, the level access connects to an intake air ventilation raise and on the southeast side connects to an exhaust air raise. Figure 16-31 shows a typical level section.



Source: SRK, 2019

Figure 16-31: Typical Level Section

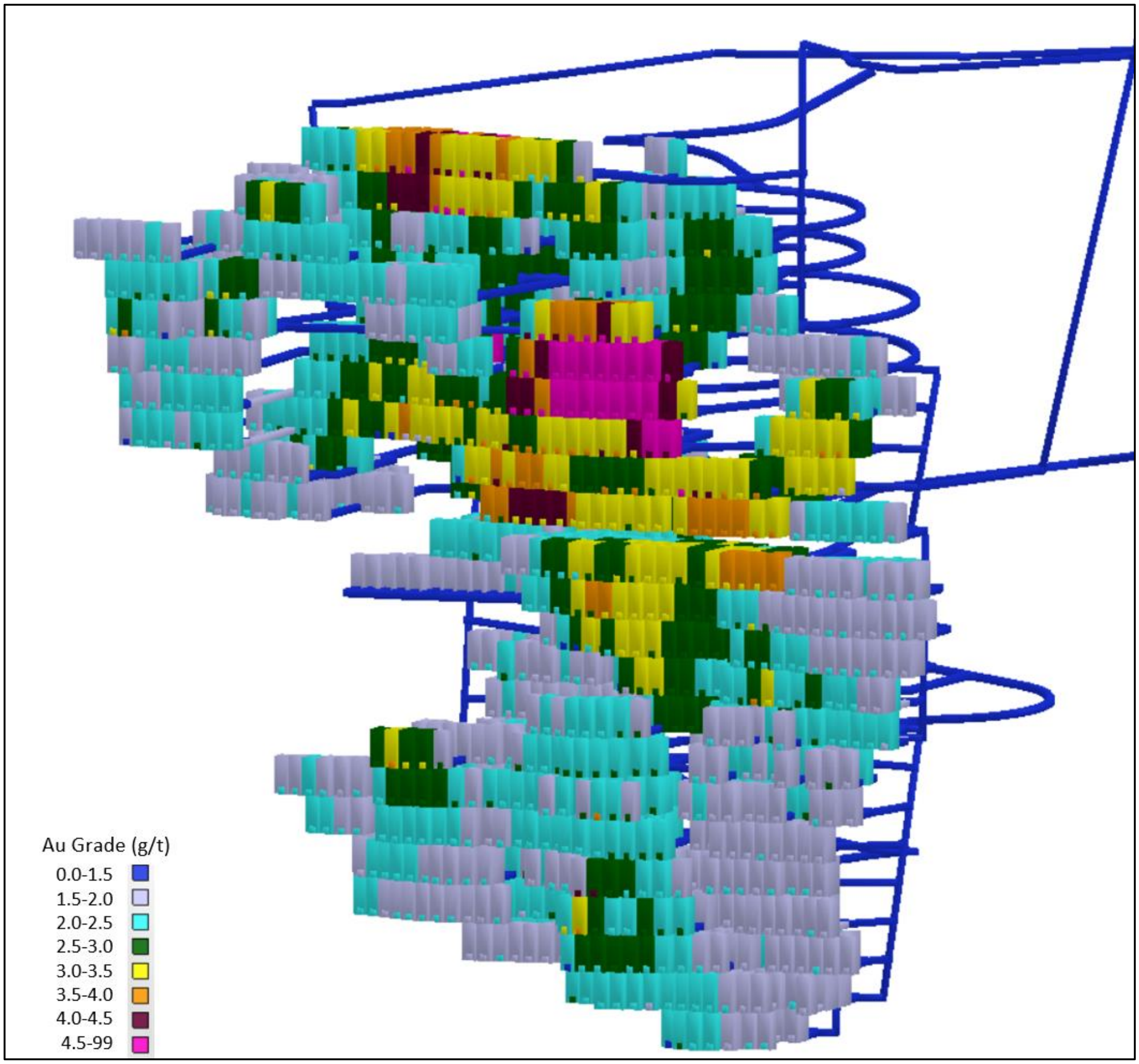
Infrastructure items underground, such as shops, sumps, etc. have not been designed at this time. Figure 16-32 shows the completed mine design.



Source: SRK, 2019

Figure 16-32: Completed Mine Design (Looking Towards the Southwest)

Figure 16-33 shows the mine design colored by Au grade. There is a distinct higher grade area in the upper mining block. This higher grade core will be sequenced first in the production schedule.



Source: SRK, 2019

Figure 16-33: Mine Design Colored By Au Grade g/t (Looking Towards the Northeast)

16.4.5 Mine Plan Resource

The underground mine design process for the MDZ area results in mine plan resources of 20.8 Mt (diluted) with an average grade of 2.5 g/t Au and 3.38 g/t Ag. This estimate is based on a mine design using a 1.75 g/t Au cut-off for the design of stopes and applying a 1.5 g/t Au cut-off to material within the design. These numbers include a 93% to 100% mining recovery based on type of opening (stope, development, etc.) to the designed wireframes in addition to a 0% to 7% unplanned waste dilution. An additional development allowance of 15% was applied to main ramps to account for detail currently not in the design. Dilution for stopes was applied at zero grade.

Table 16-16 summarizes the MDZ mine plan resources.

Table 16-16: MDZ Mine Plan Resource Classification (1)

Description	Tonnes (kt)	Au (g/t)	Ag (g/t)	Contained Au Oz (koz)	Contained Ag Oz (koz)
Indicated	3,360	2.80	4.66	302	503
Inferred	17,462	2.45	3.14	1,374	1,761
Total	**20,821**	**2.50**	**3.38**	**1,677**	**2,264**

Source: SRK, 2019
(1) Includes Indicated and Inferred material reported using a 1.5g/t Au cut-off.

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, in that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that the PEA will be realized.

16.4.6 Production Schedule

The production schedule is based on the rate assumptions shown in Table 16-17.

Table 16-17: Productivity Rates

Activity Type	Dimensions	Rate (1)
Main Ramps (single headings)	5.5 m by 5.5 m	4.6 m/d
Footwall Accesses (single headings)	5 m by 5 m	4.9 m/d
Stope Drifts (multiple headings)	4.5 m by 4.5 m	6.0 m/d
Ventilation Drifts (single headings)	4.5 m by 4.5m	6.0 m/d
Stoping (2)	-	2,125 tpd
Raisebored Raise	5.0 m diameter	3.4 m/d
Blasted Raise	4 m by 4 m	7.5 m/d
Backfilling	-	2,000 m^3/d

Source: SRK, 2019
(1) All rates are per face. Multiple areas/faces are mined together to generate the production schedule.
(2) Includes drilling, blasting, and mucking.

Backfill has been sequenced and typically includes a 14-day lag prior to mining stopes adjacent to backfill. The backfill sequence needs further refinement and detail at the next level of study.

The mining operation schedule is based on 365 days per year, seven days per week, with two 12 hour shifts each day. A production rate of 4,000 tpd was targeted with ramp-up to full production as quickly as possible.

Development drifts begin in January of 2021 and production commences mid-year 2023. Table 16-18 presents the annual mining schedule based on these assumptions. The annual schedule was completed using iGantt scheduling software.

Table 16-18: MDZ Annual Mining Schedule

Year	Mineralized Tonnes (kt)	Waste Tonnes (kt)	Mineralization Au (g/t)	Mineralization Ag (g/t)	Development Length (m)	Backfill Volume (m^3)
2021	-	180	-	-	2,790	-
2022	52	375	2.96	3.17	6,987	-
2023	831	318	3.21	3.79	9,966	222,594
2024	1,400	172	3.08	3.73	4,339	528,620
2025	1,401	67	3.48	4.09	2,273	536,495
2026	1,400	129	3.16	4.65	4,490	519,407
2027	1,397	242	2.59	3.57	5,968	514,815
2028	1,342	292	2.30	4.20	7,143	494,940
2029	1,400	214	2.26	5.19	6,272	511,787
2030	1,400	189	2.16	4.67	5,842	504,879
2031	1,400	175	2.05	2.49	5,780	502,542
2032	1,400	86	2.09	1.64	3,757	520,709
2033	1,400	92	2.18	1.79	3,871	516,445
2034	1,400	77	2.18	2.02	3,531	517,480
2035	1,400	55	2.22	2.82	3,261	522,577
2036	1,400	76	2.36	3.27	4,090	502,255
2037	1,205	2	2.47	3.06	122	488,351
2038	592	-	2.57	3.14	-	246,480
Totals	**20,821**	**2,739**	**2.50**	**3.38**	**80,482**	**7,650,376**

Source: SRK, 2019

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves.

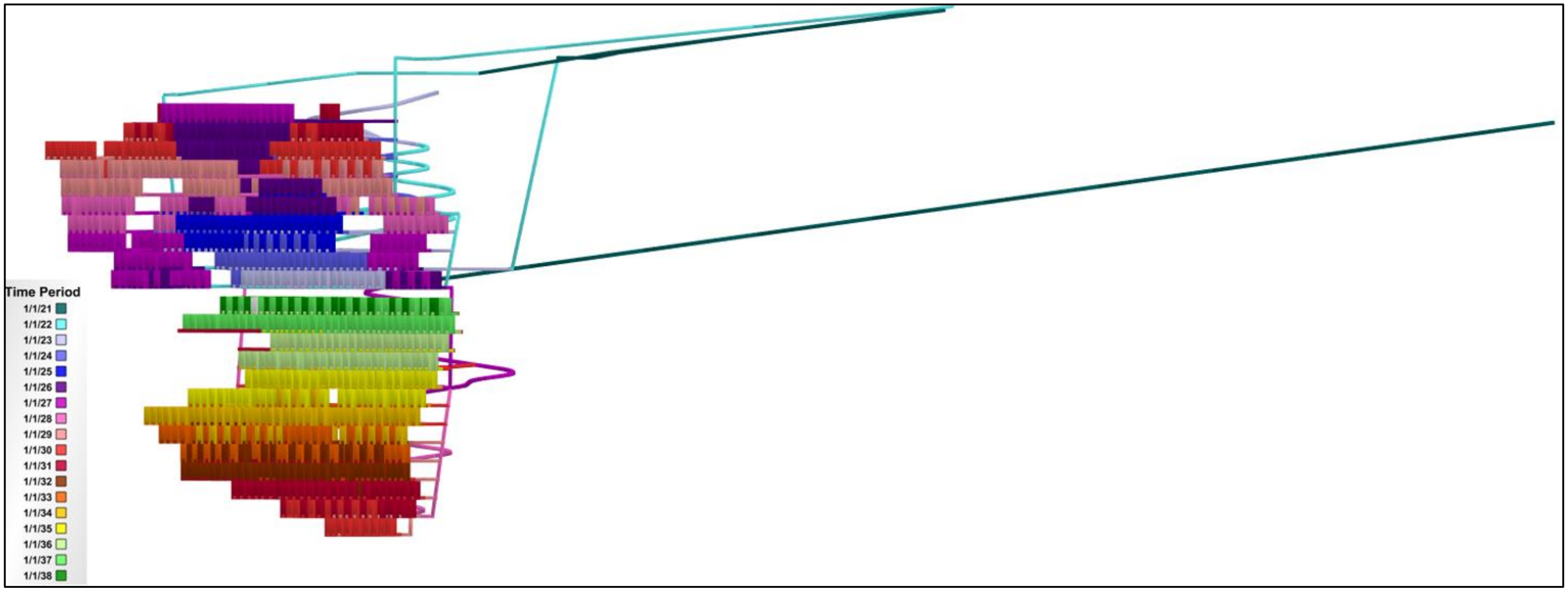
Table 16-19 summarizes the production schedule totals by development type.

Table 16-19: Production Schedule Totals by Activity Type

Develop/Production Type	Length (m)	Total Tonnes (kt)
Main Ramp (5.5x5.5)	8,671	615
Footwall Access (5x5)	10,091	681
Stope Drift (4.5x4.5)	55,894	3,056
Ventilation Drift (4.5x4.5)	4,171	228
Raisebore	1,458	74
Ventilation Slot Raise	102	4
Stoping		18,896
Total		**23,555**

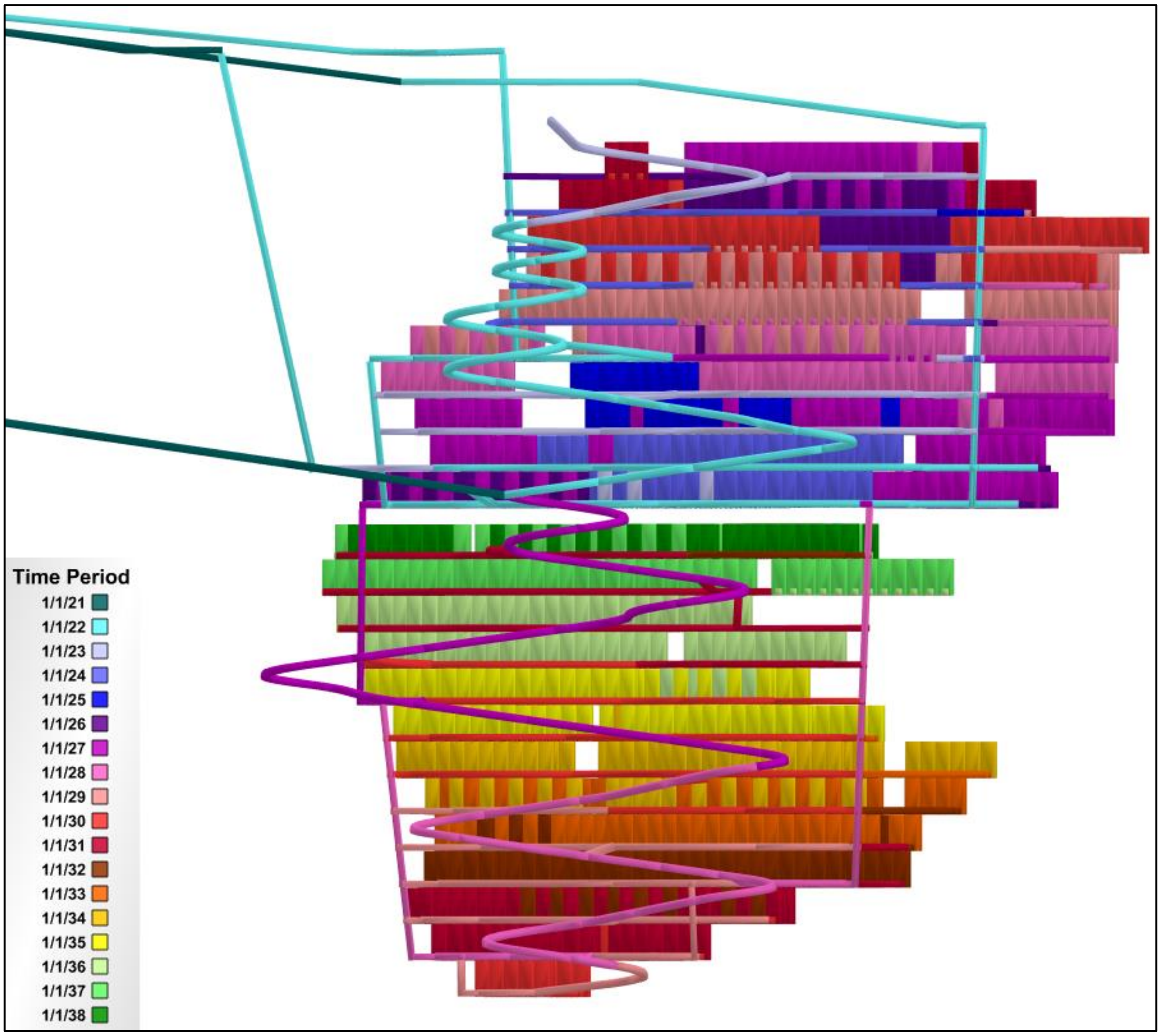
Source: SRK, 2019

Figure 16-34 and Figure 16-35 show the mine production schedule colored by year.



Source: SRK, 2019

Figure 16-34: Mine Production Schedule Colored by Year, Rotated View Looking Toward the Footwall (Northeast)



Source: SRK, 2019

Figure 16-35: Mine Production Schedule Colored by Year, Rotated View Looking Toward the Hangingwall (Southwest)

16.4.7 Mining Operations

Stoping

Stope lengths vary throughout the deposit ranging from 5 m to a maximum of 20 m giving a range of approximately 3,000 to 12,500 t per stope. After bottom and top accesses are established a slot raise will be developed at the far end of the stope (hangingwall side). Drilling will continue with the longhole drill using a fan shaped pattern. Holes will be loaded with bulk emulsion and stope blasting will commence in the slot and subsequently rings will be blasted retreating toward the level access.

Remote mucking will be required for the majority of stope mucking so the load-haul-dump (LHD) operator can remain behind the brow of the stope. Stope material will be mucked primarily into a muck bay near the level access or the adjacent stope access. The material will then be loaded into trucks and hauled to surface. Once the stope is emptied a bulkhead will be placed in the 4.5 m by 4.5 m access and the stope void will be filled with paste backfill from the top access or via a drillhole at the top of the stope.

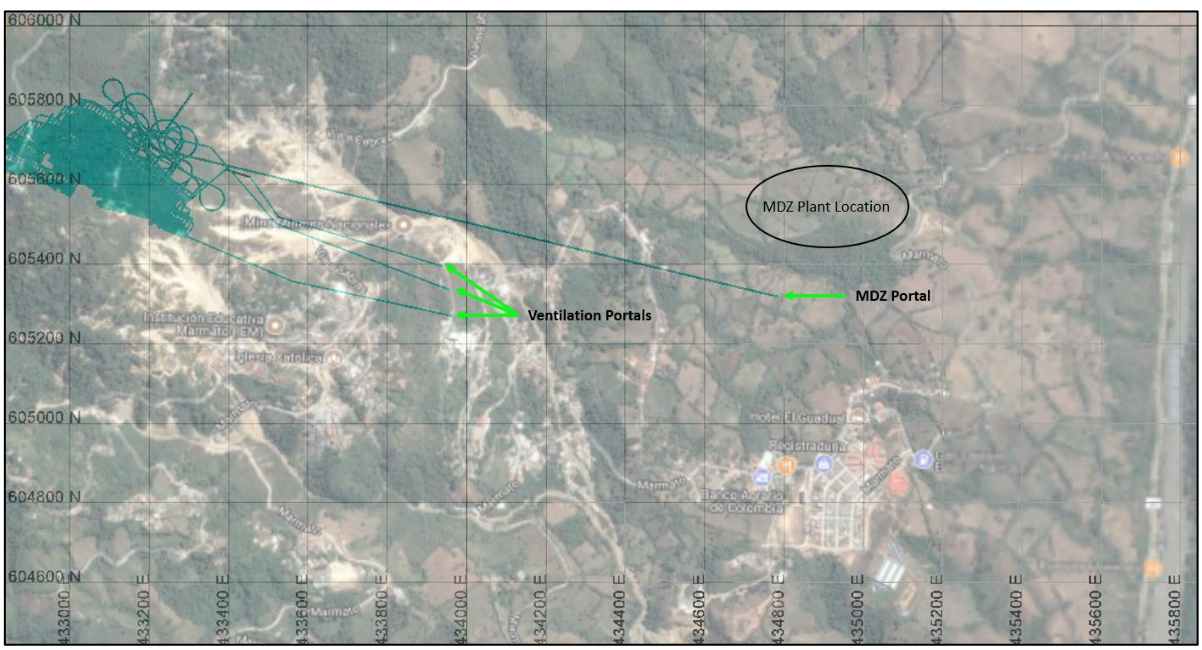
Development

Main ramps as 5.5 m by 5.5 m openings with an arched back. Footwall accesses are 5.0 m by 5.0 m with a flat back. Stope accesses and ventilation drifts are sized as 4.5 m by 4.5 m flat back openings. These dimensions provide enough room for equipment, ventilation ducting, and utilities where necessary. Main ramps are typically a single heading environment. Level accesses are also typically single heading environments. Stope accesses are multiple heading environments. All development will be mined using a double boom jumbo taking 4 m rounds. Blasted material will typically be mucked into a muck bay near the heading. The waste muck will subsequently be loaded into trucks and transported to surface or for disposal in the secondary stopes. As some stope access development is in mineralized material grade control will be required to determine material destination on a round by round basis.

The ramp system is designed at a maximum gradient of 14%. A turning radius of 30 m was used which is suitable for any underground truck.

Mine Access

The underground MDZ area will be accessed via a ramp from the process area facility. Three additional dedicated ventilation drifts will be developed from near the existing veins process facility as shown in Figure 16-36.



Source: SRK, 2019

Figure 16-36: Portal and Ventilation Drift Locations

An emergency escape system (bullet hoist) is envisioned in the fresh air raise system. The main ramp connects to existing workings and could also be used for emergency egress.

Haulage

The mine will incorporate the use of four 17 t load-haul-dump (LHD) loaders that muck material from stopes and development headings to either a muck bay or into 50 t underground trucks for haulage to surface. Early in the mine life average one-way haulage distances are approximately 2,000 m and five trucks are required. As haulage distance increases the truck count increases to nine. Where possible, waste material from development will be moved from muck bays and hauled to secondary stopes as backfill in conjunction with the pastefill.

Table 16-20 shows the maximum one-way haul distance by mining block and the number of trucks required. Note that the truck count includes waste haulage.

Table 16-20: Trucks and Haul Distance for Mineralized Material

Block	Max. One-Way Haul Distance (m)	Number of Trucks
Lower Portion of Upper Block	1,825	5
Upper Portion of Upper Block	3,230	7
Lower Portion of Lower Block	4,205	9

Source: SRK, 2019

Backfilling

A paste backfill plant will be located on surface and the paste backfill product will be made from tailings and using cement will be produced by the backfill plant and then placed underground through a piping system that will place the pastefill directly into the stopes. Barricades will be constructed in the stopes prior to placement. The primary/secondary extraction sequence requires the primaries to be backfilled with cemented pastefill having a minimum 14-day UCS strength of 1.0 MPa for a single face fill exposure during mining of the adjacent secondary stope in high-stress conditions. In lower stress conditions the minimum 14-day UCS strength can be 0.5 MPa. In secondary stopes where the backfill will never be exposed only sufficient binder is required to prevent liquefication of the backfill during mining operations. The paste backfill plant has been designed to fill 2,000 m^3/day when operating. Any waste rock mined underground and not hauled to surface will also be used as backfill material.

Ground Support

The current knowledge of the geotechnical characteristics indicates that ground support will be required in the ramps and primary access drifts as well as some of the stope access drifts. The ground support plan will use split set bolts as a standard. The bolting will be supplemented with wire mesh, shotcrete, and additional support where required. Cable bolts are expected to be utilized on the brows of the stopes and in certain larger intersections or infrastructure locations where warranted. A standard bolter and cable bolter will be utilized as normal practice and shotcrete equipment and transmixer are included in the estimate.

Grade Control

As the main ramp is developed, drill stations from the main ramp allow for fan drilling of the deposit prior to developing levels. This confirmatory drilling should be used to update the long-term block model and provide confidence in expected tonnage and grades prior to level development.

Once a level is being developed, level and stope accesses will be sampled to determine material destination. Once stope accesses are developed vertical holes will be drilled through the anticipated stopes and cuttings will be sampled to determine stope extents and estimated stope grades. Any samples tested in the lab should be used to update short term planning block models to better estimate tonnages and grades in the short term mine plan.

16.4.8 Ventilation

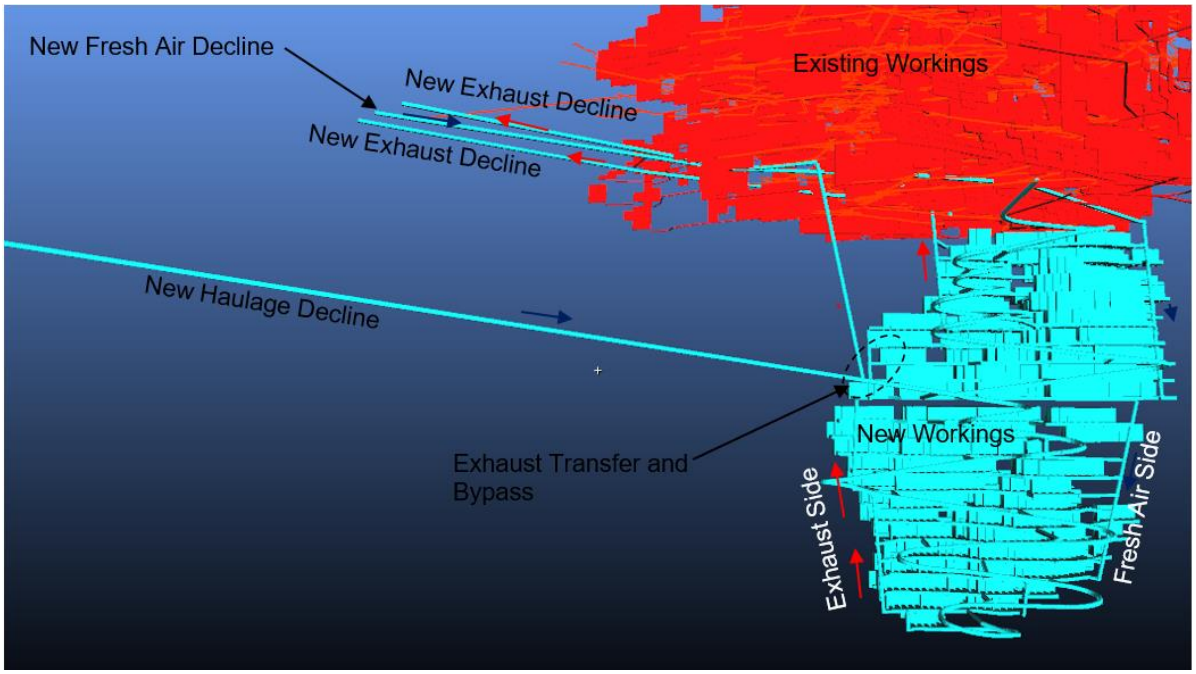
The overall ventilation design is developed to meet the Colombian mining regulations and general best practices. The main regulations for this mine are identified in in “Reglamento de Suguridad en Las Labores Mineras Subterraneas, Decreto 1886 del 21 de Septembre de 2015”. Main relevant points from this regulation are:

- Article 56 identifies the maximum allowable air velocity in an area where personnel are working or transiting through as 6 m/s. This is used to develop the dimensions of the main access decline and to provide the split between the fresh air system and the ramp air system.
- Article 54 dictates an airflow requirement for 0.09 m^3/s/kW (for newer low CO producing diesels). The MDZ Project envisions the implementation of a new equipment fleet consisting of Tier 3 or Tier 4 equipment and a lower dilution value of 0.06 m^3/s/kW may be considered. A variance would need to be applied for and as such, this PEA aligns with the higher airflow requirement.
- Article 42 states that the separation between intake and exhaust portals should be no less than 50 m.

The ventilation system has been initially developed for this PEA as an isolated system. The existing upper workings will only be connected to the lower workings for the purposes of egress and the access points will be isolated with air doors.

Ventilation Layout

The layout of the MDZ area below the existing workings is shown in Figure 16-37. Four new accesses will be developed; one 5.5 m by 5.5 m haulage/access decline to the middle of the deposit, and three 4.5 m by 4.5 m declines to the top of the MDZ area.



Source: SRK, 2019

Figure 16-37: General Ventilation Layout

Two exhaust fan installations are required, one in each of the upper access declines either on the surface with vertical discharges or in the decline. The exhaust system will have a transfer drift so that both systems will operate in parallel.

Equipment Load and Airflow Requirement

The equipment list is based on other similar mining operations with a basic airflow requirement of 0.09 $m^3/s/kW$ (370.5 m^3/s) as per Colombian regulations. These values are shown in Table 16-21.

A separate calculation was performed for a dilution rate of 0.06 $m^3/s/kW$ (261.7 m^3/s) as per Canadian and European requirements, shown in Table 16-22. These values are shown but would require a variance from Colombian regulators. With the high-power demand required for the system, it may be practical to investigate the variance for the next phase of study.

Table 16-21: Equipment Load and Airflow Requirement for 0.09 $m^3/s/kW$ Dilution Rate

Equipment Type	Number of Pieces	Number of Personnel	Power (kW)	Availability	Operating on the Surface (Diesel)	Operating in the Mine (Diesel)	Applied Power in the Mine (Diesel)	Airflow Requirement (m^3/s)
Drill Longhole In the Hole - Sandvik DU 412i	1		130	85%		10%	11	1.0
Drill Longhole Top Hammer - Sandvik DL 431	1		110	85%		10%	9	0.8
Explosives Charger (large)-Stopes/Raises - Orica MaxiCharger 5344	1		*120*	85%		20%	20	1.8
LHD (14T/6.2m3) - Sandvik LH 514	4		256	85%		75%	653	58.8
Drill Jumbo - 2 Boom - Sandvik DD 422i	4		119	85%		10%	40	3.6
Explosives Charger (small)-Drifts - Orica Handiloader 1120 (U100)	1							
Bolter - Sandvik DS 411 Mechanical Bolter	3		110	85%		10%	28	2.5
Scissor Lift - Getman A64	2		129	85%		10%	22	2.0
Haul Truck (50T) - Sandvik TH550	8		405	85%	40%	60%	1652	148.7
Cable Bolter - Atlas Copco Cabletec L	1		120	85%		10%	10	0.9
LHD (7T) - Sandvik LH307	3		160	85%		75%	306	27.5
Boom Truck - Getman A64	1		129	85%		20%	22	2.0
Lube Truck - Getman A64	1		129	85%		20%	22	2.0
Fuel Truck - Getman A64	1		129	85%		20%	22	2.0
Flat Bed - Getman A64 (for small explosives charger and misc. use)	1		129	85%		10%	11	1.0
Shotcrete Sprayer Trailer - (38.2 m3/hr)	1							
Transmixer Truck - Getman A64 HD R60	1		129	85%		50%	55	4.9
Telehandler - CAT 1255D	1		106	85%		20%	18	1.6
Skid Steer - CAT 272D	2		73	85%		10%	12	1.1
Personnel Carriers - Kubota RTV 1120D	10		19	85%		50%	81	7.3
Personnel Carrier - Getman A64 (16 person)	1		129	85%		10%	11	1.0
Underground Core Drill - Sandvik DE130i or equivalent	2		0	85%		10%	0	0.0
Grader - CAT UG20M	1		105	85%		10%	9	0.8
Light Vehicles (4x4 Pickup Trucks, included in surface)	5		75	85%		10%	32	2.9
982M CAT Front End Loader - 6.7m3/8.75 dy3	1		325	85%		75%	207	18.6
Charger base - Getman A64 (for small and large chargers)	2		129	85%		20%	44	3.9
Total Airflow Required for Diesel Dilution (0.09 $m^3/s/kW$)							3298	296.8
Personnel (0.05 $m^3/s/person$)		200						10.0
Shop/Facilities								30.0
Leakage (10%)								33.7
Total Airflow Requirement								**370.5**

Source: SRK, 2019

Table 16-22: Equipment Load and Airflow Requirement for 0.06 m³/s/kW Dilution Rate

Equipment Type	Number of Pieces	Number of Personnel	Power (kW)	Availability	Operating on the Surface (Diesel)	Operating in the Mine (Diesel)	Applied Power in the Mine (Diesel)	Airflow Requirement (m^3/s)
Drill Longhole In the Hole - Sandvik DU 412i	1		130	85%		10%	11	0.7
Drill Longhole Top Hammer - Sandvik DL 431	1		110	85%		10%	9	0.6
Explosives Charger (large)-Stopes/Raises - Orica MaxiCharger 5344	1		*120*	85%		20%	20	1.2
LHD (14T/6.2m3) - Sandvik LH 514	4		256	85%		75%	653	39.2
Drill Jumbo - 2 Boom - Sandvik DD 422i	4		119	85%		10%	40	2.4
Explosives Charger (small)-Drifts - Orica Handiloader 1120 (U100)	1							
Bolter - Sandvik DS 411 Mechanical Bolter	3		110	85%		10%	28	1.7
Scissor Lift - Getman A64	2		129	85%		10%	22	1.3
Haul Truck (50T) - Sandvik TH550	8		405	85%	40%	60%	1652	99.1
Cable Bolter - Atlas Copco Cabletec L	1		120	85%		10%	10	0.6
LHD (7T) - Sandvik LH307	3		160	85%		75%	306	18.4
Boom Truck - Getman A64	1		129	85%		20%	22	1.3
Lube Truck - Getman A64	1		129	85%		20%	22	1.3
Fuel Truck - Getman A64	1		129	85%		20%	22	1.3
Flat Bed - Getman A64 (for small explosives charger and misc. use)	1		129	85%		10%	11	0.7
Shotcrete Sprayer Trailer - (38.2 m3/hr)	1							
Transmixer Truck - Getman A64 HD R60	1		129	85%		50%	55	3.3
Telehandler - CAT 1255D	1		106	85%		20%	18	1.1
Skid Steer - CAT 272D	2		73	85%		10%	12	0.7
Personnel Carriers - Kubota RTV 1120D	10		19	85%		50%	81	4.8
Personnel Carrier - Getman A64 (16 person)	1		129	85%		10%	11	0.7
Underground Core Drill - Sandvik DE130i or equivalent	2		0	85%		10%	0	0.0
Grader - CAT UG20M	1		105	85%		10%	9	0.5
Light Vehicles (4x4 Pickup Trucks, included in surface)	5		75	85%		10%	32	1.9
982M CAT Front End Loader - 6.7m3/8.75 dy3	1		325	85%		75%	207	12.4
Charger base - Getman A64 (for small and large chargers)	2		129	85%		20%	44	2.6
Total Airflow Required for Diesel Dilution (0.09 $m^3/s/kW$)							3298	197.9
Personnel (0.05 m^3/s/person)		200						10.0
Shop/Facilities								30.0
Leakage (10%)								23.8
Total Airflow Requirement								**261.7**

Source: SRK, 2019

Main Decline Auxiliary Ventilation System

A reduced ventilation load, with a single haul truck operating in the decline, was considered with equipment specified in Table 16-23. A service vehicle is assumed to be in the decline because of the slower use of the haul trucks.

Table 16-23: Reduced Equipment Load for Decline Development

Equipment	Power (kW)	Airflow (m^3/s)
Haul Truck TH550	405	36.5
LHD Sandvik LH307	160	14.4
Service Equipment	129	11.6

Source: SRK, 2019

This places the delivered airflow requirement at 62.5 m3/s for a diesel dilution rate of 0.09 m3/s/kW. The required ventilation system is shown in Table 16-24.

Table 16-24: Suggested Auxiliary Ventilation System for Decline Development

Area	Quantity	Duty Point			Length	Diameter
		(m^3/s)	(kPa)	(kW)	(m)	(m)
Main Decline	2	40.9	7.4	465	1800	1.6

Source: SRK, 2019

A total of four fans will be required at 40.9 m^3/s at 3.7kPa, 41 m^3/s, 200 kW. The fan system could be applied in two different configurations; all fan pressure could be applied at the surface outside of the portal, or the fan pressure could be applied in two locations (outside the portal and approximately 1/3 of the way into the drift). The staged approach would be preferred because the reduced differential pressure across the duct would provide a decreased level of leakage. By installing the separated fans, the duct between the portal and the second fan will need to be rigid; either steel, fiberglass, or reinforced plastic duct.

Stope Ventilation Systems

The stope ventilation systems are identified for two different orientations; main body, and fringe area, with the difference being the length of the auxiliary ventilation systems. The main body stopes would take airflow from the footwall directly, the fringe stopes have a longer access and require a longer duct. Haul trucks are assumed to be loaded on the footwall which is naturally ventilated in the main area, however, the footwall is not directly ventilated from the fresh air raise which will require the installation of a separate auxiliary system.

The auxiliary ventilation system required in the main area is assumed to ventilate three stopes off of a single ventilation duct. The general length of the access is assumed to be approximately 100 m. The general operating equipment load for a stope area is assumed as shown in Table 16-25.

Table 16-25: Main Body Stope Ventilation Airflow and Equipment

Equipment	Power (kW)	Airflow (m^3/s)
LHD Sandvik LH307	160	14.4
Service Equipment	129	11.6
Drilling/Support		10.0

Source: SRK, 2019

This places the delivered airflow requirement at 36 m^3/s for a diesel dilution rate of 0.09 $m^3/s/kW$. This equipment load represents a maximum configuration with the simultaneous operation of the equipment. There will be other configurations representing different parts of the mining cycle; bolting, blasting, etc.

The required ventilation system for this area is shown in Table 16-26

Table 16-26: Main Body Stope Auxiliary Ventilation System

Scenario	Fan Pressure (kPa)	Fan Airflow Modelled Leakage (m^3/s)	Additional 15% Leakage (m^3/s)	Total Duct Airflow at Fan (m^3/s)	Fan Power at 75% efficiency (kW)
Single 1.3m Duct	1.4	40.0			75

Source: SRK, 2019

The fringe stope areas have a longer than average access length which will likely require ventilation to support the operation of haul trucks in the footwall in addition to the stope production equipment. The auxiliary ventilation system required for a series of stopes is assumed to ventilate three stopes off of a single ventilation duct

This places the delivered airflow requirement at 36 m^3/s for a diesel dilution rate of 0.09 $m^3/s/kW$. The average length of the extended accesses would be approximately 200 m. The required ventilation system for this area is shown in Table 16-27.

Table 16-27: Fringe Stope Auxiliary Ventilation System

Scenario	Fan Pressure (Kpa)	Total Duct Airflow at Fan (M^3/S)	Fan Power at 75% Efficiency (Kw)
Single 1.3m Duct	2.4	42.3	140

Source: SRK, 2019

An additional auxiliary ventilation system is required to support the operation (loading) of a haul truck in the long fringe access as shown in Table 16-28. This auxiliary system would be used in addition to the auxiliary ventilation system for the stope ventilation. The system could be turned on or off depending upon the operation of the haul truck.

Table 16-28: Fringe Stope Haulage Airflow Requirement

Equipment	Power (kW)	Airflow (m^3/s)
Haul Truck TH550	405	36.5

Source: SRK, 2019

This places the delivered airflow requirement at 36.5 m^3/s for a diesel dilution rate of 0.09 $m^3/s/kW$. The auxiliary ventilation system required for the haul trucks in the fringe drive is shown in Table 16-29.

Table 16-29: Fringe Stope Haulage Auxiliary Ventilation System

Scenario	Fan Pressure (kPa)	Fan Airflow Modelled Leakage (m^3/s)	Additional 15% Leakage (m^3/s)	Total Duct Airflow at Fan (m^3/s)	Fan Power at 75% efficiency (kW)
Single 1.3m Duct	3.3	36.8	5.5	42.3	190

Source: SRK, 2019

It is assumed that there would be four active areas in the main body and four active areas in the extended or fringe areas. A summary of the systems is shown in Table 16-30.

Table 16-30: Stope Auxiliary Ventilation System Summary

Area	Quantity	Duty Point			Length	Diameter
		(m^3/s)	(kPa)	(kW)	(m)	(m)
Main Zone Stope	4	40.0	1.4	75	100	1.3
Fringe Area Stope	4	42.3	2.4	140	200	1.3
Fringe Area Haulage	4	42.3	3.3	190	200	1.3

Source: SRK, 2019

Main Ventilation System

The pressure losses through the ventilation system were determined by calculating the basic resistance of the ventilation route and applying the square to the system with a generalized airflow distribution. Two levels of airflow were analyzed based on different diesel dilution.

The primary system was based on the higher level of diesel dilution. All airways were sized based on this criterion. When considering the lower airflow criteria, the raises and level drift dimensions were held constant with the only change being the airflow quantity.

The pressure losses along the highest resistance circuit are summed together to determine the fan operating pressure. The parallel airways that have a lower pressure loss would be controlled by the addition of a regulator to the level which would balance the pressures between circuits. This represents a very basic analysis for the determination of the fan operating pressure. An additional 15% was added to the calculated operating pressure to account for additional shock losses and transient conditions. It is assumed that the fans will be installed underground for both control, security and for the minimization of surface structures. An additional 750 Pa was added to the operating pressure to account for entry and discharge losses for the fan installation. It was assumed that inlet bells and discharge cones would be added to each installation. The actual value of this loss would be calculated based on the fan diameter and discharge air velocity.

Two analyses were developed and are summarized in Table 16-31 and Table 16-32. The two summaries relate to the two different diesel dilution criteria. For the PEA the higher diesel dilution criteria should be used, however, the utilization of a lower criteria should be considered for the next level of the study (PFS).

It was assumed that the fan efficiency would be approximately 75%, and an additional leakage of 5% of the total airflow would be drawn from the upper mine workings.

Table 16-31: Exhaust Fan Pressure, Airflow, and Power Summary (0.09 $m^3/s/kW$)

Category	Pressure (Pa)
Fresh Air Side	
System Pressure	896
15% Additional Loss	134
Total	**1,030**
Exhaust Air Side	
System Pressure	1,723
15% Additional Loss	258
Total	**1,982**
Total Fan Pressure (Pa)	3,012
Installation Losses (Pa)	750
Fan Airflow (m^3/s)	389
Fan Power (kW) Total	1,951

Source: SRK, 2019

With two fans mounted in parallel drifts (one for each of the two exhaust portals) each fan would be required to provide 195 m^3/s at 3.76 kPa and have a power capacity of at least 975 kW.

Table 16-32: Exhaust Fan Pressure, Airflow, and Power Summary (0.06 $m^3/s/kW$)

Category	Pressure (Pa)
Fresh Air Side	
System Pressure	896
15% Additional Loss	134
Total	**1,030**
Exhaust Air Side	
System Pressure	839
15% Additional Loss	126
Total	**965**
Total Fan Pressure (Pa)	1,995
Installation Losses (Pa)	750
Fan Airflow (m^3/s)	275
Fan Power (kW) Total	1,006

Source: SRK, 2019

For the lower diesel dilution criteria two fans are required to be mounted in parallel with each fan required to provide 138 m^3/s at 2.75 kPa and have a power capacity of at least 500 kW.

16.4.9 Mine Services

Pumping

Based on the water inflow work discussed in section 16.4.2, a 220 L/s (3,200 gpm) capacity, 110 L/s 1600 gpm) operating, dewatering system is planned.

The system will be built in phases as the mine develops and consists of a portable development system and permanent level pump stations that will be constructed above the sill levels as the mine production levels are constructed. The system will pump from the underground workings through steel and SDR11 8-inch HDPE pipes to the MDZ plant where the water will be used at the processing plant.

The portable development system will be used from the portal to the level of Upper Block. Once the permanent pump system is completed on the MDZ upper zone base level and piping out through the decline to the surface is completed, the portable system will be removed from the decline and utilized for the development decline progressing to the MDZ lower zone. A second permanent pump station will be established on the base of the MDZ lower zone that will pump from the MDZ lower zone level to the MDZ upper zone base permanent pumping sump, where the MDZ upper zone level pumps will pump the water to the surface for discharge to the processing plant.

The portable development system will consist of four complete pump skid stations that provide 110 L/s (1,600 gpm) at 59 m each with a 28.4 m^3 capacity tank with electrical motor control center (MCC) mounted on the skid.

Permanent pump stations will be established at the bottom of the first mine block near the bottom of the initial access ramp and at the bottom of the bottom block once development is complete in that area.

Face pumps will be employed at the development faces and will pump to the stage pumping skids that feed the main block sump.

Electrical Supply

All surface facilities will be supplied by power from a new MDZ Project main substation that will tie into an existing power infrastructure in the area. The main surface substation will be located near the new plant facility and will feed a new mine substation located on the surface near the portal. The mine substation will feed the underground electrical systems, infrastructure, and equipment power down the main haulage decline via a 13.2 kV feed line. The 13.2 kV power will feed throughout the mine to main load centers where the power will be stepped down to 480v for underground equipment use. Feeds will be provided at 220v for auxiliary use in the shops and for smaller loads such as fans, pumps, and auxiliary lighting.

A diesel backup generator at the surface will supply backup power for the required ventilation systems to maintain minimum ventilation requirements in the case of emergency.

Health & Safety

The mine design incorporates Colombian safety standards and includes emergency egress through the fresh air ventilation decline at the upper MDZ zone of the mine as well as access into the active mine above. The MDZ bottom zone egress plan will include an emergency hoist in the return air raise. A stench air system will be available to notify employees of the need to exit the mine. Additionally, multi-person refuge chambers are included that will be located in active working areas over the LoM.

The mine will have a communications system that has both mine phones and wireless communication through a leaky feeder system. The mine safety program will integrate with local providers in case of any mine emergency. A stench gas emergency warning system will be installed in the mine's intake ventilation system. This system can be activated to warn underground employees of a fire situation or other emergency whereupon emergency procedures will be followed.

Workforce

Workforce levels are estimated based on the production schedule and equipment needs. The productivities used reflect a mix of local and skilled labor with an experienced management team.

Manpower levels are estimated based on the production schedule and equipment needs. The productivities used reflect a mix of local and skilled labor with an experienced management team.

The estimate is based on owner mining using an operating schedule consisting of 12 hours per shift, two shifts per day, and seven days per week. The 12 hour shift is supported by a four crew rotation. The management and technical team are planned to work five 8 hour days per week.

Table 16-33 shows the estimated MDZ mine workforce. The rotating crews will have a split of 66 people underground and 8 people on the surface. It is expected that the maximum personnel underground would be 72 per shift. The workforce will increase over time through the addition of staff to operate additional equipment.

Table 16-33: Typical Mining Labor by Shift

Day Shift (salaried)		**Days**
Mine Superintendent		3
Mine Planner		2
Maintenance Superintendent		1
Maintenance Planner		2
Maintenance Technician		1
Senior Mining Engineer		1
Geotechnical Engineer		2
Mine Planning Engineer		3
Surveyor		3
Mine Technologist		2
Geologist		5
Total		**25**
Rotating Shift (hourly)	**Per Shift**	**Total**
Mine Supervisor/Shift Boss	3	15
Grade Control Geologist	2	8
Geotechnical Engineer	2	8
Safety/Mine Rescue/Training Supervisor	2	8
Surface Equipment Operator	2	8
Blasters	4	16
Ground Support, Hanging Services	6	24
Fuel/Lube/Boom/Grader/Telehandler	2	8
LHD & Truck Operator	8	32
Longhole and Jumbo Operator	4	16
Laborer	4	16
Diamond Driller	4	16
Backfill Crew - Bulkheads, Piping, Monitor	6	24
Paste Backfill Plant Operators-Surface	2	8
Mine Maintenance Supervisor/Lead Hand	1	4
Mechanic	6	24
Mechanic Helper	6	24
Electrician	4	16
Total	74	299
Grand Total	**99**	**324**

Source: SRK, 2019

Equipment

The underground equipment used, shown in Table 16-34, is typical for a sublevel stoping mining method with the number of pieces of equipment calculated from the production rates and typical availabilities for underground mines. The later years include additional trucks and LHDs due to increasing haul distance.

Table 16-34: Mobile Equipment Life of Mine Summary

Type	2021	2022	2023	2024	2025	2026	2027	Total Units
Drill Longhole In the Hole - Sandvik DU 421		1	1					2
Explosives Charger (large)-Stopes/Raises - Orica MaxiCharger 5344		1						1
LHD (17T/7.3m3) - Sandvik LH 517		3	1					4
Drill Jumbo - 2 Boom - Sandvik DD 422i		3	1					4
Explosives Charger (small)-Drifts - Orica Handiloader 1120 (U100)		1						1
Bolter - Sandvik DS 411 Mechanical Bolter		2	1					3
Scissor Lift - Getman A64		2						2
Haul Truck (50T) - Sandvik TH551i		4	1		2		2	9
Cable Bolter - Sandvik DS421		1						1
LHD (7T) - Sandvik LH307		2	1					3
Boom Truck - Getman A64		1						1
Lube Truck - Getman A64		1						1
Fuel Truck - Getman A64		1						1
Flat Bed - Getman A64 (for small explosives charger and misc. use)		1						1
Shotcrete Sprayer Trailer - (38.2 m3/hr)		1						1
Transmixer Truck - Getman A64 HD R60		1						1
Telehandler - CAT 1255D		1						1
Skid Steer - CAT 272D		2						2
Personnel Carriers - Kubota RTV 1120D		5	5					10
Personnel Carrier - Getman A64 (16 person)		2						2
Underground Core Drill - Sandvik DE130i or equivalent		2						2
Mobile Trailer Compressors (350cfm)		2						
Grader - CAT UG20M		1						1
Light Vehicles (4x4 Pickup Trucks, included in surface)	2	5						7
982M CAT Front End Loader - 6.7m3/8.75 dy3		1						1
Charger base - Getman A64 (for small and large chargers)		1	1					2

Source: SRK, 2019

The mine will also have surface equipment and facilities that is summarized in Table 16-35. The underground facilities and equipment are summarized in Table 16-36.

Table 16-35: Mine Surface Equipment and Facilities Summary

Surface Facilities and Equipment
Portal
Power Supply and Mine Substation
Preproduction Fan System
Shipping Container Storage
Engineering Equipment Allowance
Main power line to UG yard (estimate 1km)
Mine Substation
Backup Generator for ventilation system
Dry and Office Building
Maintenance Shop (Truck Shop)
Tool Allowance for Shop
Communications Infrastructure and Mine automation
Mine Water Tank
Paste Backfill Plant

Source: SRK, 2019

Table 16-36: Underground Facilities and Equipment

UG Shop
Tool allowance for UG Shop
Main Ventilation Fans and Install
Refuge Chambers (12 person)
Refuge Chambers (8 person)
Mobile 200 amp welders
Mobile 400 amp welder
Portable Power Center 500 kVA (for mobile equipment)
Portable Power Center 1500 kVA (for pump stations - use 2 for portal)
Pump System for main level
Pump System for level -bottom of mine
Pump Skid System during development
Stope Fans
Ventilation misc (air doors, regulators, bulkheads)
Misc. Piping
Paste Backfill Piping and System
Powder and Primer Magazines
Fuel Delivery System (lines and pump system)

Source: SRK, 2019

17 Recovery Methods

GCM operates a 1,200 tpd process plant to recover gold and silver values from material produced from current Marmato mining operations in the Upper Zone. In addition, GCM is evaluating the development of the MDZ, which is below the current mining operations and the construction of a new 4,000 tpd plant to process material soley from the MDZ. Recovery methods currently in use for processing Marmato material and recovery methods being evaluated for processing MDZ material are presented and discussed in this section.

17.1 Marmato Process Plant (Current Operations)

The Marmato process plant flowsheet incorporates unit operations that are standard to the industry and includes:

- Three-stage crushing;
- Closed circuit ball mill grinding;
- Gravity concentration;
- Flotation;
- Flotation and gravity concentrate regrind;
- Cyanidation of the flotation and gravity concentrates;
- Counter-current-decantation;
- Merrill-Crowe zinc precipitation; and
- Smelting of precipitates to produce final dore’ product.

The Marmato process plant flowsheet is shown in Figure 17-1 and a list of major equipment is shown in Table 17-1. Each of the process unit operations is briefly described in this section.

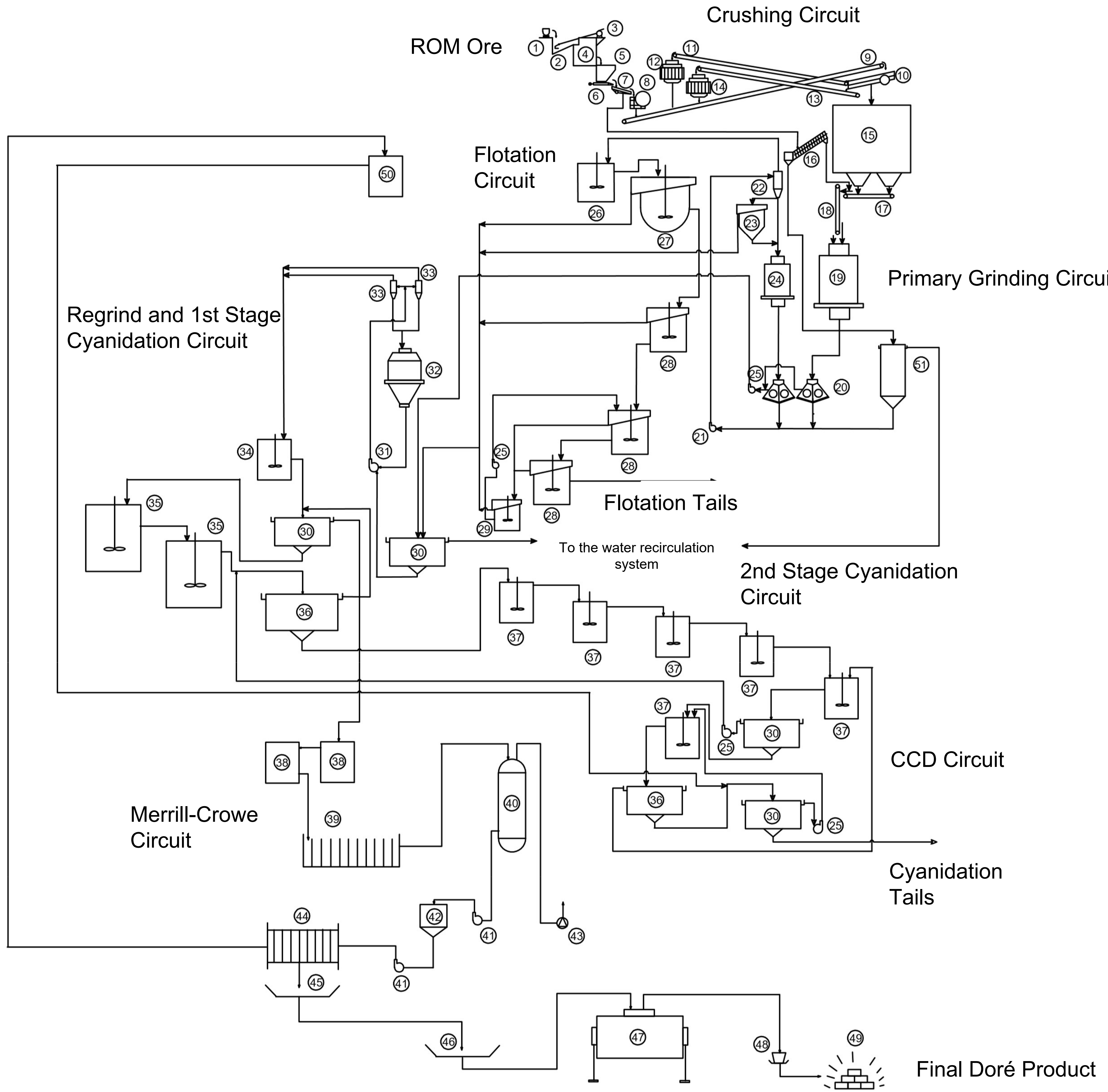


Figure 17-1: Marmato Process Flowsheet

Table 17-1: Equipment List for Marmato Process Plant

Flowsheet No.	Equipment	Description	Quantity	Hp
1	Mine rail cars	1.5 t		
2	Pre-hopper	100 t	1	
3	Winch		2	60
4	Feed hopper	5 m x 7 m	1	
5	Feed hopper with hydraulic gate		1	
6	Hydraulic gate feeder		1	
7	Vibrating grizzly	5 ft x 13 ft (5/16")	1	20
8	Primary jaw crusher	25" x 40"	1	125
9	Conveyor belt	30"	1	25
10	Vibrating screen (double-deck)	20 ft x 8 ft (7/8" x 3/8")	1	40
11	Conveyor belt	24" x 14 m	1	12
12	Secondary cone crusher	Omincone -1352	1	250
13	Conveyor belt	24" x 14 m	1	12
14	Tertiary cone crusher	HP300	1	300
15	Fine ore bin	7 m x 5.8 m	1	
16	Spiral Classifier	30" x 17 ft	1	7.5
17	Conveyor belt	24" x 7 m	1	7
18	Conveyor belt (with belt scale)	24" x 9 m	1	7
19	Primary ball mill (Allis Chalmers)	9.5 ft x 14 ft	1	600
20	Tapezoidal jig	17 ft2	1	7
21	Cyclone feed pump	6" x6"	2	75
22	Hydrocyclone	20"	2	
23	Flash flotation cell	SK-80	1	20
24	Secondary ball mill	7.5 ft x 10 ft	1	300
25	Gravity concentrate pumps	3" x 3"	6	20
26	Flotation conditioner	12 ft x 12 ft	1	30
27	Rougher flotation cell	KCF/KYF 30	1	75
28	Scavenger flotation cell (circular)	10 ft x 10 ft	3	30
29	Cleaner flotation cell	2 m x 2 m	2	7.5
30	Thickener	24 ft x 10 ft	4	3
31	Regrind cyclone feed pump	Wilfley 5K	2	60
32	Regrind ball mill (Hardinge)	7 ft x 5 ft	1	200
33	Regrind hydrocyclone	6"	2	
34	Pretreatment agitated tank	12 ft x 12 ft	2	12
35	Leach tank	20 ft x 20 ft	2	30
36	Thickener	30 ft x 10 ft	2	5
37	Leach tank	12 ft x 12 ft	6	12
38	PLS tank	12 ft x 12 ft	2	
39	Clarifier		1	
40	Deaeration tower		1	
41	Precipitate filter feed pump		1	25
42	Zinc dust dosing cone		2	
43	Vacuum pump		1	12
44	Precipitate filter press		1	
45	Precipitate receiving tray		1	
46	Flux mixing tray		1	
47	Precipitate smelting furnace		1	
48	Ingot molds		2	
49	Dore’			
50	Barren solution tank	12 ft x 12 ft	1	
51	Thickener	12 ft x 35 ft	1	

Source: GCM, 2019

17.1.1 Crushing Circuit

RoM material is hauled by rail from the mine and dumped into a hopper where a slusher is used to move the material to a 5 m by 7 m feed hopper that feeds a vibrating grizzly to remove the -3/8 inch material prior to feeding the primary jaw crusher. The discharge from the jaw crusher is conveyed to a double-deck vibrating screen fitted with a 7/8 inch upper deck and a 3/8 inch lower deck. The screen oversize from both decks is conveyed to a Nordberg HP300 cone crusher, which is operated in closed circuit with the vibrating screen. The -3/8 inch screen undersize discharges to the fines bin. The -3/8 inch undersize from the vibrating grizzly is further classified in a spiral classifier. The classifier oversize is fed directly into the primary ball mill and the classifier undersize is thickened and then pumped to the primary hyrocyclones. A Nordberg Omincone 1352 serves as a standby secondary cone crusher.

17.1.2 Grinding and Gravity Concentration Circuit

Crushed material (-3/8 inch) is fed from the fines bin and then transported on a conveyor fitted with a belt scale to the primary ball. The primary ball mill discharges to a jig which serves to recover coarse gravity recoverable gold. The jig tailing is pumped to the cyclones where a size separation at P_{50} 75 µm is made. The cyclone underflow discharges to the secondary ball mill, which is operated in closed circuit with the cyclones and the overflow advances to the flotation circuit. The jig concentrate is combined with the flotation concentrates prior to advancing to the regrind and cyanidation circuits.

17.1.3 Flotation and Concentrate Regrind Circuit

The cyclone overflow from the grinding circuit is advanced to the flotation circuit where it is first conditioned with the required flotation reagents and then subjected to one stage of rougher flotation followed by one stage of scavenger flotation, which provides a total flotation retention time of 40 minutes to recover the contained gold and silver values. The scavenger flotation concentrate is upgraded in one stage of cleaner flotation and combined with the rougher flotation concentrate. The rougher+scavenger cleaner flotation concentrates are combined with the gravity concentrate, and then thickened to about 55% solids and reground to about 80% passing (P_{80}) 44 µm. A portion of the flotation tailings are pumped to an agitated storage tank and then pumped back underground with a positive displacement pump for use as hydraulic backfill in the mine.

17.1.4 Cyanidation and Counter-Current-Decantation (CCD) Circuit

The reground gravity and flotation concentrates are re-thickened and then advanced to a conventional two-stage cyanidation circuit which provides a total of 30 hours of leach retention time. The first stage of leaching consists of two 20 foot (ft) by 20 ft agitated leach tanks operated in series at a cyanide concentration of 700 mg/L NaCN and at a pH of 10.5 adjusted with lime. The second-stage leach circuit consists of six 12 ft by 12 ft agitated leach tanks in which the cyanide concentration is allowed to attenuate through the circuit from 700 to 400 mg/L NaCN. The leached slurry is then passed through a counter-current-decantation circuit (CCD) which serves to wash the pregnant leach solution (PLS) from the leached solids. The leached solids are discharged from the last CCD thickener and then pumped to the tailing storage facility. The PLS is processed in the Merrill-Crowe circuit to recover the solubilized gold and silver values.

17.1.5 Merrill-Crowe Circuit and Smelter

The PLS is pumped to the Merrill-Crowe circuit where it is first clarified to remove any remaining suspended solids and then deaerated to less than 1 mg/L dissolve oxygen in a vacuum tower. Zinc dust is then added to the deaerated PLS in a controlled manner which results in the precipitation of the gold and silver values, which are then recovered in a filter press. The resulting gold and silver precipitate is removed from the filter press on a scheduled basis and then smelted using a flux with the following composition:

- Borax: 40%
- Sodium nitrate: 30%
- Soda ash: 20%
- Silica: 10%

Approximately 600 kg of flux is blended with 600 kg of precipitate and smelted in a diesel-fired furnace to produce a final dore' product.

17.1.6 Process Plant Consumables

Process plant consumables are shown in Table 17-2 and includes grinding media, wear materials and process reagents. Consumable costs during 2019 (Jan to July) averaged US$3.05/t processed.

Table 17-2: Marmato Process Plant Consumables

Item	kg/t	US$/Kg	US$/t
Grinding Balls (1.5 inch)	0.165	1.13	0.19
Grinding Balls (2 inch)	0.312	1.15	0.36
Grinding Balls (3 inch)	0.392	1.15	0.45
Wear Liners			0.32
Sodium Cyanide	0.370	2.48	0.92
Zinc Dust	0.020	5.03	0.10
Lime	0.625	0.18	0.11
Copper Sulfate	0.015	2.35	0.04
Xanthate (Z-11)	0.011	3.58	0.04
Aerofroth (A65)	0.028	5.56	0.16
Collector MX 5160	0.005	11.12	0.06
Aero AR1404	0.003	11.22	0.03
Lead Acetate	0.002	6.49	0.01
Flocculant (EGA 1203)	0.015	5.09	0.08
Silica	0.095	1.17	0.11
Borax	0.015	0.74	0.01
Soda Ash	0.008	0.60	0.00
Potassium Carbonate	0.001	1.65	0.00
Soloun K	0.015	1.37	0.02
ACPM	0.017	2.47	0.04
Total Consumables			**3.05**

Source: GCM, 2019

17.1.7 Operating Performance

The current Marmato process plant performance is summarized in Table 17-3 for the period from 2013 to 2019 (January to July). During this period mineralized tonnes processed has increased from 274,191 to 338,902 tpy while grades have declined slightly from 2.90 g/t Au in 2013 to 2.45 g/t Au in 2019 and silver grades have ranged from 12.36 to 9.13 g/t Ag. Overall gold recovery has ranged from 89.0 to 83.7% and has averaged about 86.5% over the past three years. Silver recovery has ranged from 41.1 to 31.5% and has averaged 33.2% over the past three years. Gold production has increased from 22,566 ounces in 2013 to 24,909 ounces in 2018.

Table 17-3: Summary of Marmato Plant Operating Performance

Parameter	2013	2014	2015	2016	2017	2018	2019 (Jan to Jul)
Mineralized Tonnes	274,191	295,023	303,279	341,309	365,119	338,902	211,817
Mineralization Grade							
Au (g/t)	2.90	2.85	2.79	2.56	2.48	2.67	2.45
Ag (g/t)	12.36	9.13	9.33	9.24	9.61	10.53	10.41
Metal Recovery							
Au (%)	88.6	89.0	88.0	83.7	86.8	85.5	87.2
Ag (%)	36.6	41.1	37.9	35.8	34.9	33.2	31.5
Metal Produced							
Au (Ounces)	22,566	24,113	23,954	23,449	25,163	24,909	14,538
Ag (Ounces)	39,916	34,753	34,490	36,318	39,524	37,522	22,878

Source: GCM, 2019

17.1.8 Operating Costs

Marmato process plant operating costs reported for 2019 (Jan to July) are summarized in Table 17-4. During this period operating costs averaged US$13.08/t processed (exchange rate: 3,000 COP per US$).

Table 17-4: Summary of Marmato 2019 Process Plant Operating Costs (January to July)

Cost Area	**COP**							
	January	**February**	**March**	**April**	**May**	**June**	**July**	**Total**
Crushing	240,724,049	261,324,792	252,355,448	223,144,847	228,189,512	231,127,744	245,642,011	**1,682,508,403**
Grinding	414,319,416	411,151,704	480,682,035	391,170,440	457,363,520	441,570,671	441,171,448	**3,037,429,234**
Flotation	178,736,120	199,589,193	191,049,133	189,186,472	192,761,316	175,173,963	220,898,954	**1,347,395,151**
Cyanidation and Merrill-Crowe	277,422,528	265,688,731	302,358,644	257,520,281	270,019,391	287,406,736	322,249,721	**1,982,666,032**
Smelting	43,974,090	45,764,157	41,608,733	42,517,945	37,338,644	47,447,376	43,405,216	**302,056,161**
Total	1,155,176,203	1,183,518,577	1,268,053,993	1,103,539,985	1,185,672,383	1,182,726,490	1,273,367,350	**8,352,054,981**
Cost Area	**US$/tonne**							
	January	**February**	**March**	**April**	**May**	**June**	**July**	**Average**
Crushing	2.64	3.19	2.49	2.33	2.44	2.72	2.73	2.64
Grinding	4.55	5.03	4.74	4.09	4.89	5.20	4.90	4.76
Flotation	1.96	2.44	1.89	1.98	2.06	2.06	2.45	2.11
Cyanidation and Merrill-Crowe	3.05	3.25	2.98	2.69	2.89	3.38	3.58	3.11
Smelting	0.48	0.56	0.41	0.44	0.40	0.56	0.48	0.47
Total	**12.69**	**14.47**	**12.51**	**11.54**	**12.67**	**13.92**	**14.14**	**13.08**
Mineralized Tonnes	30,346	27,267	33,776	31,889	31,195	28,327	30,017	212,817
Exchange Rate	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000

Source: GCM, 2019

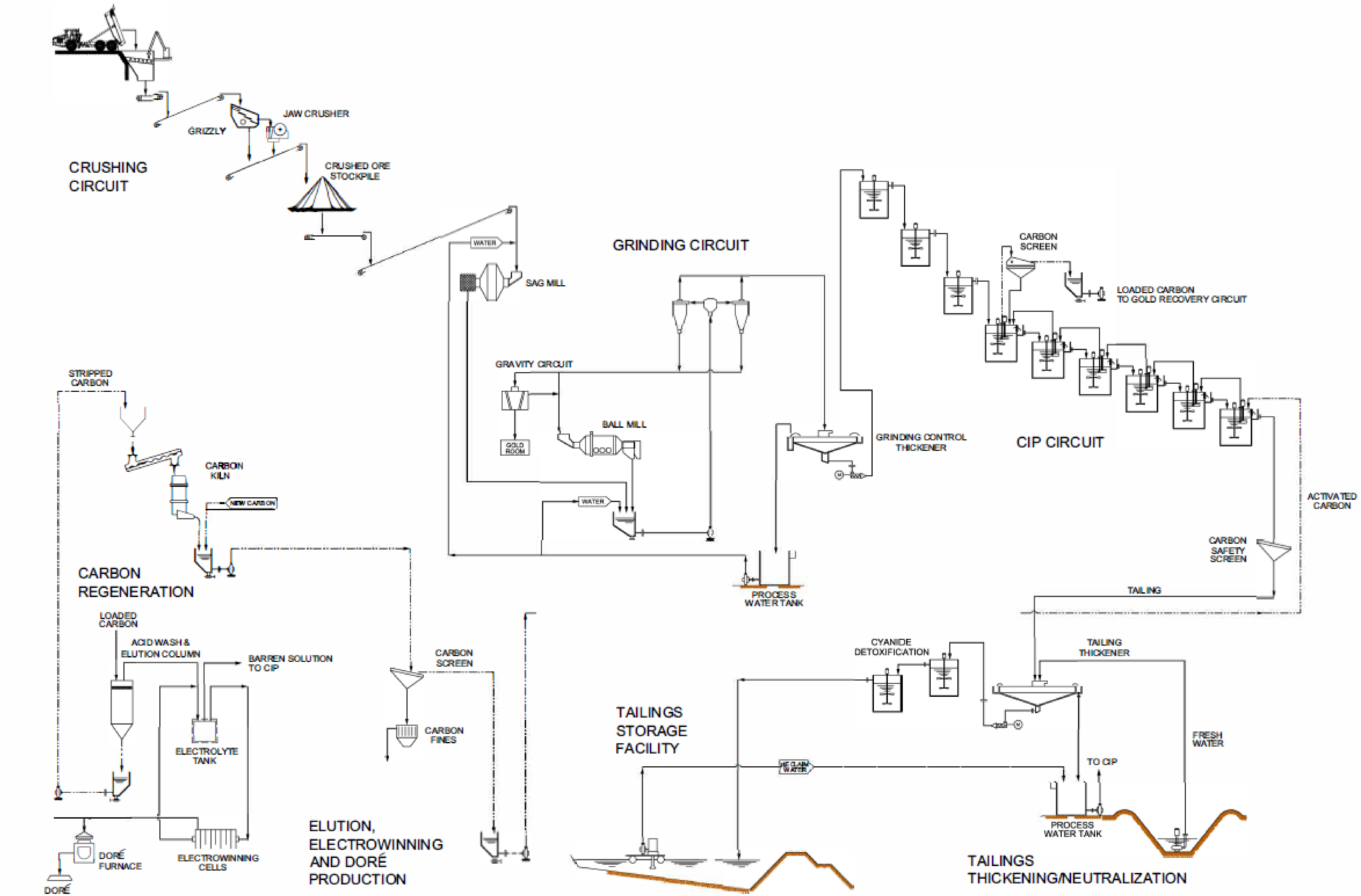
17.2 MDZ Process Plant

Metallurgical testwork was conducted to evaluate three different process flowsheet options including:

- Whole-ore cyanidation;
- Gravity concentration followed by cyanidation of the gravity tailing; and
- Gravity concentration followed by gold and silver flotation from the gravity tailing and cyanidation of the flotation concentrate.

After conducting a trade-off study the process flowsheet that includes gravity concentration followed by cyanidation of the gravity tailing was selected as this flowsheet option offers higher overall gold recoveries along with lower estimated capital and operating costs than the flowsheet that includes flotation followed by cyanidation of the flotation concentrate.

The process flowsheet will incorporate process unit operations that are standard to the industry, including: primary crushing, SAG mill/ball mill grinding, agitated cyanide leaching, gold and silver carbon-in-pulp (CIP) adsorption onto activated carbon, gold and silver desorption, electrowinning and refining. A conceptual process flowsheet is shown in Figure 17-2. Preliminary process design criteria are presented in Table 17-5 and a major equipment list is provided in Table 17-6.



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DESIGN:	DRAWN:	REVIEWED: EO
SCALE: NOT TO SCALE		DATE: 10/07/2019
FILE:		

MDZ CONCEPTUAL FLOWSHEET

REPORT TITLE:

ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO**

FIGURE 17-2

REVISION NO. **A**

SRK JOB NO.: **544400.020**

Table 17-5: Preliminary Design Criteria for the MDZ Process Plant

Area	Units	Criteria
General		
Mine production rate, tpy	tpy	1,400,000
Operating days per year	days	350
Mill design capacity	tpd	4,000
Gold grade	g/t	2.5
Silver grade	g/t	3.4
Gold recovery	%	95.0
Silver Recovery	%	40.0
Crushing		
Operating days per year	days	350
Shifts per day		2
Hours per shift	hours	12
Availability	%	70
Operating hours per day	hours	17
Crushing Rate	tph	238
Feed F80	mm	1,200
Product P80	mm	150
Grinding		
Operating days per year	days	350
Hours per day	hours	24
Shifts per day		2
Hours per shift	hours	12
Availability	%	92
Feed Rate	tph	181
Ball Mill Work Index, Bond, BWi	kWh/t	20
SMC, Axb		29
Abrasion Index , Ai		0.65
Ball Mill, F80	µm	9,500
Ball Mill, P80	µm	100
Pre-Leach Thickener		
Slurry Feed Density	w/w%	35
Underflow density	w/w%	60
Specific Settling Area	m2/tpd	0.09
Hydraulic Loading	m3/m2.hr	5.1
Flocculant (Magnafloc 10)	g/t	15
Flocculant (Magnafloc 1687)	g/t	10
Cyanidation		
Preaeration	hr	4
Slurry Density	w/w%	45
Retention Time	hr	48
Cyanide Leach Concentration (NaCN)	ppm	500
PH		10.5
Cyanidation Tailing Thickening		
Slurry Feed Density	w/w%	45
Underflow density	w/w%	55
Specific Settling Area	m2/tpd	0.09
Hydraulic Loading	m3/m2.hr	5.10
Flocculant (Magnafloc 10)	g/t	15
Flocculant (Magnafloc 1687)	g/t	10
Cyanide Detoxification		
Slurry density	w/w%	55
retention time	hours	1.5
Feed CNT	ppm	150
Discharge CNT	ppm	10
SO2 dosage	g/g CNwad	8
CuSO4	g/g CNwad	0.20

Table 17-6: Preliminary Major Equipment List

	Quantity	Units	Size	KW	Comment
Crushing Circuit					
ROM Ore Bin	1	tonne	250		
Stationary Grizzly	1	mm	600 x 600		spacing
Apron Feeder	1	mm	1,100 x TBD	20	
Fines Vibrating Screen	1	m2	20	50	
Primary Jaw Crusher	1	mm	700 x1060	110	
Crushed Ore Stockpile	1	tonnes	2,000		One shift live capacity
Grinding Circuit					
SAG Mill	1	meter	7.3 x 2.7	2,300	
Ball Mill	1	meter	4.0 x 8.5	4,000	
Cyclones	6	inch	D-15		
Pebble Crusher	1		HP100	90	
Centrifugal Gravity Concentrator	2		KC-XD30	15	Knelson or equivalent
Grinding Control Thickener	1	meter	30	7.5	
Cyanidation Circuit					
Agitated Leach Tanks	9	meter	13 x13	100	
Interstage Screen	6		TBD		Vertical air swept
Carbon Recovery Screen	1		TBD		
Carbon Safety Screen	1	meter	3 x 8	2 x 15	
Tailings Thickener	1	meter	30	7.5	
Detox Circuit					
Agitated Detoxification Tanks	2	meter	6 x 6	50	
Gold Room					
Acid Wash Column	2	meter	1.2 x 9.5		4.5t carbon capacity
Elution Column	2	meter	1.2 x 9.5		4.5t carbon capacity
Carbon Activation Kiln	1		250 kg /hr		Horizontal, diesel fired
Electrowinning Cells	4	Ampere	1,500		SS 304/polylined
Dore' Furnace	1				Diesel Fired

Source: SRK, 2019

17.2.1 Crushing Circuit

RoM material will be loaded into the RoM feed hopper by haul truck or front-end loader. A grizzly will be fitted to the RoM hopper to protect the downstream equipment from oversize material. A rock breaker will be provided to reduce oversize rock retained on the grizzly. RoM material will be drawn from the hopper at a controlled rate by a variable speed apron feeder and discharged onto a vibrating grizzly ahead of the jaw crusher. The grizzly oversize (+6 inches) will feed the jaw crusher. The crusher product and vibrating grizzly undersize will discharge onto a conveyor belt and be transported to the crushed material stockpile. Crushed material will be withdrawn from the stockpile at a controlled rate by variable speed apron feeders and conveyed to the grinding circuit.

17.2.2 Grinding, Classification and Gravity Circuit

The grinding circuit will consist of a SAG mill discharging to a pebble screen with the screen oversize conveyed to a pebble crusher and then returned to the SAG mill. The screen undersize will be advanced to a ball which will be operated in closed circuit with hydrocyclones to produce a final grind size of P_{80} 100 µm. The cyclone underflow will flow to either the Knelson-type semi-continuous centrifugal gravity concentrator or be recirculated back to the ball mill. The cyclone overflow will gravitate to the pre-leach thickener where it will be thickened prior to being pumped to the cyanidation circuit. The gravity concentrate produced from the Knelson concentrators will be subjected to batch intensive cyanide leaching. The tails from the Knelson concentrators will flow back to the ball mill.

17.2.3 Grinding Control Thickener

Cyclone overflow will be screened to remove trash such as any coarse particles, wood fragments, organic material, plastics and lime slurry grits that could otherwise blind the inter-tank screens. The screen oversize (trash) will be collected in a bin, and the screen undersize (slurry) will flow to the grinding control thickener where it will be dosed with flocculant in the feed well. Flocculant fed to the thickener will be diluted with water in a static mixer to ensure good dispersion throughout the feed stream. Thickener underflow will be pumped to the agitated cyanidation circuit, and thickener overflow will report to the process water tank.

17.2.4 Leach and Carbon Adsorption Circuit

The thickener underflow will be pumped to the leach distributor feed box where it will be sampled by a two-stage cross cut feed sampler. The sampler will be used to take representative samples of the cyanide circuit feed head grade for metallurgical accounting purposes. It is anticipated that the cyanidation circuit will consist of one stage of preaeration followed by standard agitated cyanidation and CIP. The leach tanks will be interconnected with launders, and slurry will flow by gravity through the tank train. Each tank will be fitted with a dual impeller mechanical agitator to ensure uniform mixing and dispersion. Oxygen required for leaching will be provided by air sparging through the bottom of the agitator shaft into the slurry. The CIP adsorption tanks will each be fitted with an air swept woven wire inter-tank screen to retain the carbon. All tanks will be fitted with bypass facilities to allow any tank to be removed from service for agitator or screen maintenance.

Sodium cyanide solution will be metered into the leach feed distribution box, as required, to achieve the desired initial cyanide concentration (500 ppm NaCN), which will be allowed to attenuate to about 200 ppm NaCN at the discharge of the circuit. Compressed air will be distributed to the circuit and sparged down the shafts of the agitators to allow a high dissolved oxygen profile to be maintained in the circuit. Fresh and regenerated carbon will be returned to the circuit at the last CIP Tank, and will be advanced counter-currently through the circuit. The inter-tank screen in each CIP tank will retain the carbon and allow the slurry to gravity flow to the next CIP tank. This counter-current process will be repeated until the carbon reaches CIP Tank 1 at which point an air lift will be used to transfer loaded carbon to the loaded carbon recovery screen. The loaded carbon will be washed and dewatered on the recovery screen prior to reporting to the acid wash/elution column. The recovery screen undersize will return to the CIP circuit.

Slurry from the last CIP tank (leach tails) will flow to the vibrating carbon safety via the tails sampler for metallurgical accounting. The safety screen will recover any carbon passing through worn inter-tank screens or overflowing the tanks. Screen underflow will then flow to the cyanide destruction circuit. Barren carbon returning to the adsorption circuit from the carbon regeneration kiln will be screened on the sizing screen to remove fine carbon and prevent associated gold losses. The CIP tanks will be located in a bunded area with a sloping concrete floor. Any spillage from the circuit will report to one of two sumps and can be returned to the circuit or to the carbon safety screen ahead of the cyanide destruction circuit.

17.2.5 Elution and Gold Room Operations

The following operations will be carried out in the elution and gold room areas:

- Acid washing of carbon;
- Stripping of gold from loaded carbon;
- Electrowinning of gold from pregnant solution;
- Smelting; and
- Carbon regeneration.

Acid Wash

Loaded carbon will be recovered on the loaded carbon recovery screen and directed to the acid wash column. Acid washing of the carbon will commence after carbon transfer and drain down is complete. The acid wash solution, 3% w/w HCl in fresh water, will be mixed in the dilute acid tank and transferred to the acid wash column. The acid wash process removes contaminants, primarily calcium, from the loaded carbon and prevents carbon fouling which reduces the effectiveness of the carbon. After the prescribed acid soak period, the carbon will be rinsed with fresh water. Approximately three bed volumes of fresh water will be pumped through the column to displace any residual acid from the carbon. Dilute acid and rinse water will be neutralized and disposed of with the tailings. Acid-washed carbon will be transferred to the elution column for stripping.

Pre-Soak and Elution

Strip solution will be pumped from the strip solution tank through in-line heat exchangers into the base of the elution column. Sodium hydroxide and sodium cyanide solutions will be pumped from the respective storage tanks into the strip solution tank. The loaded carbon will be pre-soaked in the 2% cyanide / 2% caustic strip solution for 30 minutes to prepare the gold for elution. The carbon will then be eluted with a hot strip solution (120^0 C), which will then flow to the pregnant solution tank. Outgoing strip solution will pass through the recovery heat exchanger to heat the incoming strip solution.

Electrowinning

Direct current will be passed through stainless steel anodes and stainless steel wool mesh cathodes to deposit gold and silver sludge on the cathodes. Solution discharging from the electrowinning cells will return by gravity to the pregnant solution tank. Electrowinning will continue until the solution exiting the electrowinning cells is depleted of gold.

Gold Room

The electrowinning cells will be located within the security area of the gold room. Rectifiers, one per cell, will be located in a non-secure area below the cells allowing maintenance access without breaching gold room security. The electrowon gold and silver will be removed from the cathodes in-situ by washing with high pressure water. The resulting sludge will be filtered in laboratory-style pressure filters and dried in an oven. The sludge will then be smelted with fluxes in an HFO or diesel fired furnace to produce doré bars. Slag from smelting operations will be returned to the milling circuit. Fume extraction equipment will be provided to remove gases from the electrowinning cells, drying oven and smelting furnace.

Carbon Regeneration

After completion of the elution process, the barren carbon will be transferred from the elution column to the carbon dewatering screen to dewater the carbon prior to entering the feed hopper of the horizontal carbon regeneration kiln. Any residual water will be drained from the carbon in the kiln feed hopper before it enters the kiln. It is anticipated that only 75% of the carbon will be regenerated each cycle. In the kiln, the carbon will be heated to 650°C to 750°C for 20 minutes to allow regeneration to occur. Regenerated carbon from the kiln will be quenched and report to the carbon sizing screen. The screen oversize (regenerated and sized carbon) will return to the cyanidation circuit.

17.2.6 Carbon Safety Screen

Tailings slurry from the final CIP tank will gravitate through the metallurgical sampler to the carbon safety screen. Recovered carbon will be collected in the fine carbon bin for separate handling to recover the contained precious metal values. The safety screen undersize will be advanced to the cyanide destruction circuit.

17.2.7 Cyanide Destruction Circuit

The carbon safety screen undersize slurry will report to the SO_2/air cyanide destruction circuit, which utilizes SO_2 and air in the presence of a soluble copper catalyst to oxidize cyanide to cyanate (OCN). The SO_2 source will be sodium metabisulfite (SMBS). Copper sulfate pentahydrate will be added to supply the necessary copper in solution. Air will be sparged into the cyanide destruction tanks through the agitator shaft. Slaked lime will be added to neutralize the sulfuric acid formed in the reaction and maintain the slurry at about pH 9.

The cyanide destruction circuit will reduce the weak acid dissociable cyanide (CNwad) concentration in the CIP discharge to less than 10 ppm, which is expected to self-attenuate to about 2.5 ppm in the tailing storage facility (TSF). The cyanide destruction circuit will consist of two agitated tanks, which will provide 90 minutes total retention time.

17.2.8 Tailings Disposal

Tailings from the cyanide detoxification circuit and other miscellaneous waste streams from the process plant will be combined in the tailings collection sump and pumped to the TSF or mine backfill plant where they will be filtered for dry stack disposal at the TSF or used as backfill underground.

17.3 Consumable Requirements

Process plant consumable requirements are shown in Table 17-7 and based on metallurgical studies conducted to-date.

Table 17-7: MDZ Process Plant Consumables

Cost Area	Usage (sets/year)	kg/t
Wear Materials		
Crusher Liners	6	
SAG Mill Grinding Media		0.75
SAG Mill Liners	2	
Ball Mill Grinding Media		0.75
Ball Mill Liners	2	
Reagents		
Lime (agitated leach)		1.0
Lime (CN-destruct)		0.30
Cyanide (agitated leach)		0.15
Cyanide (gravity conc. leach)		0.15
Flocculant		0.04
Sodium Metabisulfite		1.6
Copper Sulfate		0.03
Activated Carbon		0.05
Power		kWh/t
Grinding Power (kWh/t)		27
Plant Power- Other (kWh/t)		16

Source: SGS and SRK, 2019

17.3.1 Lime

Quicklime will be delivered to the site in bulk by pneumatic tanker and stored in the lime silo. It is anticipated that the quicklime will be slaked in a vendor supplied package accompanying the silo. The slaked lime will be pumped to the grinding circuit and the cyanide destruction circuit in a ring main. A dust collector will minimize dust emissions during silo filling.

17.3.2 Sodium Cyanide

Sodium cyanide will be delivered as briquettes in shipping containers containing approximately 1 t of cyanide each. The containers will be emptied into the cyanide mixing tank and mixed with water to dissolve the cyanide to a target strength of 20% NaCN. Sodium hydroxide will be added to the mixing tank prior to cyanide addition in order to maintain a solution pH of 11 to prevent HCN generation. The mixed cyanide solution will be transferred to the storage tank for dosing to the process. Empty cyanide containers will be returned to the vendor.

17.3.3 Activated Carbon

Activated carbon will be delivered in 500 kg bulk bags. Carbon will be added to the carbon quench vessel as required for carbon make-up to the CIP inventory. This addition point will allow removal of carbon fines prior to entering the CIP tanks.

17.3.4 Flocculant

Flocculant for use in the grinding control and tailings thickeners will be delivered to site in 25 kg bags. The vendor supplied flocculant mixing system will automatically mix batches of flocculant and transfer the mixed flocculant to the aging tank after each mixing cycle is complete. Flocculant will be distributed to the thickeners using positive displacement dosing pumps.

17.3.5 Copper Sulfate

Copper sulfate will be delivered in 1 t bulk bags and will be added to the mixing tank using an electric hoist and bag breaker. Fresh water will be added to the mixing tank to dilute the copper sulfate. The solution will be metered to the cyanide destruction and flotation circuits directly from the mixing tank.

17.3.6 Sodium Metabisulfite

Sodium metabisulfite will be delivered in 1 t bulk bags and will be added to the mixing tank using an electric hoist and bag breaker. An air exhaust fan will draw dust and fumes away from this area as SO_2 gas is evolved and the dust can cause skin irritation. Fresh water will be used to mix the sodium metabisulfite. The solution will be pumped from the mixing tank to the storage tank for metering to the cyanide destruction circuit by dosing pump.

17.3.7 Grinding Media

Grinding balls will be delivered to site in bulk or 200 L steel drums.

17.4 Metallurgical Accounting

A weightometer on the primary crusher discharge conveyor will measure the primary crushed tonnage and a weightometer on the SAG mill feed conveyor will determine mill feed tonnes.

Density and flow meters on the leach feed will allow the dry tonnage of solids to be determined as a cross check on the mill feed tonnage determined from the mill feed weightometer. In conjunction with the leach feed and tails samplers, the mass flow measurements will allow the gold recovered in the CIP to be calculated.

Routine sampling of the leach feed stream and the final leach tailings will ensure reliable composite shift samples for leach head grade and tails solution and residue grades. Regular in-circuit gold surveys will allow reconciliation of precious metals in feed compared to doré production.

17.5 Operating Cost Estimate

Process operating costs are summarized in Table 17-8 and are estimated at US$13.34/t processed. Operating costs have been estimated by major category (labor, power, consumables, etc.) and are based on a throughput capacity of 4,000 tpd. The major contributors to operating cost are labor, reagents, comminution consumables and power.

Table 17-8: Operating Cost Summary MDZ Process Plant

Cost Area	Gravity-Cyanidation (US$/t)
Labor	1.14
Comminution Consumables	2.76
Reagents	2.54
Power	4.30
Maintenance & Lubrication Supplies	0.60
Laboratory	0.50
Other	1.50
Total	**13.34**

Source: SRK, 2019

17.5.1 Labor

Labor costs are estimated US$1.14/t processed and are based on GCM's actual labor rates and the manpower schedule shown in Table 17-9. A total of 92 process plant employees (operations and maintenance) has been identified. The labor cost estimate is based on the process plant operating two 12-hour shifts per day and includes a 60% burden rate. GCM's labor rates have been converted to US$ at an exchange rate of 3,000 COP per US$.

Table 17-9: MDZ Process Plant Manpower Schedule and Labor Cost Estimate

Position	Number	Annual (US$)	Burden	Burdened (US$)	Total(US$)
Plant Manager	1	90,000	0.60	144,000	144,000
Administrative Assistant	1	8,000	0.60	12,800	12,800
Subtotal	**2**	**98,000**		**156,800**	**156,800**
Operating Crews					
Shift Supervisors	4	20,000	0.60	32,000	128,000
Crusher/Conveying Area Lead Operator	4	8,500	0.60	13,600	54,400
Assistant Crusher/Conveying Area Operator	4	4,500	0.60	7,200	28,800
Grinding/Gravity Area Lead Operator	4	8,500	0.60	13,600	54,400
CIP Operator	4	8,500	0.60	13,600	54,400
Gold Room Supervisor	2	20,000	0.60	32,000	64,000
Gold room Operator	4	8,500	0.60	13,600	54,400
Reagent Area Operator	2	4,500	0.60	7,200	14,400
General Laborer	8	4,500	0.60	7,200	57,600
Tailings Operator	4	4,500	0.60	7,200	28,800
Control Room Operator	4	6,000	0.60	9,600	38,400
Subtotal	**44**	**98,000**		**156,800**	**577,600**
Senior Metallurgist	1	36,000	0.60	57,600	57,600
Junior Metallurgist	1	10,000	0.60	16,000	16,000
Metallurgical Technician	2	7,000	0.60	11,200	22,400
Chemist	2	10,000	0.60	16,000	32,000
Sample Preparers	6	4,500	0.60	7,200	43,200
Analytical Technicians	6	6,000	0.60	9,600	57,600
Subtotal	18	73,500		117,600	228,800
Maintenance - Process plant					
Maintenance Foreman	4	44,000	0.60	70,400	281,600
Maintenance Planner / Foreman	2	9,000	0.60	14,400	28,800
Mechanics	8	9,000	0.60	14,400	115,200
Welders	2	9,000	0.60	14,400	28,800
Electrician	6	9,000	0.60	14,400	86,400
Instrument Technician	2	14,000	0.60	22,400	44,800
Trades Assistants	4	7,000	0.60	11,200	44,800
Subtotal	28	101,000		161,600	630,400
Total Processing Plant + Maintenance	92	370,500		592,800	1,593,600
Mineralized Tonnes Per Year	1,400,000			US$/t	1.14

Source: GCM, 2019

17.5.2 Consumables

Estimated consumable costs are shown in Table 17-10. Comminution consumables are estimated at US$2.76/t and are based on wear liner and grinding media consumption rates that are typical for this type of process facility. Reagent costs are estimated at US$2.54/t and are based on reagent consumption rates established during metallurgical testing and current reagent pricing reported by GCM. Other costs are estimated at US$0.20/t and are intended to cover costs for miscellaneous consumables, such as sodium hydroxide, hydrochloric acid and fluxing agents.

Table 17-10: MDZ Process Plant Consumable Operating Cost Estimate

Cost Area	Usage Sets/year	kg/t	US$/set or kg (FOB Site)	US$/t
Wear Materials				
Crusher Liners	6		25,000	0.11
SAG Mill Grinding Media		0.75	1.20	0.90
SAG Mill Liners	2		300,000	0.43
Ball Mill Grinding Media		0.75	1.20	0.90
Ball Mill Liners	2		300,000	0.43
Subtotal Wear Materials				2.76
Reagents				
Lime (agitated leach)		1.0	0.25	0.25
Cyanide (agitated leach)		0.15	2.50	0.38
Cyanide (gravity conc. leach)		0.15	2.50	0.38
Flocculant		0.04	5.00	0.20
Na-Metabisulfite		1.6	0.50	0.82
Copper Sulfate		0.03	2.30	0.07
Lime (CN-destruct)		0.3	0.25	0.08
Activated Carbon		0.05	3.50	0.18
Other				0.20
Subtotal Reagents				2.54
Power		kWh/t	US$/kWh	US$/t
Grinding Power (kWh/t)		27	0.10	2.70
Plant Power- Other (kWh/t)		16	0.10	1.60
Subtotal Power				4.30
Mineralized Tonnes Per Year	1,400,000			

Source: SGS and SRK, 2019

17.5.3 Power

The process plant power cost is estimated at US$4.30/t and is based on power supplied from the grid at US$0.10/kWh and a total unit power consumption of 43 kWh/t. The estimated unit power consumption includes an estimate of 27 kWh/t to grind feed material to the target grind of P_{80} 100 µm prior to cyanidation, which is based on SAG and ball mill comminution index (SMC and BWi) determinations conducted during the metallurgical investigation. An allowance of 16 kWh/t for other process unit operations is also included.

17.5.4 Maintenance Supplies

Maintenance supply costs are estimated at US$0.60/t and are based on 3% of estimated equipment capital expenditure.

17.5.5 Laboratory Cost

Laboratory cost is estimated at US$0.50/t, which is similar to other operations of this size

17.6 Capital Cost Estimate

The capital cost for the 4,000 tpd process plant is summarized in Table 17-11 and is estimated at US$65.6 million and is considered at a conceptual level with a +/-50% level of accuracy. The capital cost estimate is based on Infomine's CostMine Model for a CIP processing plant, and includes the following adjustments:

- Capital cost has been adjusted to the 4,000 tpd design using the industry accepted Cost-Capacity relationship; $Cost_{p2} = Cost_{p1} \times (Capacity_{p2}/Capacity_{p1})^{0.65}$;
- TSF capital cost has been excluded (included as a separate cost area);
- Working capital excluded (included in the technical economic model);
- Contingency excluded (a global contingency factor is included in the technical economic model); and
- Process plant capital cost has been increased by 30% based on SRK's experience with the CostMine models.

Table 17-11: Preliminary Process Capital Cost Estimate

Cost Area	**Tonnes Per Day**
	4,000
Equipment	19,069,499
Installation Labor	13,241,430
Concrete	1,600,360
Piping	5,946,814
Structural Steel	1,641,741
Instrumentation	1,276,735
Insulation	623,637
Electrical	2,046,104
Coatings & Sealants	259,755
Mill Building	2,027,888
Engineering and Design	10,022,380
Construction Management	7,795,122
Total CIP (US$)	**65,551,466**

Source: SRK, 2019
Capex Estimate Excludes:

- Initial TSF
- Working Capital
- Contingency

17.7 Significant Factors

GCM currently operates a 1,200 tpd process plant (rated capacity) to recover gold and silver values from mineralization produced from current Marmato mining operations. The current Marmato process plant includes gravity concentration followed by gold flotation from the gravity tailing and cyanidation of the flotation concentrate.

- During the period from 2013 to 2019 overall gold recovery has ranged from 89.0 to 83.7% and has averaged about 86.5% over the past three years. Silver recovery has ranged from 41.1 to 31.5% and has averaged 33.2% over the past three years.
- Marmato process plant operating costs reported for 2019 (Jan-July) averaged US$13.08/t processed (exchange rate: 3,000 COP per US$).

MDZ material will be processed at the rate of 4,000 tpd using a flowsheet that includes gravity concentration followed by cyanidation of the gravity tailing:

- The MDZ material is significantly harder than Marmato mineralization and will incur higher comminution costs;
- Gold recovery is estimated at 95% and silver recovery is estimated at 40%;
- Process plant operating costs are estimated at US$13.34/t; and
- Process plant capital cost is estimated at US$65.6 million at an accuracy of +/-50%.

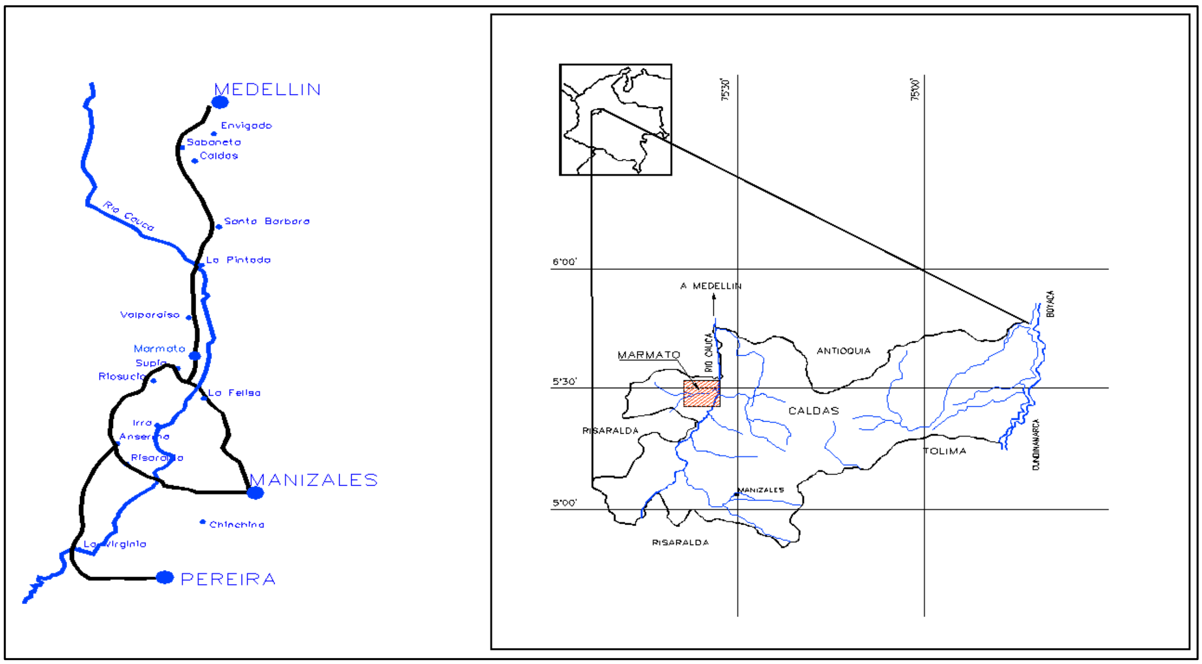
18 Project Infrastructure

18.1 General Site Access

The Project is in the Municipality of Marmato in Caldas. The concessions of the Marmato Project are located on the eastern side of the Western Cordillera (Cordillera Occidental) of Colombia on the west side of the Cauca River.

The Marmato Project is located approximately 125 km south of Medellín, the capital of the department of Antioquia, Colombia. Medellín is the second largest city in Colombia with a population of approximately 2.5 million. The Project is located in the department of Caldas near El Llano.

Figure 18-1 shows the location of the Project.



Source: GCM, 2017

Figure 18-1: Marmato Project Location

Primary access to the site is via the Pan American Highway, Colombia Highway 25. The road is a paved two lane improved highway that is the winds through the mountainous area south of Medellín and then follows the Cauca River to the turn off to the Project. The route from Medellín is via Itaguí (7 km), Caldas (12 km), Alto de Minas (13 km), Santa Barbara (27 km), La Pintada (26 km), La Guaracha del Rayo (32 km), and then a turn onto a secondary road to the community of El Llano that is the community closest to the Project. From El Llano, the road is paved but partially single lane another 2 km to the project security gate. An improved dirt road continues up the mountain another 4 km to the community of Marmato, where the artisinal miners are working the Zona Alta portion of the Project. Approximately 40% of the 1,200 employees currently employed by the Project live in El Llano, with the remainder traveling to work from various communities in the area.

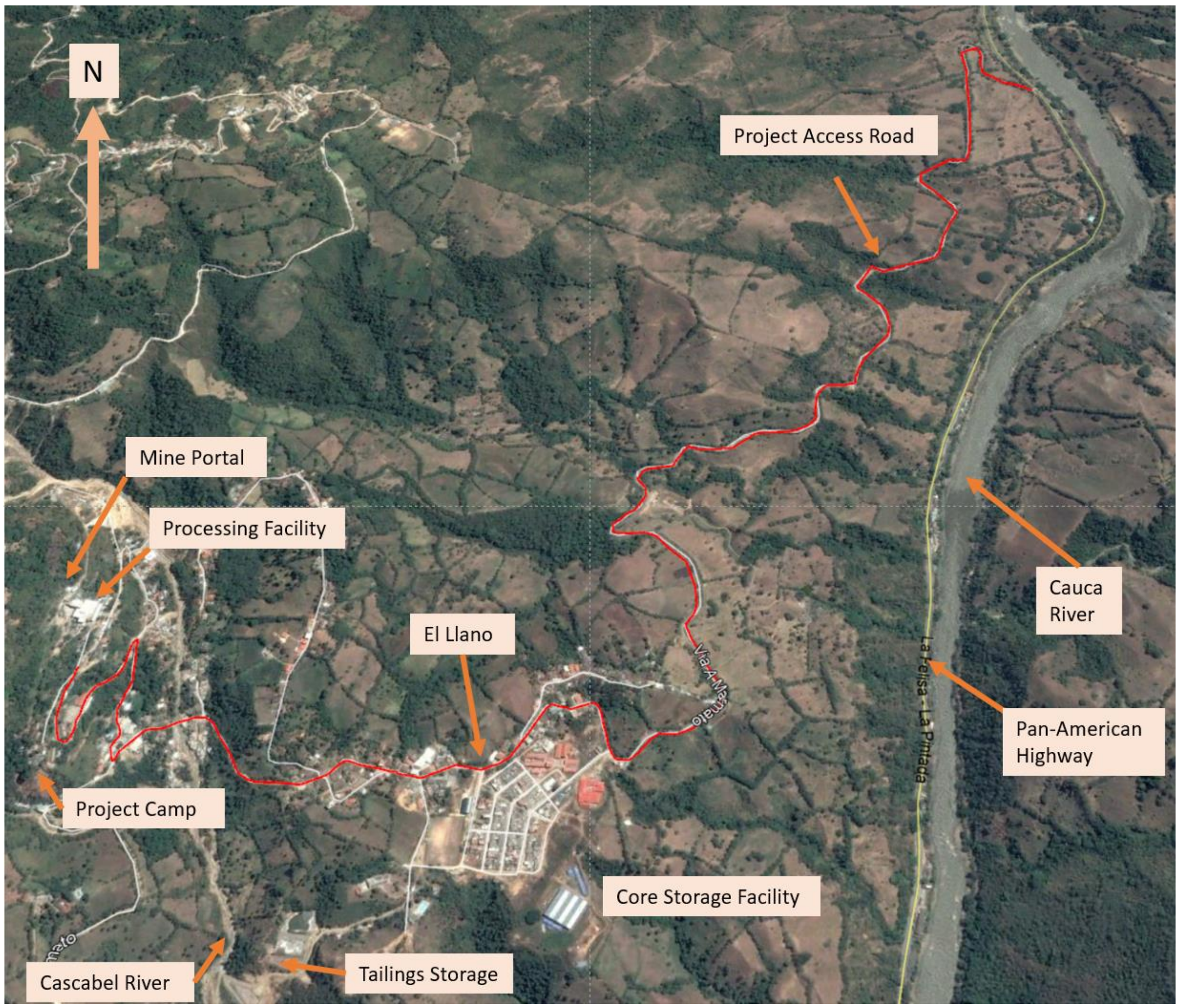
The Pan American Highway continues south and east 90 km to another large regional city, Manizales, and then on another 270 km to Bogotá, the capital of Colombia. Air access through international and regional airports is available in Medellín, Manizales, and Bogatá.

Field personnel for the exploration program have been employed from the towns of Marmato and El Llano and neighboring municipalities. In the long term, personnel currently working on the large number of small scale mines and from the surrounding region would be able to supply the basic workforce for any future mining construction and operation.

18.2 Marmato Existing Operations Infrastructure

18.2.1 Existing Project Access

The general project access is described in section 18.1. The secondary road to the existing operation and general layout of the facilities is shown in Figure 18-2.

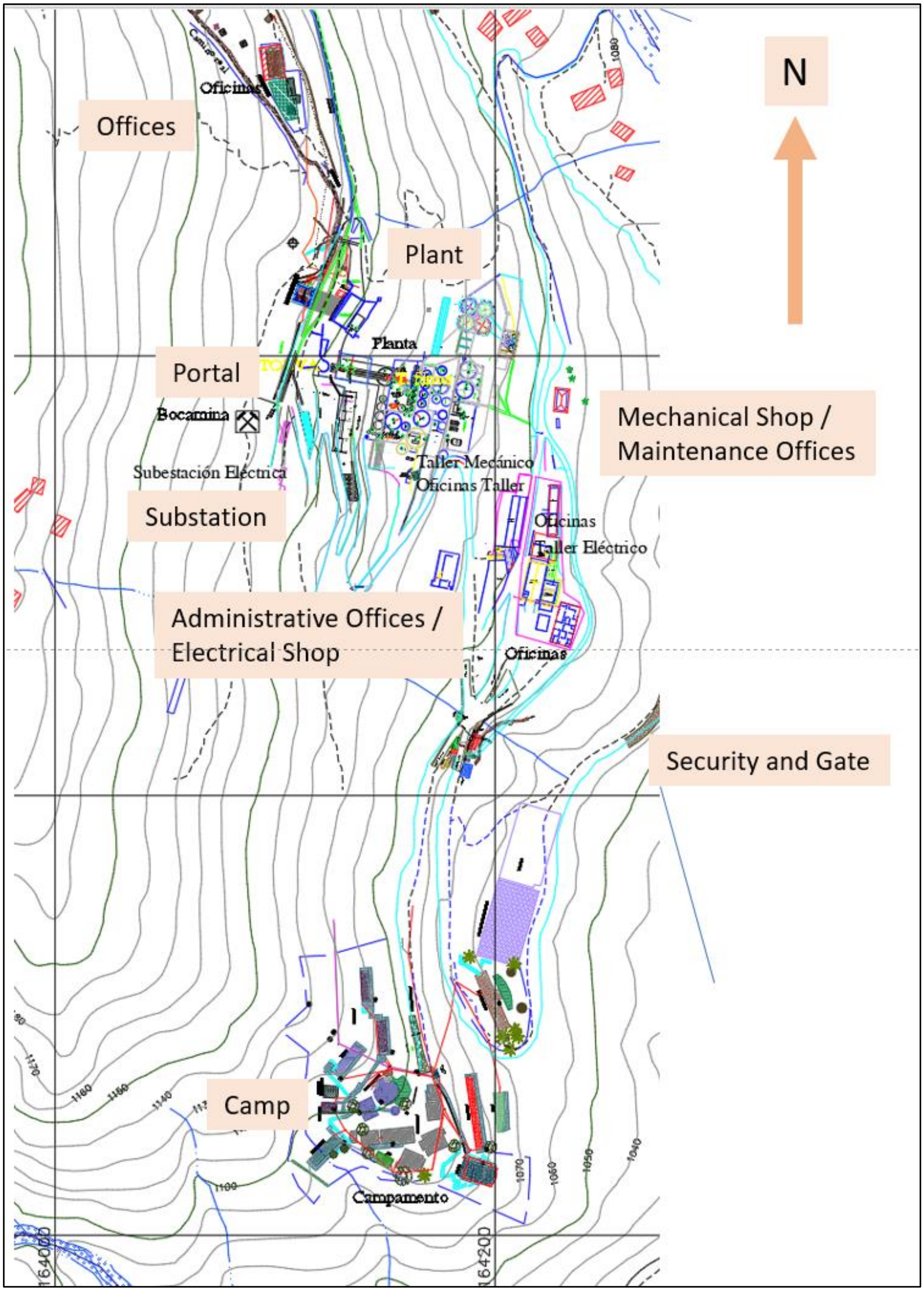


Source: SRK, 2019

Figure 18-2: Marmato General Access and Major Facilities

18.2.2 Existing Project Facilities

The Project has a fully developed infrastructure and facilities that include a security checkpoint that provides access to the office and administrative office area. The facilities include employee motorcycle parking, meeting area, multiple shops and warehouses, a camp with cafeteria, exercise and sports field, equipment storage yards, compressor station, welding shop, a 500 kW backup generator, processing plant, underground mine, explosives storage a short distance from the mine that is managed by the military, main power substation and distribution powerlines with motor control centers at key loads. The site has three portals that access the mine workings. A yard that has rail through it near the portal allows servicing of the mine cars and locomotives. Figure 18-3 shows the overall plant and camp area.



Source: GCM, Modified by SRK, 2019

Figure 18-3: Marmato Existing Project Site Map

18.2.3 Energy Supply and Distribution - Existing Marmato Project

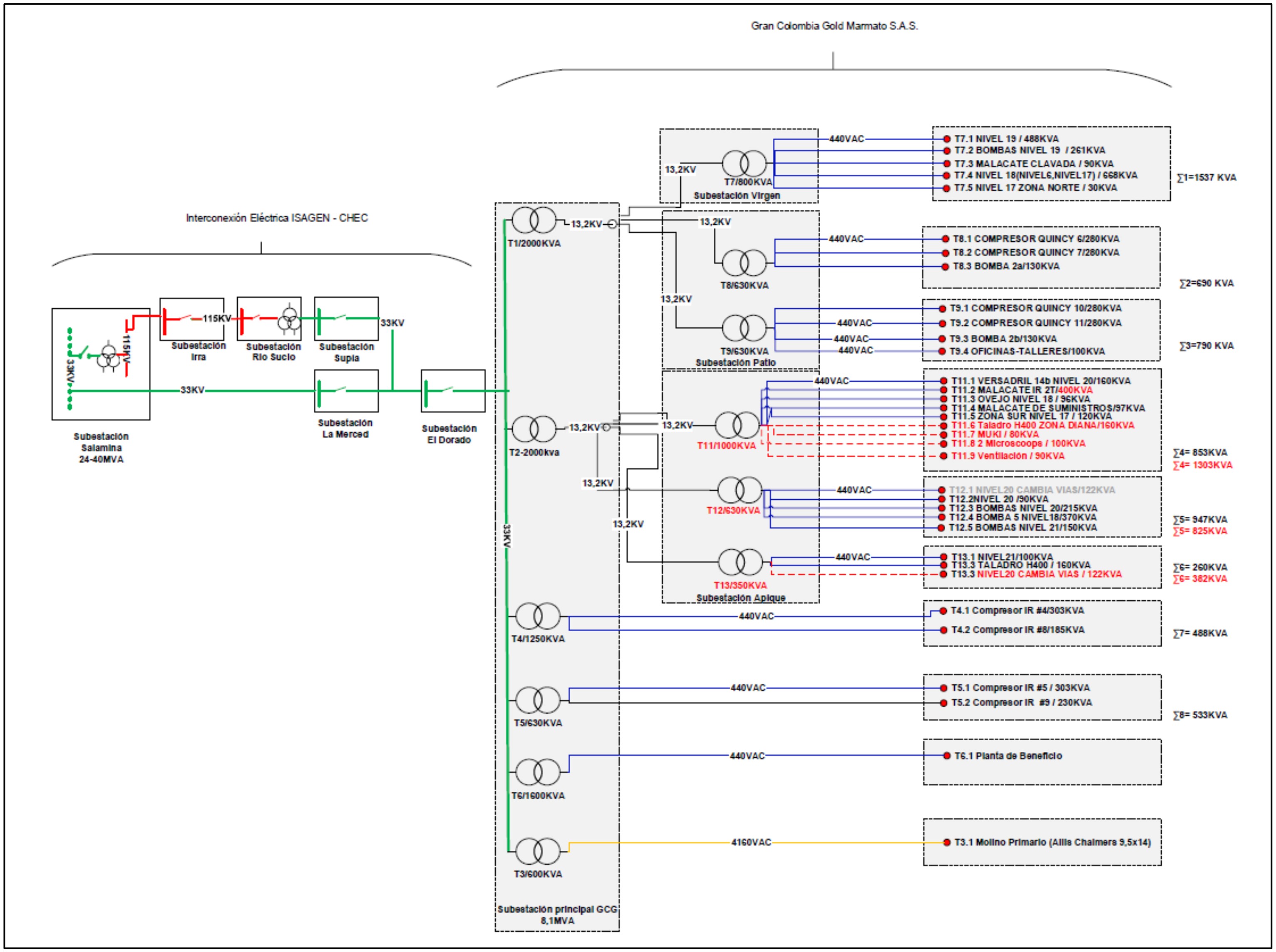
Power to the project site is provided through the Colombian power company Central Hidroeléctrica de Caldas (CHEC), a subisidiary of Empresas Públicas de Medellín (EPM) through existing local substations. Substantial transmission capacity is available in the region around the Project, with energy provided over the transmission system by the third largest electricity producer in Colombia, ISAGEN.

A 115 kV ISAGEN powerline feeds two main substations that transform the 115 kV to 33 kV. The main substation feeding the Project has a capacity of 40 MVA and supplies the Project at 33 kV through the El Dorado substation to the GCM principal substation. The 8.1 MVA main GCM substation has six transformers that provide power to the mine, mill, and other facilities.

The loads supported by each transformer are provided as follows:

- Transformer 1 (2,000 KVA) steps the 33 kV power down to 13.2 kV and feeds the three mine substations that in turn feed the compressors, pumps and offices/shops 440 VAC;
- Transformer 2 (2,000 KVA) feeds the mine at 13.2 kV through three separate mine transformers that in turn feed the various mine levels, hoists, pumps, and mine equipment. The equipment operates on 440 VAC;
- Transformer 3 (600 KVA) feeds the ball mill at the processing plant at 4,160 VAC;
- Transformer 4 (1,250 KVA) and 5 (630 KVA) feeds two compressors each at 440 VAC; and
- Transformer 6 (1,600 KVA) provides 440 VAC for the beneficiation plant.

The Project one-line electrical diagram is shown in Figure 18-4.



Source: GCM, 2019

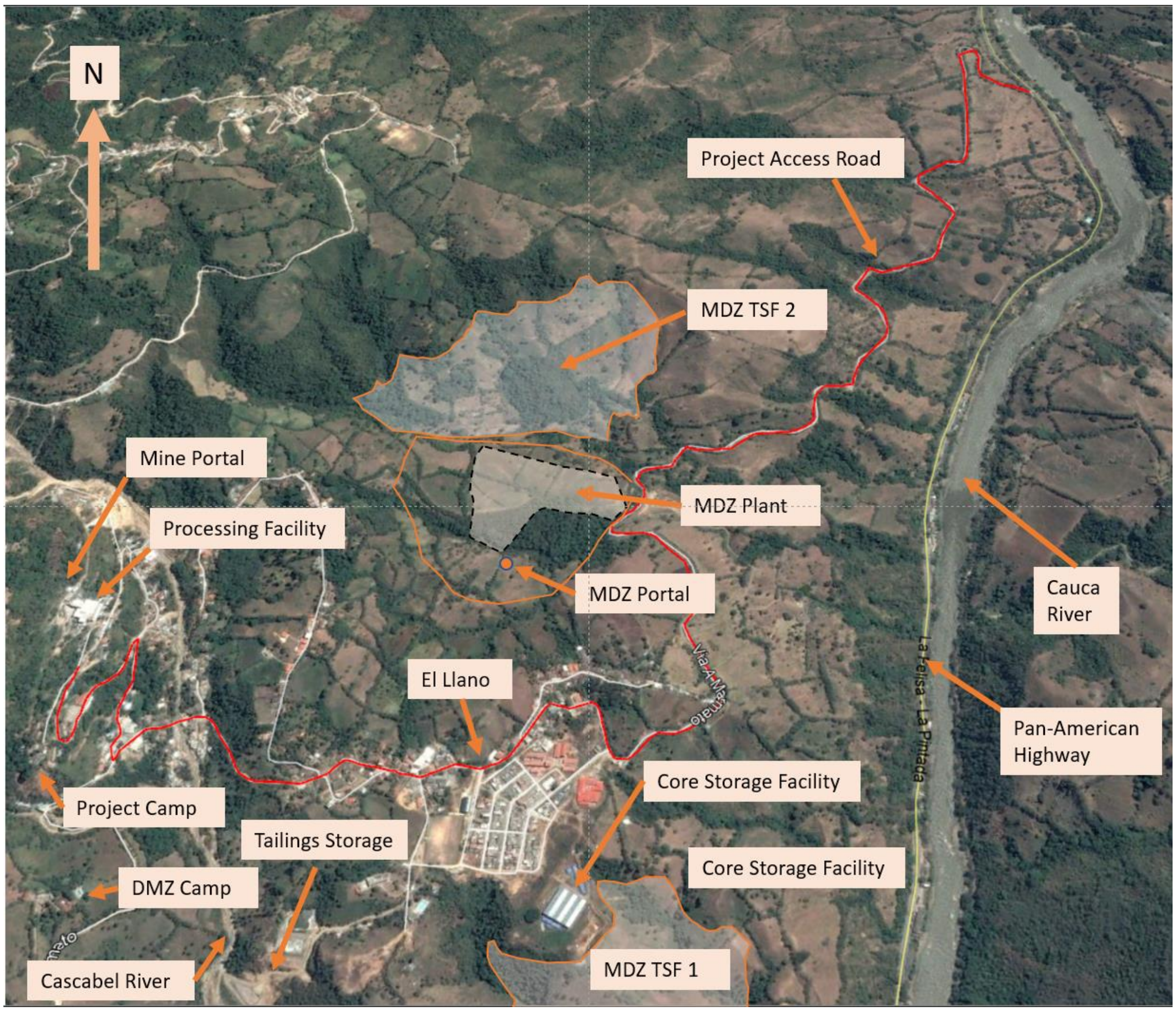
Figure 18-4: Marmato Electrical System Schematic

18.2.4 Site Water Supply

Water supply for the existing Marmato mining activities is currently provided by a combination of underground dewatering and reclaim from the existing TSF. The current Project has adequate water supplies, but can be challenged during dry portions of the year. The Project has contingency plans to draw water from either the Cascabel or Cauca River, with the Cauca River being the preference.

18.3 MDZ Project Infrastructure

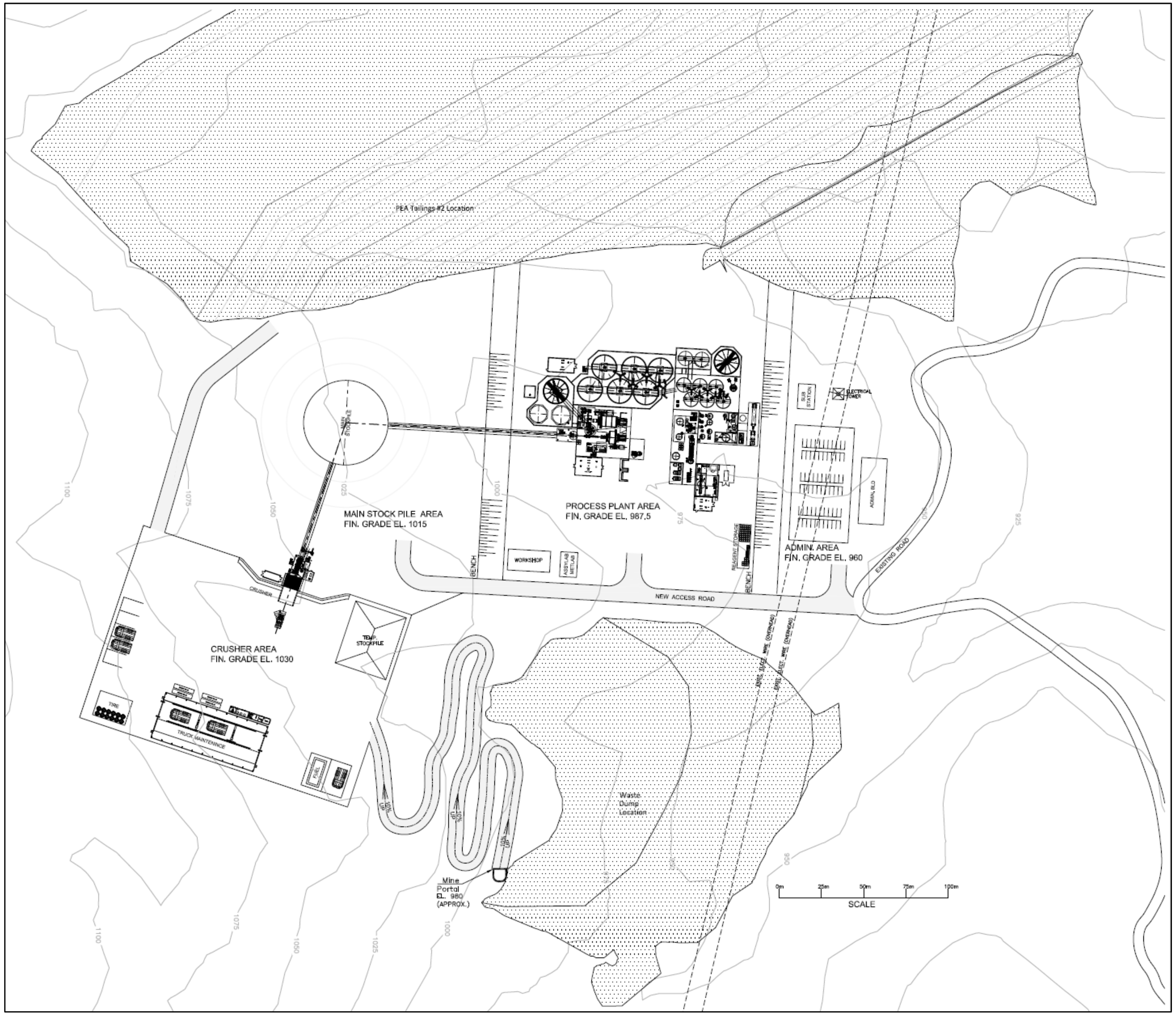
The MDZ Project will require new infrastructure to support the development of the new mine and processing facility. The new facilities will be located northeast of the existing Marmato Project and approximately 2.7 km from the existing site by the existing road through El Llano. Figure 18-5 shows the general location of the new processing facilities, camp, TSF, and MDZ mine portal.



Source: SRK, 2019

Figure 18-5: Marmato MDZ General Infrastructure Location

Additional detail is provided in the conceptual layout of the processing plant, mine portal, mine dump, and access to the MDZ processing site (Figure 18-6).



Source: SRK/Ausenco, 2019

Figure 18-6: MDZ Processing and Mine Portal Site General Arrangement

18.3.1 MDZ Access

The new MDZ plant site will be accessed with a new access road off of the main paved road that connects El Llano to the Pan American Highway. The new road will allow access to the plant facilities and the new portal.

18.3.2 MDZ Project Surface Facilities

The MDZ Project will include a security access at the junction of the El Llano highway. The plant area will have a parking area and an office facility. The plant site will have reagent storage, workshops, laydown areas, warehousing, laboratory facilities, a new MDZ substation and power distribution to the mine portal and the plant facility including the crusher and main stockpile. The portal site will be adjacent to the plant with access to the mine, mine truck shop, tire pad, fuel storage and fueling facility, parking, and mine RoM storage. The mine waste dump will also be located near the portal. The TSF will be in several locations and is discussed in Section 18.4

The MDZ camp facility will be located near the existing camp with access to the existing camp facilities.

The new infrastructure will include all necessary communications systems required to operate the mine and mill effectively.

18.3.3 MDZ Energy Supply and Distribution

The MDZ site will include a new 25 MVA substation that will be fed from the existing power system in the area. The new substation will feed the processing facility and will also feed the mine through a mine substation at the mine portal. The mine substation will feed power at 13.2 kV down the mine decline to the main mineralized zone. Backup generation will be required for key systems at the plant and ventilation systems and pumps at the mine.

18.3.4 Water Supply

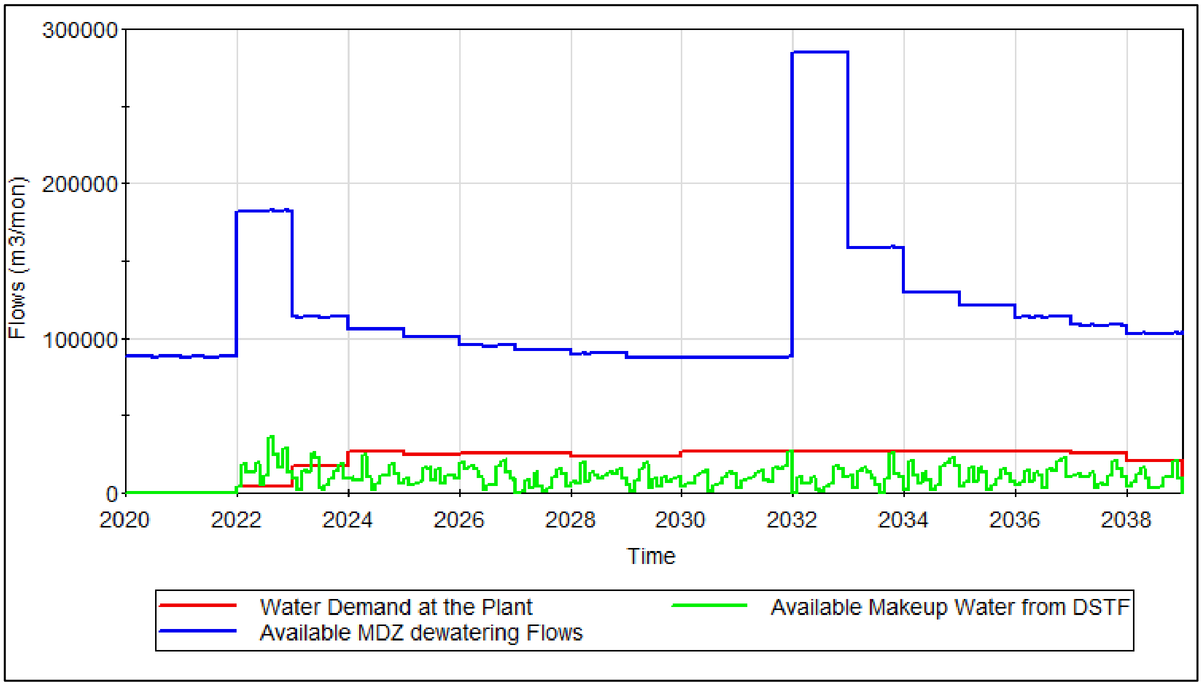
The water for the MDZ plant and mine will be supplied by a combination of groundwater from the mine, recycled water from the TSF and new dry stack tailings storage facility (DSTF), and fresh water from the Cauca River.

Water Balance Modeling

The water balance was developed to produce a makeup demand for water at the MDZ processing plant which could be sourced from the contact water produced by the DSTF, underground dewatering as described in Section 16.3.2, or an external freshwater source, prioritized in that order. The model also estimated if excess water would result from the DSTF contact water and/or the underground dewatering exceeding the makeup demand. SRK assumed that discharges from the underground could be discharged if monitored, and discharges from the DSTF can be discharged with monitoring and control of suspended solids and cyanide. This is further addressed in Section 20, Environmental Studies.

The water balance indicated that the water produced by the DSTF will occasionally exceed that required by the process plant as makeup, with the most excess during the first few years of the Project (25,000 m^3/mon), as production (and thus demand) is still ramping up but the DSTF footprint is relatively large. With limited storage available at site, excess water from the DSTF would need to be discharged.

Similarly, projected underground dewatering flows exceed the demand for water from the underground by a factor of 3 or more and excess underground dewatering is expected to be discharged at rates of 80,000 to 270,000 m^3/mon. Makeup demand vs available water from the DSTF and MDZ dewatering predicted by the model is shown in Figure 18-7.



Source: SRK, 2019

Figure 18-7: Makeup and Demand at the MDZ Process Plant

Overall, the water balance model of the Project indicates a net surplus of water for the Project due largely to an overall increase in available water from the MDZ dewatering and a decrease in makeup demand resulting from the lower moisture contents in the dewater tailings.

Although the water balance does not indicate there will be a need for additional water supply sources, during dry periods the underground dewatering will be the only available source of water. A back-up supply from the nearby Cauca river is recommended as a secondary supply source for time when underground dewatering flows are unavailable, especially during milling ramp-up stage.

18.4 Tailings Management Facilities

The following sections summarize reasonably available information regarding existing tailings generation and management and describe the conceptual design and operation of a new DSTF for filtered (dewatered) tailings. The existing TSF is estimated to provide three to four years of additional capacity, after which the new DSTF would need to accommodate tailings through the currently-estimated 16-year LoM. Where appropriate, recommendations for additional investigation(s), or expansion of existing baseline data collection programs, are provided.

On December 1, 2016, Mark Willow, a Qualified Person in accordance with Companion Policy 43-101CP to NI 43-101, conducted a personal inspection of the Marmato site under Section 6.2 of the Instrument. This inspection was intended to familiarize Mr. Willow with the conditions at the mine site and any potentially available material information that could affect mine development/expansion in this location. Information collected on site in 2016 was supplemented by GCM during 2019, as necessary. Mark Willow shared his site knowledge with his SRK colleague Joshua Sames, who has directly overseen the conceptual DSTF design described herein. In addition, SRK contracted with in-country geotechnical consulting company Dynami to perform site reconnaissance and coordinate with site personnel.

18.4.1 Existing Tailings Facilities

Currently, the processing plant sends tailings from the cyanide leach circuit to unlined settling ponds. The underdrain water from these ponds is directed to small collection basins downgradient of the tailings disposal piles. Flocculant is added to this water on an as-needed basis to remove residual suspended solids. The solids from this process are pumped back to the tailings settling ponds, while clarified overflow water is pumped back to the plant for use in the process. Excess water not needed at the plant is discharged under permit to the adjacent stream, Quebrada (Qda.) Cascabel. Once sufficiently dewatered to allow mechanical handling, the tailings are excavated from the ponds and transported via truck to the existing DSTF.

18.4.2 Tailings Storage Facility Siting Study

A number of potential tailings storage sites were evaluated during the PEA. During the site selection process, potential sites for both conventual slurry TSFs and DSTFs were evaluated.

The facilities were evaluated from technical, environmental, social, and cost perspectives. Other criteria that influenced the site selection were: topography; extent of heavy vegetation; proximity of communities; existing land use in and around the footprint; existing creeks, springs and streams; and, distances between the proposed process plant location and the prospective tailings storage facilities.

The conventional slurry tailings storage facilities typically required very large embankments and footprints/impacted areas and generally could not provide the required storage volume. Due to the steep terrain within the available land holding and the anticipated high capital and operating costs associated with a traditional slurry TSF, SRK concluded that a DSTF would be a more favorable tailings management method for the Marmato site. Recent advances in tailings filtration technology and performance favors tailings filtration and dry stacking for this project. Maximizing recycling and reuse of process water and thereby minimizing makeup water requirements is considered an additional benefit of the DSTF methodology.

To achieve the required storage for the LoM, two preferred locations were identified as the most feasible locations for DSTF construction for consideration in the PEA. GCM is currently evaluating additional siting options that may be evaluated as part of completion of a PFS.

18.4.3 Dry Stack Tailings Storage Facility

The preferred DSTF locations are shown in Figure 18-8 below. DSTF-1 would be constructed first and receive dewatered tailings from both the upper and the MDZ for the first one to two years after commissioning of the facility. After the filter plant is commissioned, tailings generated from mining in the MDZ would be filtered, trucked and stacked at DSTF-1 until it reaches capacity at approximately year 7, at which point filtered tailings would be hauled to DSTF-2, adjacent to the plant location.

Construction of DSTF-2 would commence one year prior to reaching capacity in DSTF-1 so no loss in production would be observed.

Source: SRK, 2019

Figure 18-8: Preferred DSTF Locations

18.4.4 Design Criteria

Conceptual designs for both DSTF-1 and DSTF-2 are based on an assumed 16-year LoM with a total mined tonnage of 26 Mt, and a maximum annual mining rate of 2 Mtpy for 16 years. Of the tailings produced, approximately 11.7 Mt (approximately 45% of total future mill throughput) would be mixed with cement and sent back to underground workings as paste backfill. The remaining 14.3 Mt (55% of total mill throughput) of tailings would be dewatered using filter presses to a target moisture content of about 15%, then trucked to the DSTFs, mixed with about 0.5% to 1% cement by weight, and placed and compacted in controlled lifts to a specified minimum compacted density. The dry density of the placed tailings was assumed to be 1.8 t/m^3 for volumetric calculations. Key DSTF design criteria are summarized in Table 18-1 below.

For the purposes of the PEA, the DSTF outer slope was designed at a 2H:1V (horizontal:vertical) slope. Cement addition is assumed to be required at this stage to achieve global stability of the tailings mass. The actual required amount of cement addition should be determined through laboratory testing, overall slope angles may be revised for global stability in the next stage of the Project.

Due to the tailings being filtered and placed and compacted with cement amendment, it is not anticipated that an appreciable amount of draindown from the tailings would contribute flows to the underdrain. The DSTFs are therefore assumed to be unlined, but would have an underdrain system for interception of any potential upwelling groundwater or flows from springs or seeps.

The rock starter embankments were designed with 2H:1V upstream and downstream slopes and would be constructed from mined NAG waste rock from underground workings or a nearby approved borrow source.

Table 18-1: DSTF Design Criteria

Criteria	Units	Description
General		
Total Annual Mined Tonnes	tpy	2,000,000
Total Mineralization Mined	t	26,000,000
Life of Mine	yr	13
Paste Backfill - percent of total tailings generated	%	45
Required Capacity	t	14,300,000
	m^3	7,840,000
100-year 24-hour Storm Event	mm	143
Filtered Tailings		
Placed Dry Density	t/ m^3	1.8
Moisture Content	w/w	15%
Tailings Acid Generation Potential	PAG/NAG	NAG
Tailings Transport and Stacking System		
Tailings Transport System from Filter to DSTF		Truck
Tailings Spreading within DSTF	-	Dozer
Tailings Cement Amendment	-	Tractor with Discs
Compaction of Tailings	-	Dozer and Vibrating Smooth Drum Roller
Overall Slope Angle	XH:1V	2:1
Rock Starter Embankment		
Rock Source		Waste Rock or Local Borrow
Downstream/Upstream Slopes	XH:1V	2:1

Source: SRK, 2019

18.4.5 Rock Starter Embankments

It was assumed that there would be sufficient and suitable Net Acid Generation (NAG) waste rock available to construct the rock starter embankments at the initial stages of the Project to ensure tailings stability and containment. The rock embankments would be constructed between ridges that form the primary valleys within which the DSTFs would be located. These ridges should consist of bedrock and constitute solid foundations for embankments foundation, and unsuitable soils should be removed from the embankment footprints. A geotechnical site investigation is recommended for completion during the next stage of study to confirm suitable foundation conditions.

The rock embankment fill areas would require foundation preparation prior to placement of the waste rock fill materials, to include clearing and grubbing and removal and stockpiling of salvaged topsoil materials for later use in reclamation.

Drainage systems would be provided in the foundation of the DSTF and continue under the rock starter embankment foundation. An allowance for the drainage systems is included in the cost estimate and consists of a longitudinal underdrain system with a number of connecting transverse drains under each DSTF footprint.

18.4.6 Tailings Stacking

Trucks would transport the filtered tailings from the filter plant to the DSTFs via dedicated haul roads, as shown in Figure 18-9. The dewatered tailings would be end dumped, spread in approximately 30-cm-thick loose lifts with a dozer, mixed with cement with discs, and compacted using a vibrating smooth drum roller or sheepsfoot compactor, as appropriate. It is currently anticipated that the cement amendment would be required to achieve global slope stability with 2H:1V slopes. Global slope stability should be evaluated following completion of the geotechnical site investigation and tailings materials characterization programs.

18.4.7 Underdrain and Surface Water Management

Seeps, springs or upwelling groundwater from within the DSTF footprint would be controlled by the installation of an underdrain system consisting of central header pipes and connecting transverse drains. Any seepage that is intercepted by the underdrain would be considered non-contact water and routed to a sediment pond below each DSTF's rock starter embankment and then released to the natural drainage downstream.

For the purposes of PEA cost estimation, the underdrain was considered to consist of a network of 1m diameter perforated and solid wall pipes surrounded by gravel drain and wrapped in a geotextile.

The upper operational deck surface would be graded so that contact runoff reports to a lined operational pond to facilitate pumping back to the plant site process water pond for reuse.

Concrete-lined diversion channels would be constructed around the DSTFs before any dewatered tailings are placed within the facility. These channels would direct non-contact stormwater runoff around the DSTFs and back into natural drainages downstream. As part of the current conceptual design, two set of channels would be constructed. The first at around the mid-height of the DSTF's planned valley fill to capture stormwater for the first half of the design life of the facility. The second would be the ultimate closure channels that would capture surface runoff for the second half of the DSSTF design life and serve as the final closure channel. Table 18-2 summarizes the currently-assumed size of the trapezoidal channels used in preparing the PEA cost estimate.

DSTF reclamation activities would be carried out concurrently during active operations per Section 18.4.9 and therefore do not require management of contact water.

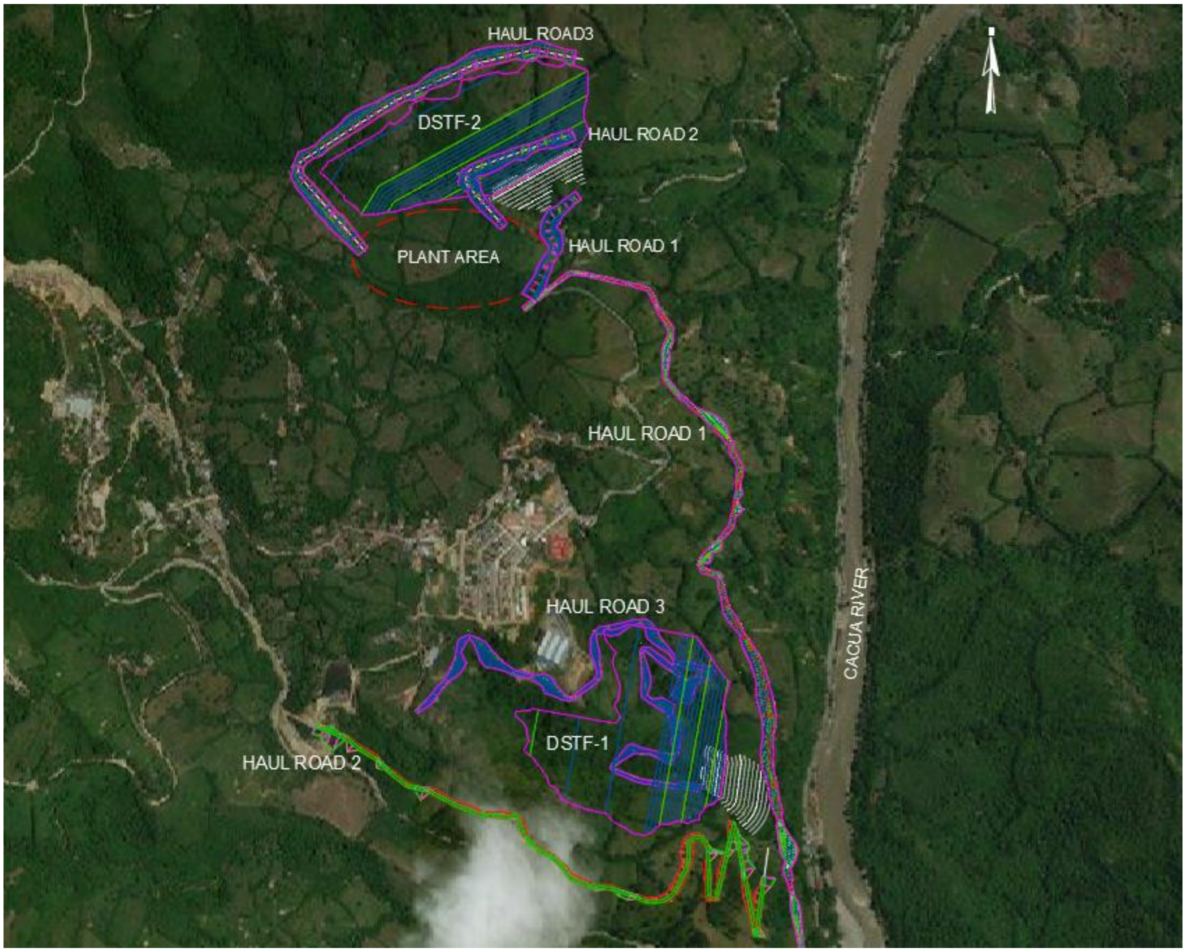
Table 18-2: Stormwater Diversion Channel Summary for 100-yr 24-hr Storm

Location	Length (m)	Bottom Width (m)	Depth (m)	Sideslope (m)
DSTF-1				
Mid-Bench SW Diversion Channel	1,600	1	1	1H:1V
FNL SW Diversion Channel	1,700	1	1	1H:1V
DSTF-2				
Mid-Bench SW Diversion Channel	1,400	1	2	1H:1V
FNL SW Diversion Channel	1,500	1	2	1H:1V

Source: SRK, 2019
Note: All channel are assumed to be concrete lined.

18.4.8 Access Road

The DSTFs would be accessed by dedicated haul roads (Figure 18-9) that include additional width to support stormwater management and safety berms. For the purposes of PEA cost estimation, the platform for both the road and drainage channels was assumed to be 20 m wide with a maximum grade of 15%.



Source: SRK 2019

Figure 18-9: DSTF Haul Roads

18.4.9 Reclamation

Reclamation of the DSTFs would be being undertaken concurrent with DSTF construction. As successive lifts of dewatered tailings are placed and compacted, the outer slope face would be covered with topsoil, vegetated and armored with rock cladding. The final top surface of the DSTF would be graded back to the natural slope and stormwater diversion channels at not less than 2%, covered with topsoil and revegetated.

18.5 Off-Site Infrastructure and Logistics Requirements

The Project has no significant off-site infrastructure needs and this section is provided for reference only.

18.5.1 Port

Marmato is 200 km west of the Pacific Ocean and 300 km south of the Caribbean Sea and Atlantic Ocean. The nearest port is Buenaventura on the Pacific Ocean (320 km by the Pan American Highway). The port will not be used for the Project other than as support for delivery of out of country equipment.

18.5.2 Rail

There is an abandoned railway cutting along the east side of the Cauca River opposite Marmato, which previously formed part of a railway network between the Pacific and Atlantic Oceans which ran between Buenaventura and Puerto Berrio on the navigable Magdalena River. The middle section between Medellín and La Felisa (Caldas, 10 km south of Marmato) was completed in 1942 and closed in 1972. Ferrocarriles del Suroeste S. A. (Southwestern Railways) applied for a concession to rebuild this 185 km line between La Felisa and Envigado, near Medellín, in November 2007, at a cost of US$140 million. This would become integrated with the national railway network. Ferrocarriles del Oeste S. A. (Western Railways) were awarded the contract to operate the 499 km Buenaventura to La Felisa railway in November 2007. The concession is in two stages. In July 2009 the 388 km railway between Buenaventura and Cartago and La Tebiada which has been rehabilitated was opened. In the second stage the new concessionaire will take over operation of the 119 km section between Cartago and La Felisa once this has been rebuilt by Tren de Occidente (Western Train). Currently the concession contractor (Western Railway) is in liquidation, so another company would have to develop the rail if needed for the Project. Currently there are no plans to use the rail.

19 Market Studies and Contracts

19.1 Commodity Price Projections

Gold and silver markets are mature, global markets with reputable smelters and refiners located throughout the world. Demand is presently high with prices for gold showing an increase during the past year. Markets for doré are readily available. Marmato possess a gold room for the production of doré.

Assumed prices are based on the long-term outlook for gold and silver. This projection is below the three-year trailing averages and current spot prices and are in-line with long term view of relevant market analysts in the precious metal sector. Table 19-1 presents the prices used for the cash flow modelling and resources estimation.

Table 19-1: Marmato Price Assumptions

Description	Value	Unit
Gold	1,300	US$/oz
Silver	17.00	US$/oz

Source: Gran Colombia Gold, 2019

19.2 Contracts and Status

Marmato currently has a long-term supply agreement for the sale of its products to an international refinery who take delivery of dore from the mine at designated transfer points in Colombia. The refinery is responsible for shipping the products abroad. The refining costs and discounts associated with the sales of the products are based on this agreement. This study was prepared under the assumption that the Project will sell doré containing gold and silver.

Treatment charges and net smelter return (NSR) terms are summarized in Table 19-2.

Table 19-2: Marmato Net Smelter Return Terms

Description	Value	Units
Doré		
Payable Gold	100%	
Doré Smelting & Refining Charges	6.38	US$/oz-Au

Source: Gran Colombia, 2019

20 Environmental Studies, Permitting and Social or Community Impact

The following section discusses reasonably available information on environmental, permitting and social or community factors related to the Marmato Project and the proposed expansion. Where appropriate, recommendations for additional investigation(s), or expansion of existing baseline data collection programs, are provided.

On December 1, 2016, Mark Willow, a Qualified Person in accordance with Companion Policy 43-101CP to NI 43-101, conducted a personal inspection of the Marmato site under Section 6.2 of the Instrument. This inspection was intended to familiarize Mr. Willow with the conditions on the properties, and any potentially available material information that could affect mine development/expansion in this location. Information collected on site in 2016 was supplemented by GCM during 2019, as necessary.

20.1 Environmental Studies and Management

The existing Marmato Project predates the regulatory requirements to prepare an environmental impact assessment as part of the permitting process. Instead, the operations were authorized through the approval of a PMA. The PMA for Marmato was approved by the regional environmental authority (*Corporación Autónoma Regional del Caldas* or Corpocaldas) on October 29, 2001 under Resolution 0496, File No. 616. The site-specific PMA covers environmental studies and required management procedures and practices associated with:

- Reclamation in the area of the production plant;
- Reclamation and closure of the tailings settling ponds;
- Management of unstable zones (including erosion control);
- Water management in the mines;
- Management of stormwater runoff;
- Management and protection of watersheds;
- Control planning and use of explosives;
- Reforestation and revegetation programs;
- Reclamation and closure planning;
- Management of tailings;
- Containment structures;
- Cyanide destruction (detoxification);
- Management of waste water (domestic);
- Water usage;
- Air resource management;
- Physical risk management measures (including toxic substances);
- Social management; and
- Biological Management (i.e., biodiversity).

In 2017, an audit of this PMA environmental management system was prepared from which most of the following information was obtained.

20.1.1 Environmental Setting

The Marmato Project is located in the Municipality of Marmato, Department of Caldas, Republic of Colombia, and is approximately 125 km due south of the city of Medellín, the capital of the Department of Antioquia. The Town of Marmato was founded in 1540 and has a population of approximately 10,000 people. It is one of the historic gold-mining regions of the hemisphere. The Marmato Project has excellent infrastructure, being located along the Pan American Highway with access to Medellín to the north and Manizales (the capital of Caldas) to the south, and has access to the national electricity grid which runs near the property.

The west-central Colombian Department of Caldas is situated in the Cordillera Central of the Andes Mountains and is bounded by the Magdalena River on the east and the Cauca River on the west. Marmato lies at an elevation of 1,050 masl, just west of Río Cauca, which joins the Magdalena River near Magangué in Bolívar Department, before eventually flowing out into the Caribbean Sea.

The local topography is characterized by steep, incised valleys. The climate is tropical with an annual average temperature of 21°C, that typically varies from 14°C to 24°C, and average annual rainfall of approximately 2,162 mm/y, predominantly falling between April and November, with a negligible difference of 153 mm of precipitation between the driest and wettest months. The drainage pattern across Marmato is dendritic; the license area drains east into the Cauca River, which is heavily influenced by artisanal mining operations. The vegetative cover across the landscape consists of disturbed grassland (used mainly for mining and livestock rearing activities) interspersed with fragmented forest patches, mainly along drainage lines within the incised valleys. Forest patches provide important habitat for wildlife.

The operations are located within the town of Marmato, which has been a center for gold mining for more than 500 years and the environmental and social setting is strongly influenced by this. Mining, both formal and informal, is the main economic activity in Marmato and the neighboring towns of El Llano and La Garrucha. Informal processing operations using basic technology has resulted in poor health and safety conditions and widespread water contamination from discharge of tailings and waste directly into the environment. This has led to a prevalence of mercury-related health problems in the local populations. Health issues related to population influx are also common.

20.1.2 Water Quality and Monitoring

GCM has seven domestic wastewater discharges and three non-domestic wastewater discharges for which monitoring is conducted. Table 20-1 lists each discharge point with its respective analytical parameters to be measured. The results of the monitoring are provided to the regional environmental authority (Corpocaldas).

Table 20-1: Water Discharges

ID	Waste Water Discharge	Discharge Point	Parameters
1	Non-Domestic	Tailings dam	Turbidity, pH, temperature, total solids, total suspended solids, DB05, DQ0, grease and oils, flow, conductivity, sedimentary solids (for treatment systems – bulk tails, if total cyanide and lead are taken into account).
2		Thickener	
3		Sedimentation Ponds	
4	Domestic	Camp 1	pH, temperature, total solids, total suspended solids, DB05, DQ0, grease and oils, flow, fecal coliform and total coliform.
5		Camp 2	
6		Camp 3	
7		Offices	
8		Mines	
9		Mine Dry	
10		Contractor Lodging	

Source: GCM S.A.S.

20.1.3 Air Quality and Monitoring

Air quality emissions from stationary sources at Marmato are regulated and monitored by the Air Pollution Unit (*Unidad de contaminación Atmosférica* or UCA) according to Table 20-2.

Table 20-2: Stationary Emission Sources

Unit	Parameter	UCA	Degree of Significance	Monitoring Frequency	Next Measurement Date	Observations
Metallurgical Laboratory	Particulate matter (PM)	0.01	Very low	3 years	03/07/2017	
	Sulphur dioxide (SO_2)	0.00	Very low	3 years	03/07/2017	
	Nitrogen oxides (NO_X)	0.04	Very low	3 years	18/02/2019	NO_X measurement took place on 18/02/2016
	Lead (Pb)	0.014	Very low	3 years	03/07/2017	
Smelter / Foundry	Particulate matter (PM)	0.05	Very low	3 years	04/08/2017	
	Sulphur dioxide (SO_2)	0.37	Low	2 years	04/08/2016	Measurement took place in 29/11/2016
	Nitrogen oxides (NO_X)	0.21	Very low	3 years	04/08/2017	
	Lead (Pb)	0.47	Low	2 years	04/08/2016	Measurement took place on 29/11/2016

Source: GCM S.A.S.

20.1.4 Environmental Procedures and Permissions

Environmental protection measures and procedures followed by GCM at the Marmato operations include those shown in Table 20-3.

Table 20-3: Environmental Procedures

ID	Resource	Environmental Procedure	Location	Approval	Current State of Process					Competent Authority
					New	Valid	Valid for (years)	Renewal or Modification	Filing Date	
1	Air	Atmospheric emission permit	Smelter	Resolution 270 of April 27, 2009			5	×	February 21, 2014	Corpocaldas
			Laboratory furnace							
			Shedding filter bag							
2	Water	Surface water concession	La Maruja portal			×	5			Corpocaldas
			Aguas Claras	Resolution 0046 of March 09, 2004, amended by Resolution 127 of May 5, 2004				×	February 07, 2014	
			Zaparillo							
			Guineo							
		Domestic water discharge permit	Camp 1	Resolution 270 of April 27, 2009 amended by resolution 254 of February 28, 2014				×	February 21, 2014	
			Camp 2							
			Camp 3							
			Office							
			Mine							
			Contractor							
			Mine dry							
		Non-domestic water discharge permit (industrial)	Tailings							
			Thickener							
			Sediment ponds							
		Channel Occupation	Charco Hondo	Resolution 0062 of February 15, 2006						Corpocaldas

Source: GCM, S.A.S.

20.1.5 Environmental Management

Environmental management programs for the current Marmato operations and associated costs, are provided in Table 20-4.

Table 20-4: Environmental Management Budget

Item	Description	Annual Cost (Peso)	Annual Cost (USD)
1	Tailings Management	$26,909,709	$7,848
2	Forest Nursery Management	$5,894,500	$1,719
3	Order and Cleanliness (Housekeeping)	$12,237,000	$3,568
4	Potable Water Plant Operation	$10,541,644	$3,074
5	Septic Tank Maintenance	$37,951,500	$11,068
6	Environmental Monitoring Samples	$33,506,200	$9,771
7	Environmental Education Program	$33,506,200	$9,771
8	Reclamation of Dumps/Ponds 1 & 2	$16,124,000	$4,702
9	Reforestation	$2,920,000	$851
10	Unstable areas	$8,120,000	$2,368
11	Bulk Tailings Effluent Treatment	$538,991,160	$157,193
12	Tailings Effluent Cyanide Treatment	$94,981,814	$27,700
13	Water pumping and recirculation	$190,228,000	$55,478
14	Tailings Dam Operation	$251,780,000	$73,430
15	Labor	$187,646,400	$54,726
16	Administration and Payment of Services	$6,600,000	$1,924
ENVIRONMENTAL MANAGEMENT BUDGET*		$1,457,938,128	$425,199

Based on exchange rate of 3,455 to 1
Source: GCM S.A.S., 2017

20.1.6 Geochemistry

The objective of this part of the program is to identify, at a preliminary level, aspects of environmental geochemistry that might produce impacts that could negatively affect the project economics. At the PEA stage of a project, and for a deposit of this type (i.e., precious or base metal), the evaluation typically includes the following:

- Acid rock drainage and metal leaching (ARDML) potential of tailings;
- Chemical composition of tailings supernatant and the potential for seepage to the environment. The assessment of tailings impacts is often preliminary because processing methods and TSF construction plans might be modified as the Project advances;
- Acid rock drainage and metal leaching potential of waste rock;
- Chemical composition of inflowing underground mine water and the need for management of the inflow based on the chemistry; and
- Background groundwater and surface water quality, and an assessment of potential impacts to ambient groundwater and surface water quality due to mining activities; additionally, how the ambient groundwater and surface water might affect mining operations.

At this stage of a project, SRK relies on existing data and information for the environmental geochemistry evaluation rather than collect new data. The primary sources of information for this assessment were the Updated Mineral Resource Estimate (SRK, 2017), the Hatch report (2012), and the Mineral Processing and Metallurgical Testing section of this report. SRK was not provided copies of the mining waste characterization reports, so we are unable to comment on that aspect of the program.

Findings

Characteristics of the Marmato deposit that are significant with respect to environmental geochemistry include the following:

- Acid-generating sulfide minerals identified in the deposit include pyrite, arsenopyrite, iron-bearing sphalerite, pyrrhotite, and chalcopyrite (SRK, 2017). Sulfide concentrations are locally elevated, as demonstrated by the Terra Mineralogical Services study (2019) conducted for the SRK metallurgical program, which reported a pyrite concentration of 23.1% in mineralization, along with lesser concentrations of sphalerite, pyrrhotite, chalcopyrite, and arsenopyrite.
- Samples of groundwater discharging in the underground, collected by Colombia Goldfields Ltd for the baseline study, indicated that 13 of the 14 samples reported acidic pH. Four samples reported pH less than 3, and another eight reported pH between 3 and 4.
- The underground water samples carried elevated metal(loid)s, many of which exceed World Bank target guideline levels for mining discharges, including: Arsenic (2.64 mg/L against target level of 1.0 mg/L), cadmium (2.12 mg/L against target level of 0.1 mg/L), copper (2.64 mg/L against target level of 0.3 mg/L), iron (332 mg/L against target level of 2.0 mg/L), lead (7.06 mg/L against target level of 0.6 mg/L), mercury (4.88 µg/L against target level of 2.0 µg/L), and zinc (663 mg/L against target level of 1.0 mg/L).
- Geochemical properties of current and future waste rock are unknown, as no test data have been provided. Knight Piésold reportedly started a geochemistry characterization program of waste rock and tailings, but the status of the program is unknown. Based on the characteristics of recently tested mineralization, it is reasonable to assume that future waste rock will contain mineralogical characteristics capable of generating ARDML.
- Field leach test cells were reportedly started but the type of rocks that were tested and the status of the tests are unknown.
- Waste rock has been used to backfill underground or placed in unused headings. The program is expected to continue. Non-acid generating waste rock has reportedly been used on the surface for construction purposes.
- The tailings backfill is expected to continue, with approximately an estimated 50% of tailings to be assigned to backfill.
- There is conflicting information in the Hatch report regarding the geochemical leaching properties of the hydraulic backfilled tailings. Water samples collected underground, and believed to be sourced from backfilled material, produced neutral pH. However, onsite humidity cell tests conducted by Knight Piésold (2012), reportedly indicated that tailings from the Mineros Nacionales mine generated acidic pH.
- Potential environmental liabilities reported by SRK (2017) include:
  - Historical dumping of waste rock down mountain slopes and dumping of tailings into watersheds represents a potential liability;
  - Contamination of water by cyanide, acid drainage, heavy metals and solids;
  - Contamination of water by untreated sewage from the Marmato and El Llano towns and by agricultural chemicals and waste from cultivation of coffee, bananas etc.; and
  - Potential contamination from the Mineros Nacionales operation which has:
    - Unprotected waste rock dumps; and
    - A reportedly unlined tailings facility that discharges overflow directly into Cascabel creek and subsequently the Cauca River.

Risks and Uncertainties

SRK's review identified the following risks and uncertainties related to environmental geochemistry of the future mine.

- The abundance of sulfide minerals in the deposit combined with an apparent scarcity of acid-neutralizing minerals (e.g., carbonate minerals such as calcite, dolomite) indicates a potentially elevated threat of ARDML, both on a regional and local scale depending on the concentration and distribution of acid-generating versus acid neutralizing minerals. The data and reports provided to SRK contain insufficient information to make this determination, and SRK recommends further investigation in the next phase of work.
- Based on metallurgical testwork completed for this report, all mineralization except for recovered gold and silver will report to tailings, including acid-generating sulfide minerals. If the reported pyrite concentration of 23.1% in Marmato material is representative of tailings in general, then the ARDML potential of future tailings could be considerable.
- The acidic and metalliferous characteristic of groundwater discharging in the underground workings may be evidence of a significant mass of reactive acid-generating sulfide minerals with elevated trace metal content in both mineralized material and sub-mineralized grade rock. Additional data collection is recommended in the next phase of work to determine the ARDML properties of waste rock that could potentially be brought to surface for disposal.
- The ARDML characteristics and leaching potential of historical tailings, waste rock, and artisanal tailings in the area is an unknown that could pose an environmental risk. Detailed characterization is recommended in the next phase of work.
- As reported in SRK (2017), the Marmato Project holds the potential for significant adverse environmental and social impacts which are diverse. Extensive assessment of the physical, biophysical and socioeconomic features of the area are required and careful planning of project infrastructure should occur.

20.2 Mine Waste Management

20.2.1 Waste Rock Management

Very little waste rock is generated by the underground operations at Marmato. What little waste rock is generated is used as structural backfill in the underground workings or on the surface for small construction projects, such as maintenance of roads. SRK is unaware of any geochemical or geotechnical testing of the waste rock for use as construction material, so the potential impact associated with ARDML cannot be assessed at this time. However, since so little, if any waste rock is brought to the surface, a comprehensive geochemical characterization program is probably not warranted on this surface located material, and only opportunistic sampling and testing of construction materials is probably necessary. At the moment, Corpocaldas does not require the testing of mine waste materials; only effluent discharges. This is likely to change in the future, as source control becomes more of the norm in this jurisdiction.

20.2.2 Tailings Management

The Marmato gold processing plant is fed with mill feed which is milled and processed through a cyanide (CN) leach circuit using underground dewatering water and make-up water from the surface. Currently, approximately 70% of the tailings from the operations is returned to the underground workings as structural backfill. The remaining approximately 30% (including fines) are slated for surface disposal.

The CN leach circuit includes a hydrogen peroxide (H_2O_2) and copper sulfate ($CuSO_4$) destruction unit on the tail end to reduce CN concentrations to below the Colombian mine effluent discharge limit of 1.0 mg/L (Article 10 of Resolution 0631, dated March 17, 2015) before sending the residual tailings to the unlined settling ponds ("Cascabel 1" and "Cascabel 2"). The underdrain water from these ponds is directed to small collection basins downgradient of the tailings surface disposal piles. Flocculant is added to this water on an as-needed basis to remove residual suspended solids; the underflow (solids) from this process is directed back to the tailings settling ponds, while the clarified overflow water is pumped back to the plant for use in the process. Excess water, not needed at the plant, is discharged under permit to the adjacent stream, Quebrada (Qda.) Cascabel.

Once sufficiently dewatered to allow mechanical handling, the tailings are excavated from the ponds and transported via truck to the final disposal location. Monitoring of the residual tailings to determine whether or not they are classifiable as 'hazardous' is accomplished through Corrosive, Reactive, Explosive, Toxic, Inflammable, Pathogen [biological] (CRETIP) analyses. Toxicity analyses were carried out by the Universidad Pontificia Bolivariana on cyanides and metals (chromium, mercury and lead). The results support the classification of the tailings as non-toxic for the metals based on comparisons to the maximum concentration thresholds established by Decree 4741 of 2005. The analyses also showed that Total CN was below the threshold allowed in Decree 1594 of 1984 for water discharges. SRK did not perform a comprehensive audit of all testing data; however, given the vintage of these results, SRK suggests that more frequent sampling and analysis be conducted by the operation going forward, and especially for the expansion project.

Limited data and information are available pertaining to the geochemistry of tailings. However, general conclusions can be drawn based on the geochemistry of the orebody and the process methodology. Until detailed data are collected on the acid generating potential of the tailings (as recommended by SRK), the conservative assumption will be to assume acid generating tailings with appropriate management.

For the deep zone expansion of the Marmato Project, GCM intends to continue with a similar approach to tailings and tailings water management. However, rather than using less efficient settling ponds, the tailings will be filter pressed. The "dry stack" tailings will be transported to the new disposal facility, mixed with cement, and stacked in a configuration that minimizes surface runoff. To the extent practicable, contact water collected on the deck of the DSTF will be reused in the process; excess water will be discharged under permit.

A critical driver of environmental impacts from tailings is whether contact water will be contained. The current and predicted future quality of contact water needs to be determined. Two components of potential chemical loading need to be estimated: 1) loading to surface water due to tailings runoff, and loading to groundwater through seepage from the base of the tailings facilities.

Future metallurgical testing should include environmental geochemistry, to provide data that will assist in forecasting tailings solids and potential runoff chemistry. Water re-use/recycling is recommended to the extent feasible. A zero-discharge policy should be adopted so that water recycling is maximized, and water treatment and discharge is minimized.

20.2.3 Site Monitoring

Various mitigation and monitoring programs are discussed in the approved PMA (per Section 20.1).

20.2.4 General Water Management

Operational water for the Marmato operation is provided through a combination of underground mine dewatering water, and several surface freshwater sources. Even with the high precipitation experienced by the site, only nominal effort appears to be directed toward stormwater management and the prevention of contact with mine equipment and facilities. Some concrete channels and energy dissipation structures for the management of run-off are already constructed, and some others are being considered.

Surface water runoff is likely to represent a significant water management challenge to the Project considering the difficulties in distinguishing between the impacts from the artisanal mining activities and those of the Project.

Insufficient work has been undertaken on groundwater hydrogeology and surface water to establish the true level of risk associated with potential groundwater contamination. A detailed evaluation, including a groundwater model, would provide information that would assist in forecasts of post-closure mine water discharge and possible treatment criteria.

The geochemical and hydrogeological/hydrological impacts should be evaluated at closure when dewatering ceases and water levels rebound. Of critical importance is the possibility of mine water discharging to surface water or groundwater and potentially impacting users. There are reports that dewatering effluent carries elevated concentrations of metals. The water quality of dewatering effluent must be well characterized in the event that treatment is needed before it is used or discharged. A forecast of closure water quality is needed.

20.2.5 Off-Site Impacts

All mining wastes releasing uncontained runoff could be contributing to offsite impacts, and mitigation measures should be implemented.

Informal processing operations in this location, using basic technology, has resulted in poor health and safety conditions and widespread water contamination from discharge of tailings and waste directly into the environment. This has led to a prevalence of mercury-related health problems in the local populations.

The largest uncertainty regarding closure costs is associated with the potential need for long-term treatment of water from the post-closure mine workings.

A comprehensive baseline surface and groundwater sampling program will be important to establish the baseline condition and try to quantify the contributions from artisanal or pre-mining conditions, especially with respect to mercury from artisanal mining.

20.3 Project Permitting

20.3.1 General Mining Authority

Since 1940, the Ministry of Mines and Energy (MME), formerly the Mines and Petroleum Ministry, has been the main mining authority with the legal capacity to regulate mining activities in accordance with the laws issued by the Colombian Congress. The MME can delegate its mining related powers to other national and departmental authorities. Mining regulations in Colombia follow the principle that (except for limited exceptions) all mineral deposits are the property of the state and, therefore, may only be exploited with the permission of the relevant mining authority, which may include the MME, the National Agency for Mining or the regional governments designated by law.

In 2001, the Congress issued Law 685 (the Mining Code). This law established that the rights to explore and exploit mining reserves would only be granted through a single mining concession agreement (the 2001 Concession Agreement). This new form of contracting did not affect the pre-existing mining titles (licenses, aportes and concessions) which continue to be in force until their terms lapse. The 2001 Concession Agreement includes the exploration, construction, exploitation and mine closure phases and are granted for periods of up to 30 years. This term may be extended upon request by the title holder for an additional 30-year term. According to the Mining Code, the initial term was divided into three different phases:

- **Exploration** – During the first three years of the concession agreement, the title holder will have to perform the technical exploration of the concession area. This term may be extended for two additional years upon request;
- **Construction** – Once the exploration term lapses, the title holder may begin the construction of the necessary infrastructure to perform exploitation and related activities. This phase has an initial three-year term which may be extended for one additional year; and
- **Exploitation** – During the remainder of the initial term minus the two previous phases, the title holder will be entitled to perform exploitation activities.

20.3.2 Environmental Authority

In 1993, Law 99 created the Environmental Ministry and then in 2011 the Decree 3570 modified its objectives and structure and changed the name to Environment and Sustainable Development Ministry. The Ministry is responsible for the management of the environment and renewable natural resources, and regulates the environmental order of the territory. Also, the Ministry defines policies and regulations related to rehabilitation, conservation, protection, order, management, use, sustainable use of natural resources. Article 33 of the same Law created the regional environmental authorities (including Corpocaldas) with the responsibility to manage the environment and renewable natural resources.

In 2011, Decree 3533 created the National Authority of Environmental Licenses (*Autoridad Nacional de Licencias Ambientales*, ANLA). ANLA is responsible that all project, works or activities subject to licensing, permit or environmental procedures comply with the environmental regulations and contribute to the sustainable development of the Country. ANLA will approve or reject licenses, permits or environmental procedures according to the law and regulations, and will enforce compliance with the licenses, permits and environmental procedures.

With regard to the licensing process of mining projects, the competence of either ANLA or Corpocaldas is determined by the annual volume of material to be exploited. For projects exploiting more than 2 Mtpy the responsibility will be with ANLA. Both ANLA and Corpocaldas can enforce project compliance with the terms of their licenses or permits. Up to now, based on the annual production and transport of materials at Marmato, the environmental authority that controls operations is Corpocaldas.

20.3.3 Environmental Regulations and Impact Assessment

Colombian laws have distinguished between the environmental requirements for exploration activities, and those that have to be fulfilled for construction and exploitation works. During the exploration phase, the concession holder is not required to obtain an environmental license. However, the concession holder requires environmental permits which will be obtained from the regional environmental authority. The concession holder will have to comply with the mining and environmental guidelines issued by the MME and the Environmental Ministry.

In order to begin and perform construction and exploitation operations, the concession holder must obtain an environmental license or the approval of an existing Environmental Management Plan (EMP) either from ANLA if the Project exploits more than 2 Mtpy or from the regional environmental authority (Corpocaldas) if the mineral exploitation is less than 2 Mtpy.

The approval process begins with the request for Terms of Reference (ToR) to prepare an Environmental Impact Statement (EIS) or update an existing PMA. The approval of the EIS and PMA by the jurisdictional environmental authority includes all environmental permits, authorizations and concessions for the use, exploitation or affectation, or all of the above, of natural resources necessary for the development and operation of the Project, work or activity. Additionally, other permits and requirements (non-environmental) are required in order to begin construction and operation of the Project.

Non-Governmental Organizations (NGOs) and the local communities have the opportunity to participate in the environmental administrative procedures leading up to the issuance of an environmental license. The environmental process will include participation of, and information to, all communities in the project area including indigenous communities and Afro-descendant communities.

20.3.4 Water Quality and Water Rights

The Colombian regulations that principally govern water quality, including discharge permitting and requirements, are Decree 2811 of 1974, Decree 1541 of 1978, Decree 1594 of 1984, Decree 3930 of 2010 and Resolution 631 of 2015, that establish the maximum permissible limits for discharges to surface water.

Currently, the Decree 1594 of 1984 is under review, which will modify the maximum concentrations, allowed in industrial effluents. Resolution 631 of 2015 (new parameters and maximum limits on point discharges) is being used as a guideline for this project. The regional environmental authority (Corpocaldas) enforces compliance with these regulations.

Water rights for mining activities are granted by means of a water concession which is granted by Corpocaldas and which is independent to the mining concession or to land ownership. The water rights related to mining activities are included in the environmental licenses or in the approved PMA and are normally granted for five years. The terms and conditions under which a water concession is granted may depend, amongst others, on the amount of water available in the specific region, the possible environmental impact of the concession, water demand, the ecological flow and the different users that the water source services. The water concession is accompanied with a discharge permit.

Water concessions held by GCM for the Marmato Project include:

Table 20-5: Surface Water Concessions

Location	Approval	Term (Years)	Renewal of Modification	Filing Date	Competent Authority
Bocamina La Maruja	Resolution 345 of March 17, 2014	5	×	January 29, 2019 (In process)	Corpocaldas
Aguas Claras	Resolution 0046 of March 9, 2004 (Amended by Resolution 127 of May 5, 2004)	10	×	February 7, 2014 (In process)	
Zaparillo					
Guineo					

Source: GCM S.A.S.'

20.3.5 Air Quality

Decree 948 of 1995, Resolution 650 of 2010 and Resolution 2154 of 2010 provide the main regulations on protection and control of air quality. These regulations set forth the general principles and regulations for the atmospheric protection, prevention mechanisms, control and attention of pollution episodes from fixed, mobile or diffused sources. These regulations also provide emission levels or standards. Among the emission sources regulated are: controlled open burnings, discharge of fumes, gases, vapors, dust or particles through stacks or chimneys; fugitive emissions or dispersion of contaminants by open pit mining exploitation activities; solid, liquid and gas waste incineration; operation of boilers or incinerators by commercial or industrial establishments, etc.

Also, Resolution 627 of 2006 regulates noise emissions in terms of ambient noise. The parameters regulated are: SO_2, NO_2, CO, TSP, PM_{10}, O_3, and noise. Corpocaldas enforces compliance with these regulations at Marmato.

20.3.6 Fauna and Flora Protection

The main regulations for the protection of fauna and flora are contained in the Natural Resources Code and the Agreement about Biological Diversity entered into in Rio de Janeiro on June 5, 1992, within the framework of the Rio Convention. Also, forest management and use is regulated by Decree 1791 of 1996 and the compensation for biodiversity loss is regulated by Resolution 15717 of 2012. In addition, there are other important regulations on the matter such as the Cartagena Protocol on Biotechnology Security of the Agreement about Biological Diversity entered into in Montreal on January 29, 2000, and the Convention on International Trade of Threatened Wild Fauna and Flora Species (CITES). Endangered species are protected by environmental and criminal law.

In order to perform biodiversity studies, a permit for scientific investigation must first be obtained from Corpocaldas.

20.3.7 Protection of Riparian Areas and Drainages

Resolution No. 077 of March 2, 2011, regarding riparian and water channel protection, strictly prohibits the filling of perennial water courses except under very specific terms: road and pipeline crossings, bank and slope protection measures, and installation of public service networks (Title III, Article 9). The backfilling of intermitted or ephemeral channels can be authorized under permit by Corporcaldas, provided that the design is appropriate for the conditions, and that surface water and groundwater are properly managed. Application of this prohibition directly influenced the siting of the future tailings disposal areas, in that it(they) cannot be located in perennial drainages.

20.3.8 Protection of Cultural Heritage or Archaeology

Cultural and natural heritage protection in Colombia is stated in the political constitution and developed through several international treaties and laws of the state. There are strict legal provisions, such as Law 397 of 1997 and Decree 763 of 2009, whereby the heritage is safeguarded and protected. For example, if a citizen finds an archeological specimen, he or she must inform the Ministry of Culture of the discovery within 24 hours; otherwise he or she could be sanctioned by the competent authority.

20.3.9 Marmato Permitting

The Marmato Project is authorized under a number of resolutions issued by the regional environmental authority (Corpocaldas) in the name of GCM's predecessor, Mineros Nacionales S.A.S. These are identified in the Environmental Studies and Management section (above), and include, among others:

- Environmental Management Plan or PMA (Resolution No. 0496);
- Various water concessions; and
- Discharge permits (Resolutions 270 and 255).

According to GCM, all of the resolutions for operation of the existing Marmato Project are current and valid, or in the process of being renewed. Most will need to be amended as part of the authorization of the MDZ expansion of the Marmato Project to include the increases tonnage to be mined, new process facilities, and tailings management area.

The current PMA authorizes the mining and processing of up to 1,500 tpd of ore. The current processing plant has capacity of 1,200 tpd. The PMA will need to be modified to allow for the proposed MDZ expansion project, which envisions a rate of an extra 4,000 tpd in a second processing plant to be constructed. By regulation, the total of mined material (including waste and material) cannot exceed 2Mtpy in order for Corpocaldas (regional Environmental Authority) to remain as the permitting authority. If more than 2Mtpy is mined, then the PMA will need to be submitted to and authorized by the federal authority, ANLA (Environmental License National Authority). During construction, Channel Occupancy Permits will most likely need to be obtained for the new tailings site, the tailings pipeline corridor, the process plant site, and the site of the underground portal (bocamina). Likewise, a Forest Exploitation Permit may be needed for areas of proposed surface disturbance with trees (DBH> 10 cm).

These new facilities and operation will be subject to the environmental licensing process described above, which will require the submittal of comprehensive design reports, hydraulic and hydrogeological investigation reports, geotechnical reports on stability, and an environmental impact assessment will need to be prepared. This will require that adequate baseline data be collected from which the significance of potential impacts can be assessed.

Operationally, the existing discharge, emissions (if applicable), and water concession permits may also require modification to suit the new mining conditions. GCM estimates that a minimum of six to 12 months will be required for review of the complete application and issuance of the Environmental License by Corpocaldas for the MDZ expansion of the Marmato Project.

Additional baseline data collection will need to be completed on the new areas proposed for the plant site, portal, ancillary facilities, and tailings disposal area(s). GCM is currently in the process of bidding out the baseline programs, and also the modifications to the PMA to an environmental consultant (under a single contract). The plan is to award this contract in Q4 2019, with an anticipated completion by end of Q1 2020 and submission to Corpocaldas.

20.3.10 Performance and Reclamation Bonding

The termination of a mining concession can happen for several reasons: resignation, mutual agreement, and expiration of the term, the concession holder's death, free revocation and reversion. In all cases, the concession holder is obliged to comply or guarantee the environmental obligations payable at the time the termination becomes effective.

The 2001 Mining Code requires the concession holder to obtain an Insurance Policy to guarantee compliance with mining and environmental obligations which must be approved by the relevant authority, annually renewed, and remain in effect during the life of the Project and for three years from the date of termination of the concession contract. The value to be insured will be calculated as follows:

- During the exploration phase of the Project, the insured value under the policy must be 5% of the value of the planned annual exploration expenditures;
- During the construction phase, the insured value under the policy must be 5% of the planned investment for assembly and construction; and
- During the exploitation phase, the insured value under the policy must be 10% of the value resulting from the estimated annual production multiplied by the pithead price established annually by the government.

According to the Law, the concession holder is liable for environmental remediation and other liabilities based on actions and or omissions occurring after the date of the concession contract, even if the actions or omissions are by an authorized third-party operator on the concession. The owner is not responsible for environmental liabilities which occurred before the concession contract, from historical activities, or from those which result from non-regulated mining activity, as has occurred on and around the Marmato Project site.

According to GCM, an Environmental Insurance Policy is in effect for the Marmato Project, though no other information was provided at this time.

20.4 Social Management

The 2001 PMA for Marmato specifically requires the management of the social component of the Project. GCM is required to maintain records on all community activities (including number of participants, topics, duration, etc.), which is to be turned over to Corpocaldas as part of the ongoing monitoring programs.

20.4.1 Stakeholder Engagement

As part of the social management and monitoring program, GCM has developed a social investment model which seeks to promote the development of communities in the area of influence, with the purpose of contributing to the consolidation of society and fostering economic development (Economic Development), guaranteeing the care and respect for the environment (Environmental Development), and supporting and participating in actions aimed at improving the quality of life and well-being of its inhabitants (Social Development and Promotion of Solidarity Actions). Activities in 2017 included, among others (for example):

- Direct economic compensation in excess of 3,850,000,000 pesos (US$1.123 million) (including state royalties and a payment of 2,079,000,000 pesos [US$606,330] to the municipality of Marmato);
- Support of education programs, such as Mining Training School with the SENA National Center, Mine Rescue Training, Nutrition and Safety Training, etc.
- Support for traditional festivities of local municipalities like Ferias de El Oro and Fiesta of El Barequero;
- Support for the Afro-Colombian meeting; and
- Health programs including vaccination days, sexual education workshops, drug addiction prevention workshops, etc.

According to GCM, the company has a complaints and petitions handling procedure to record grievances both at the company offices and community office in El Llano. The grievance recording and response procedures follow international good practice.

20.4.2 Artisanal and Small-Scale Mining Operations

The Marmato district has been exploited since pre-Colonial times by the Quimbaya people. The Spanish colonists assumed control of the Marmato mines in 1527 and the area has been in almost continuous production ever since. This mining is informal/artisanal in nature (sometimes referred to as "traditional"), which is the general characteristic of the mining sector in Colombia. A recent census revealed that 72% of all mining operations in Colombia are classed as 'artisanal and small-scale mining' (ASM), and 63% are 'informal', lacking a legal mining concession or title. Large-scale mining (LSM) only accounts for 1% of operations. Over 340,000 Colombians depend directly on ASM and medium-scale mining (MSM) for their income. This informality deprives the state of important financial resources, while the current poor conditions (environmental, social, health and safety, labor, technical and trading) prevent the sector from delivering on important social objectives, such as generating formal employment and improving the quality of life in mining communities (Echavarria 2014).

In 2013, a decree (933) was enacted to address the legal void for almost 4,000 requests for formalization from Law 1382 of 2010, which was promulgated, in part, with the objective of combating illegal mining, while recognizing the traditional nature of informal ASM. This decree redefined traditional mining as a form of informal mining. It set out formalization procedures for ASM in LSM mining concessions and titles, notably including procedures for concession-owners to cede areas to ASM, and included tax incentives. For the first time, it also provided options for areas returned to the state to be reserved for ASM formalization. In addition, Mercury Law No. 1658 of 2013, introduced incentives for the formalization of ASM such as: granting of soft credits and financing programs to facilitate access to resources; and created a sub-contract intended to formalize illegal mining activities with the registered license-holder. Under Article 11 of Law 1658, concession owners can sign subcontracts with ASM operating in their concessions without the liability associated with normal operating contracts. These subcontracts will legally allow these ASM to operate in an agreed upon area with no oversight by the concession owner. Instead these ASM will be under the control of the Colombian mining and environmental authorities.

GCM has since changed course with respect to the open pit concept, and has determined that it is preferential to develop the concession as an underground operationwithout impacting the town.

20.5 Mine Closure and Reclamation

Article 209 of Law 685 of 2001 requires that the concession holder, upon termination of the agreement, shall undertake the necessary environmental measures for the proper reclamation and closure of the operation. To ensure that these activities are carried out, the Environmental Insurance Policy shall remain in effect for three years from the date of termination of the contract. Little else regarding the specifics of mine closure is provided in the Law. Decree 2820 Article 40 Paragraph 2 of 2010 specifically indicates that the concession holder must submit a plan for dismantling and abandonment of the Project.

While a formal closure plan is not legally required at this stage of the operation, currently there is a closure plan for Marmato (May 2019) which discusses basic reclamation and closure actions including aspects of temporary, progressive, and final closure. More detailed, site-wide closure actions have not yet been defined, as these will be developed through five-year updates to help identify potential closure risks that GCM may need to manage and finalized closer to the end of operations. The below discussion focuses on final closure and post-closure.

Some surface facilities (e.g., tailings disposal facility) will be progressively reclaimed over the duration of the mine site operations, albeit on a limited basis, as there are relatively few surface facilities suitable for concurrent reclamation and closure. In addition, progressive reclamation and closure can result in the development of expertise on the most appropriate reclamation methods. Progressive reclamation and closure will be undertaken, however, without posing impediments on day-to-day operations of the site. Final closure of the mine site will be undertaken following completion of all mining operations.

Final closure of the facility will entail the following activities, if not undertaken during progressive closure phases:

- Reclamation of tailings dam:
  - The dam will be covered with growth media and revegetated.
  - Concrete structures will be properly decommissioned.
  - Metal fences will be removed.
- Underground workings:
  - All equipment with resale value will be removed and salvaged.
  - All portals, ventilation shafts, etc., will be sealed to exclude public access.
- Plant and other buildings:
  - Plant equipment will be decommissioned and removed for transportation to, and storage in, Medellín.
  - Buildings will be demolished.
- Erosion control measures will be taken where there is evidence of erosion.
- Human resources: mine workers' contracts will not be renewed (no extra costs to be included in the reclamation and closure cost estimate) or the contracts will be terminated (which would incur additional costs).

The May 2019 closure plan discusses post-closure activities which include monitoring for physical and chemical stability. Physical stability monitoring will include monitoring for ground movements which would indicate subsidence. The plan currently assumes monitoring of physical stability twice annually for three years and then annually for three years if no movement is detected. Chemical stability will include monitoring of water quality of mine effluent as well as tailings draindown. Monitoring of water quality will continue twice a year for the first three years and then once annually for at least three years afterwards until such time that permissible limits are met, or flows diminish (in the case of draindown from tailings).

20.5.1 Reclamation and Closure Costs

Reclamation and closure costs for the current operation are provided in the May 2019 reclamation and closure plan. These costs are based on percentages of costs to build the facilities. The plan does not provide the basis for the percentages. The reclamation and closure cost estimate provided totals 20,128,000,000 pesos (US$5.8 million based on exchange rate of 3,455 to 1). A requirement for long-term post-closure water treatment, if any, would significantly increase this estimate.

Given the conceptual design nature of the deep zone expansion of the Marmato Project, a detailed closure cost assessment cannot be made at this time. However, it is not unreasonable to assume that the closure cost would be 1.5 to 2.0 times the current estimate, given the increase in production anticipated for the new operations and the construction of a new plant and tailings disposal area(s). A cost of US$6.1 million was included in the technical economic model to account for a second tailings disposal facility. These costs will need to be revisited more accurately during the PFS phase of project development.

21 Capital and Operating Costs

SRK visited Marmato's mine site and office various times in 2019, during these visits SRK reviewed budget estimates and site-specific cost data, this review included data regarding both capital and operating costs. These reviewed budgets and site-specific cost data are the base support for the capital and operating cost models prepared for this PEA.

The cost models prepared for Marmato Upper Zone operation are mostly based on the reviewed budgets, as this will be a continuation of the current operation. MDZ cost estimates are based on cost models prepared by SRK, which are based on SRK's experience with similar operations and site-specific data provided by Marmato's staff.

The mine is currently owner operated and the projections prepared for this PEA assume that it will be maintained. Common prices for consumables, labor, fuel, lubricants and explosives were used by all engineering disciplines to derive capital and operating costs. Included in the labor costs are shift differentials, vacation rotations, all taxes and the payroll burdens.

21.1 Capital Cost Estimates

21.1.1 Marmato Upper Zone

Marmato Upper Zone is a currently operating underground mine, the estimate of capital includes only sustaining capital to maintain the equipment and all supporting infrastructure necessary to continue operations until the end of the projected production schedule. The estimate of capital is divided into the following main areas:

- Drilling;
- Development;
- Mine Equipment;
- Other Mine Infrastructure;
- Surface Facilities; and
- Plant.

The sustaining capital cost estimates developed for this mining area includes the costs associated with the engineering, procurement, construction and commissioning. The cost estimate is based on budgetary estimates prepared by Marmato and reviewed by SRK. The estimate indicates that the Project requires a sustaining capital of US$40.5 million to support the projected production schedule throughout the LoM. Table 21-1 summarizes the LoM sustaining capital estimate and Table 21-2 and Table 21-3 present the same estimate by year.

Table 21-1: Marmato Upper Zone Sustaining Capital (LoM)

Description	LoM ($000's)
Drilling	1,450
Development	19,501
Mine Equipment	11,545
Other Mine	2,690
Surface	4,330
Plant	1,010
Total	**40,526**

Source: Gran Colombia/SRK, 2019

Table 21-2: Marmato Upper Zone Sustaining Capital (2019 to 2026)($000's)

Description	2019	2020	2021	2022	2023	2024	2025	2026
Drilling	100	100	100	100	100	100	100	100
Development	1,186	2,014	1,374	1,545	1,395	1,268	1,873	1,623
Mine Equipment	920	2,155	1,640	725	580	515	450	645
Other Mine	280	550	720	250	330	300	80	-
Surface	220	480	300	330	250	250	350	250
Plant	50	50	50	50	50	50	50	260
Total	**2,756**	**5,349**	**4,184**	**3,000**	**2,705**	**2,483**	**2,903**	**2,878**

Source: Gran Colombia/SRK, 2019

Table 21-3: Marmato Upper Zone Sustaining Capital (2027 to 2034)($000's)

Description	2027	2028	2029	2030	2031	2032	2033	2034
Drilling	100	100	100	100	100	100	50	-
Development	816	1,159	2,096	1,284	966	315	289	297
Mine Equipment	410	540	470	550	565	590	390	400
Other Mine	130	-	-	50	-	-	-	-
Surface	350	250	250	150	250	250	150	250
Plant	50	50	50	50	50	50	50	50
Total	**1,856**	**2,099**	**2,966**	**2,184**	**1,931**	**1,305**	**929**	**997**

Source: Gran Colombia/SRK, 2019

Most of this sustaining capital estimate is supported by a budget forecast prepared by Marmato and reviewed by SRK, the following items are covered by this budget:

- A yearly infill drilling expenditure of US$100,000;
- Mine equipment maintenance and replacement schedule;
- Other mine sustaining capital includes improvements and maintenance of existing mine infrastructure and stationary equipment;
- Surface sustaining capital includes improvements and maintenance of infrastructure located at surface, such as the camp, laboratory, filter presses, detox system, etc.; and
- Plant sustaining capital is an estimate of a yearly maintenance cost.

Development costs are derived from the mining schedule prepared by SRK. The prepared mining schedule includes meters of development in waste, this schedule of meters was combined with unit costs, based on site specific data, to estimate the cost of this development operation. This cost was then capitalized into the sustaining capital.

Table 21-4 presents the assumed unit costs for this development, while Table 21-5 and Table 21-6 present the yearly schedule of capital development meters.

Table 21-4: Marmato Upper Zone Capital Development Unit Costs

Description	US$/m
Veins Dev in Waste (2.2 by 2.2)	1,050
Ramp (3.5 by 3.5)	1,200
Apique (Widening)	2,300

Source: Gran Colombia/SRK, 2019

Table 21-5: Marmato Upper Zone Capital Development Meters (2019 to 2026)($000's)

Description	2019	2020	2021	2022	2023	2024	2025	2026
Veins Dev in Waste	418	1,139	1,309	1,472	1,329	1,207	1,784	1,546
Ramp Meters	459	367	-	-	-	-	-	-
Apique	86	164	-	-	-	-	-	-

Source: Gran Colombia/SRK, 2019

Table 21-6: Marmato Upper Zone Capital Development Meters (2027 to 2034)($000's)

Description	2027	2028	2029	2030	2031	2032	2033	2034
Veins Dev in Waste	777	1,104	1,996	1,223	920	300	275	282
Ramp Meters	-	-	-	-	-	-	-	-
Apique	-	-	-	-	-	-	-	-

Source: Gran Colombia/SRK, 2019

21.1.2 Marmato Deeps Zone

The MDZ is a lower part of the deposit that is undeveloped. Before Marmato can exploit this part of the deposit it will have to expand the existing operation. The expansion is planned to be executed between the years of 2021 and 2022 and the following areas will require an investment of capital for the following areas:

- Exploration Drilling;
- Mine Development;
- Mining Equipment;
- Surface Facilities and Equipment;
- Underground Facilities and Equipment;
- Power Supply;
- Access Road;
- Camp;
- Other Supporting Infrastructure;
- Mineral Processing Plant; and
- Tailings Storage Facility.

The capital cost estimates prepared for the expansion into this mining area also include estimates for Engineering, Procurement and Construction Management (EPCM) and the owner's cost to manage it. The cost estimate is based on cost models prepared by SRK with site specific inputs from Marmato. The estimate indicates that the expansion will require an investment of US$268.9 million, this includes an estimated capital of US$215.1 million plus 25% contingency of US$53.8 million. Table 21-7 summarizes the expansion capital estimate.

Table 21-7: Marmato Deeps Zone Construction Capital ($000's)

Description	Total	2021	2022
Exploration Drilling	2,600	1,300	1,300
Development	52,021	14,790	37,232
Mining Equipment	26,956	923	26,033
Surface Facilities and Equipment	12,114	1,264	10,850
UG Facilities and Equipment	499	-	499
Power Supply	675	169	506
Access Road	225	169	56
Camp	750	188	563
Pump Station	200	-	200
Process Plant	65,551	26,221	39,331
Tailings Storage Facility	35,953	-	35,953
EPCM	7,562	268	7,294
Owners	10,000	5,000	5,000
Sub-Total	**215,107**	**50,291**	**164,816**
Contingency (25%)	53,777	12,573	41,204
Total	**268,884**	**62,864**	**206,020**

Source: Gran Colombia/SRK, 2019

The MDZ construction capital is supported by a mix of budgetary estimates and cost models developed for the installation of the expansion. Budget estimates were prepared by Marmato and reviewed by SRK and cost models were prepared by SRK. The following items are covered by this budget:

- Exploration drilling to improve resource estimate as a yearly expenditure of US$1.3 million;
- Schedule of mine equipment purchases prepared by SRK;
- Cost model estimate from SRK to install surface facilities like portal, ventilation system, power generation and distribution, ancillary building, etc.;
- Cost model estimate from SRK to install underground facilities like shops, ventilation systems, refuge chambers, pumping systems, paste distribution, fuel distribution, ancillary equipment, etc.;
- Cost model from SRK to install other infrastructure including power supply, access road, camp and pumping station;
- Cost model estimate from SRK to install mineral processing plant, which already includes EPCM costs;
- A scoping study prepared by SRK build a tailings storage facility that estimated the initial cost around US$36 million;
- Additional EPCM costs estimated as 15% of surface and underground facilities, power supply, access road, camp, pumping station, tailings and tailings storage facility; and
- An estimate of US$10 million for owner's team to supervise the expansion.

Development costs are derived from the mining schedule prepared by SRK. The prepared mining schedule includes meters of development during pre-production, this schedule of meters was combined with unit costs, based on site specific data, to estimate the cost of this development operation. The pre-production development costs assume that this operation will be performed by a contractor. Table 21-8 presents the assumed development unit costs and Table 21-9 presents the scheduled meters for the pre-production period.

Table 21-8: Marmato Deeps Zone Pre-Production Development Unit Costs (Contractor)($000's)

Description	US$/m
Main Ramp (5.5 by 5.5 arched back)	5,500
Footwall Access (5 by 5)	5,000
Stope Drift in Ore(4.5 x 4.5)	5,000
Stope Drift in Waste (4.5 x 4.5)	5,000
Ventilation Drift (4.5 x 4.5)	5,000
Raisebore (5 m dia)	6,500

Source: Gran Colombia/SRK, 2019

Table 21-9: Marmato Deeps Zone Pre-Production Development Meters

Description	2021	2022
Main Ramp (5.5 by 5.5 arched back)	1,679	1,968
Footwall Access (5 by 5)	-	1,271
Stope Drift in Ore (4.5 by 4.5)	-	944
Stope Drift in Waste (4.5 by 4.5)	-	335
Ventilation Drift (4.5 by 4.5)	1,111	1,594
Raisebore (5 m dia)	-	875

Source: Gran Colombia/SRK, 2019

Additionally, the MDZ will require sustaining capital to maintain the equipment and all supporting infrastructure necessary to continue operations until the end of its projected production schedule. The estimate of sustaining capital is divided into the following main areas:

- Drilling;
- Development;
- Mine Equipment;
- UG Facilities and Equipment
- Surface Facilities;
- Plant;
- Tailings;
- Other Sustaining; and
- Project Closure.

The sustaining capital cost estimate developed for this mining area includes the costs associated with the engineering, procurement, construction and commissioning. The cost estimate is based on cost models prepared by SRK with site specific inputs from Marmato. The estimate indicate that the Project requires a sustaining capital of US$140.7 million to support the projected production schedule through the LoM. Table 21-10 summarizes the LoM sustaining capital estimate and Table 21-11 and Table 21-12 present the same estimate by year.

Table 21-10: Marmato Deeps Zone Sustaining Capital (LoM)

Description	LoM ($000's)
Drilling	14,500
Development	55,848
Mine Equipment	22,245
UG Facilities and Equipment	8,150
Surface	280
Plant	7,000
Tailings	22,804
Other Sustaining	3,812
Project Closure	6,100
Total	**140,739**

Source: Gran Colombia/SRK, 2019

Table 21-11: Marmato Deeps Zone Sustaining Capital (2023 to 2030)($000's)

Description	2023	2024	2025	2026	2027	2028	2029	2030
Drilling	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000
Development	9,701	5,878	829	877	8,385	10,434	6,486	5,866
Mine Equipment	5,727	800	3,564	3,536	935	2,318	2,352	1,004
UG Facilities and Equipment	4,603	1,440	463	375	200	75	920	75
Surface	280	-	-	-	-	-	-	-
Plant	-	500	500	500	500	500	500	500
Tailings	842	842	842	871	842	842	7,608	5,730
Other Sustaining	219	237	242	247	250	251	262	263
Project Closure	-	-	-	-	-	-	-	-
Total	**22,371**	**10,696**	**7,440**	**7,406**	**12,112**	**15,419**	**19,128**	**14,438**

Source: Gran Colombia/SRK, 2019

Table 21-12: Marmato Deeps Zone Sustaining Capital (2031 to 2038)($000's)

Description	2031	2032	2033	2034	2035	2036	2037	2038
Drilling	1,000	1,000	1,000	1,000	1,000	1,000	500	-
Development	6,010	1,029	353	-	-	-	-	-
Mine Equipment	1,004	1,004	-	-	-	-	-	-
UG Facilities and Equipment	-	-	-	-	-	-	-	-
Surface	-	-	-	-	-	-	-	-
Plant	500	500	500	500	500	500	500	-
Tailings	1,245	871	842	842	585	-	-	-
Other Sustaining	263	263	263	263	263	263	263	-
Project Closure	-	-	-	-	-	-	-	6,100
Total	**10,023**	**4,668**	**2,958**	**2,605**	**2,348**	**1,763**	**1,263**	**6,100**

Source: Gran Colombia/SRK, 2019

The assumptions supporting this estimate of sustaining capital are mostly the same as described for the MDZ construction capital, except for the development costs and closure costs.

In the case of development sustaining capital, these costs are derived from the mining schedule prepared by SRK. The prepared mining schedule includes meters of development in waste, this schedule of meters was combined with unit costs, based on site specific data, to estimate the cost of this development operation. The development in waste costs assume that this operation will be performed by mix of contractors and the owner. Table 21-13 presents the assumed development unit costs and Table 21-14 and Table 21-15 present the scheduled meters for the sustaining period.

Table 21-13: Marmato Deeps Zone Development Sustaining Capital Unit Costs

Description	US$/m
Raisebore (5 m dia) - Contractor	6,500
Ventilation Slot Raise (4x4) - Owner	3,500
Main Ramp (5.5 by 5.5 arched back) - Owner	3,800
Footwall Access (5 by 5) - Owner	3,300
Ventilation Drift (4.5 by 4.5) - Owner	2,250

Source: Gran Colombia/SRK, 2019

Table 21-14: Marmato Deeps Zone Development Sustaining Capital Meters (2023 to 2030)($000's)

Description	2023	2024	2025	2026	2027	2028	2029	2030
Raisebore (5 m dia)	-	-	-	-	135	350	97	-
Ventilation Slot Raise (4 by 4)	-	-	-	-	-	-	-	77
Main Ramp (5.5 by 5.5 arched back)	948	280	-	-	1,556	1,368	365	266
Footwall Access (5 by 5)	1,683	1,272	241	128	439	863	1,113	1,300
Ventilation Drift (4.5 by 4.5)	2,264	931	905	1,998	1,672	2,119	1,675	1,249

Source: Gran Colombia/SRK, 2019

Table 21-15: Marmato Deeps Zone Development Sustaining Capital Meters (2031 to 2037)($000's)

Description	2031	2032	2033	2034	2035	2036	2037
Raisebore (5 m dia)	-	-	-	-	-	-	-
Ventilation Slot Raise (4 by 4)	-	25	-	-	-	-	-
Main Ramp (5.5 by 5.5 arched back)	241	-	-	-	-	-	-
Footwall Access (5 by 5)	1,374	299	107	-	-	-	-
Ventilation Drift (4.5 by 4.5)	960	1,181	1,542	1,413	1,012	1,387	40

Source: Gran Colombia/SRK, 2019

Project closure costs are based on a budget estimate prepared by Marmato and reviewed by SRK.

21.2 Operating Cost Estimates

SRK and Marmato prepared the estimate of operating costs for the PEA's production schedule. These costs were subdivided into the following categories:

- Mining Operating Expenditure;
- Processing Operating Expenditure; and
- Site G&A Operating Expenditure.

Marmato Upper Zone LoM cost estimate is presented in Table 21-16 and MDZ LoM cost estimate is presented in Table 21-17

Table 21-16: Marmato Upper Zone Operating Costs Summary

Description	LoM (US$/t-Ore)	LoM (US$000's)
Mining	46.71	258,918
Process	15.88	88,003
G&A	11.85	65,667
Total Operating	**74.44**	**412,587**

Source: Gran Colombia/SRK, 2019

Table 21-17: Marmato Deeps Zone Operating Costs Summary

Description	LoM (US$/t-Ore)	LoM (US$000's)
Mining	32.43	675,150
Process	16.14	336,005
G&A	3.46	72,000
Total Operating	**52.02**	**1,083,155**

Source: Gran Colombia/SRK, 2019

21.2.1 Basis for Operating Cost Estimates

The prepared estimates that compose the operating costs consist of domestic and international services, equipment, labor, etc. Where required, the following were included:

- Value added tax;
- Freight; and
- Duty.

It was assumed that the mill operates 350 days per year under a daily schedule of two shifts of 12 hours.

The operating cost estimates are based on the quantities associated with the production schedule, including the following:

- Development meters;
- Stope ore tonnage; and
- Ore tonnage.

All operating costs include supervision staff, operations labor, maintenance labor, consumables, electricity, fuels, lubricants, maintenance parts and any other operating expenditure identified by contributing engineers.

Site-specific 2018 budget estimates were used to estimate the LoM operating costs of Marmato Upper Zone. The following costs were used to estimate the operating cost of this mining area:

- Vein mining: US$46.25/t-stope, this includes backfill costs;
- Level 21 mining: US$42.00/t-stope, this includes backfill costs;
- Mineral Processing: US$13.08/t-ore;
- Tailings Deposition: US$2.80/t-ore; and
- G&A: US$4,000,000/year.

Additionally, development operating costs are derived from the mining schedule prepared by SRK. The prepared mining schedule includes meters of development in ore, this schedule of meters was combined with unit costs, based on site specific data, to estimate the cost of this development operation. The development in ore costs assume that this operation will be performed by the owner. Table 21-18 presents the assumed development unit cost, while Table 21-19 and Table 21-20 present the scheduled meters for the operating years.

Table 21-18: Marmato Upper Zone Operating Development Unit Costs

Description	US$/m
Veins Dev in Ore (2.2 by 2.2)	1,050
Dev Access (3.5 by 3.5)	1,200

Source: Gran Colombia/SRK, 2019

Table 21-19: Marmato Upper Zone Operating Development Meters (2019 to 2027)

Description	2019	2020	2021	2022	2023	2024	2025	2026	2027
Veins Dev in Ore	249	525	444	513	483	477	485	522	503
Dev Access	260	687	494	-	-	-	-	-	-

Source: Gran Colombia/SRK, 2019

Table 21-20: Marmato Upper Zone Operating Development Meters (2028 – 2035)

Description	2028	2029	2030	2031	2032	2033	2034	2035
Veins Dev in Ore	685	675	506	511	360	343	238	28
Dev Access	-	-	-	-	-	-	-	-

Source: Gran Colombia/SRK, 2019

Cost models prepared by SRK based on site-specific inputs from Marmato were used to estimate the LoM operating costs of MDZ. The following costs were used to estimate the operating cost of this mining area:

- Stope mining: US$29.23/t-stope, this includes backfill costs;
- Mineral Processing: US$13.34/t-ore;
- Tailings Deposition: US$2.80/t-ore; and
- G&A: US$4,500,000/year.

Additionally, development operating costs are derived from the mining schedule prepared by SRK. The prepared mining schedule includes meters of development in ore, this schedule of meters was combined with unit costs, based on site specific data, to estimate the cost of this development operation. The development in ore costs assume that this operation will be performed by the owner. Table 21-21 presents the assumed development unit cost, while Table 21-22 and Table 21-23 and present the scheduled meters for the operating years.

Table 21-21: Marmato Deeps Zone Operating Development Unit Costs

Description	US$/m
Stope Drift in Ore (4.5 by 4.5)	2,250
Stope Drift in Waste (4.5 by 4.5)	2,250

Source: Gran Colombia/SRK, 2019

Table 21-22: Marmato Deeps Zone Operating Development Meters (2023 to 2030)

Description	2023	2024	2025	2026	2027	2028	2029	2030
Stope Drift in Ore (4.5 by 4.5)	4,828	1,582	1,112	2,161	2,101	2,393	2,711	2,775
Stope Drift in Waste (4.5 by 4.5)	2,264	931	905	1,998	1,672	2,119	1,675	1,249

Source: Gran Colombia/SRK, 2019

Table 21-23: Marmato Upper Zone Operating Development Meters (2031 to 2037)

Description	2031	2032	2033	2034	2035	2036	2037
Stope Drift in Ore (4.5 by 4.5)	2,970	2,258	2,222	2,118	2,248	2,703	82
Stope Drift in Waste (4.5 by 4.5)	960	1,181	1,542	1,413	1,012	1,387	40

Source: Gran Colombia/SRK, 2019

22 Economic Analysis

The financial results presented here are based on monthly inputs from the production schedule prepared by SRK. All financial data is first quarter 2019 and currency is in U.S. dollars (US$), unless otherwise stated.

22.1 External Factors

Marmato currently has a long-term supply agreement for the sale of its products to an international refinery who take delivery of dore from the mine at designated transfer points in Colombia. The refinery is responsible for shipping the products abroad.. The refining costs and discounts associated with the sales of the products are based on this agreement. This study was prepared under the assumption that the Project will sell doré containing gold and silver.

Treatment charges and net smelter return (NSR) terms are summarized in Table 22-1.

Table 22-1: Marmato Net Smelter Return Terms

Description	Value	Units
Doré		
Payable Gold	100%	
Doré Smelting & Refining Charges	6.38	US$/oz-Au

Source: Gran Colombia, 2019

22.2 Production Assumptions

Marmato's operation is supported by the production of doré bars containing gold and silver. The doré bars result from the mineral processing of RoM containing gold and silver from an underground mining operation. Table 22-2 presents this PEA's production summary.

Table 22-2: Marmato Production Summary

Description	Unit	Value
Total Ore Mined	tonnes	26,363,978
Total Waste Mined	tonnes	303,387
Total Material Mined	tonnes	26,667,365
Average Mined Grade	g/t Au	2.78
Average Mined Grade	g/t Ag	5.91
Contained Gold Mined	ounces	2,355,994
Contained Silver Mined	ounces	5,006,222
Gold Recovered	ounces	2,180,395
Silver Recovered	ounces	1,816,020

Source: SRK, 2019

The Marmato operation was broken down into two major mining areas, namely the Marmato Upper Zone, which is represented by the current operation that is composed by a mining area at the upper portion of the deposit and an existing mineral processing plant, and the MDZ. No pre-production period has been considered for the Marmato Upper Zone mine schedule, as this part of the operation is currently producing.

The MDZ is a lower portion of the deposit that will support an expansion of the Marmato operation. This will require expansion of the mining operation and the installation of a new mineral processing circuit to extract doré bars from this material. The mine schedule for MDZ assumes a pre-production period of four years, considering 2019, which consists of a delay period of two years followed by a construction period of another two years.

The Marmato Upper Zone mine production is based on a LoM assumed average mining rate of 1,020 tpd (350 days/year basis), including movement of both ore and waste. The MDZ mine production is based on a LoM assumed average mining rate of 4,100 tpd (350 days/year basis), including movement of both ore and waste. The combined total mine movement is limited to a maximum of 2 Mtpy.

The mine schedule does not include any major stockpiling, as most of the blending of RoM is done in the mine. The schedule does include small quantities of stockpiling, the maximum size of this stock is 51,620 t. Table 22-3 to Table 22-5 present the yearly LoM mine production assumptions by area.

Table 22-3: Marmato Yearly (2019 to 2026) Mine Production Assumptions

	2019	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**
Marmato Upper Zone								
Ore Tonnes (t)	109,383	349,997	402,491	350,004	350,079	350,055	350,078	350,005
Au Head Grade (g/t)	3.95	3.77	3.53	3.64	3.63	3.64	3.75	3.92
Ag Head Grade (g/t)	14.32	12.16	10.95	14.41	15.28	15.06	14.81	15.29
Au Contained Gold (oz)	13,878	42,459	45,642	40,913	40,883	41,013	42,164	44,114
Ag Contained Gold (oz)	50,363	136,852	141,679	162,136	172,004	169,480	166,735	172,064
Waste (t)	28,231	42,600	29,587	21,012	18,972	17,240	25,470	22,074
Total (t)	137,614	392,597	432,078	371,016	369,051	367,295	375,548	372,079
Marmato Deeps Zone								
Ore Tonnes (t)	-	-	-	-	830,693	1,400,364	1,400,872	1,400,216
Au Head Grade (g/t)	-	-	-	-	3.21	3.08	3.48	3.17
Ag Head Grade (g/t)	-	-	-	-	3.79	3.73	4.09	4.65
Au Contained Gold (oz)	-	-	-	-	85,828	138,498	156,950	142,538
Ag Contained Gold (oz)	-	-	-	-	101,291	167,975	184,183	209,262
Waste (t)	-	-	-	-	317,934	171,585	66,588	123,795
Total (t)	-	-	-	-	1,148,627	1,571,949	1,467,460	1,524,011
Total								
Ore Tonnes (t)	109,383	349,997	402,491	350,004	1,180,772	1,750,419	1,750,950	1,750,221
Au Head Grade (g/t)	3.95	3.77	3.53	3.64	3.34	3.19	3.54	3.32
Ag Head Grade (g/t)	14.32	12.16	10.95	14.41	7.20	6.00	6.23	6.78
Au Contained Gold (oz)	13,878	42,459	45,642	40,913	126,711	179,511	199,114	186,652
Ag Contained Gold (oz)	50,363	136,852	141,679	162,136	273,295	337,455	350,919	381,325
Waste (t)	28,231	42,600	29,587	21,012	336,906	188,825	92,058	145,869
Total (t)	**137,614**	**392,597**	**432,078**	**371,016**	**1,517,678**	**1,939,244**	**1,843,008**	**1,896,090**

Source: SRK, 2019

Table 22-4: Marmato Yearly (2027 to 2034) Mine Production Assumptions

	2027	2028	2029	2030	2031	2032	2033	2034
Marmato Upper Zone								
Ore Tonnes (t)	350,065	350,080	350,031	350,048	350,030	350,049	350,061	350,046
Au Head Grade (g/t)	4.08	4.08	3.89	3.58	3.82	4.04	4.03	3.94
Ag Head Grade (g/t)	15.57	16.42	17.68	16.32	17.03	18.08	17.29	16.14
Au Contained Gold (oz)	45,928	45,955	43,750	40,324	43,006	45,413	45,364	44,396
Ag Contained Gold (oz)	175,209	184,807	198,972	183,703	191,698	203,469	194,565	181,683
Waste (t)	11,089	15,767	28,500	17,467	13,130	4,283	3,933	4,032
Total (t)	361,154	365,847	378,531	367,515	363,160	354,332	353,994	354,078
Marmato Deeps Zone								
Ore Tonnes (t)	1,396,679	1,342,363	1,400,182	1,399,950	1,400,341	1,400,202	1,400,406	1,400,358
Au Head Grade (g/t)	2.59	2.29	2.26	2.16	2.04	2.09	2.18	2.18
Ag Head Grade (g/t)	3.57	4.20	5.19	4.67	2.49	1.64	1.79	2.03
Au Contained Gold (oz)	116,430	99,000	101,855	97,389	92,041	93,901	98,054	98,302
Ag Contained Gold (oz)	160,229	181,333	233,685	210,353	112,222	73,841	80,550	91,196
Waste (t)	241,877	291,664	213,666	188,674	174,910	85,794	91,527	77,264
Total (t)	1,638,556	1,634,027	1,613,848	1,588,624	1,575,251	1,485,996	1,491,933	1,477,622
Total								
Ore Tonnes (t)	1,746,744	1,692,443	1,750,213	1,749,998	1,750,371	1,750,251	1,750,467	1,750,404
Au Head Grade (g/t)	2.89	2.66	2.59	2.45	2.40	2.48	2.55	2.54
Ag Head Grade (g/t)	5.97	6.73	7.69	7.00	5.40	4.93	4.89	4.85
Au Contained Gold (oz)	162,359	144,955	145,605	137,713	135,046	139,314	143,418	142,698
Ag Contained Gold (oz)	335,437	366,140	432,657	394,056	303,920	277,310	275,115	272,878
Waste (t)	252,966	307,431	242,166	206,141	188,040	90,077	95,460	81,296
Total (t)	1,999,710	1,999,874	1,992,379	1,956,139	1,938,411	1,840,328	1,845,927	1,831,700

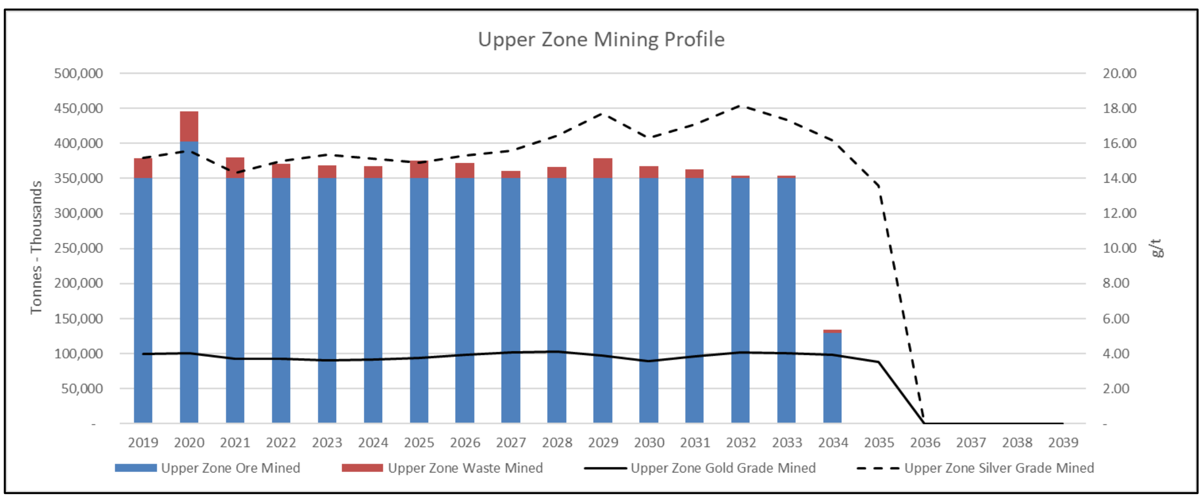
Source: SRK, 2019

Table 22-5: Marmato Yearly (2035-2038) Mine Production Assumptions

	2035	2036	2037	2038
Marmato Upper Zone				
Ore Tonnes (t)	130,095	-	-	-
Au Head Grade (g/t)	3.54	-	-	-
Ag Head Grade (g/t)	13.57	-	-	-
Au Contained Gold (oz)	14,794	-	-	-
Ag Contained Gold (oz)	56,771	-	-	-
Waste (t)	-	-	-	-
Total (t)	130,095	-	-	-
Marmato Deeps Zone				
Ore Tonnes (t)	1,399,974	1,400,328	1,204,825	592,005
Au Head Grade (g/t)	2.21	2.36	2.47	2.57
Ag Head Grade (g/t)	2.82	3.28	3.06	3.14
Au Contained Gold (oz)	99,600	106,133	95,652	48,916
Ag Contained Gold (oz)	126,821	147,481	118,584	59,765
Waste (t)	55,342	75,852	2,191	-
Total (t)	1,455,316	1,476,180	1,207,016	592,005
Total				
Ore Tonnes (t)	1,530,069	1,400,328	1,204,825	592,005
Au Head Grade (g/t)	2.33	2.36	2.47	2.57
Ag Head Grade (g/t)	3.73	3.28	3.06	3.14
Au Contained Gold (oz)	114,394	106,133	95,652	48,916
Ag Contained Gold (oz)	183,593	147,481	118,584	59,765
Waste (t)	55,342	75,852	2,191	-
Total (t)	**1,585,411**	**1,476,180**	**1,207,016**	**592,005**

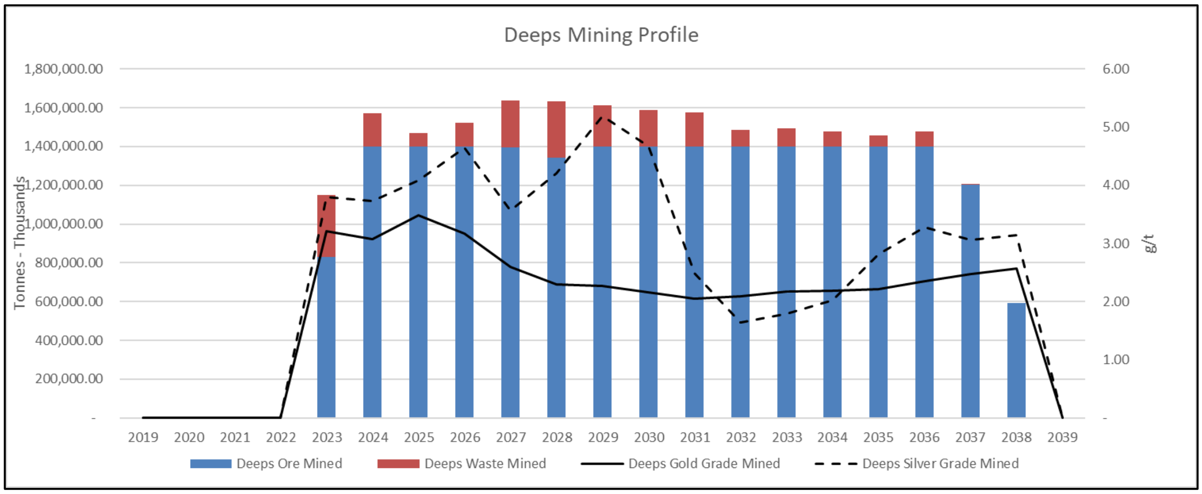
Source: SRK, 2019

Figure 22-1 and Figure 22-2 show each mine area's RoM ore production. Ore production and development waste are mostly stable and gold and silver grades are either stable or declining over the LoM.



Source: SRK, 2019

Figure 22-1: Marmato Upper Zone Mine Production Profile



Source: SRK, 2019

Figure 22-2: Marmato Deeps Zone Mine Production Profile

Marmato's Upper Zone operation is supported by an existing mineral processing facility, this plant has a capacity of 525,000 tpy (based on 1,500 tpd and an availability of 350 days/year). Table 22-6 presents the projected LoM plant production for Marmato Upper Zone.

Table 22-6: Marmato Upper Zone LoM Mill Production Assumptions

Description	Unit	Value
Processing Capacity	tpy	525,000
Ore Processed	tonnes	5,542,597
Average Feed Grade	g/t Au	3.82
Average Feed Grade	g/t Ag	15.39
Gold Recovery	%	87%
Silver Recovery	%	33%
Gold Recovered	ounces	588,196
Silver Recovered	ounces	910,407
Average Gold Per year	ounces/yr	34,600
Average Silver Per year	ounces/yr	53,553

Source: SRK, 2019

The MDZ will require the installation of an additional mineral processing facility, this plant will have a capacity of 1,400,000 tpy (based on 4,000 tpd and an availability of 350 days per year). Table 22-7 presents the projected LoM plant production for MDZ.

Table 22-7: Marmato Deeps Zone LoM Mill Production Assumptions

Description	Unit	Value
Processing Capacity	tpy	1,400,000
Ore Processed	tonnes	20,821,381
Average Feed Grade	g/t Au	2.50
Average Feed Grade	g/t Ag	3.38
Gold Recovery	%	95%
Silver Recovery	%	40%
Gold Recovered	ounces	1,592,199
Silver Recovered	ounces	905,613
Average Gold Per year	ounces/yr	99,512
Average Silver Per year	ounces/yr	56,601

Source: SRK, 2019

All figures presented above are based on the production derived from the resources disclosed in this report.

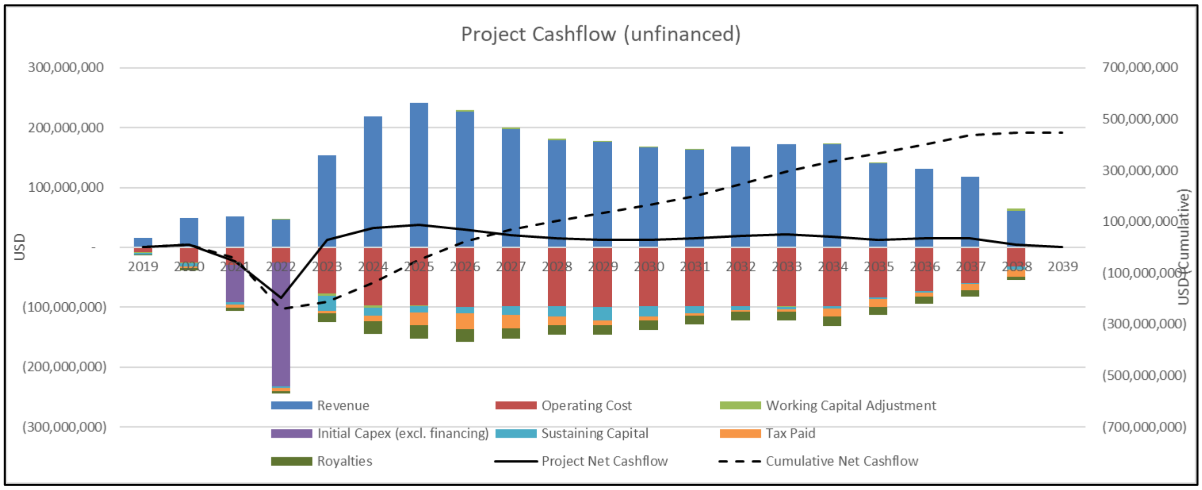
22.3 Taxes, Royalties and Other Interests

The analysis of the Marmato Project includes an effective corporate income tax rate of 30%. A depreciation schedule was calculated by SRK assuming a 10 year straight-line depreciation.

Royalties are also deducible from taxable income. The Project includes payment of a governmental production royalties on both gold and silver sales as outlined in Section 4.3. The total royalty due, excluding the inter-company royalty to Croesus, is calculated as 9.2% of gross metal sales deducted by the costs of transportation and metal refining.

22.4 Results

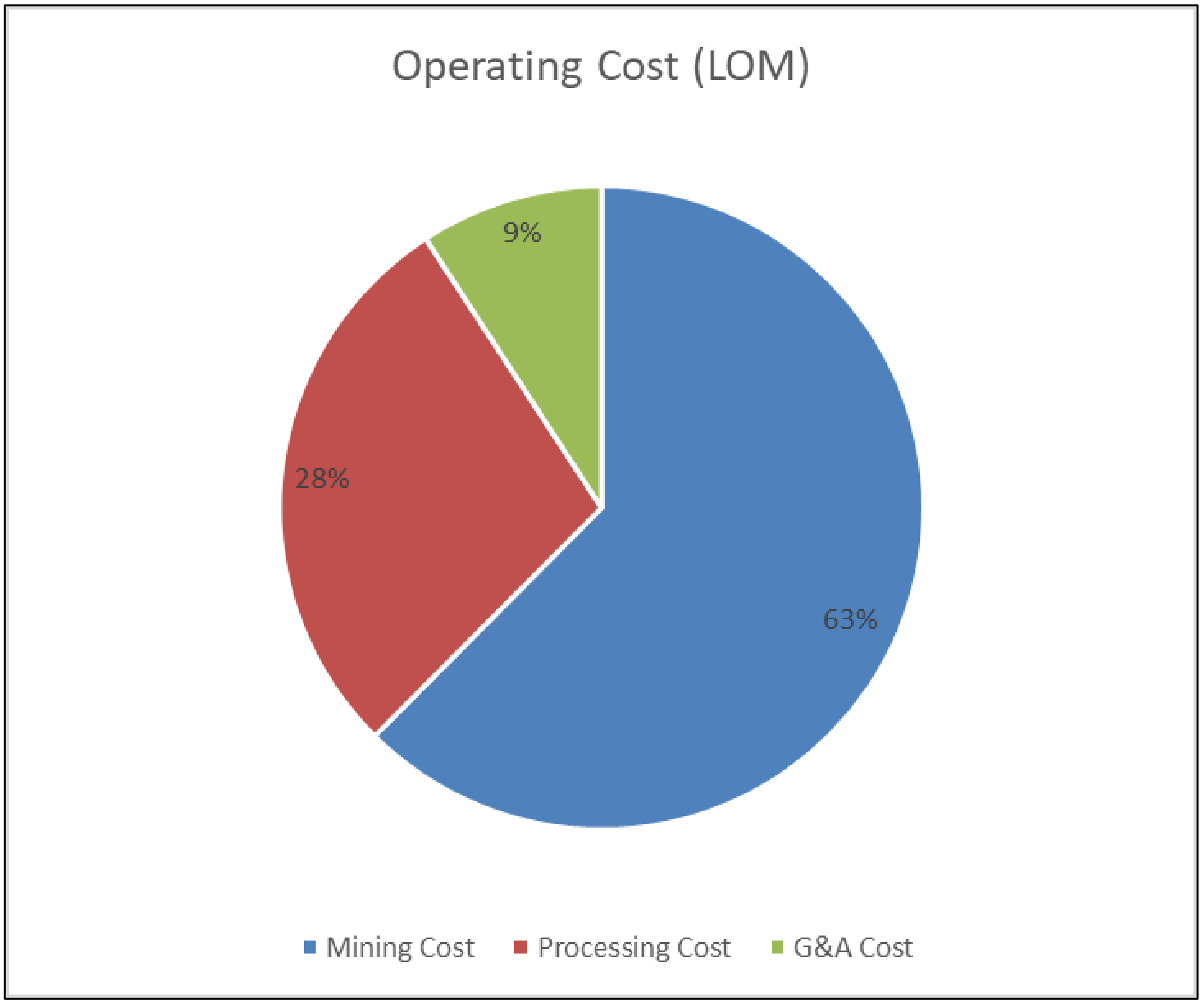
The valuation results of the Marmato Project indicate that the Project has an after-tax Net Present Value (NPV) of approximately US$206.8 million, based on a 5% discount rate and gold and silver prices of US$1,300/oz and US$17.00/oz respectively. The operation is projected to have negative cash flows during the years 2021 and 2022, when the MDZ is installed, with payback for the expansion expected by 2026. The annual free cash flow profile of the Project is presented in Figure 22-3. The full annual TEM is in Appendix D.



Source: SRK, 2019

Figure 22-3: Marmato After-Tax Free Cash Flow, Capital and Metal Production

Indicative economic results are presented in Table 22-8 . The Project is a gold operation with a sub-product of silver, where gold represents 99% of the total projected revenue and silver the remaining 1%. The underground mining cost is the heaviest burden on the operation representing 63% of the operating cost, as presented in Figure 22-4.



Source: SRK, 2019

Figure 22-4: Marmato Operating Cost Break-Down

Table 22-8: Marmato Indicative Economic Results

LoM Cash Flow		
Gold Price	USD/oz	1,300
Silver Price	USD/oz	17.00
Total Revenue	USD	2,851,475,063
Mining Cost	USD	(934,068,009)
Processing Cost	USD	(424,007,603)
G&A Cost	USD	(137,666,667)
Total Opex	USD	(1,495,742,278)
Operating Margin	USD	1,355,732,785
Operating Margin Ratio	%	48%
Taxes Paid	USD	(195,035,875)
Free Cashflow	USD	717,096,368
LoM Capital		
Expansion CAPEX	USD	(268,884,037)
Sustaining CAPEX	USD	(181,264,836)
Total LOM CAPEX	USD	(450,148,872)
Before Tax		
Free Cash Flow	USD	643,248,206
NPV @ 5%	USD	322,995,835
NPV @ 8%	USD	214,836,234
NPV @ 10%	USD	163,296,934
IRR	%	28%
After Tax		
Free Cash Flow	USD	448,212,331
NPV @ 5%	USD	206,821,360
NPV @ 8%	USD	126,595,938
NPV @ 10%	USD	88,893,940
IRR	%	20%
Payback	Year	2026

Source: SRK, 2019

Table 22-9 shows annual production and revenue forecasts for the life of the Project. All production forecasts, material grades, plant recoveries and other productivity measures were developed by SRK and GCM.

Table 22-9: Marmato LoM Annual Production and Revenues

Year	Ore Tonnes (t)	Au Head Grade (g/t)	Ag Head Grade (g/t)	Recovered Gold (oz)	Recovered Silver (oz)	Free Cash Flow (US$)
2019	109,383	3.95	14.32	12,004	16,721	2,280,989
2020	349,997	3.77	12.16	36,727	45,435	9,680,557
2021	402,491	3.53	10.95	39,480	47,037	8,747,239
2022	350,004	3.64	14.41	35,390	53,829	8,483,242
2023	1,190,079	3.33	7.15	117,433	97,682	28,640,806
2024	1,750,055	3.19	6.00	167,169	123,453	74,044,112
2025	1,750,078	3.53	6.23	184,969	128,793	89,142,295
2026	1,750,005	3.32	6.76	173,944	140,508	70,662,159
2027	1,750,065	2.90	5.99	151,357	122,995	47,078,629
2028	1,732,890	2.66	6.66	137,026	135,762	34,694,943
2029	1,750,031	2.59	7.69	134,593	159,521	30,519,081
2030	1,750,048	2.45	7.00	127,403	145,135	30,334,978
2031	1,750,030	2.40	5.40	124,618	108,525	34,163,408
2032	1,750,049	2.48	4.93	128,474	97,089	45,176,084
2033	1,750,061	2.55	4.89	132,362	96,805	49,971,149
2034	1,750,046	2.54	4.85	131,765	96,784	41,240,974
2035	1,530,095	2.33	3.73	107,417	69,563	29,297,203
2036	1,400,000	2.36	3.28	100,797	58,970	36,731,400
2037	1,206,566	2.47	3.06	90,995	47,507	35,767,475
2038	592,005	2.57	3.14	46,470	23,906	10,439,649
Total	**26,363,978**	**2.78**	**5.91**	**2,180,395**	**1,816,020**	**717,096,368**

Source: SRK, 2019

The estimated All-in Sustaining Costs (AISC), including sustaining capital, is US$882/Au-oz. Table 22-10 presents the breakdown of the Marmato cash costs.

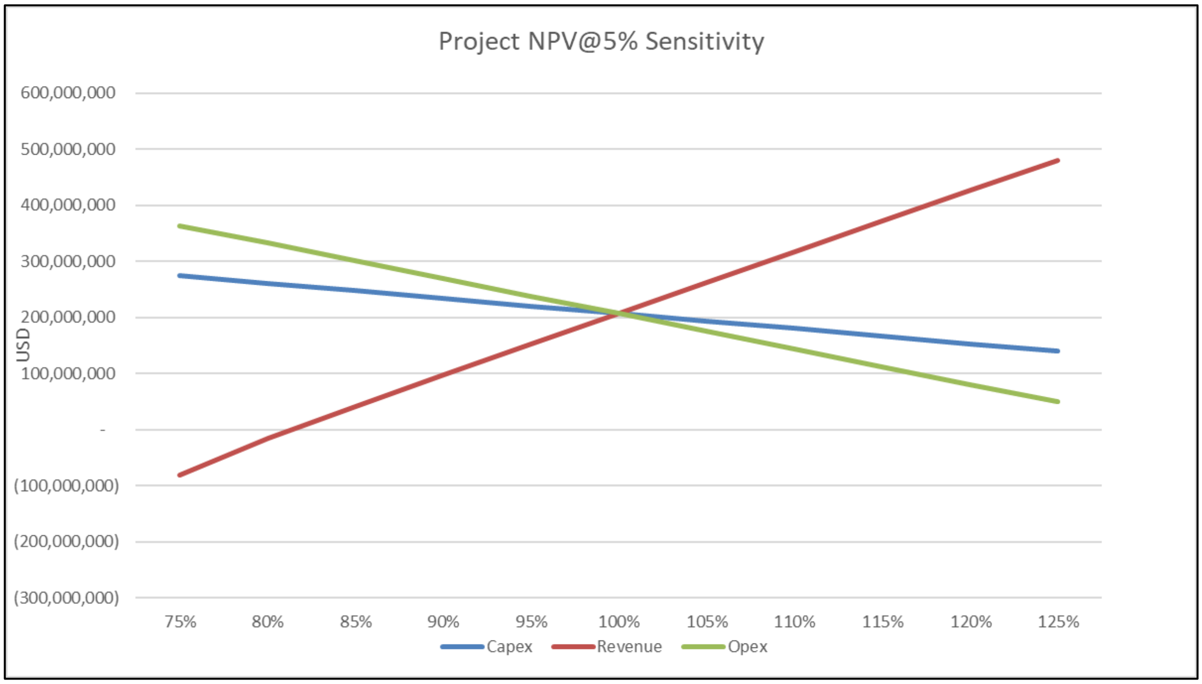
Table 22-10: LOM All-in Sustaining Cost Breakdown

Description	Unit	Value
Mining	USD/Au	428
Processing	USD/Au	194
G&A	USD/Au	63
Refining	USD/Au	6
Royalty	USD/Au	120
Sustaining Capital	USD/Au	83
Silver Credit	USD/Au	(14)
AISC	**USD/Au**	**882**

SRK's standard Cash Cost reporting methodology for NI 43-101 reports includes smelting/refining costs; whereas Gran Colombia's basis of reporting treats these costs as a reduction of realized gold price (the refinery discounts the selling price by a factor to cover these charges) and excludes them from its reported "total cash cost per ounce".
Source: SRK, 2019

22.5 Sensitivity Analysis

A sensitivity analysis on variation of Project costs, both capital and operating, and metal prices indicated that the cash generation is most sensitive to reduction in metal prices, or possibly loss on metal recovery, and secondly to an increase in operating costs.



Source: SRK, 2019

Figure 22-5: Marmato NPV Sensitivity

23 Adjacent Properties

There are no properties or other operating mines with NI 43-101 compliant resources adjacent to Marmato.

24 Other Relevant Data and Information

There is no additional information or explanation necessary to make the technical report understandable and not misleading.

25 Interpretation and Conclusions

25.1 Property Description and Ownership

The Company currently owns sufficient ground to operate and explore all the currently defined Mineral Resources. In the Zona Alta area within the Company -owned licenses a number of small scale operations exist..

SRK notes within the transfer of licenses from the previous owner, there is a gap between the existing licenses for Zona Baja and Echandia. This ground is under application from the Company with the Colombia government for formal approval to continue mining. SRK has reviewed the application within the government website and notes that the status is defined as “in progress”, which has been in place since September 30, 2009. The Company is taking steps to get the approval finalized.

Mining by GCM within this area has historically been conducted through the current operations and it has been reported to SRK that this area is under application for adjustment of the status for inclusion in the current mining operations of GCM. As the area represented by this gap contains approximately 8.7% of Measured & Indicated resources in Zona Baja and 1.9% of Inferred resources in Zona Baja, SRK recommends that GCM accelerate its effort to obtain government resolution on this gap as it could potentially result in loss of Mineral Resources (and future Reserves). Only approximately 2.3% of the total LOM gold production included in the proposed mine plan for Zona Baja contained herein is sourced from the area represented by this gap.

25.2 Exploration

SRK has been supplied with electronic databases covering the sampling at the Project, all of which have been validated by the Company. The databases comprise of a combination of historical and recent diamond core and underground channel samples. In total, there are some 1,317 diamond drillholes for a combined length of 266,390 m; plus 24,824 individual underground channel samples, inclusive of current mine sampling contained in the databases. Isolated historical channel samples in the upper portion of the mines have a degree of uncertainty on spatial location and quality as they have not been independently verified by SRK during site visits. SRK has excluded these samples from estimation of the porphyry units but has used them to guide the geological interpretation of the veins at higher elevations.

Historic underground channel samples are typically taken across the width of the vein, with limited sampling into the hanging-wall and footwall where possible. GCM geologists have made great efforts to position the sampling as accurately as possible, but SRK has noted some limitations still exist. These samples have been allocated separate geological codes in the modelling process as to not influence the geological model.

SRK is of the opinion that the exploration and assay data is sufficiently reliable to support evaluation and classification of Mineral Resources in accordance with generally accepted CIM Estimation of Mineral Resource and Mineral Reserve Best Practices Guidelines (2014).

25.3 Mineral Resource Estimate

The mineralization occurs in parallel, sheeted and anastomosing veins (vein domain), all of which follow a regional structural control, with minor veins forming splays of the main structures (splays) which often have limited strike or dip extent. The vein domain intersects broader zones of intense veinlet mineralization (termed “porphyry domain” for the purpose of this report) and is hosted by a lower grade mineralized porphyry. In addition, a discrete, relatively high-grade core, or feeder zone, to the main mineralization (MDZ), has been identified at depth by GCM geologists. The upper portion of the MDZ has been exposed in Level 21 of the existing mining operations, while deeper sections have been observed in drillcore, both of which have been confirmed as a separate style of mineralization. The main differences between the porphyry domain and the MDZ is that the veins and veinlets in the upper mine pyrite has replaced pyrrhotite. The lowest levels of the mine have currently intersected a combination of the porphyry domain which is where the gold is associated with veinlets with pyrite, and the MDZ where gold (Au) is associated with pyrrhotite. There is a small transition between the two domains, which is observed to some extent in the current mine workings but is not clearly defined from the current drilling. Underground mining at the Company operated mine remains focused on the vein structures located in the central portion (Zona Baja) of the Marmato deposit.

Stillitoe (2019) concluded the only geological parameter than can be used to constrain the grade model is veinlet intensity, although the presence of visible native gold also acts as a useful grade indicator. SRK has used this assumption as the basis for the mineralization model, by using a an indicator gold (1.7 g/t) grade to act as a proxy to higher grades of vein density (which have been logged consistently in older holes in the area).

SRK has produced block models using Datamine™ Studio RM Software (Datamine™). The procedure involved import from Leapfrog™Geo of wireframe models for the fault networks, veins, definition of resource domains (high-grade sub-domains), data conditioning (compositing and capping) for statistical analysis, geostatistical analysis, variography, block modelling and grade interpolation followed by validation. Grade estimation has for the veins has been based on block dimensions of 5 m by 5 m by 5 m for the Porphyry and MDZ units. Sub-blocking to 0.5 m by 1 m by 1 m has been allowed to reflect the narrow nature of the geological model. The block size reflects the relatively close-spaced underground channel sampling and spacing within veins compared to the wider drilling spacing, with the narrower block size used in the MDZ at depth to reflect the proposed geometry of the mineralization (steeply dipping feeder zone).

SRK is of the opinion that the Mineral Resource Estimate has been conducted in a manner consistent with industry best practices and that the data and information supporting the stated mineral resources is sufficient for declaration of Measured, Indicated and Inferred classifications of resources. SRK considers currently the veins (including splays) and the MDZ to be of sufficient confidence for use in a mining study, but recommends further work on the short scale variability within the porphyry be completed to confirm the current interpretation within areas of the existing mining infrastructure prior to use in any mining studies.

The exclusive structural control of the Lower Zone orebody implies that additional examples could exist elsewhere within the P1 stock and that they represent a priority exploration target, but further exploration will be required to test this theory and there is no gaurentee of exploration success.

25.4 Mining and Mineral Reserve Estimate

No Mineral Reserves have been estimated for the Project. The available data indicate that underground operations are viable for the Project. The upper zone is currently being mined using a conventional cut and fill stope method. The MDZ zone is porphyry material below 1,025 m elevation, and will be mined using a longhole stoping method. The MDZ zone is currently not developed.

Upper zone mining begins with a production of 700 tpd (245,000 tpy) from the vein stopes for the first two years, and then increases to 1,000 tpd (350,000 tpy) for the remaining 15 years of the mine life. The MDZ zone material will feed a new process facility with a capacity of 4,000 tpd (1.4 Mtpy). Higher grade material from the MDZ area is targeted early in the mine life.

25.5 Metallurgy and Processing

GCM currently operates a 1,200 tpd process plant (rated capacity) to recover gold and silver values from mineralization produced from current Marmato mining operations. The current Marmato process plant includes gravity concentration followed by gold flotation from the gravity tailing and cyanidation of the flotation concentrate.

- During the period from 2013 to 2019 overall gold recovery has ranged from 89.0 to 83.7% and has averaged about 86.5% over the past three years. Silver recovery has ranged from 41.1 to 31.5% and has averaged 33.2% over the past three years.
- Marmato process plant operating costs reported for 2019 (Jan-July) averaged US$13.08/t processed (exchange rate: 3,000 COP per US$).

MDZ material will be processed at the rate of 4,000 tpd in a new plant using a flowsheet that includes gravity concentration followed by cyanidation of the gravity tailing:

- The MDZ material is significantly harder than Marmato mineralization and will incur higher comminution costs;
- Gold recovery is estimated at 95% and silver recovery is estimated at 40%;
- Process plant operating costs are estimated at US$13.34/t; and
- Process plant capital cost is estimated at US$65.6 million at an accuracy of +/-50%.

25.6 Infrastructure

- **Existing Project Facilities and Support Infrastructure:** The existing infrastructure at the existing project is fully developed and functioning. There are no significant additions to the surface infrastructure other than potential pumping station to supplement the water supply to the existing operation during dry seasons;
- **Existing Project Tailings:** The final and third phase of the existing TSF will follow the design prepared by IRYS and includes the addition of a third settling pond at El Guacio and an estimated two to three years of additional capacity at current production rates. Per GCM, the method of dewatering the tailings will stay consistent with the current employed methods.
- **MDZ Infrastructure:** The new project will require a new access road, a connection to the existing power system and the associated power infrastructure at the new site, new water supply, and a new camp site. The new site infrastructure concepts and preliminary designs are adequate to address the MDZ Project needs at a PEA level.

- **MDZ Tailings:** The MDZ Project will require the construction of at least two new DSTFs to accommodate the estimated tailings volume required to be managed through the LoM. These facilities will require new access roads, haul roads, pipeline corridors and culverts, perimeter stormwater channels and associated settling ponds, a connection to the existing power system for filter presses, and a building to house the filter presses and thickeners. Based on the currently-available data, the conceptual new site infrastructure and DSTF designs would be adequate to address the MDZ Project needs through the currently-envisioned mine life. However, a significant effort is required to adequately characterize the proposed DSTF foundations and the tailings material itself, in addition to identifying and testing a suitable borrow source for rock starter embankment construction and outer slope cladding for erosion protection. Based on the limited site and tailings characterization data currently available, there is significant uncertainty with the DSTF design and construction costs presented herein.

25.7 Environmental Studies and Permitting

The following interpretations and conclusions have been drawn with respect to the currently available information provided for the Marmato Project:

- **Permitting**: Operations are permitted through the posting of an Environmental Management Plan (PMA) and secondary permits for use of water abstraction, forest use, air emissions, discharges and river course (channel) construction. The PMA for the current operations was approved in 2001.
- **Environmental and Social Management**: Environmental and social issues are currently managed in accordance with the approved PMA, and may need to be updated and/or modified for the proposed expansion project.
- **Monitoring:** GCM has seven domestic wastewater discharges and three non-domestic (industrial) wastewater discharges for which monitoring is conducted. They also monitor air quality emissions from the metallurgical laboratory and smelter for particulate matter (PM), sulphur dioxide (SO_2) nitrogen oxides (NO_X) and lead (Pb). The tailings are infrequently monitored for hazard classification purposes through a Corrosive, Reactive, Explosive, Toxic, Inflammable, Pathogen [biological] (CRETIP) program. The results of the monitoring are provided to Corpocaldas. This monitoring program will require significant modification to include the facilities for the proposed expansion project, and to bring it up to international best practices.
- **Geochemistry:** Acid-generating sulfide minerals identified in the deposit include pyrite, arsenopyrite, iron-bearing sphalerite, pyrrhotite, and chalcopyrite (SRK, 2017). Samples of groundwater discharging into the underground are predominantly acidic. The underground water samples contain elevated metal(loid) concentrations. Geochemical properties of current and future waste rock and tailings are currently unknown.
- **Health and Safety:** The entire area around Marmato is heavily contaminated as a result of ‘artisanal and small-scale mining’ (ASM) alongside the formal GCM concession operation. Significant health issues related to this contamination are occurring in the artisanal mines and are expected to continue under current conditions. Whereas in many other parts of Colombia, illegal armed groups (i.e., FARC, ELN, etc.), as well as armed criminal groups (i.e., BACRIM) are tied to the ASM operations, the ASM operations at Marmato have become generational, with only a few families controlling mines in the area that have been in existence for almost 500 years.

- **Stakeholder Engagement**: GCM has conducted stakeholder identification and analysis programs and has set stakeholder engagement objectives and goals to develop communications plans with government, community, media and small miners but the company does not currently have a formal stakeholder engagement plan.
- **Closure Costs**: The reclamation and closure cost estimate provided for the current operations is approximately US$5.8 million, though there is uncertainty surrounding the basis for this estimate. A requirement for long-term post-closure water treatment, if any, could significantly increase this estimate. A cost of US$6.1 million was included in the technical economic model to account for a second tailings disposal facility.

Although additional studies are recommended to further develop the proposed mine expansion and tailings management strategies, there do not appear to be any other known environmental issues that could materially impact GCM's ability to conduct mining and milling activities at the site. Preliminary mitigation strategies have been developed to reduce environmental impacts to meet regulatory requirements and the conditions of the PMA.

Geochemistry

A substantial effort is needed to bring the mine into conformity with international best practices of data collection, management, and geochemical characterization. Implementation of a more comprehensive data collection and management program will form the quantitative basis for understanding the current status, forecasting future impacts, and designing concurrent and post-closure mitigation measures to minimize environmental impacts.

25.8 Projected Economic Outcomes

The cash flow valuation of the Marmato Project indicate that the Project has an after-tax Net Present Value (NPV) of approximately US$206.8 million, based on a 5% discount rate, gold price of US$1,300/oz and silver price of US$17.00/oz. The operation is projected to have negative cash flows during the years 2021 and 2022, when the MDZ is installed, with payback for the expansion expected by 2026. Life of mine is projected to end in 2038 resulting in a total production of 2.18 million ouces of gold and 1.82 million ounces of silver in the form of doré bars containing both precious metals. Indicative economic results are presented in Table 25-1.

Table 25-1: Marmato Indicative Economic Results

LoM Cash Flow		
Gold Price	USD/oz	1,300
Silver Price	USD/oz	17.00
Total Revenue	USD	2,851,475,063
Mining Cost	USD	(934,068,009)
Processing Cost	USD	(424,007,603)
G&A Cost	USD	(137,666,667)
Total Opex	USD	(1,495,742,278)
Operating Margin	USD	1,355,732,785
Operating Margin Ratio	%	48%
Taxes Paid	USD	(195,035,875)
Free Cashflow	USD	717,096,368
LoM Capital		
Expansion CAPEX	USD	(268,884,037)
Sustaining CAPEX	USD	(181,264,836)
Total LOM CAPEX	USD	(450,148,872)
Before Tax		
Free Cash Flow	USD	643,248,206
NPV @ 5%	USD	322,995,835
NPV @ 8%	USD	214,836,234
NPV @ 10%	USD	163,296,934
IRR	%	28%
After Tax		
Free Cash Flow	USD	448,212,331
NPV @ 5%	USD	206,821,360
NPV @ 8%	USD	126,595,938
NPV @ 10%	USD	88,893,940
IRR	%	20%
Payback	Year	2026

Source: SRK, 2019

The Project is a gold operation with a sub-product of silver, where gold represents 99% of the total projected revenue and silver the remaining 1%. The underground mining cost is the heaviest burden on the operation representing 63% of the operating cost, while processing costs represent 28% and G&A costs the remaining 9%.

The estimated All-in Sustaining Costs (AISC), including sustaining capital, is US$882/Au-oz.

25.9 Foreseeable Impacts of Risks

25.9.1 Water Supply

The water balance indicates that adequate water supply is available from the underground dewatering flows and additional water supply will be available from contact water runoff and seepage flows from the DSTF. However, the contact water supply will be erratic and uncertain and should not considered a reliable water supply. The underground dewatering flows represent a single source for a reliable water supply and a backup supply is recommended. Either sufficient water storage to span a foreseeable shut-down period of the underground dewatering system or a secondary water supply from the nearby Cauca River is recommended to ensure adequate water supply for the Project.

The water balance of the Project indicated that both contact water from the DSTF and underground dewatering flows are likely to exceed the makeup requirements at the processing plant at certain times in the LoM. Discharges from both water sources are expected and should be addressed with appropriate discharge structures as part of general infrastructure. Additionally, discharges of contact water should be monitored to ensure environmental compliance is maintained.

25.9.2 Mining

Upper Area

The mine plan for the Veins area is based on economic areas above a CoG. This grade is higher than the as-mined reported grade from Marmato due to factors such as Marmato mining outside of planned areas, mining marginal areas below the CoG and uncontrolled dilution.

The Level 21 represents a new mining method for Marmato and carries risks and uncertainties. Marmato will likely require a ramp up period to take full advantage of the bulk mining from this area.

25.9.3 Infrastructure

The infrastructure for the existing Project is mature and functioning. There are no significant additions required for the existing Project to operate other than the potential to supplement the water supply.

The following risks are identified with respect to new DSTFs:

- Filter Plant – It is currently envisioned that a filter plant will be installed and will achieve a tailings moisture content of around 15% by weight. If a filter plant is not installed, or if the target moisture content is not achieved, management of the tailings and amendment with cement may require more extensive and expensive handling than is currently envisioned.
- Seepage – Stormwater runon diversion, diligent stormwater runoff management, and subdrains are required to minimize infiltration into the DSTF, minimize the potential for seeps or springs to affect the DSTF foundation, and prevent the generation of an elevated phreatic surface that could impact DSTF mass stability.
- Elevated Groundwater – Groundwater elevations beneath the conceptual DSTF footprints have not been determined and may affect the design and extents of the subdrain system and the shear strength of foundation soils.
- Slope Stability – The potential for seismic loading, saturated foundations, foundation brittle failure, and unsatisfactory foundation conditions (weak soils) should be investigated and analyzed in future design phases.
- Erosion – The conceptual outer slopes of the DSTF are necessarily relatively steep at 2H:1V to maximize DSTF capacity in the steep natural terrain and will require rock cladding, cover soil placement and revegetation to prevent erosion.
- Geochemical Stability - The geochemical characteristics of the tailings, waste rock and potential borrow sources have not been established and may alter assumptions with respect to the necessity of a base liner, the availability and cost of a suitable borrow source, and the requirements to either treat or discharge contact stormwater and seepage.
- Land Position/Land Ownership – SRK completed an analysis of approximately eight potential DSTF sites within the current project boundary. There are no suitable sites available for development that do not cross ephemeral or perennial streams within the current landholding. A determination as to the ability to permit DSTF facilities across one of these drainage channels must be made and may require acquisition of additional landholdings in more permittable terrain.

- Access and Haul Roads – To minimize potential impacts of hauling and facility access on nearby communities, SRK prepared conceptual alignments and grading plans for haul and access roads associated with new DSTF areas. The ability to permit and construct these roads in the steep terrain at the site across existing perennial and ephemeral drainages, and the ability to manage stormwater flows that may intersect the road alignments, has yet to be determined and may significantly affect the estimated PEA costs associated with these structures.
- Rate-of-Rise – The ability to amend, place and compact tailings in the narrow natural drainages at the site will result in a relatively high initial rate-of-rise which may inhibit the dissipation of porewater pressure and affect the overall stability of the DSTF during early phases. Additional study is required to determine the potential impacts to DSTF operation, but additional management and associated operating costs may be required.
- Climate (rainy season) – The ability to amend, place and compact tailings during the rainy season may be curtailed. In addition, management of stormwater contacting exposed tailings is assumed to require management and routing through the process with subsequent water treatment prior to discharge. Additional characterization and discussions with permitting agencies is required to determine the ultimate management requirements for contact water.
- Material balance – it is currently assumed that a sufficient quantity of suitable waste rock or local borrow material is available for rock starter embankment construction and outer slope cladding. If waste rock or local borrow cannot be supplied when required to meet demand, additional more expensive and distant borrow sources may be required.
- Permitting – It is currently assumed that the DSTFs can be placed on prepared native subgrade without a geomembrane or imported compacted clay liner, and that non-contact stormwater captured upstream from the TSF or running off the closed outer faces of the DSTFs can be discharged into natural drainages downstream from the DSTFs. It is also assumed that subdrains could be utilized to adequately manage water from existing seeps, springs, or groundwater within natural drainages under the DSTFs. The ability to permit new unlined DSTFs in existing natural drainages with perennial surface expressions of water of any kind has yet to be established and may affect the final location and cost of design, construction and operation of the DSTFs in later design phases.

26 Recommendations

26.1 Recommended Work Programs and Costs

The Project requires a PFS level design effort to complete tradeoff studies and further evaluate the feasibility and viability of the Project.

26.1.1 Mineral Resources

SRK is currently working with GCM's geologists to optimize the remainder of the 2019 drilling program, with a focus on increasing the confidence in the MDZ. The technical studies will aim to infill the drilling spacing to a 50 by 50 m grid in the upper portions of the MDZ, and potentially increase the Inferred Resources at the end. SRK is also working on a number of engineering studies to support the future development of potential maiden Mineral Reserves for the Project. SRK anticipates the drilling to be completed during Q4 2019 with an updated Mineral Resource produced during the same time period.

Additional on-going recommendations for the Mineral Resource studies on this project, to be done prior to the PFS, should include:

- A detailed review of the vein model with GCM geologists to ensure continuity is suitably modelled;
- Review of the capping sensitivity used in the veins to ensure grades match the current mine plans on the lower levels of the mines, as GCM reported lower grades during recent months, but this could also be a function of dilution, which will also need to be addressed;
- It is recommended that GCM develop a system to flag channel samples in the database taken from the working stopes (vein channels), compared to more detailed exploration channel samples taken from cross-cuts and exploration development where possible. Any updates in the database will need to be completed prior to the PFS update, and SRK will work with the geological team to ensure the best solution can be found in the time available. The aim will be to identify areas for potential mining targets to provide additional feed to the current operation and plant;
- Continual monitoring of the MDZ drilling program with regular updates on the Leapfrog Model;
- SRK also recommends a structural review of the controlling mineralization at depth to support the PFS; and
- Comments by Richard H. Sillitoe: The conclusion that the 500 m long, west-northwest-striking Lower Zone orebody is entirely controlled by a veinlet array developed during dextral transpression raises the possibility that one or more look-alike deposits could be present elsewhere within the 6 km by 5 km P1 stock. They could be exposed at lower elevations than Marmato (say, less than 900 masl) as a result of greater degrees of erosion or, as in the case of the Lower Zone itself, remain concealed beneath and partly overprinted by a swarm of 'epithermal' massive base-metal sulfide veins. Either possibility would represent an attractive exploration opportunity, which it is considered worth pursuing. SRK considers this an important consider for future exploration on completion of the current drilling program.

26.1.2 Geotechnical

The current geotechnical design parameters are based on characterization data from 35 diamond drillholes, for which six were drilled deep in the orebody. The following recommendations are made with respect geotechnical issues regarding the Marmato Project as the design advances to the next level:

- Complete the geotechnical drilling program for the deep deposit (ongoing);
- Complete an hydrogeological investigation (ongoing);
- Prepare a numerical hydrogeological model;
- Prepare a major structural model;
- Integrate the exploration geotechnical investigation and the Operational geotechnical data collection (consolidate a unique geotechnical model)
- Conduct mine induced strength measurements;
- Conduct a 3D numerical model to understand the long-term mine stability;
- Estimate the back filling strength specifications;
- Re- evaluate the stope dimensions;
- Estimate surface subsidence magnitude and effect of mining in upper levels and infrastructure;
- Complete a hydrogeologic study for estimating pore pressures and possible inflow to new deeper levels; and
- Implement monitoring system (surface and micro seismic).

26.1.3 Tailings Management Facilities

During the next phase of study, the DSTF footprints need to be finalized and detailed characterization of the proposed DSTF foundations and the tailings material itself needs to be completed, in addition to identifying and testing a suitable borrow source for rock starter embankment construction and outer slope cladding for erosion protection. GCM is currently contracting for a new aerial survey to facilitate final DTSF siting and identification of borrow sources for construction. The ability to filter the tailings and achieve a target moisture content of about 15% by weight must also be established. The depth to groundwater and the source(s) of perennial water within each DSTF footprint must be determined, and the ability to permit the DSTFs without a base liner in natural drainages at the site must be established.

26.1.4 Mining

SRK recommends that the following areas be reviewed during the next phase of work:

- Review drift sizes to optimize and minimize waste movement;
- Review and refine the development required during the pre-production period to minimize cost and maximize faces available for development;
- Detailed sequencing of development and pre-production activities;
- Review the mine ventilation to optimize the system. This should include reviewing the impact of utilizing more electric powered equipment to reduce ventilation needs;
- Review costs, productivities, and PEA assumptions to reduce cost, optimize sizing, and reduce upfront time to production;
- Provide geotechnical and hydrogeological analysis of the decline and ventilation raise areas;
- Review the truck haulage system to see if there is a more cost effective method or options to reduce cost;
- Refine the backfill quantities, material characteristics, and placement scheme to optimize the system and confirm PEA assumptions. This should include testwork to determine paste makeup and strength;

- Perform geotechnical analysis and numerical models confirm geotechnical assumptions; and
- Optimization of the production schedule by delaying development to an as-needed basis.

26.1.5 Metallurgy and Mineral Processing

Additional metallurgical studies will need to be conducted during the next phase of study. This work has commenced and is being done at SGS Canada. This work is being conducted on master composites and variability composites that are reprensentative of the MDZ. These studies are directed at optimizing process parameters for the selected process flowsheet and establish the process design criteria for the process plant. The next phase of metallurgical testing is estimated to cost about US$250,000.

26.1.6 Recovery Methods

During the next phase of study, process plant engineering and design will need to be conducted by a qualified engineering design based on process design criteria developed during the metallurgical program. This work has commenced and is being conducted by Ausenco. Process engineering and design will be conducted to a level of detail that will support capital and operating cost estimates at a +/-25% level of accuracy. It is estimated that prefeasibility level process engineering and design will cost US$250,00 to US$300,000.

26.1.7 Infrastructure

GCM is currently working on retaining a third party engineering firm for completion of PFS level infrastructure engineering. GCM has estimated that a third party firm will be selected and contracted by the end of 2019.

26.1.8 Hydrogeology

Based on the current understanding of hydrogeological conditions, the minimum data set will be needed to meet a PFS-level hydrogeological analysis, SRK recommends the following work plan:

- Verify the field status of the 2012 piezometers presented in Table 16-7 and conduct a detailed QA/QC of the hydraulic tests performed in 2012. Based on the verification findings, a detailed work plan will be designed. The following preliminary tasks are strongly recommended:
  - Conduct short-term hydraulic tests (airlift test, slug test constant head test, etc.) in the available standpipe piezometers.
  - Conduct shutdown tests in the piezometers located La Macha Tank (Level 17).
  - Construct multiple VWP and/or standpipe piezometers upstream and low stream of planned underground mine (proposed Hydro 001 and 002).
  - Hydraulic testing of three NQ exploration/geotechnical holes from Level 21, (packer tests, flow/shut-in testing if flowing conditions or slug testing if water level is below underground developments) with a bottom target from 421 to 635 m amsl (boreholes SRK 3, SRK 13 and SRK 16).
  - Install piezometers/monitoring wells in three geotechnical holes (SRK 13 and SRK 16) for water level measurements (string of vibrating wire piezometers, VWPs, to define vertical hydraulic gradient or install pressure transducers in standpipes).
  - Install two deep-VWP targeting potential structures with connection to the Cauca river. (Hydro 3 Hydro 4).

- Collecting groundwater samples in proposed piezometers.
- Continue monitor flow (total, from different levels). Surveying underground developments by collecting groundwater seepage rates, seepage-water strike elevations and water quality data.

Figure 16-28 shows the location of the proposed piezometers.

26.1.9 Environmental Studies and Permitting

The following recommendations are made with respect to environmental, permitting and social issues regarding the Marmato Project:

- Prepare a more detailed site-wide closure plan from which a more accurate final closure cost estimate can be developed. This should include things like: equipment inventories; building inventories (with limited design details), portal and vent plugging details and conceptual designs, tec. This plan and cost estimate would require annual reviews and updates in order to capture the latest configurations and conditions at the mine site(s) and processing facilities.
- Insufficient work has been undertaken on groundwater hydrogeology and surface water to establish the true level of risk associated with potential groundwater contamination and underground dewatering impacts. A detailed evaluation, including a groundwater model, could provide information that would assist in forecasts of post-closure mine water discharge and possible treatment requirements. Such an investigation could also provide vital information on underground geotechnical stability, both during operations and post closure.
- There is an apparent shortage of geochemical characterization data. Without an understanding of the scope of previous investigations (i.e., Knight Piésold investigations), further work will be required for the PFS level, including ARDML characterization of future waste rock, underground wall rock, and tailings designated for backfill or disposal to the TSF. The water quality of dewatering effluent must be well characterized in the event that treatment is needed before it is used or discharged. A forecast of closure water quality is needed.
- Detailed characterization of the future backfill tailings is needed to resolve the conflicting information regarding their geochemical properties.
- Additional work is recommended to confirm groundwater and surface water quality in the mine area and enhance the data collection network through installation of water quality monitoring wells, and establishment of a periodic underground water quality monitoring program.
- The regulation prohibiting surface waste rock disposal should be further investigated. Although the volume of waste rock brought to the surface is expected to be relatively small, the regulation could represent a hindrance to operations unless a variance can be obtained.
- Co-disposal of acid-generating waste rock in the TSF should be evaluated in further detail.
- Characterization work should be completed on artisanal tailings and waste rock to understand their ARDML potential and devise a management plan.
- A comprehensive baseline surface and groundwater sampling program will be important to establish the baseline condition and try to quantify the contributions from artisanal or pre-mining conditions, especially with respect to mercury from artisanal mining.

Substantial financial resources and technical specialist support will be required to implement the environmental monitoring and mitigation measures likely to be presented in the updated PMA for the expansion project.

26.1.10 Project Economics

The following recommendations are made with respect to capital and operating cost and economic evaluation of the Marmato Project:

- Prepare first principles estimate of capital and operating costs with enough accuracy to support a future PFS study of the project, including:
    - Capital estimates and expenditure schedule based on basic engineering;
    - Prepare cash flow model based on shorter periods of production;
    - Further detail site-specific operating cost data and cost models to include fixed and variable nature of costs and detail cost models to include breakdown by area and function; and
    - Improve cost models to include currencies used to estimate each cost and prepare sensitivity to currencies variability.
- Preliminary economic valuations performed for the PEA indicate that the project could be further optimized, SRK recommends that an optimization of the valuation is performed through a series of trade-off studies during the PFS study.

26.1.11 Costs

The recommended work outlined here is targeted at the completion of a PFS for the project. The estimated costs are outlined in Table 26-1 below.

Table 26-1: Summary of Costs for Recommended Work

Recommended Work	Cost Estimate (US$)
Complete infill drilling program to approximate 50x50 m spacing	$1.7 million
Process PFS level design and metallurgical test programs.	$0.6 million
Infrastructure PFs design program	$0.1 million
NI 43-101 Compliant PFS report work program including geology, mining, hydrogeology, tailings and economics work programs	$1.8 million
Total	$4.2 million

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28 Glossary

The Mineral Resources and Mineral Reserves have been classified according to CIM (CIM, 2014). Accordingly, the Resources have been classified as Measured, Indicated or Inferred, the Reserves have been classified as Proven, and Probable based on the Measured and Indicated Resources as defined below.

28.1 Mineral Resources

A **Mineral Resource** is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

An **Inferred Mineral Resource** is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An **Indicated Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

A **Measured Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

28.2 Mineral Reserves

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

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

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

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

28.3 Definition of Terms

The following general mining terms may be used in this report.

Table 28-1: Definition of Terms

Term	Definition
Assay	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure	All other expenditures not classified as operating costs.
Composite	Combining more than one sample result to give an average result over a larger distance.
Concentrate	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
Crushing	Initial process of reducing ore particle size to render it more amenable for further processing.
Cut-off Grade (CoG)	The grade of mineralized rock, which determines as to whether or not it is economic to recover its gold content by further concentration.
Dilution	Waste, which is unavoidably mined with ore.
Dip	Angle of inclination of a geological feature/rock from the horizontal.
Fault	The surface of a fracture along which movement has occurred.
Footwall	The underlying side of an orebody or stope.
Gangue	Non-valuable components of the ore.
Grade	The measure of concentration of gold within mineralized rock.
Hangingwall	The overlying side of an orebody or slope.
Haulage	A horizontal underground excavation which is used to transport mined ore.
Hydrocyclone	A process whereby material is graded according to size by exploiting centrifugal forces of particulate materials.
Igneous	Primary crystalline rock formed by the solidification of magma.
Kriging	An interpolation method of assigning values from samples to blocks that minimizes the estimation error.
Level	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Lithological	Geological description pertaining to different rock types.
LoM Plans	Life-of-Mine plans.
LRP	Long Range Plan.
Material Properties	Mine properties.
Milling	A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease	A lease area for which mineral rights are held.
Mining Assets	The Material Properties and Significant Exploration Properties.
Ongoing Capital	Capital estimates of a routine nature, which is necessary for sustaining operations.
Ore Reserve	See Mineral Reserve.

Term	Definition
Pillar	Rock left behind to help support the excavations in an underground mine.
RoM	Run-of-Mine.
Sedimentary	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Shaft	An opening cut downwards from the surface for transporting personnel, equipment, supplies, ore and waste.
Sill	A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.
Smelting	A high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase.
Stope	Underground void created by mining.
Stratigraphy	The study of stratified rocks in terms of time and space.
Strike	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulfide	A sulfur bearing mineral.
Tailings	Finely ground waste rock from which valuable minerals or metals have been extracted.
Thickening	The process of concentrating solid particles in suspension.
Total Expenditure	All expenditures including those of an operating and capital nature.
Variogram	A statistical representation of the characteristics (usually grade).

28.4 Abbreviations

The following abbreviations may be used in this report.

Table 28-2: Abbreviations

Abbreviation	Unit or Term
A	ampere
AA	atomic absorption
A/m^2	amperes per square meter
ANFO	ammonium nitrate fuel oil
Ag	silver
Au	gold
AuEq	gold equivalent grade
°C	degrees Centigrade
CCD	counter-current decantation
CIL	carbon-in-leach
CoG	cut-off grade
cm	centimeter
cm^2	square centimeter
cm^3	cubic centimeter
cfm	cubic feet per minute
ConfC	confidence code
CRec	core recovery
CSS	closed-side setting
CTW	calculated true width
°	degree (degrees)
dia.	diameter
EIS	Environmental Impact Statement
EMP	Environmental Management Plan
FA	fire assay
ft	foot (feet)
ft^2	square foot (feet)
ft^3	cubic foot (feet)
g	gram

Abbreviation	Unit or Term
gal	gallon
g/L	gram per liter
g-mol	gram-mole
gpm	gallons per minute
g/t	grams per tonne
ha	hectares
HDPE	Height Density Polyethylene
hp	horsepower
HTW	horizontal true width
ICP	induced couple plasma
ID2	inverse-distance squared
ID3	inverse-distance cubed
IFC	International Finance Corporation
ILS	Intermediate Leach Solution
kA	kiloamperes
kg	kilograms
km	kilometer
km^2	square kilometer
koz	thousand troy ounce
kt	thousand tonnes
kt/d	thousand tonnes per day
kt/y	thousand tonnes per year
kV	kilovolt
kW	kilowatt
kWh	kilowatt-hour
kWh/t	kilowatt-hour per metric tonne
L	liter
L/sec	liters per second
L/sec/m	liters per second per meter
lb	pound
LHD	Long-Haul Dump truck
LLDDP	Linear Low Density Polyethylene Plastic
LOI	Loss On Ignition
LoM	Life-of-Mine
m	meter
m^2	square meter
m^3	cubic meter
masl	meters above sea level
MARN	Ministry of the Environment and Natural Resources
MDA	Mine Development Associates
mg/L	milligrams/liter
mm	millimeter
mm^2	square millimeter
mm^3	cubic millimeter
MME	Mine & Mill Engineering
Moz	million troy ounces
Mt	million tonnes
MTW	measured true width
MW	million watts
m.y.	million years
NGO	non-governmental organization
NI 43-101	Canadian National Instrument 43-101
OSC	Ontario Securities Commission
oz	troy ounce
%	percent
PLC	Programmable Logic Controller
PLS	Pregnant Leach Solution
PMF	probable maximum flood
ppb	parts per billion

Abbreviation	Unit or Term
ppm	parts per million
QA/QC	Quality Assurance/Quality Control
RC	rotary circulation drilling
RoM	Run-of-Mine
RQD	Rock Quality Description
SEC	U.S. Securities & Exchange Commission
sec	second
SG	specific gravity
SPT	standard penetration testing
st	short ton (2,000 pounds)
t	tonne (metric ton) (2,204.6 pounds)
t/h	tonnes per hour
tpd	tonnes per day
t/y	tonnes per year
TSF	tailings storage facility
TSP	total suspended particulates
µm	micron or microns
V	volts
VFD	variable frequency drive
W	watt
XRD	x-ray diffraction
y	year

Appendices

Appendix A: Certificates of Qualified Persons

SRK Consulting (U.S.), Inc.
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CERTIFICATE OF QUALIFIED PERSON

I, David Bird, MSc., PG, RM-SME, do hereby certify that:

1. I am Associate Principal Consultant (Geochemistry) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report").
3. I graduated with Bachelor's Degrees in Geology and Business Administration Management from Oregon State University in 1983. In addition, I obtained a Master's Degree in Geochemistry/Hydrogeology from the University of Nevada-Reno in 1993. I am a Registered Member of the Society for Mining, Metallurgy, and Exploration (SME). I am a certified Professional Geologist in the State of Oregon (G1438). I have worked full time as a Geologist and Geochemist for a total of 32 years. My relevant experience includes design, execution, and interpretation of mine waste geochemical characterization programs in support of open pit and underground mine planning and environmental impact assessments, design and supervision of water quality sampling and monitoring programs, geochemical modeling, and management of the geochemistry portion of numerous PEA, PFS and FS-level mine projects in the US and abroad.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Marmato property.
6. I am responsible for Geochemistry Section 20.1.6. and portions of Sections 1, 24, 25 and 26.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 27th Day of November, 2019.

"Signed"

"Sealed"

____

David Bird, MSc, PG, RM-SME

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CERTIFICATE OF QUALIFIED PERSON

I, David Hoekstra, BSc Civil Engineering, P.E, do hereby certify that:

1. I am Principal Consultant (Civil Engineer) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report").
3. I graduated with a degree in Civil Engineering from Colorado State University in 1986. I am a Professional Engineer of the States of Alaska, Colorado, Montana, South Carolina, and Wyoming. I have worked as an Engineer for a total of 30 years since my graduation from university. My relevant experience includes the design and implementation of mine water management systems and storm water controls for numerous PEA, PFS, FS-level and operating mine projects in the US and abroad.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Marmato property.
6. I am responsible for Hydrology Section 20.2.4 and portions of Sections 1, 24, 25 and 26.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 27th Day of November, 2019.

"Signed"

"Sealed"

David Hoekstra, BSc Civil Engineering, P.E.

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CERTIFICATE OF QUALIFIED PERSON

I, Eric Olin, MSc, MBA, RM-SME do hereby certify that:

1. I am a Principal Process Metallurgist of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia, with an Effective Date of July 31, 2019 (the "Technical Report").
3. I graduated with a Master of Science degree in Metallurgical Engineering from the Colorado School of Mines in 1976. I am a Registered Member of The Society for Mining, Metallurgy and Exploration, Inc. I have worked as a Metallurgist for a total of 40 years since my graduation from the Colorado School of Mines. My relevant experience includes extensive consulting, plant operations, process development, project management and research & development experience with base metals, precious metals, ferrous metals and industrial minerals. I have served as the plant superintendent for several gold and base metal mining operations. Additionally, I have been involved with numerous third-party due diligence audits, and preparation of project conceptual, pre-feasibility and full-feasibility studies.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Marmato property.
6. I am responsible for the preparation of Metallurgy Sections 13 and 17 and portions of Sections 1, 24, 25 and 16 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 27th Day of November, 2019.

"Signed" *"Sealed"*

____

Eric Olin, MSc, MBA, RM-SME

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CERTIFICATE OF QUALIFIED PERSON

I, Jeff Osborn, BEng Mining, MMSAQP do hereby certify that:

1. I am a Principal Consultant (Mining Engineer) of SRK Consulting (U.S.), Inc., 1125 Seventeenth, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia” with an Effective Date of July 31, 2019 (the “Technical Report”).
3. I graduated with a Bachelor of Science Mining Engineering degree from the Colorado School of Mines in 1986. I am a Qualified Professional (QP) Member of the Mining and Metallurgical Society of America. I have worked as a Mining Engineer for a total of 33 years since my graduation from university. My relevant experience includes responsibilities in operations, maintenance, engineering, management, mine costing, and construction activities.
4. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I visited the armato property on July 16 to July 18, 2019 for 3 days. I visited the Marmato property on August 22 and 23, 2017 for two days.
6. I am responsible for infrastructure and costing Sections 18.1, 18.2, 18.3, 18.5 and 21, and portions of Sections 1, 24, 25 and 26.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is contribution to the infrastructure section of the Ni 43-101 Technical Report Updated Mineral Resource Estimate Marmato Project Colombia report date November 20, 2017.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 27th Day of November, 2019.

“Signed”

“Sealed”

Jeff Osborn, BEng Mining, MMSAQP [01458QP]
Principal Consultant (Mining Engineer)

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CERTIFICATE OF QUALIFIED PERSON

I, Benjamin Parsons, MSc, MAusIMM (CP) do hereby certify that:

1. I am a Principal Consultant (Resource Geology) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia” with an Effective Date of July 31, 2019 (the “Technical Report”).
3. I graduated with a degree in Exploration Geology from Cardiff University, UK in 1999. In addition, I have obtained a Masters degree (MSc) in Mineral Resources from Cardiff University, UK in 2000 and have worked as a geologist for a total of 16 years since my graduation from university. I am a member of the Australian Institution of Materials Mining and Metallurgy (Membership Number 222568) and I am a Chartered Professional.
4. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I visited the Marmato property on March 12, 2012 for two days, August 17, 2017 for one day and June 11, 2019 for two days.
6. I am responsible for data verification, preparation of the geologic model and mineral resource Sections 4 through 12, 14, 23 and portions of Sections 1, 24, 25 and 26
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was in preparation of the NI43-101 Mineral Resource Estimates dated September 4, 2011, August 3, 2012, and the previous NI 43-101 Mineral Resource Estimate dated June 16, 2017 .
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 27th Day of November, 2019.

“Signed” *“Sealed”*

____
Benjamin Parsons, MSc, MAusIMM
Principal Consultant (Resource Geology)

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CERTIFICATE OF QUALIFIED PERSON

I, Cristian A. Pereira Farias, SME-RM, do hereby certify that:

1. I am Senior Consultant (Hydrogeologist) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report").
3. I graduated with a degree in Bachelors of Science in Geology from Universidad de Chile in 1999. I am a registered member of the Society for Mining, Metallurgy, and Exploration. I have worked as a hydrogeologist for a total of 19 years since my graduation from university. My relevant experience includes the developing conceptual and numerical hydrogeological models, the evaluation of groundwater resources, mine dewatering requirements, environmental impacts of mining, pit lake infilling, brine extraction, and pore pressure analyses.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Marmato property on August 12, 2019 for two days.
6. I am responsible for Hydrogeology Sections 16.3.2 and 16.4.2. and portions of Sections 1, 24, 25 and 26.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 27th Day of November, 2019.

"Signed"

"Sealed"

Cristian A. Pereira Farias

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CERTIFICATE OF QUALIFIED PERSON

I, Joanna Poeck, BEng Mining, SME-RM, MMSAQP, do hereby certify that:

1. I am a Principal Mining Engineer of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report").
3. I graduated with a degree in Mining Engineering from Colorado School of Mines in 2003. I am a Registered Member of the Society of Mining, Metallurgy & Exploration Geology. I am a QP member of the Mining & Metallurgical Society of America. I have worked as a Mining Engineer for a total of 16 years since my graduation from university. My relevant experience includes open pit and underground design, mine scheduling and mine optimization.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Marmato property.
6. I am responsible for MDZ Mining related portions of sections 15, 16.4 (except for section 16.4.2) and portions of Sections 1, 24, 25 and 26.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 27th Day of November, 2019.

"Signed"

"Sealed"

Joanna Poeck, BEng Mining, SME-RM[4131289RM], MMSAQP[01387QP]
Principal Consultant (Mining Engineer)

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CERTIFICATE OF QUALIFIED PERSON

I, Fernando Rodrigues, BS Mining, MBA, MMSAQP do hereby certify that:

1. I am Practice Leader and Principal Consultant (Mining Engineer) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report").
3. I graduated with a Bachelors of Science degree in Mining Engineering from South Dakota School of Mines and Technology in 1999. I am a QP member of the MMSA. I have worked as a Mining Engineer for a total of 18 years since my graduation from South Dakota School of Mines and Technology in 1999. My relevant experience includes mine design and implementation, short term mine design, dump design, haulage studies, blast design, ore control, grade estimation, database management.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Marmato property on August 22, 2019 for two days.
6. I am responsible for Upper Zone Mining and Economics related portions of Sections 16.1, 16.3 (except for section 16.3.2), 19 and 22, and portions of Sections 1, 25 and 26.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 27th Day of November, 2019.

"Signed"

"Sealed"

Fernando Rodrigues, BS Mining, MBA, MMSAQP [01405QP]

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CERTIFICATE OF QUALIFIED PERSON

I, Joshua D. Sames, BSc Civil Engineering, PE do hereby certify that:

1. I am Senior Consultant at SRK Consulting (U.S.), Inc., 5250 Neil Road, Suite 300, Reno, NV, USA 89502.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report").
3. I graduated with a degree in Civil Engineering from University of Newcastle Australia in 2005. I am a registered Professional Engineer in the State of Nevada (PE No. 22346). I have worked as an engineer for a total of 13 years. My relevant experience includes site investigations, conceptual and detailed design of tailing storage facilities, construction supervision, management and operational assessments, mine reclamation permitting and closure design and permitting at mining properties in the western United States and South and Central America.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Marmato Project property.
6. I am responsible for the preparation of tailings management facilities Section 18.4 and portions of Sections 1, 24, 25 and 26.
7. I am independent of the issuer.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 27th day of November, 2019

"Signed"

___________________________ *"Sealed"*

Joshua D. Sames P.E
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CERTIFICATE OF QUALIFIED PERSON

I, John Tinucci, PhD, PE, ISRM, do hereby certify that:

1. I am Practice Leader/Principal Consultant (Geotechnical Engineer) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled " NI 43-101 Technical Report, Preliminary Economic Assesment, Marmato Project, Colombia" with an effective Date of July 31, 2019 (the "Technical Report").
3. I graduated with a degree in B.S. in Civil Engineering from Colorado State University, in 1980. In addition, I have obtained a M.S. in Geotechnical Engineering from University of California, Berkeley, in 1983 and I have obtained a Ph.D. in Geotechnical Engineering, Rock Mechanics from the University of California, Berkeley in 1985. I am member of the American Rock Mechanics Association, a member of the International Society of Rock Mechanics, and a Professional Member of the Society for Mining, Metallurgy & Exploration. I have worked as a Mining and Geotechnical Engineer for a total of 37 years since my graduation from university. My relevant experience includes 34 years of professional experience. I have 20 years managerial experience leading project teams, managing P&L operations for 120 staff, and directed own company of 8 staff for 8 years. I have technical experience in mine design, prefeasibility studies, feasibility studies, geomechanical assessments, rock mass characterization, project management, numerical analyses, underground mine stability, subsidence, tunneling, ground support, slope design and stabilization, excavation remediation, induced seismicity and dynamic ground motion. My industry commodities experience includes salt, potash, coal, platinum/palladium, iron, molybdenum, gold, silver, zinc, diamonds, and copper. My mine design experience includes open pit, room and pillar, (single and multi-level), conventional drill-and-blast and mechanized cutting, longwall, steep narrow vein, cut and fill, block caving, sublevel caving and cut and fill longhole stoping and paste backfilling.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Marmato property on August 22, 2017 for two days.
6. I am responsible for Geotechnical Section 16.2 and portions of Sections 1, 24, 25 and 26.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report..
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 27th Day of November, 2019.

"Signed"

"Sealed"

John Tinucci, PhD, PE, ISRM

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CERTIFICATE OF QUALIFIED PERSON

I, Mark Allan Willow, MSc, CEM, SME-RM do hereby certify that:

1. I am Practice Leader/Principal Environmental Scientist of SRK Consulting (U.S.), Inc., 5250 Neil Road, Reno, Nevada 89502.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia” with an Effective Date of July 31, 2019 (the “Technical Report”).
3. I graduated with Bachelor's degree in Fisheries and Wildlife Management from the University of Missouri in 1987 and a Master's degree in Environmental Science and Engineering from the Colorado School of Mines in 1995. I have worked as Biologist/Environmental Scientist for over 25 years since my graduation from university. My relevant experience includes environmental due diligence/competent persons evaluations of developmental phase and operational phase mines through the world, including small gold mining projects in Panama, Senegal, Peru, Ecuador, Philippines, and Colombia; open pit and underground coal mines in Russia; large copper and iron mines and processing facilities in Mexico and Brazil; bauxite operations in Jamaica; and a coal mine/coking operation in the People's Republic of China. My Project Manager experience includes several site characterization and mine closure projects. I work closely with the U.S. Forest Service and U.S. Bureau of Land Management on permitting and mine closure projects to develop uniquely successful and cost-effective closure alternatives for the abandoned mining operations. Finally, I draw upon this diverse background for knowledge and experience as a human health and ecological risk assessor with respect to potential environmental impacts associated with operating and closing mining properties and have experience in the development of Preliminary Remediation Goals and hazard/risk calculations for site remedial action plans under Superfund activities according to current U.S. EPA risk assessment guidance.
4. I am a Certified Environmental Manager (CEM) in the State of Nevada (#1832) in accordance with Nevada Administrative Code 459.970 through 459.9729. Before any person consults for a fee in matters concerning: the management of hazardous waste; the investigation of a release or potential release of a hazardous substance; the sampling of any media to determine the release of a hazardous substance; the response to a release or cleanup of a hazardous substance; or the remediation soil or water contaminated with a hazardous substance, they must be certified by the Nevada Division of Environmental Protection, Bureau of Corrective Action;
5. I am a Registered Member (No. 4104492) of the Society for Mining, Metallurgy & Exploration Inc. (SME).
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.
7. I visited the Marmato property on December 1, 2016 for 1 day.
8. I am responsible for environmental studies, permitting and social or community impact, Sections 1.11, 4.4, 20 (except for section 20.1.6), portions of 24, 25.7, and 26.1.8.
9. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.

12. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 27th Day of November, 2019.

"Signed" *"Sealed"*

SME-RM# 4104492

Mark Allan Willow, MSc, CEM, SME-RM

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CONSENT OF QUALIFIED PERSON

I, David Bird, MSc., PG, RM-SME, consent to the public filing of the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report") by Gran Colombia Gold Marmato S.A.S.

I also consent to any extracts from or a summary of the Technical Report in the press release titled "Gran Colombia Gold Announces Updated Mineral Resource Estimate and Preliminary Economic Assessment for it's Marmato Project" and dated October 15, 2019 of Gran Colombia Gold Marmato, S.A.S..

I certify that I have read in the press release titled "Gran Colombia Gold Announces Updated Mineral Resource Estimate and Preliminary Economic Assessment for it's Marmato Project" and dated October 15, 2019 being filed by Gran Colombia Gold Marmato, S.A.S. and that it fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Dated this 27th Day of November, 2019.

"Signed"

"Sealed"

David Bird, MSc, PG, RM-SME

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CONSENT OF QUALIFIED PERSON

I, David Hoekstra, BSc Civil Engineering, P.E., consent to the public filing of the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report") by Gran Colombia Gold Marmato S.A.S.

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CONSENT OF QUALIFIED PERSON

I, Eric Olin, MSc, MBA, RM-SME, consent to the public filing of the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report") by Gran Colombia Gold Marmato S.A.S.

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____

Eric Olin, MSc, MBA, RM-SME

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CONSENT OF QUALIFIED PERSON

I, Jeff Osborn, BEng Mining, MMSAQP, consent to the public filing of the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report") by Gran Colombia Gold Marmato S.A.S.

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Jeff Osborn, BEng Mining, MMSAQP

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CONSENT OF QUALIFIED PERSON

I, Benjamin Parsons, MSc, MAusIMM (CP), consent to the public filing of the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report") by Gran Colombia Gold Marmato S.A.S.

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Dated this 27th Day of November, 2019.

"Signed"

"Sealed"

____

Benjamin Parsons, MSc, MAusIMM (CP)

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CONSENT OF QUALIFIED PERSON

I, Cristian A. Pereira Farias, SME-RM, consent to the public filing of the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report") by Gran Colombia Gold Marmato S.A.S.

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Dated this 27th Day of November, 2019.

"Signed"

"Sealed"

Cristian A. Pereira Farias, SME-RM

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CONSENT OF QUALIFIED PERSON

I, Joanna Poeck, BEng Mining, SME-RM, MMSAQP, consent to the public filing of the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report") by Gran Colombia Gold Marmato S.A.S.

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Dated this 27th Day of November, 2019.

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"Sealed"

____

Joanna Poeck, BEng Mining, SME-RM, MMSAQP

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CONSENT OF QUALIFIED PERSON

I, Fernando Rodrigues, BS Mining, MBA, MMSAQP, consent to the public filing of the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report") by Gran Colombia Gold Marmato S.A.S.

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Dated this 27th Day of November, 2019.

"Signed"

______________________________ *"Sealed"*

Fernando Rodrigues, BS Mining, MBA, MMSAQP

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CONSENT OF QUALIFIED PERSON

I, Joshua D. Sames, BSc Civil Engineering, PE, consent to the public filing of the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report") by Gran Colombia Gold Marmato S.A.S.

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Dated this 27th Day of November, 2019.

"Signed"

"Sealed"

____

Joshua D. Sames, BSc Civil Engineering, PE

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CONSENT OF QUALIFIED PERSON

I, John Tinucci, PhD, PE, ISRM consent to the public filing of the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report") by Gran Colombia Gold Marmato S.A.S.

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Dated this 27th Day of November, 2019.

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John Tinucci, PhD, PE, ISRM

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CONSENT OF QUALIFIED PERSON

I, Mark Allan Willow, MSc, CEM, SME-RM consent to the public filing of the technical report titled "NI 43-101 Technical Report, Preliminary Economic Assessment, Marmato Project, Colombia" with an Effective Date of July 31, 2019 (the "Technical Report") by Gran Colombia Gold Marmato S.A.S.

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Dated this 27th Day of November, 2019.

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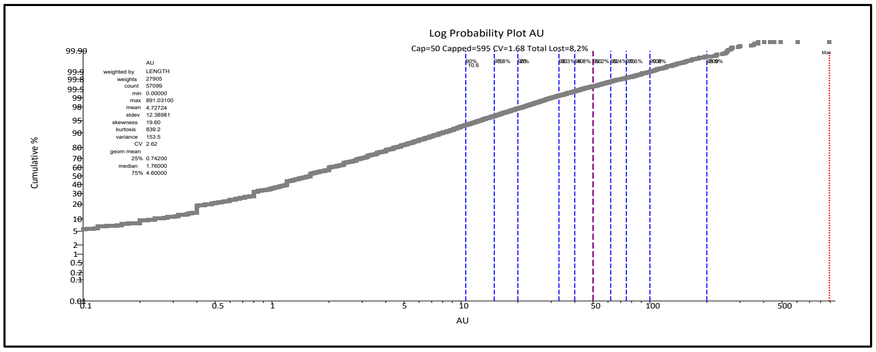
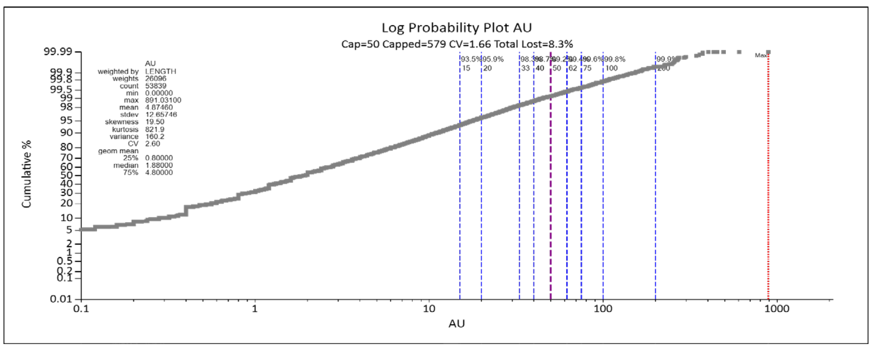
"Sealed"

Mark Allan Willow, MSc, CEM, SME-RM

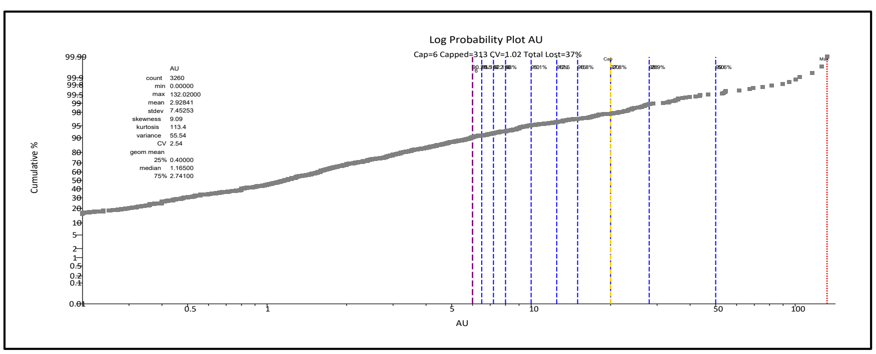
Appendix B: Capping



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AU								57099	0	891	4.73	131912	153.5	2.62
AU		200	36	99.90%	0.10%	1.30%	14%	57099	0	200	4.66	130149	110.2	2.25
AU		100	174	99.80%	0.30%	3.60%	24%	57099	0	100	4.56	127172	81.82	1.98
AU		75	293	99.60%	0.50%	5.20%	29%	57099	0	75	4.48	125033	69.28	1.86
AU		62	413	99.40%	0.70%	6.50%	32%	57099	0	62	4.42	123343	61.56	1.78
AU		50	595	99.20%	1.00%	8.20%	36%	57099	0	50	4.34	121080	53.25	1.68
AU		40	881	98.80%	1.50%	10.00%	39%	57099	0	40	4.24	118354	45.36	1.59
AU		33	1220	98.50%	2.10%	12.00%	42%	57099	0	33	4.14	115578	38.97	1.51
AU		20	2813	96.00%	4.90%	19.00%	50%	57099	0	20	3.81	106228	24.52	1.3
AU		15	4341	93.80%	7.60%	25.00%	55%	57099	0	15	3.56	99327	17.77	1.18
AU		10.6	6853	90.00%	12.00%	32.00%	60%	57099	0	10.6	3.21	89634	11.36	1.05
AU	AU > 50							595	50.09	891	97.92	22134	5735	0.77
AU	AU <= 50							56504	0	50	3.97	109778	36.52	1.52



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AU								53839	0	891	4.88	127206	160.2	2.6
AU		200	36	99.90%	0.10%	1.40%	15%	53839	0	200	4.81	125443	113.9	2.22
AU		100	169	99.80%	0.30%	3.70%	25%	53839	0	100	4.70	122517	84.02	1.95
AU		75	284	99.60%	0.50%	5.30%	30%	53839	0	75	4.62	120481	71.28	1.83
AU		62	402	99.40%	0.70%	6.60%	33%	53839	0	62	4.56	118864	63.4	1.75
AU		50	579	99.20%	1.10%	8.30%	36%	53839	0	50	4.47	116683	54.87	1.66
AU		40	861	98.70%	1.60%	10.00%	40%	53839	0	40	4.37	114039	46.71	1.56
AU		40	861	98.70%	1.60%	10.00%	40%	53839	0	40	4.37	114039	46.71	1.56
AU		33	1188	98.30%	2.20%	12.00%	43%	53839	0	33	4.27	111341	40.09	1.48
AU		20	2739	95.90%	5.10%	20.00%	51%	53839	0	20	3.92	102249	25.14	1.28
AU		15	4236	93.50%	7.90%	25.00%	55%	53839	0	15	3.66	95545	18.19	1.16
AU	AU > 50							579	50.09	891	98.13	21455	5907	0.78
AU	AU <= 50							53260	0	50	4.09	105751	37.67	1.5



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AU								3260	0	132	2.93	9547	55.54	2.54
AU		50	16	99.60%	0.50%	5.60%	20%	3260	0	50	2.77	9015	31.49	2.03
AU		28	36	98.90%	1.10%	11.00%	32%	3260	0	28	2.61	8498	20.51	1.74
AU		20	74	97.80%	2.30%	16.00%	39%	3260	0	20	2.47	8062	14.90	1.56
AU		15	105	96.80%	3.20%	20.00%	45%	3260	0	15	2.34	7622	10.87	1.41
AU		12.5	130	96.00%	4.00%	23.00%	48%	3260	0	12.5	2.25	7334	8.85	1.32
AU		10	159	95.10%	4.90%	27.00%	52%	3260	0	10	2.14	6971	6.84	1.22
AU		8	229	93.00%	7.00%	31.00%	56%	3260	0	8	2.02	6589	5.23	1.13
AU		7.2	256	92.10%	7.90%	33.00%	57%	3260	0	7.2	1.96	6396	4.57	1.09
AU		6.5	288	91.10%	8.80%	35.00%	59%	3260	0	6.5	1.90	6204	3.99	1.05
AU		6	313	90.20%	9.60%	37.00%	60%	3260	0	6	1.86	6054	3.59	1.02
AU	AU > 20							74	20.16	132	40.06	2965	702.70	0.66
AU	AU <= 20							3186	0	19.9	2.07	6582	7.94	1.36

GRANCOLOMBIAGOLD

srk consulting

DESIGN: | DRAWN: BP | REVIEWED: BP

SCALE: NOT TO SCALE | DATE: 11/26/2019

FILE: AppendixC_capping2019.xlsx

CAPPING CHARTS 1000Au

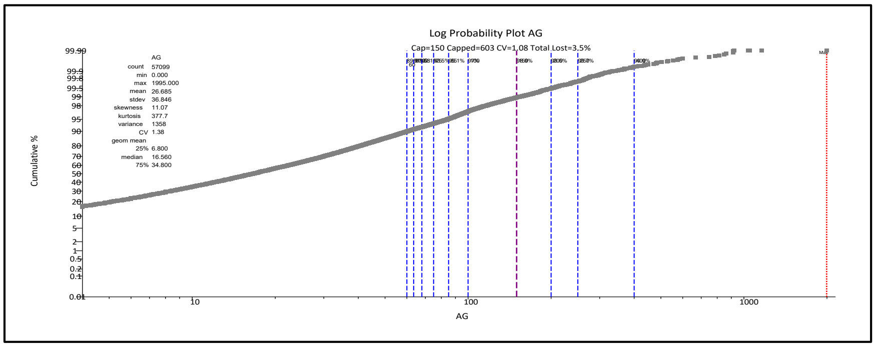
REPORT TITLE:
NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT MARMATO PROJECT

ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

APPENDIX B-1

REVISION NO. **A**

SRK JOB NO.: **544400.020**



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AG								57099	0	1995	26.69	1523709	1358	1.38
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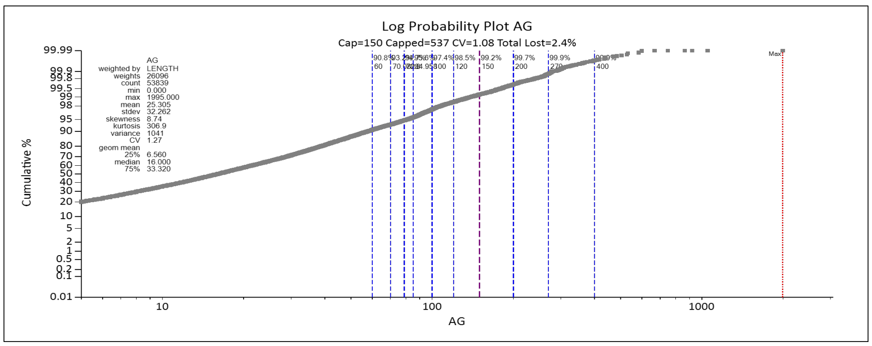
AG		250	151	99.70%	0.30%	1.40%	15%	57099	0	250	26.3	1501924	959.1	1.18
AG		200	283	99.50%	0.50%	2.10%	17%	57099	0	200	26.12	1491513	887.6	1.14
AG		150	603	98.90%	1.10%	3.50%	22%	57099	0	150	25.75	1470182	778.6	1.08
AG		100	1713	97%	3%	6.80%	28%	57099	0	100	24.87	1419745	609.9	0.99
AG		85	2789	95.10%	4.90%	9%	31%	57099	0	85	24.28	1386590	531.7	0.95
AG		75	3696	93.50%	6.50%	11%	34%	57099	0	75	23.72	1354344	468.7	0.91
AG		68	4498	92.10%	7.90%	13%	36%	57099	0	68	23.22	1325675	420.6	0.88
AG		63.5	5128	91%	9%	14%	38%	57099	0	63.5	22.84	1304042	388.3	0.86
AG		60	5737	89.90%	10%	16%	39%	57099	0	60	22.51	1285045	362.3	0.85
AG	AG > 150							603	150	1995	238.8	143977	25014	0.66
AG	AG <= 150							56496	0	150	24.42	1379732	620.3	1.02



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AG								53839	0	1995	25.31	660349	1041	1.27
AG		400	21	99.90%	0.04%	0.30%	6.40%	53839	0	400	25.22	658041	905.7	1.19
AG		270	86	99.90%	0.20%	0.70%	8.50%	53839	0	270	25.14	656007	859.6	1.17
AG		200	234	99.70%	0.40%	1.30%	11%	53839	0	200	24.98	651947	795.9	1.13
AG		150	537	99.20%	1%	2.40%	15%	53839	0	150	24.7	644665	714.1	1.08
AG		120	957	98.50%	1.80%	3.70%	19%	53839	0	120	24.37	635909	641	1.04
AG		100	1607	97.40%	3%	5.20%	22%	53839	0	100	23.98	625819	575.3	1
AG		85	2668	95.60%	5%	7.30%	25%	53839	0	84.96	23.47	612444	505.5	0.96
AG		79	3174	94.70%	5.90%	8.40%	27%	53839	0	78.8	23.17	604630	470.5	0.94
AG		70	4071	93.20%	7.60%	11%	29%	53839	0	70.08	22.65	590931	416.6	0.9
AG		60	5482	91.40%	10.20%	14%	33%	53839	0	60	21.85	570204	349	0.85
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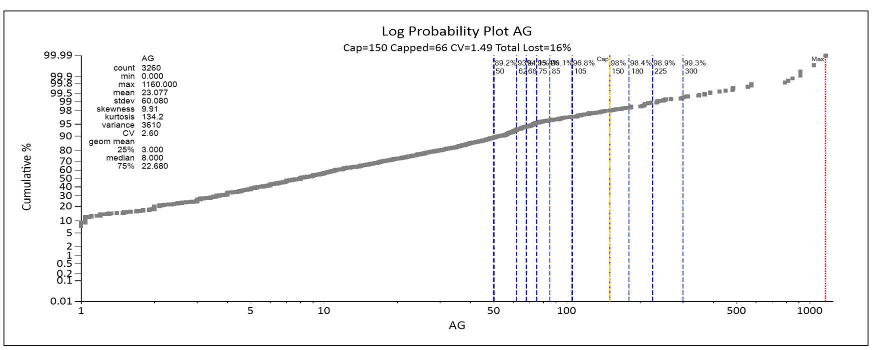

Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
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AG		225	36	98.90%	1.10%	11%	35%	3260	0	225	20.45	66655	1209	1.7
AG		180	53	98.40%	1.60%	14%	39%	3260	0	180	19.83	64646	986.5	1.58
AG		150	66	98%	2%	16%	43%	3260	0	150	19.29	62895	830.8	1.49
AG		105	102	96.80%	3.10%	21%	49%	3260	0	105	18.18	59273	592.9	1.34
AG		85	129	96.10%	4%	24%	52%	3260	0	85	17.49	57006	486.1	1.26
AG		75	152	95.40%	4.70%	26%	53%	3260	0	75	17.07	55631	433.3	1.22
AG		68	190	94.10%	5.80%	28%	54%	3260	0	68	16.69	54422	392.9	1.19
AG		62	226	93%	6.90%	29%	56%	3260	0	62	16.31	53179	355.9	1.16
AG		50	351	89.20%	10.80%	34%	59%	3260	0	50	15.23	49645	269.6	1.08
AG	AG > 150							66	151	1160	336.9	22235	54969	0.7
AG	AG <= 150							3194	0	149.8	16.59	52995	487.6	1.33

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DESIGN: | DRAWN: BP | REVIEWED: BP
SCALE: NOT TO SCALE | DATE: 11/26/2019
FILE: AppendixC_capping2019.xlsx

CAPPING CHARTS 1000Ag

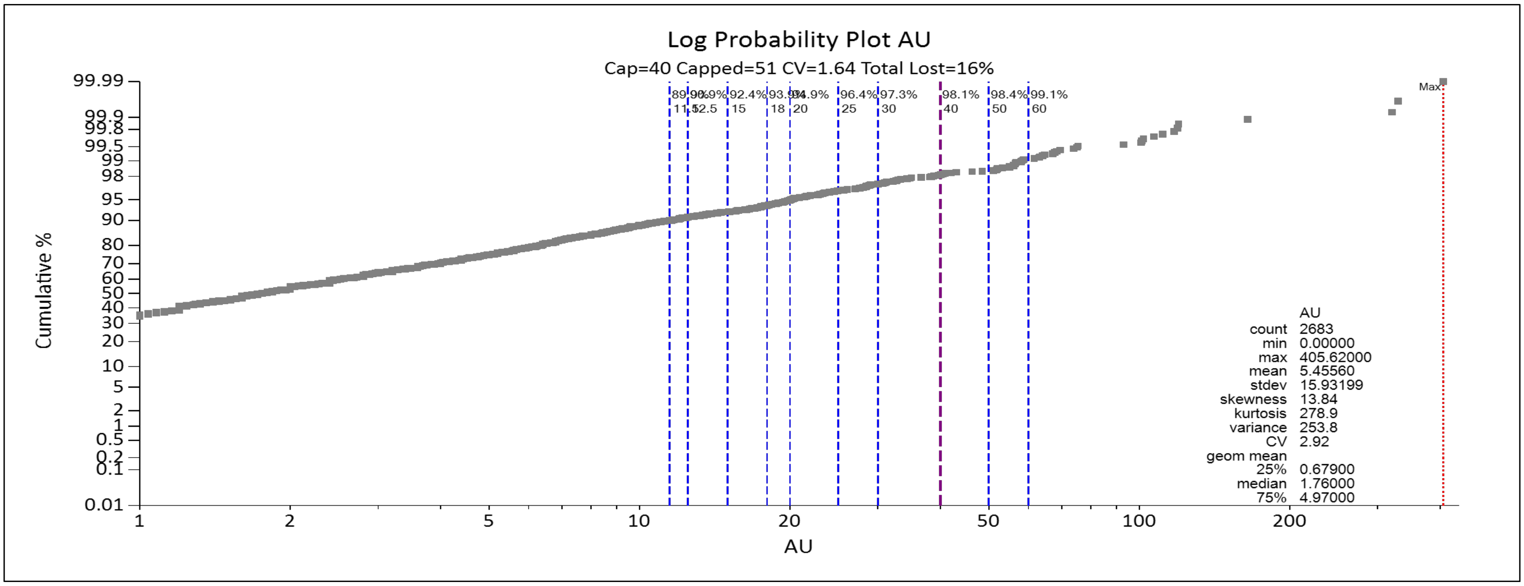
REPORT TITLE:
NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT MARMATO PROJECT

ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

APPENDIX B-2

REVISION NO. **A**

SRK JOB NO.: **544400.020**



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AU								2683	0	405.6	5.46	14637	253.80	2.92
AU		60	25	99.10%	0.90%	10%	36%	2683	0	60	4.89	13131	82.80	1.86
AU		50	43	98.40%	1.60%	13%	40%	2683	0	50	4.77	12788	70.09	1.76
AU		40	51	98.10%	1.90%	16%	44%	2683	0	40	4.60	12336	56.54	1.64
AU		30	75	97.30%	2.80%	20%	49%	2683	0	30	4.37	11734	42.97	1.5
AU		25	96	96.40%	3.60%	23%	52%	2683	0	25	4.21	11305	35.59	1.42
AU		20	136	94.90%	5.10%	27%	55%	2683	0	20	4.00	10739	27.88	1.32
AU		18	163	93.90%	6.10%	29%	56%	2683	0	18	3.89	10437	24.49	1.27
AU		15	205	92.40%	7.60%	33%	59%	2683	0	15	3.68	9878	19.22	1.19
AU		12.5	242	90.90%	9%	36%	62%	2683	0	12.5	3.48	9323	15.02	1.12
AU		11.5	270	89.90%	10.10%	38%	63%	2683	0	11.5	3.38	9066	13.39	1.08
AU	AU > 40							51	40.09	405.6	85.13	4341	5286.00	0.85
AU	AU <= 40							2632	0	39.72	3.91	10296	32.87	1.47

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FILE: AppendixC_capping2019.xlsx

CAPPING CHARTS 3000Au

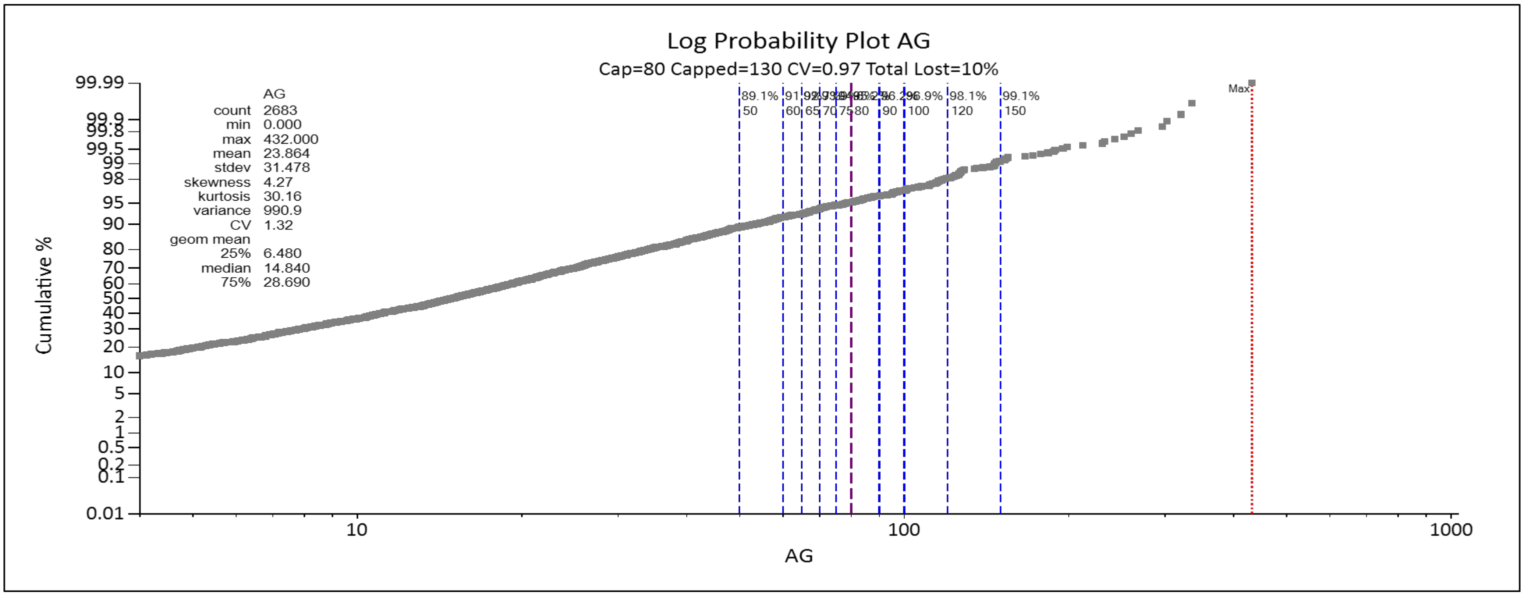
REPORT TITLE:
NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT MARMATO PROJECT

ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

APPENDIX B-3

REVISION NO. **A**

SRK JOB NO.: **544400.020**



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AG								2683	0	432	23.86	64028	990.9	1.32
AG		150	24	99.10%	0.90%	2.90%	12%	2683	0	150	23.17	62160	717.7	1.16
AG		120	51	98.10%	1.90%	4.60%	16%	2683	0	120	22.77	61093	629.8	1.1
AG		100	84	96.90%	3.10%	6.70%	21%	2683	0	100	22.27	59739	542.3	1.05
AG		90	103	96.20%	3.80%	8.20%	23%	2683	0	90	21.92	58801	491.4	1.01
AG		80	130	95.20%	4.80%	10%	26%	2683	0	80	21.49	57650	437.3	0.97
AG		75	144	94.60%	5.40%	11%	28%	2683	0	75	21.23	56961	408.4	0.95
AG		70	164	93.90%	6.10%	12%	30%	2683	0	70	20.95	56201	379.3	0.93
AG		65	196	92.70%	7.30%	14%	31%	2683	0	65	20.62	55309	348.3	0.91
AG		60	217	91.90%	8.10%	15%	33%	2683	0	60	20.23	54285	316.2	0.88
AG		50	292	89.10%	10.90%	19%	38%	2683	0	50	19.28	51729	249.5	0.82
AG	AG > 80							130	80.4	432	129.1	16778	3416	0.45
AG	AG <= 80							2553	0	80	18.51	47250	276.2	0.9

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SCALE: NOT TO SCALE | DATE: 11/26/2019

FILE: AppendixC_capping2019.xlsx

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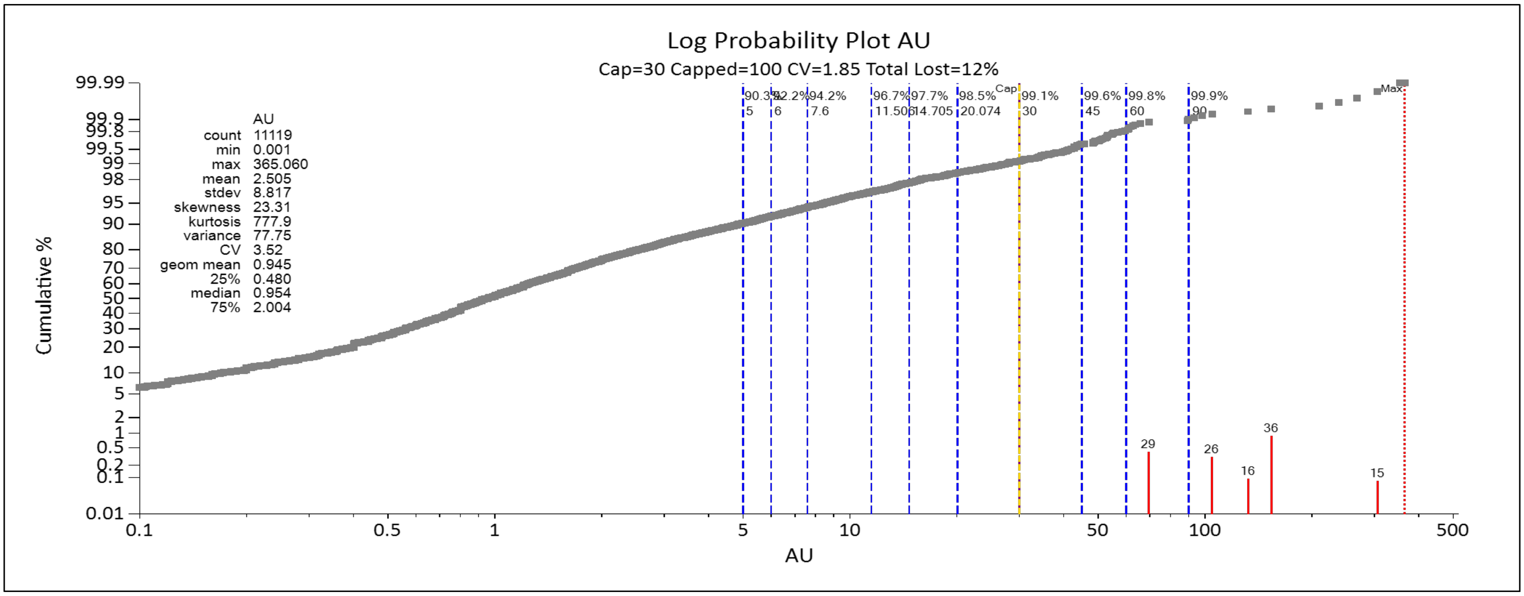
REPORT TITLE: NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT MARMATO PROJECT

ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

APPENDIX B-4

REVISION NO. **A**

SRK JOB NO.: **544400.020**



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AU								11119	0.001	365.1	2.51	27851	77.75	3.52
AU		90.0	11	99.90%	0.10%	4.80%	32%	11119	0.001	90.0	2.39	26518	32.82	2.40
AU		60.0	21	99.80%	0.20%	6.30%	37%	11119	0.001	60.0	2.35	26102	27.45	2.23
AU		45.0	43	99.60%	0.40%	8.00%	41%	11119	0.001	45.0	2.31	25628	23.26	2.09
AU		30.0	100	99.10%	0.90%	12.00%	47%	11119	0.001	30.0	2.21	24576	16.77	1.85
AU		20.1	167	98.50%	1.50%	16.00%	54%	11119	0.001	20.1	2.10	23289	11.57	1.62
AU		14.7	256	97.70%	2.30%	20.00%	58%	11119	0.001	14.7	2.00	22191	8.57	1.47
AU		11.5	367	96.70%	3.30%	24.00%	62%	11119	0.001	11.5	1.91	21204	6.61	1.35
AU		7.6	639	94.20%	5.70%	31.00%	67%	11119	0.001	7.6	1.74	19320	4.05	1.16
AU		6.0	865	92.20%	7.80%	35.00%	70%	11119	0.001	6.0	1.63	18118	2.96	1.06
AU		5.0	1076	90.30%	9.70%	38.00%	72%	11119	0.001	5.0	1.54	17145	2.27	0.98
AU	AU > 30							100	30.08	365.1	62.74	6274	3937.00	1.00
AU	AU <= 30							11019	0.001	29.8	1.96	21576	9.85	1.60

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SCALE: NOT TO SCALE DATE: 11/26/2019

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CAPPING CHARTS 4000Au

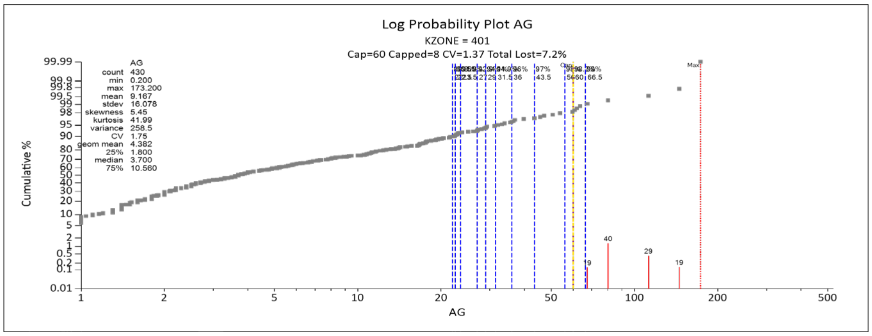
REPORT TITLE:
NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT MARMATO PROJECT

ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

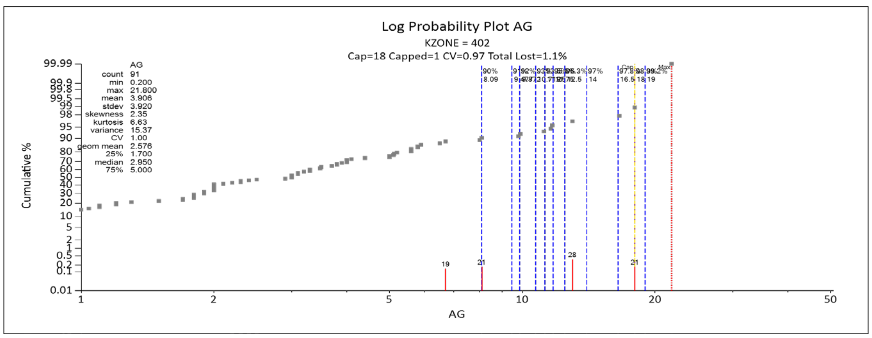
APPENDIX B-5

REVISION NO. **A**

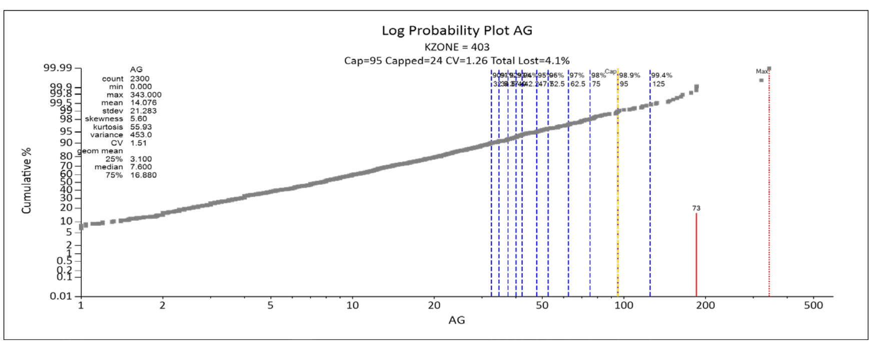
SRK JOB NO.: **544400.020**



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AG	KZONE = 401							430	0.2	173.2	9.17	3942	258.5	1.75
AG	KZONE = 401	66.5	5	99.00%	1.20%	6.20%	20%	430	0.2	66.5	8.59	3696	146	1.41
AG	KZONE = 401	60	8	98.20%	1.90%	7.20%	22%	430	0.2	60	8.50	3657	136.2	1.37
AG	KZONE = 401	56	9	98.00%	2.10%	8.10%	23%	430	0.2	56	8.42	3621	127.9	1.34
AG	KZONE = 401	43.5	13	97.00%	3.00%	12.00%	29%	430	0.2	43.5	8.11	3485	101.9	1.25
AG	KZONE = 401	36	17	96.00%	4.00%	14.00%	33%	430	0.2	36	7.85	3375	85.67	1.18
AG	KZONE = 401	31.5	22	94.90%	5.10%	17.00%	35%	430	0.2	31.5	7.64	3285	74.81	1.13
AG	KZONE = 401	29	25	94.10%	5.80%	18.00%	37%	430	0.2	29	7.50	3225	68.49	1.1
AG	KZONE = 401	27	31	92.90%	7.20%	20.00%	39%	430	0.2	27	7.37	3169	63.05	1.08
AG	KZONE = 401	23.5	35	91.90%	8.10%	23.00%	41%	430	0.2	23.5	7.10	3052	53.28	1.03
AG	KZONE = 401	22.5	41	90.50%	9.50%	24.00%	42%	430	0.2	22.5	7.01	3014	50.47	1.01
AG	KZONE = 401	22	44	89.80%	10.20%	24.00%	43%	430	0.2	22	6.96	2993	48.92	1.01
AG	KZONE = 401 - AG > 60							8	60.8	173.2	95.62	765	1880	0.45
AG	KZONE = 401 - AG <= 60							422	0.2	59.64	7.53	3177	87.41	1.24



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AG	KZONE = 402							91	0.2	21.8	3.91	355.4	15.37	1
AG	KZONE = 402	19	1	99.20%	1.10%	0.80%	2.60%	91	0.2	19	3.88	352.6	14.34	0.98
AG	KZONE = 402	18	1	98.90%	1.10%	1.10%	3.50%	91	0.2	18	3.86	351.6	14.01	0.97
AG	KZONE = 402	16.5	3	97.80%	3.30%	1.90%	5.90%	91	0.2	16.5	3.83	348.5	13.09	0.94
AG	KZONE = 402	14	3	97.00%	3.30%	4.10%	11%	91	0.2	14	3.75	341	11.18	0.89
AG	KZONE = 402	12.5	4	96.30%	4.40%	5.50%	14%	91	0.2	12.5	3.69	336	10.13	0.86
AG	KZONE = 402	11.75	4	95.60%	4.40%	6.30%	16%	91	0.2	11.75	3.66	333	9.57	0.85
AG	KZONE = 402	11.25	6	93.50%	6.60%	7.10%	17%	91	0.2	11.25	3.63	330.2	9.08	0.83
AG	KZONE = 402	10.719	7	93.00%	7.70%	8.10%	19%	91	0.2	10.72	3.59	326.6	8.48	0.81
AG	KZONE = 402	9.872	8	92.00%	8.80%	9.80%	22%	91	0.2	9.872	3.52	320.6	7.59	0.78
AG	KZONE = 402	9.477	9	91.00%	9.90%	11.00%	24%	91	0.2	9.477	3.49	317.1	7.11	0.77
AG	KZONE = 402	8.09	10	90.00%	11.00%	14.00%	29%	91	0.2	8.09	3.35	304.6	5.62	0.71
AG	KZONE = 402 - AG > 18							1	21.8	21.8	21.80	21.8	0	0
AG	KZONE = 402 - AG <= 18							90	0.2	18	3.71	333.6	11.9	0.93



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AG	KZONE = 403							2300	0	343	14.08	32374	453	1.51
AG	KZONE = 403	125	14	99.40%	0.60%	2.40%	12%	2300	0	125	13.74	31596	333.6	1.33
AG	KZONE = 403	95	24	98.90%	1.00%	4.10%	17%	2300	0	95	13.50	31040	287.6	1.26
AG	KZONE = 403	75	45	98.00%	2.00%	6.20%	21%	2300	0	75	13.20	30364	246	1.19
AG	KZONE = 403	62.5	68	97.00%	3.00%	8.40%	25%	2300	0	62.5	12.90	29664	212.3	1.13
AG	KZONE = 403	52.5	92	96.00%	4.00%	11.00%	29%	2300	0	52.5	12.55	28853	180.9	1.07
AG	KZONE = 403	47.7	115	95.00%	5.00%	12.00%	31%	2300	0	47.7	12.33	28361	164.8	1.04
AG	KZONE = 403	42.2	138	94.00%	6.00%	15.00%	34%	2300	0	42.2	12.03	27673	145.2	1
AG	KZONE = 403	40	160	93.00%	7.00%	16.00%	35%	2300	0	40	11.89	27344	136.9	0.98
AG	KZONE = 403	37.4	184	92.00%	8.00%	17.00%	36%	2300	0	37.4	11.69	26891	126.3	0.96
AG	KZONE = 403	34.6	207	91.00%	9.00%	19.00%	38%	2300	0	34.6	11.45	26334	114.5	0.93
AG	KZONE = 403	32.5	230	90.00%	10.00%	20.00%	40%	2300	0	32.5	11.25	25869	105.5	0.91
AG	KZONE = 403 - AG > 95							24	96	343	150.60	3614	3882	0.41
AG	KZONE = 403 - AG <= 95							2276	0	95	12.64	28760	219.8	1.17

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FILE: AppendixC_capping2019.xlsx

CAPPING CHARTS 4000Ag

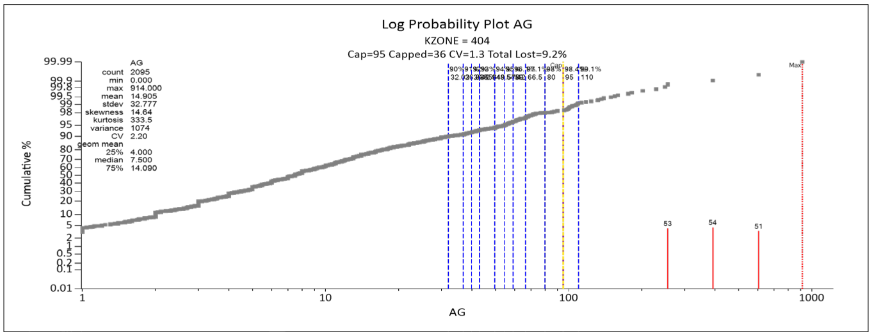
REPORT TITLE:
NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT MARMATO PROJECT

ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

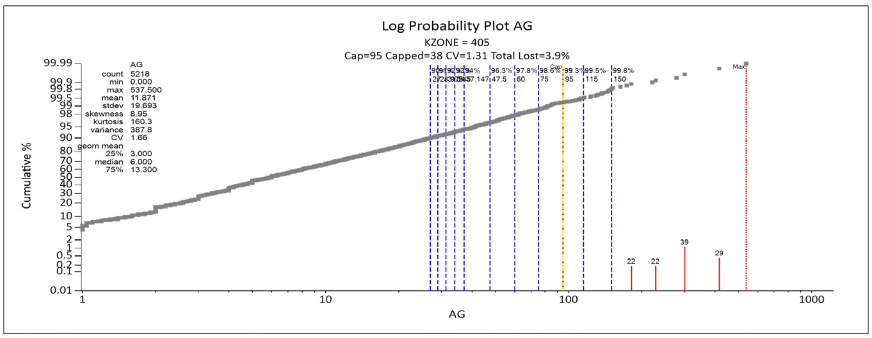
APPENDIX B-6

REVISION NO. **A**

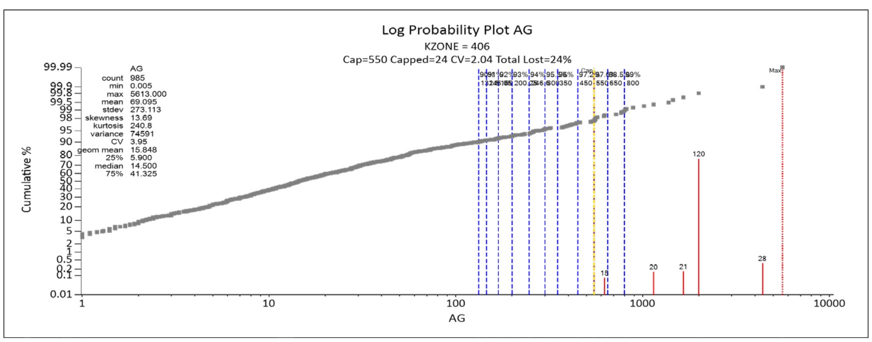
SRK JOB NO.: **544400.020**



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AG	KZONE = 404							2095	0	914	14.91	31226	1074	2.2
AG	KZONE = 404	110	19	99.10%	0.90%	7.90%	38%	2095	0	110	13.73	28753	345.6	1.35
AG	KZONE = 404	95	36	98.40%	1.70%	9.20%	41%	2095	0	95	13.53	28345	311.4	1.3
AG	KZONE = 404	80	42	98.00%	2.00%	11.00%	44%	2095	0	80	13.25	27764	270.3	1.24
AG	KZONE = 404	66.5	62	97.10%	3.00%	13.00%	46%	2095	0	66.5	12.94	27113	233.2	1.18
AG	KZONE = 404	59	81	96.10%	3.90%	15.00%	48%	2095	0	59	12.69	26578	207.8	1.14
AG	KZONE = 404	54.42	105	95.00%	5.00%	16.00%	50%	2095	0	54.42	12.48	26146	189.6	1.1
AG	KZONE = 404	49.578	126	94.00%	6.00%	18.00%	52%	2095	0	49.58	12.21	25589	168.5	1.06
AG	KZONE = 404	42.948	147	93.00%	7.00%	21.00%	54%	2095	0	42.95	11.79	24697	139.4	1
AG	KZONE = 404	39.856	168	92.00%	8.00%	22.00%	56%	2095	0	39.86	11.56	24214	125.7	0.97
AG	KZONE = 404	36.835	189	91.00%	9.00%	24.00%	57%	2095	0	36.84	11.30	23678	111.9	0.94
AG	KZONE = 404	32.02	210	90.00%	10.00%	27.00%	60%	2095	0	32.02	10.85	22722	90.67	0.88
AG	KZONE = 404 - AG > 95							36	96	914	175.00	6301	25763	0.92
AG	KZONE = 404 - AG <= 95							2059	0	90	12.11	24925	198.7	1.16



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AG	KZONE = 405							5218	0	537.5	11.87	61941	387.8	1.66
AG	KZONE = 405	150	11	99.80%	0.20%	1.90%	14%	5218	0	150	11.65	60789	274.4	1.42
AG	KZONE = 405	115	26	99.50%	0.50%	2.90%	18%	5218	0	115	11.53	60140	244.9	1.36
AG	KZONE = 405	95	38	99.30%	0.70%	3.90%	21%	5218	0	95	11.40	59500	222.1	1.31
AG	KZONE = 405	75	71	98.60%	1.40%	5.60%	25%	5218	0	75	11.21	58497	194.2	1.24
AG	KZONE = 405	60	113	97.80%	2.20%	7.70%	29%	5218	0	60	10.96	57164	165.7	1.17
AG	KZONE = 405	47.5	196	96.30%	3.80%	11.00%	34%	5218	0	47.5	10.60	55302	135.4	1.1
AG	KZONE = 405	37.147	314	94.00%	6.00%	15.00%	39%	5218	0	37.15	10.11	52732	104.3	1.01
AG	KZONE = 405	34	362	93.00%	6.90%	17.00%	41%	5218	0	34	9.90	51670	93.95	0.98
AG	KZONE = 405	31.245	418	92.00%	8.00%	18.00%	43%	5218	0	31.25	9.69	50581	84.43	0.95
AG	KZONE = 405	28.975	470	91.00%	9.00%	20.00%	44%	5218	0	28.98	9.50	49574	76.52	0.92
AG	KZONE = 405	27	519	90.00%	9.90%	22.00%	46%	5218	0	27	9.31	48601	69.59	0.9
AG	KZONE = 405 - AG > 95							38	95.8	537.5	159.20	6051	8129	0.57
AG	KZONE = 405 - AG <= 95							5180	0	92.92	10.79	55890	172.1	1.22



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AG	KZONE = 406							985	0.005	5613	69.10	68059	74591	3.95
AG	KZONE = 406	800	11	99.00%	1.10%	18.00%	43%	985	0.005	800	56.55	55703	16315	2.26
AG	KZONE = 406	650	15	98.50%	1.50%	21.00%	46%	985	0.005	650	54.42	53605	13475	2.13
AG	KZONE = 406	550	24	97.60%	2.40%	24.00%	48%	985	0.005	550	52.62	51826	11510	2.04
AG	KZONE = 406	450	28	97.20%	2.80%	28.00%	52%	985	0.005	450	49.91	49163	9086	1.91
AG	KZONE = 406	350	40	96.00%	4.10%	33.00%	56%	985	0.005	350	46.39	45693	6627	1.75
AG	KZONE = 406	300	44	95.50%	4.50%	36.00%	58%	985	0.005	300	44.30	43636	5459	1.67
AG	KZONE = 406	246.6	60	94.00%	6.10%	40.00%	61%	985	0.005	246.6	41.57	40946	4206	1.56
AG	KZONE = 406	200.25	69	93.00%	7.00%	44.00%	63%	985	0.005	200.3	38.57	37994	3110	1.45
AG	KZONE = 406	169	78	92.00%	7.90%	48.00%	66%	985	0.005	169	36.23	35681	2419	1.36
AG	KZONE = 406	146.05	89	91.00%	9.00%	50.00%	68%	985	0.005	146.1	34.24	33722	1932	1.28
AG	KZONE = 406	132.5	99	90.00%	10.10%	52.00%	69%	985	0.005	132.5	32.95	32458	1661	1.24
AG	KZONE = 406 - AG > 550							24	554	5613	1226.00	29433	1530930	1.01
AG	KZONE = 406 - AG <= 550							961	0.005	545	40.19	38626	5459	1.84

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DESIGN: | DRAWN: BP | REVIEWED: BP

SCALE: NOT TO SCALE | DATE: 11/26/2019

FILE: AppendixC_capping2019.xlsx

CAPPING CHARTS 4000Ag

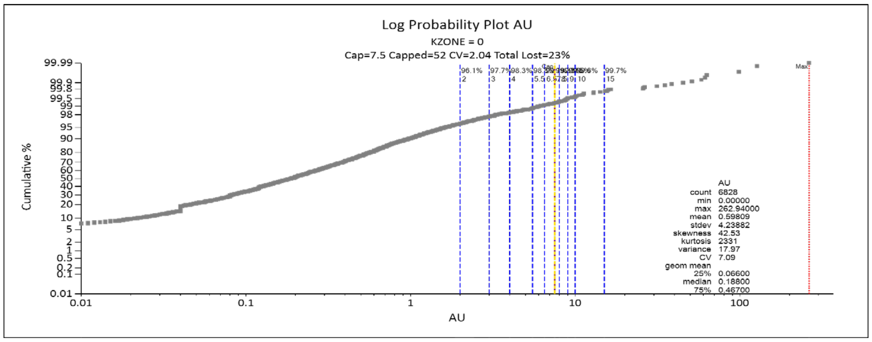
REPORT TITLE:
NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT MARMATO PROJECT

ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

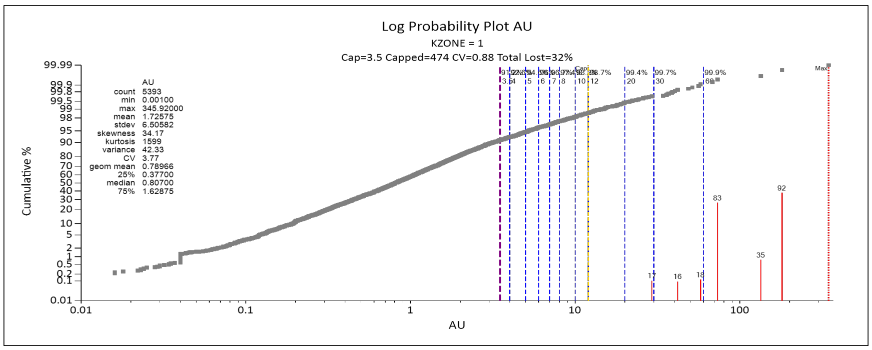
APPENDIX B-7

REVISION NO. **A**

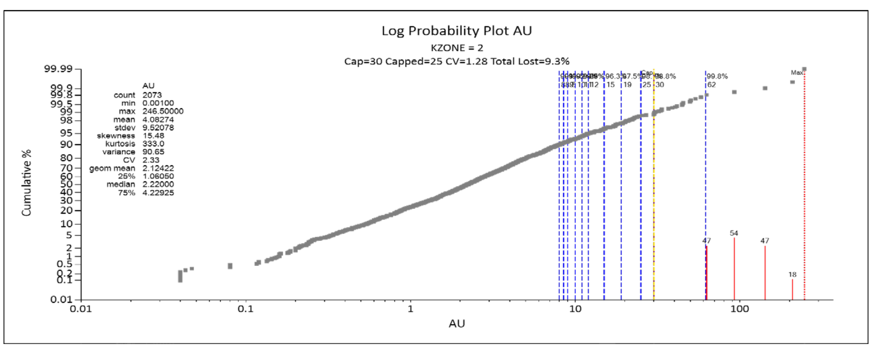
SRK JOB NO.: **544400.020**



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AU	KZONE = 0							6828	0	262.9	0.598	4084	17.97	7.09
AU	KZONE = 0	15	18	99.70%	0.30%	18%	65%	6828	0	15	0.489	3340	1.47	2.48
AU	KZONE = 0	10	29	99.60%	0.40%	21%	69%	6828	0	10	0.474	3234	1.11	2.22
AU	KZONE = 0	9	34	99.50%	0.50%	22%	70%	6828	0	9	0.469	3203	1.02	2.16
AU	KZONE = 0	8	45	99.30%	0.70%	23%	71%	6828	0	8	0.463	3161	0.93	2.08
AU	KZONE = 0	7.5	52	99.30%	0.80%	23%	71%	6828	0	7.5	0.459	3137	0.87	2.04
AU	KZONE = 0	6.5	62	99.10%	0.90%	25%	73%	6828	0	6.5	0.451	3080	0.77	1.94
AU	KZONE = 0	5.5	82	98.70%	1.20%	26%	74%	6828	0	5.5	0.441	3010	0.65	1.83
AU	KZONE = 0	4	115	98.30%	1.70%	30%	77%	6828	0	4	0.419	2864	0.47	1.63
AU	KZONE = 0	3	159	97.70%	2.30%	33%	79%	6828	0	3	0.4	2732	0.35	1.48
AU	KZONE = 0	2	267	96.10%	3.90%	38%	82%	6828	0	2	0.371	2531	0.23	1.29
AU	KZONE = 0 - AU > 7.5							52	7.52	262.9	25.71	1337	1691	1.6
AU	KZONE = 0 - AU <= 7.5							6776	0	7.44	0.405	2747	0.5	1.74



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AU	KZONE = 1							5393	0.001	345.9	1.726	9307	42.33	3.77
AU	KZONE = 1	60	5	99.90%	0.10%	5.40%	42%	5393	0.001	60	1.633	8806	12.58	2.17
AU	KZONE = 1	30	17	99.70%	0.30%	8.70%	52%	5393	0.001	30	1.576	8499	7.99	1.79
AU	KZONE = 1	20	30	99.40%	0.60%	11%	57%	5393	0.001	20	1.534	8273	6.07	1.61
AU	KZONE = 1	12	74	98.70%	1.40%	15%	63%	5393	0.001	12	1.463	7887	4.07	1.38
AU	KZONE = 1	10	100	98.20%	1.90%	17%	65%	5393	0.001	10	1.431	7717	3.47	1.3
AU	KZONE = 1	8	138	97.40%	2.60%	20%	68%	5393	0.001	8	1.387	7478	2.8	1.21
AU	KZONE = 1	7	176	96.70%	3.30%	21%	69%	5393	0.001	7	1.358	7322	2.45	1.15
AU	KZONE = 1	6	216	96%	4%	23%	71%	5393	0.001	6	1.321	7125	2.07	1.09
AU	KZONE = 1	5	292	94.60%	5.40%	26%	73%	5393	0.001	5	1.275	6874	1.68	1.02
AU	KZONE = 1	4	398	92.60%	7.40%	30%	75%	5393	0.001	4	1.211	6529	1.27	0.93
AU	KZONE = 1 - AU > 3.5							474	3.508	345.9	9.821	4655	404.3	2.05
AU	KZONE = 1 - AU <= 3.5							4919	0.001	3.5	0.946	4652	0.59	0.81



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AU	KZONE = 2							2073	0.001	246.5	4.083	8464	90.65	2.33
AU	KZONE = 2	62	5	99.80%	0.20%	5.20%	34%	2073	0.001	62	3.869	8021	35.45	1.54
AU	KZONE = 2	30	25	98.80%	1.20%	9.30%	45%	2073	0.001	30	3.701	7673	22.61	1.28
AU	KZONE = 2	25	28	98.70%	1.40%	11%	48%	2073	0.001	25	3.637	7540	19.57	1.22
AU	KZONE = 2	19	52	97.50%	2.50%	14%	52%	2073	0.001	19	3.526	7310	15.55	1.12
AU	KZONE = 2	15	76	96.30%	3.70%	17%	56%	2073	0.001	15	3.405	7059	12.3	1.03
AU	KZONE = 2	12	104	95%	5%	20%	59%	2073	0.001	12	3.274	6788	9.66	0.95
AU	KZONE = 2	11	126	93.90%	6.10%	21%	61%	2073	0.001	11	3.219	6674	8.75	0.92
AU	KZONE = 2	10	148	92.90%	7.10%	23%	62%	2073	0.001	10	3.153	6537	7.79	0.89
AU	KZONE = 2	9	174	91.60%	8.40%	25%	64%	2073	0.001	9	3.077	6379	6.82	0.85
AU	KZONE = 2	8	207	90%	10%	27%	65%	2073	0.001	8	2.986	6190	5.82	0.81
AU	KZONE = 2 - AU > 30							25	30.09	246.5	61.64	1541	3110	0.9
AU	KZONE = 2 - AU <= 30							2048	0.001	29.88	3.38	6923	14.33	1.12

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SCALE: NOT TO SCALE | DATE: 11/26/2019

FILE: AppendixC_capping2019.xlsx

CAPPING CHARTS 5000Au

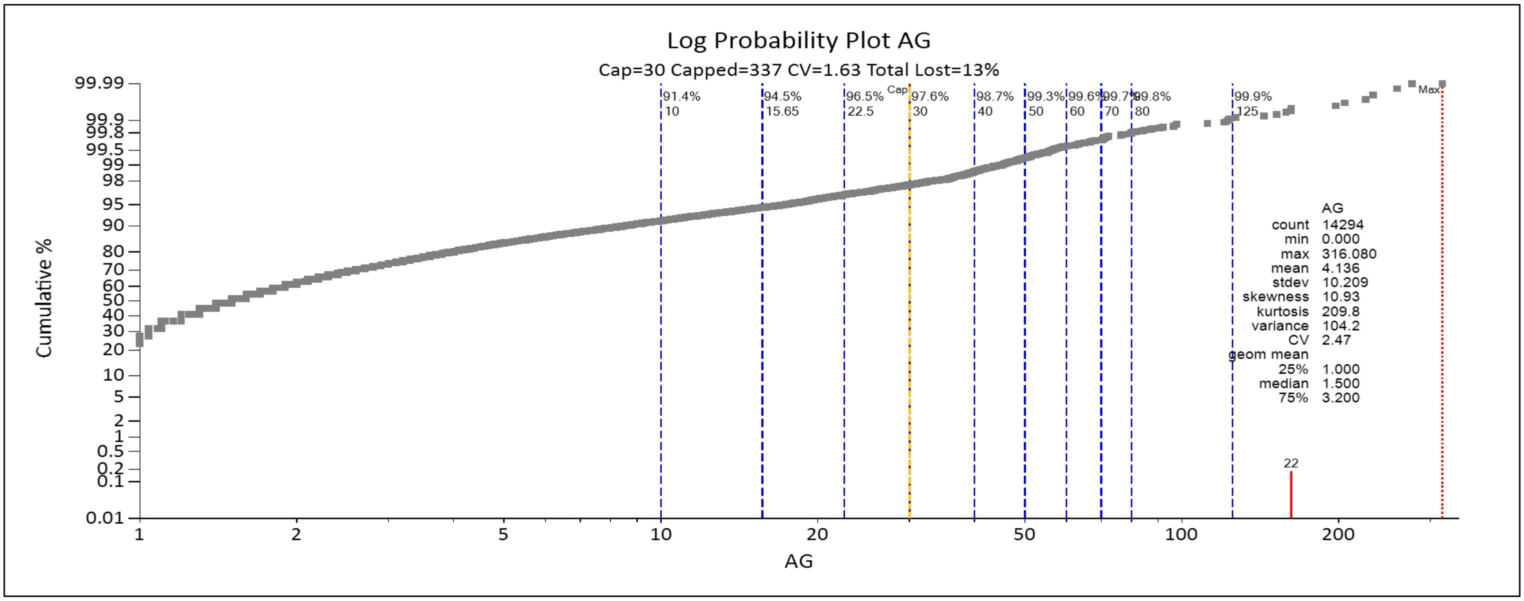
REPORT TITLE:
NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT MARMATO PROJECT

ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

APPENDIX B-8

REVISION NO. **A**

SRK JOB NO.: **544400.020**



Column	_Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max	Mean	Total	Variance	CV
AG								14,294	0	316.1	4.14	59,121	104.2	2.47
AG		125.0	13	99.9%	0.10%	1.70%	11%	14,294	0	125	4.07	58,128	79.33	2.19
AG		80.0	28	99.8%	0.20%	3.20%	17%	14,294	0	80	4.01	57,243	67.42	2.05
AG		70.0	43	99.7%	0.30%	3.80%	19%	14,294	0	70	3.98	56,902	64.04	2.01
AG		60.0	59	99.6%	0.40%	4.60%	21%	14,294	0	60	3.95	56,402	59.79	1.96
AG		50.0	103	99.3%	0.70%	5.90%	23%	14,294	0	50	3.89	55,622	54.26	1.89
AG		40.0	192	98.7%	1.30%	8.30%	27%	14,294	0	40	3.79	54,190	46.12	1.79
AG		30.0	337	97.6%	2.40%	13%	34%	14,294	0	30	3.61	51,534	34.64	1.63
AG		22.5	497	96.0%	3.50%	18%	40%	14,294	0	22.5	3.39	48,465	24.98	1.47
AG		15.7	785	94.5%	5.50%	25%	48%	14,294	0	15.65	3.09	44,142	15.56	1.28
AG		10.0	1219	91.4%	8.50%	35%	57%	14,294	0	10	2.71	38,669	8.12	1.05
AG	AG > 30							337	30.1	316.1	52.51	17,697	1271	0.68
AG	AG <= 30							13,957	0	29.88	2.97	41,424	18.25	1.44

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DESIGN:	DRAWN: BP	REVIEWED: BP
SCALE: NOT TO SCALE		DATE: 11/26/2019
FILE: AppendixC_capping2019.xlsx		

CAPPING CHARTS 5000Ag

REPORT TITLE:
NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT MARMATO PROJECT

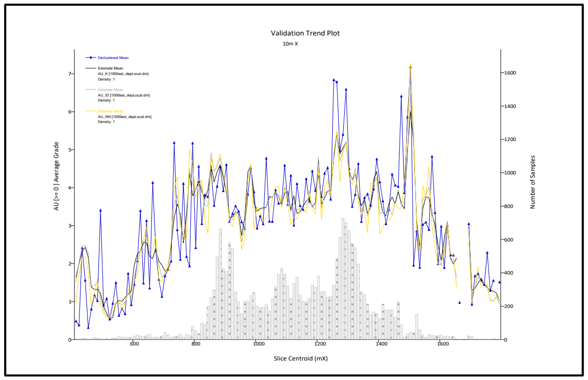
ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

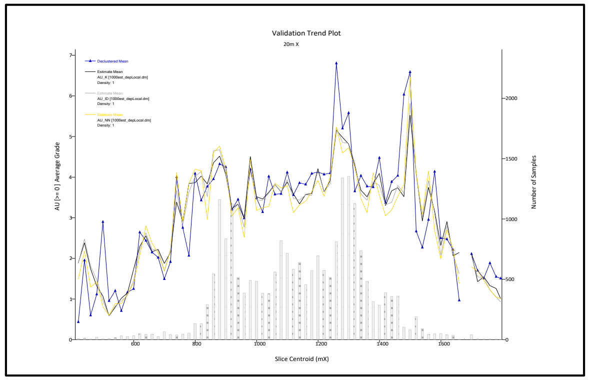
APPENDIX B-9

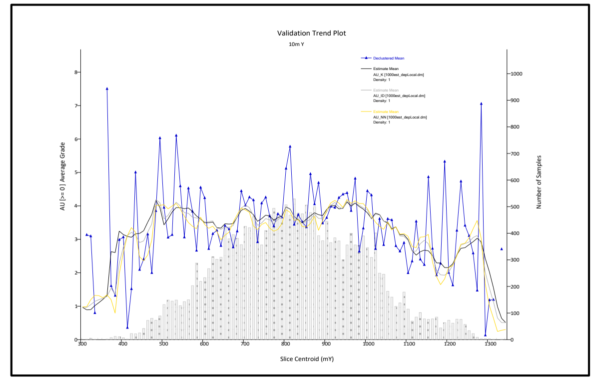
REVISION NO. **A**

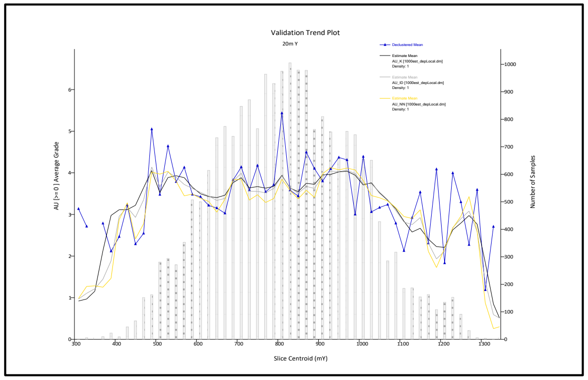
SRK JOB NO.: **544400.020**

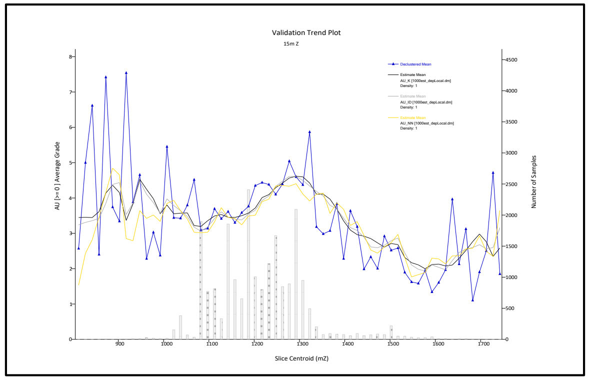
Appendix C: Swath Plots











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DESIGN:	DRAWN: BP	REVIEWED: BP
SCALE: NOT TO SCALE		DATE: 11/25/2019
FILE: Appendix-B-validation_stats_swath.xlsx		

SWATH PLOTS
1000Au

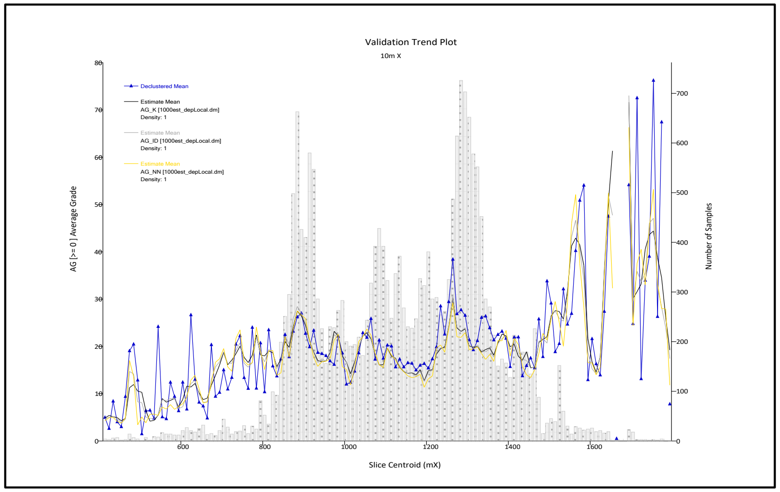
REPORT TITLE:
NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT
MARMATO PROJECT

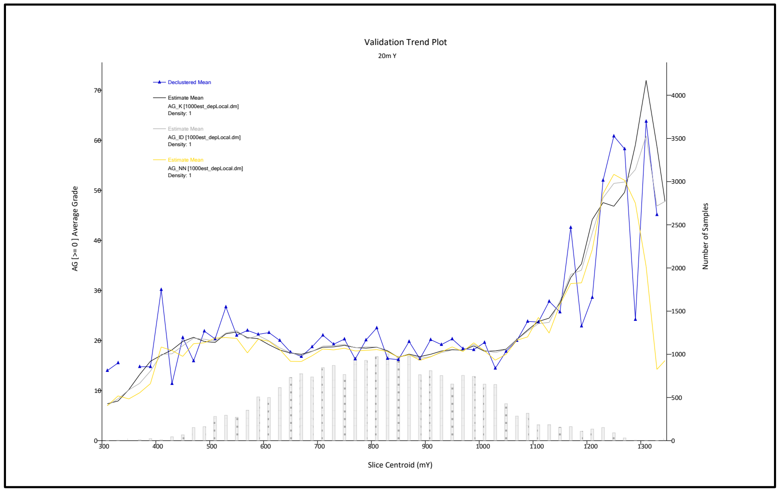
ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

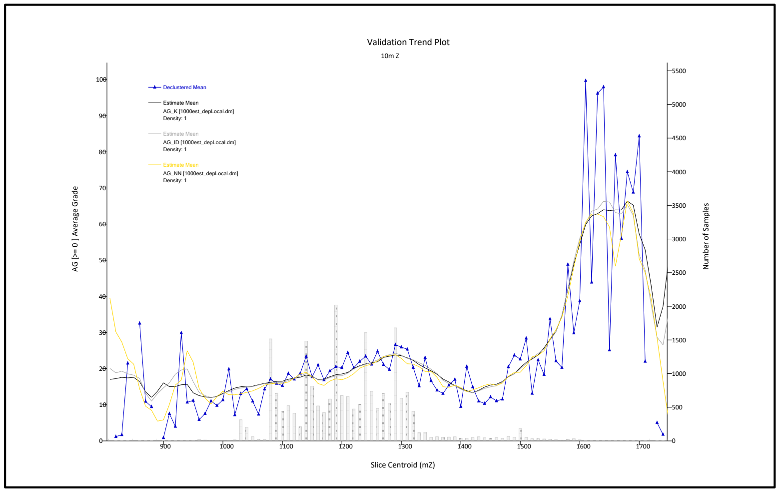
APPENDIX C-1

REVISION NO. **A**

SRK JOB NO.: **544400.020**







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DESIGN:	DRAWN: BP
REVIEWED: BP	
SCALE: NOT TO SCALE	DATE: 11/25/2019
FILE: Appendix-B-validation_stats_swath.xlsx	

SWATH PLOTS
1000Ag

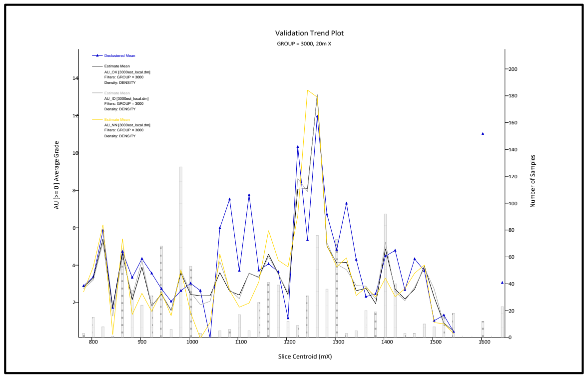
REPORT TITLE:
NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT
MARMATO PROJECT

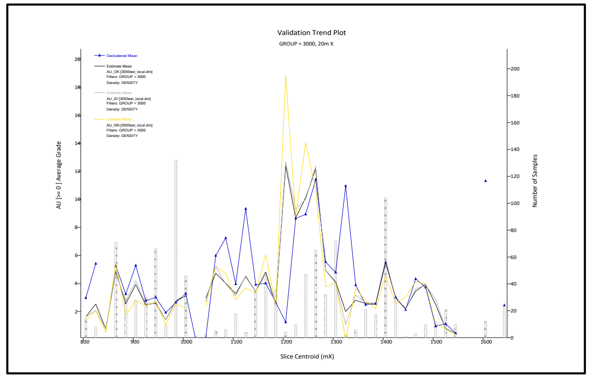
ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

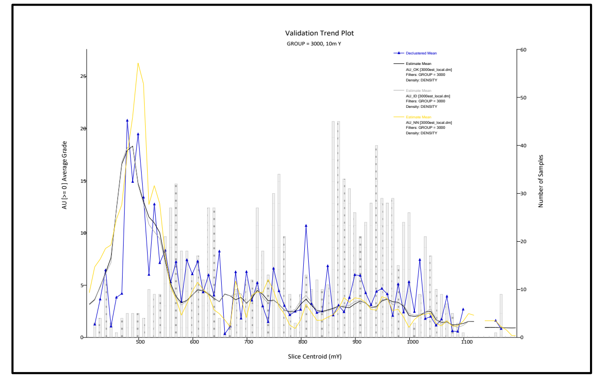
APPENDIX C-2

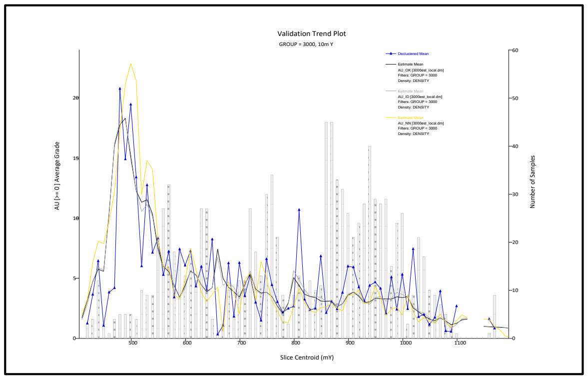
REVISION NO. **A**

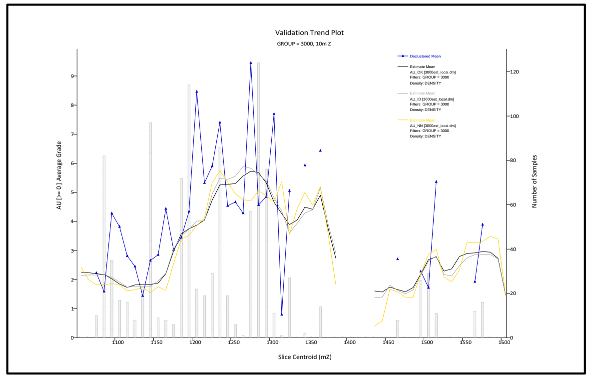
SRK JOB NO.: **544400.020**

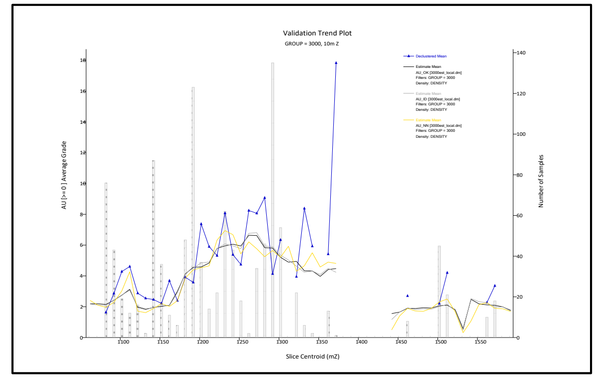












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DESIGN:	DRAWN: BP
REVIEWED: BP	
SCALE: NOT TO SCALE	DATE: 11/25/2019
FILE: Appendix-B-validation_stats_swath.xlsx	

SWATH PLOTS
3000Au

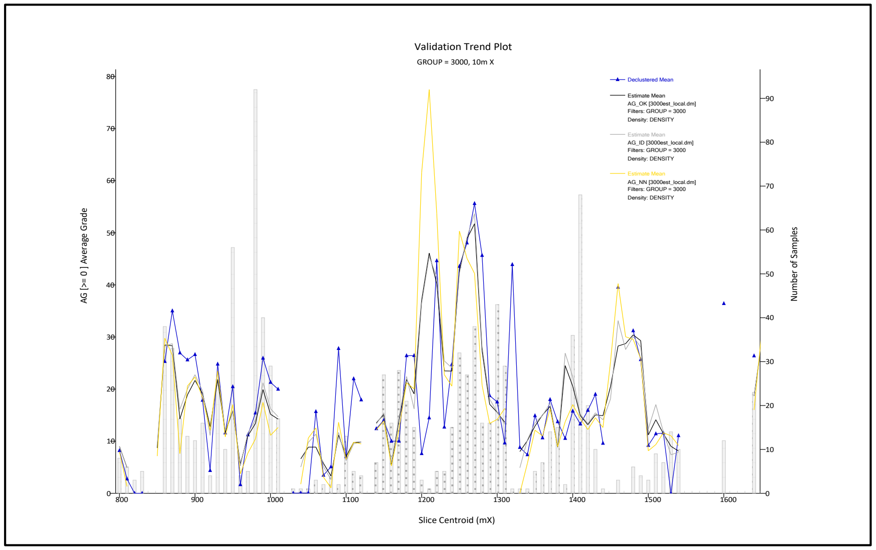
REPORT TITLE:
NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT
MARMATO PROJECT

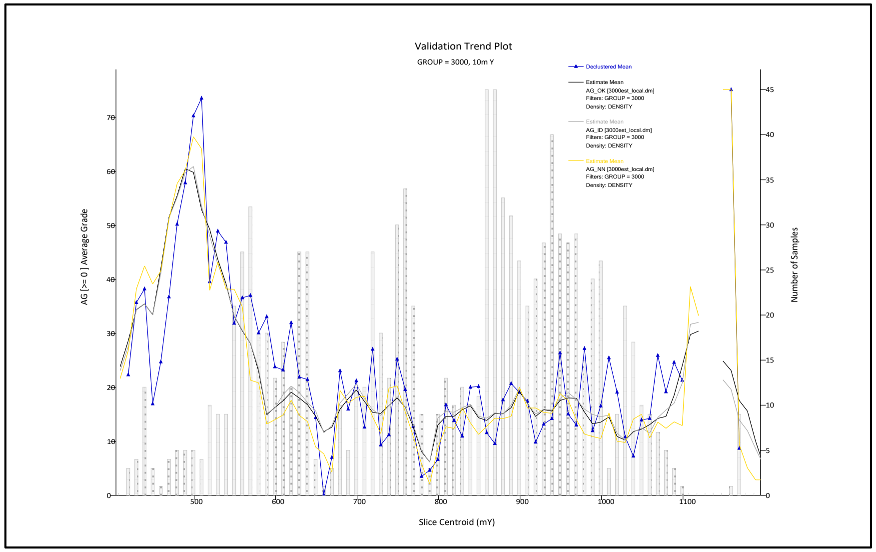
ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

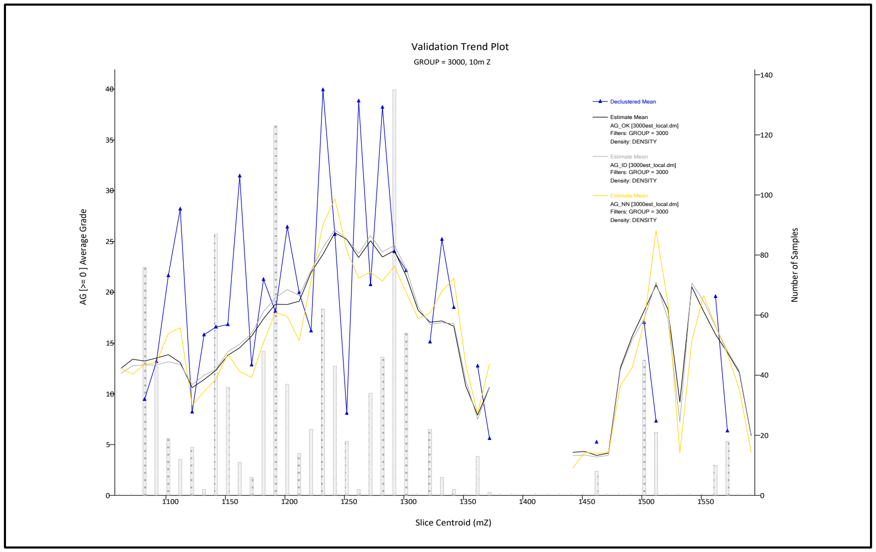
APPENDIX C-3

REVISION NO. **A**

SRK JOB NO.: **544400.020**







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DESIGN:	DRAWN: BP	REVIEWED: BP
SCALE: NOT TO SCALE		DATE: 11/25/2019
FILE: Appendix-B-validation_stats_swath.xlsx		

SWATH PLOTS
3000Ag

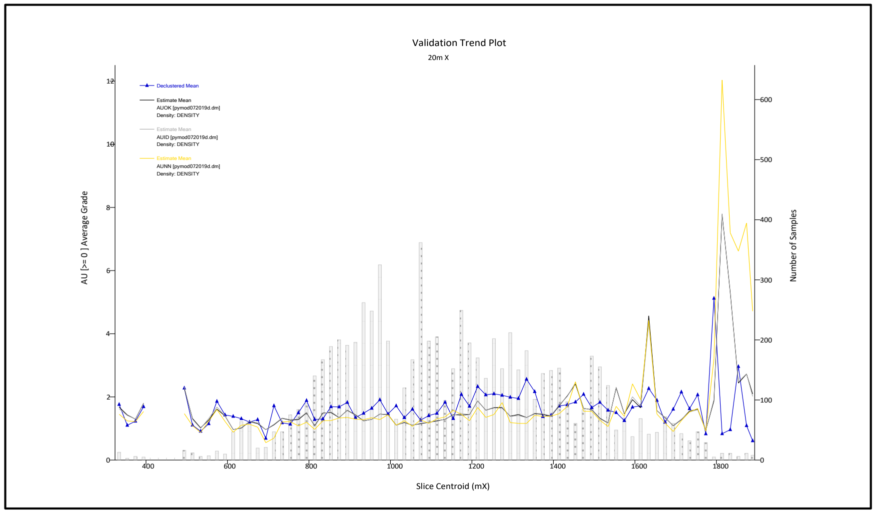
REPORT TITLE:
NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT
MARMATO PROJECT

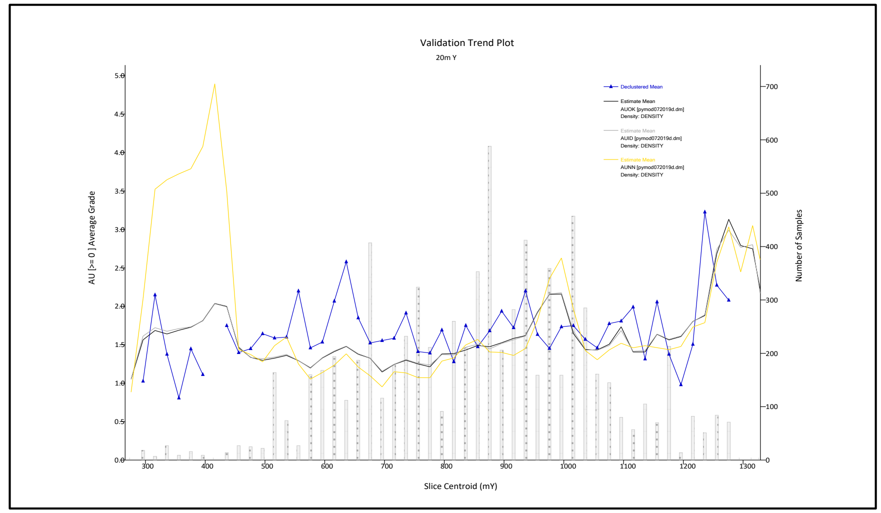
ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

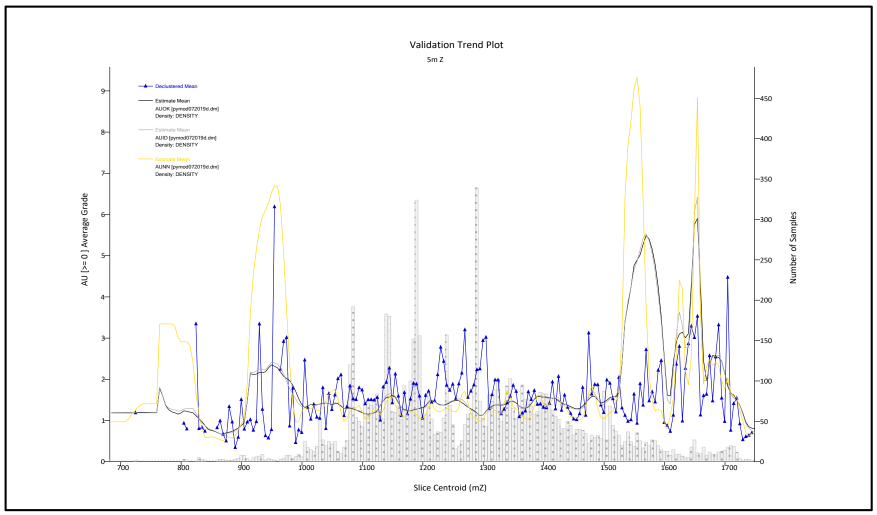
APPENDIX C-4

REVISION NO. **A**

SRK JOB NO.: **544400.020**







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DESIGN: | DRAWN: BP | REVIEWED: BP

SCALE: NOT TO SCALE | DATE: 11/25/2019

FILE: Appendix-B-validation_stats_swath.xlsx

SWATH PLOTS
4000Au

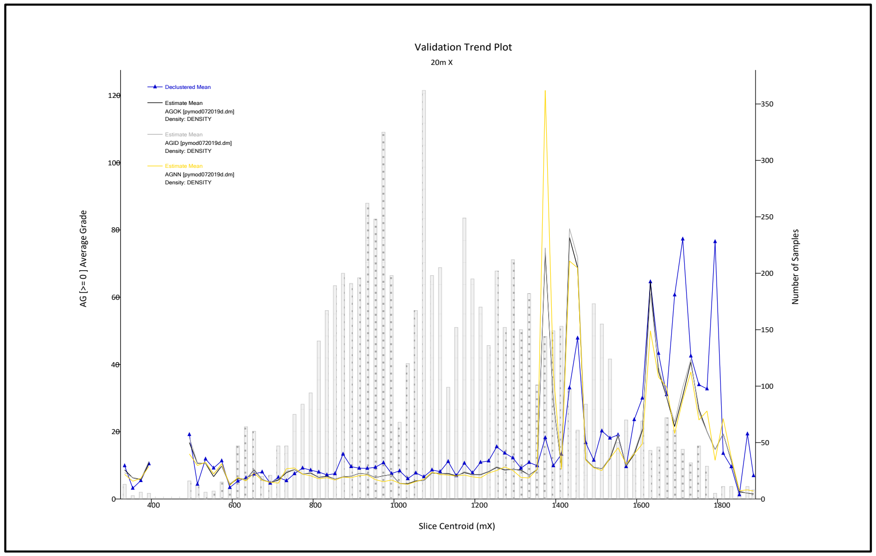
REPORT TITLE:
NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT
MARMATO PROJECT

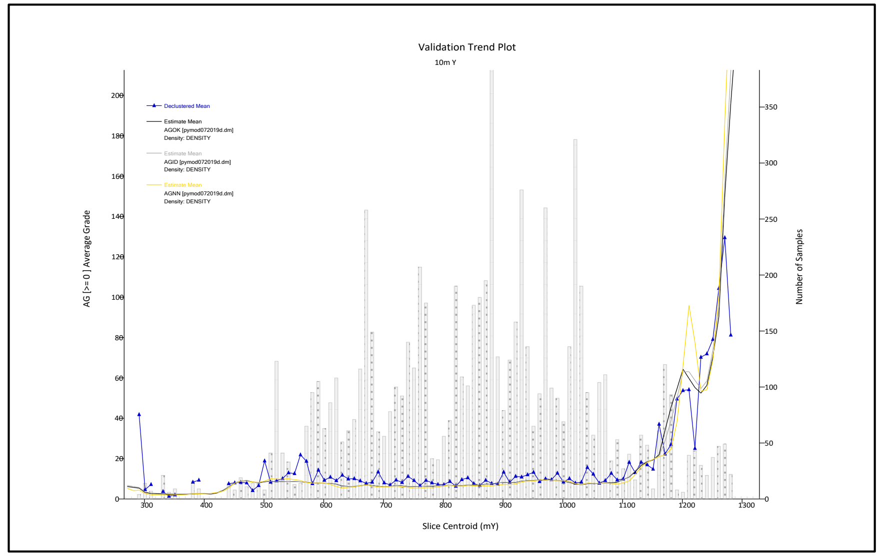
ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

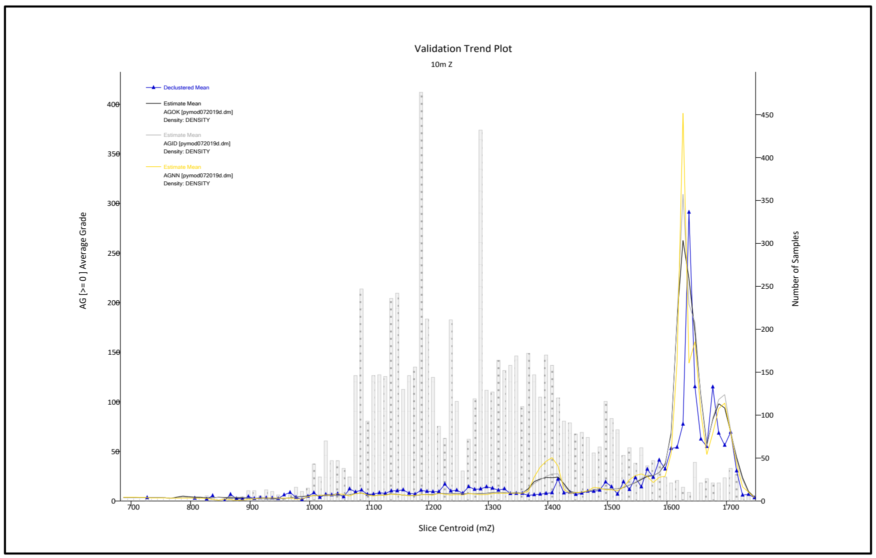
APPENDIX C-5

REVISION NO. **A**

SRK JOB NO.: **544400.020**







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DESIGN:	DRAWN: BP	REVIEWED: BP
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FILE: Appendix-B-validation_stats_swath.xlsx		

SWATH PLOTS
4000Ag

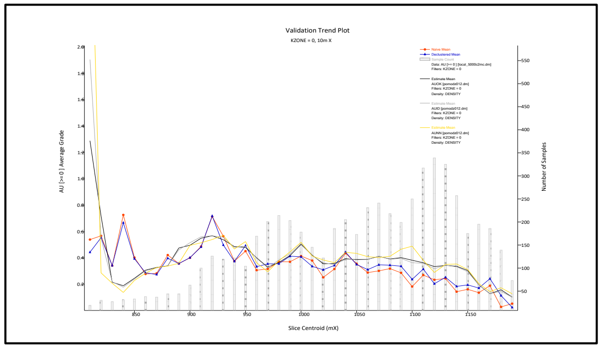
REPORT TITLE:
NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT
MARMATO PROJECT

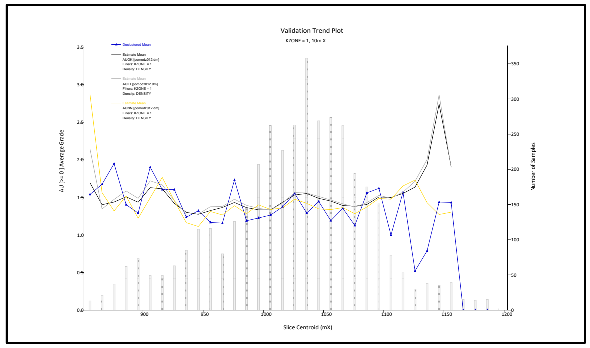
ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

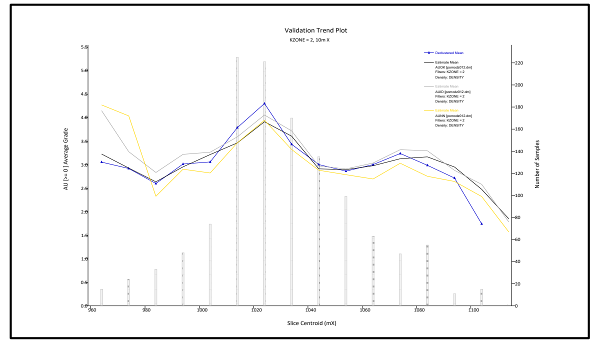
APPENDIX C-6

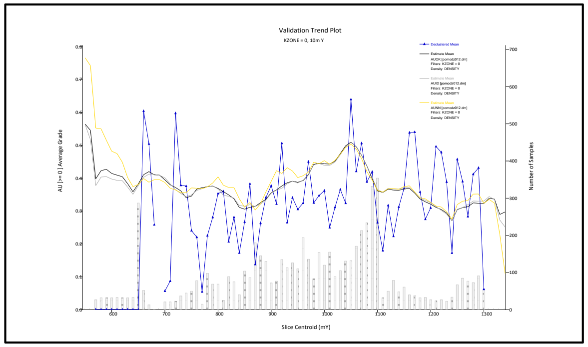
REVISION NO. **A**

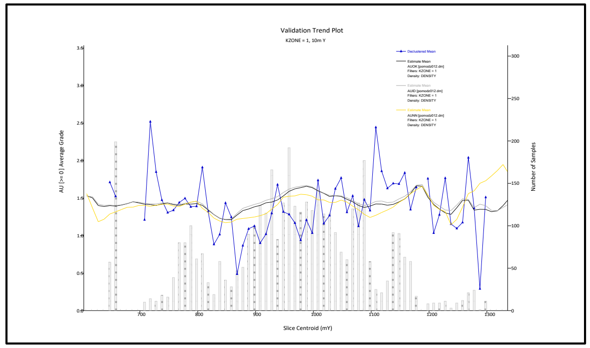
SRK JOB NO.: **544400.020**

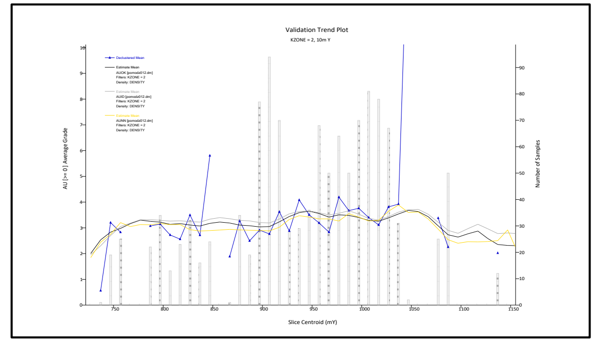


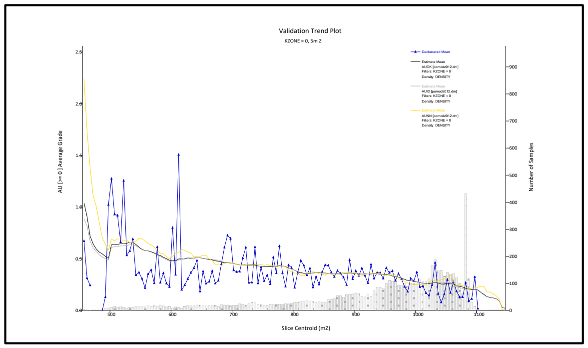


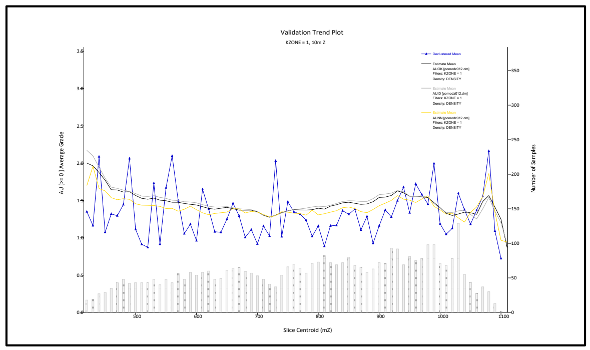


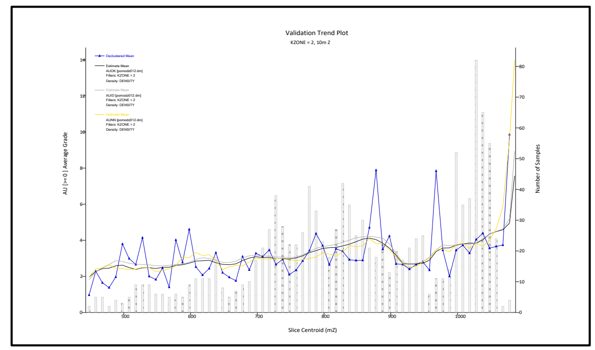












GRANCOLOMBIAGOLD

srk consulting

DESIGN: | DRAWN: BP | REVIEWED: BP

SCALE: NOT TO SCALE | DATE: 11/25/2019

FILE: Appendix-B-validation_stats_swath.xlsx

SWATH PLOTS
5000Au

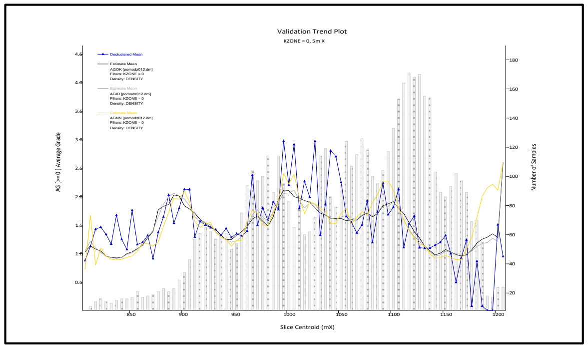
REPORT TITLE:
NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT
MARMATO PROJECT

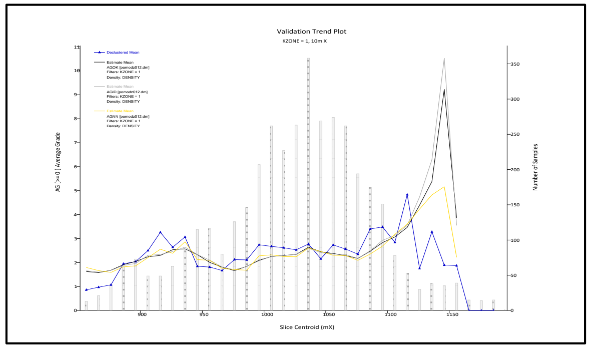
ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

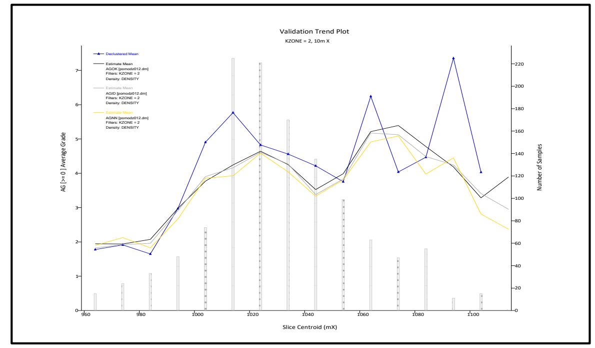
APPENDIX C-7

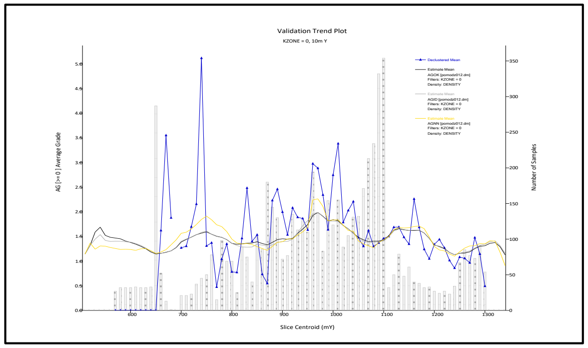
REVISION NO. **A**

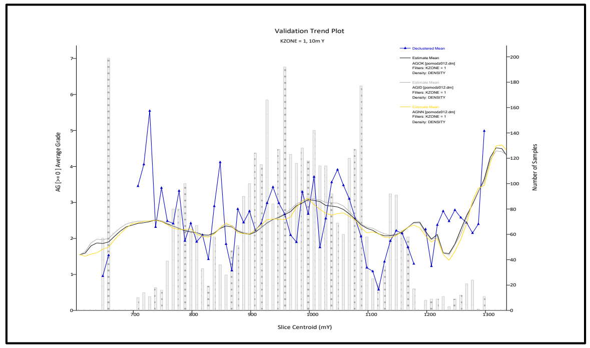
SRK JOB NO.: **544400.020**

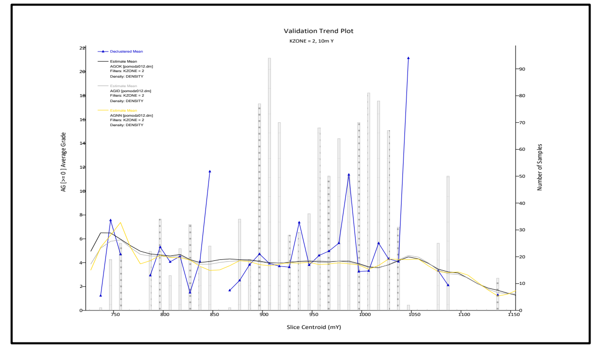


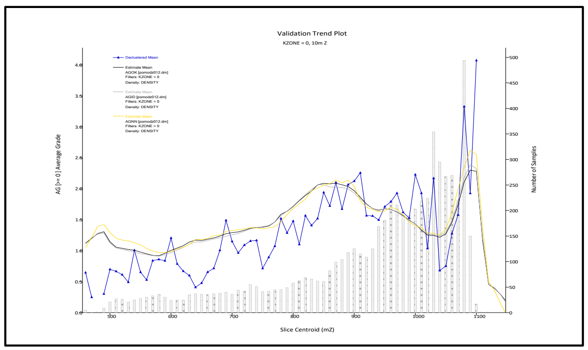


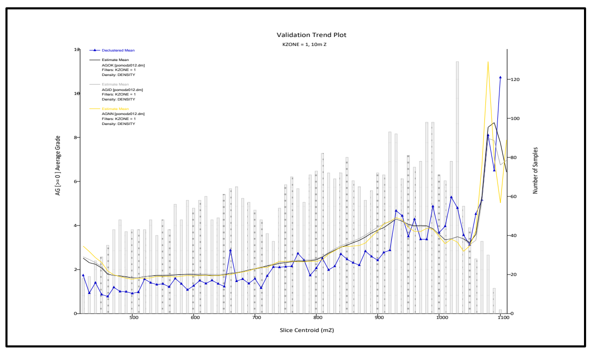


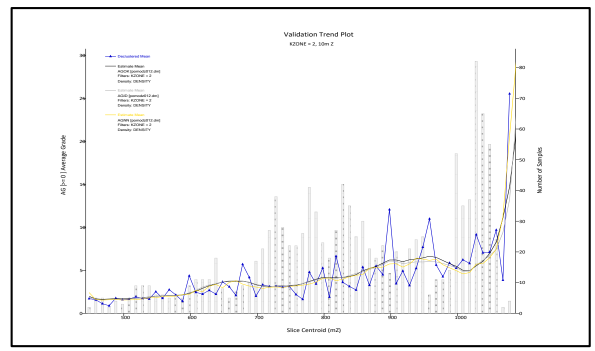












GRANCOLOMBIAGOLD

srk consulting

DESIGN: | DRAWN: BP | REVIEWED: BP
SCALE: NOT TO SCALE | DATE: 11/25/2019
FILE: Appendix-B-validation_stats_swath.xlsx

SWATH PLOTS
5000Ag

REPORT TITLE:
NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT
MARMATO PROJECT

ISSUED FOR: **GRAN COLOMBIA GOLD MARMATO S.A.S**

APPENDIX C-8

REVISION NO. **A**

SRK JOB NO.: **544400.020**

Appendix D: Annual TEM

	Total	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028
Tonnes milled	11,135,047	109,383	349,997	402,491	350,004	1,190,079	1,750,055	1,750,078	1,750,005	1,750,065	1,732,890
Head grade (g/t Au)	3.20	3.95	3.77	3.53	3.64	3.33	3.19	3.53	3.32	2.90	2.66
Contained gold (ozs)	1,147,106	13,878	42,459	45,642	40,913	127,272	179,637	198,477	187,046	163,433	148,350
Recovery	92%	87%	87%	87%	87%	92%	93%	93%	93%	93%	92%
Gold produced (ozs)	1,055,500	12,004	36,727	39,480	35,390	117,433	167,169	184,969	173,944	151,357	137,026
Silver produced (ozs)	912,215	16,721	45,435	47,037	53,829	97,682	123,453	128,793	140,508	122,995	135,762
Spot gold price ($/oz)		1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Spot silver price ($/oz)		17	17	17	17	17	17	17	17	17	17
Cash cost/oz	743					758	688	631	681	760	822
AISC/oz	843					971	767	687	740	853	950
Revenue (net of refining)	1,380,923,791	15,813,051	48,283,143	51,872,351	46,695,954	153,574,457	218,352,138	241,469,593	227,406,045	197,889,404	179,567,654
Opex	(656,841,906)	(8,739,562)	(25,869,897)	(28,801,035)	(25,853,105)	(76,512,666)	(97,079,719)	(96,735,577)	(99,868,753)	(98,989,206)	(98,392,385)
Royalties	(127,044,989)	(1,454,801)	(4,442,049)	(4,772,256)	(4,296,028)	(14,128,850)	(20,088,397)	(22,215,203)	(20,921,356)	(18,205,825)	(16,520,224)
Income taxes paid	(111,269,598)	-	(1,685,606)	(5,308,670)	(5,246,550)	(4,595,358)	(10,354,667)	(21,077,710)	(27,082,786)	(22,001,626)	(13,916,624)
Working capital adjustments	(6,653,711)	(581,383)	(1,255,769)	(59,121)	183,162	(4,620,735)	(3,606,516)	(1,955,584)	1,413,429	2,353,734	1,475,073
Operating cash flow	485,767,299	5,037,306	15,029,822	12,931,269	11,483,433	53,716,848	87,222,839	99,485,520	80,946,579	61,046,481	52,213,493
Sustaining Capital	(105,658,618)	(2,756,317)	(5,349,265)	(4,184,030)	(3,000,191)	(25,076,042)	(13,178,727)	(10,343,224)	(10,284,421)	(13,967,852)	(17,518,551)
Free cash flow	380,108,681	2,280,989	9,680,557	8,747,239	8,483,242	28,640,806	74,044,112	89,142,295	70,662,159	47,078,629	34,694,943
Expansion capex	(268,884,037)	-	-	(62,863,892)	(206,020,145)	-	-	-	-	-	-
Project cash flow	111,224,644	2,280,989	9,680,557	(54,116,653)	(197,536,903)	28,640,806	74,044,112	89,142,295	70,662,159	47,078,629	34,694,943

	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Tonnes milled	1,750,031	1,750,048	1,750,030	1,750,049	1,750,061	1,750,046	1,530,095	1,400,000	1,206,566	592,005	-
Head grade (g/t Au)	2.59	2.45	2.40	2.48	2.55	2.54	2.33	2.36	2.47	2.57	-
Contained gold (ozs)	145,592	137,717	135,024	139,300	143,387	142,673	114,394	106,102	95,784	48,916	-
Recovery	92%	93%	92%	92%	92%	92%	94%	95%	95%	95%	0%
Gold produced (ozs)	134,593	127,403	124,618	128,474	132,362	131,765	107,417	100,797	90,995	46,470	-
Silver produced (ozs)	159,521	145,135	108,525	97,089	96,805	96,784	69,563	58,970	47,507	23,906	-
Spot gold price ($/oz)	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Spot silver price ($/oz)	17	17	17	17	17	17	17	17	17	17	17
Cash cost/oz	840	875	892								
AISC/oz	1,005	1,006	988								
Revenue (net of refining)	176,824,653	167,278,968	163,052,858	167,847,630	172,871,600	172,099,454	140,139,609	131,395,264	118,520,244	60,520,991	-
Opex	(99,560,941)	(98,572,560)	(98,062,200)	(98,033,478)	(98,820,995)	(98,420,706)	(83,844,281)	(72,902,504)	(59,327,021)	(31,355,686)	-
Royalties	(16,267,868)	(15,389,665)	(15,000,863)	(15,441,982)	(15,904,187)	(15,833,150)	(12,892,844)	(12,088,364)	(10,903,862)	(5,567,931)	-
Income taxes paid	(8,685,790)	(7,062,474)	(4,178,605)	(2,842,320)	(3,923,958)	(13,033,570)	(13,578,982)	(7,349,517)	(11,188,139)	(11,922,921)	-
Working capital adjustments	303,268	703,340	305,404	(380,774)	(363,880)	30,564	1,821,739	(560,440)	(70,708)	4,865,196	-
Operating cash flow	52,613,322	46,957,609	46,116,594	51,149,077	53,858,581	44,842,592	31,645,241	38,494,438	37,030,513	16,539,649	-
Sustaining Capital	(22,094,242)	(16,622,632)	(11,953,186)	(5,972,993)	(3,887,432)	(3,601,619)	(2,348,038)	(1,763,038)	(1,263,038)	(6,100,000)	-
Free cash flow	30,519,081	30,334,978	34,163,408	45,176,084	49,971,149	41,240,974	29,297,203	36,731,400	35,767,475	10,439,649	-
Expansion capex	-	-	-	-	-	-	-	-	-	-	-
Project cash flow	30,519,081	30,334,978	34,163,408	45,176,084	49,971,149	41,240,974	29,297,203	36,731,400	35,767,475	10,439,649	-