TECHNICAL REPORT AND PRELIMINARY FEASIBILITY STUDY
FOR THE

AGUABLANCA
NICKEL-COPPER MINERAL DEPOSIT

EXTREMADURA REGION
SPAIN

DATED MAY 23, 2024

EFFECTIVE DATE: MARCH 24, 2024

PREPARED FOR:

DENARIUS METALS CORP.

BY

RESOURCE DEVELOPMENT ASSOCIATES
HIGHLANDS RANCH, COLORADO

DATE AND SIGNATURE PAGE

Denarius Metals Corp.: Technical Report and Preliminary Feasibility Study for the Aguablanca Nickel-Copper Mineral Deposit, Extremadura Region, Spain.

Technical Report Effective Date: March 24, 2024

Dated May 23, 2024

(signed/sealed) *Scott E. Wilson*
Scott E. Wilson, SME-RM, CPG
Geologist

(signed/sealed) Ray "*Zeke" Blakeley*
Ray "Zeke" Blakeley, SME-RM, QP
Mining Engineer

(signed/sealed) *Rick S. Jordan*
Rick S. Jordan, SME-RM, QP
Consulting Metallurgist

AUTHOR CERTIFICATE

Scott E. Wilson

I, Scott E. Wilson, CPG, SME-RM, of Highlands Ranch, Colorado, as the author of the technical report entitled “Technical Report and Preliminary Feasibility Study for the Aguablanca Nickel-Copper Mineral Deposit, Extremadura Region, Spain” (the “Technical Report”) with an effective date of March 24, 2024 prepared for Denarius Metals Corp. (the “Issuer”), do hereby certify:

1. I am currently employed as President by Resource Development Associates, Inc., 10262 Willowbridge Way, Highlands Ranch, Colorado USA 80126.
2. I graduated with a Bachelor of Arts degree in Geology from the California State University, Sacramento in 1989.
3. I am a Certified Professional Geologist and member of the American Institute of Professional Geologists (CPG #10965) and a Registered Member (#4025107) of the Society for Mining, Metallurgy and Exploration, Inc.
4. I have been employed as both a geologist and a mining engineer continuously for a total of 31 years. My experience included resource estimation, mine planning, geological modeling, geostatistical evaluations, project development, and authorship of numerous technical reports and preliminary economic assessments of various projects throughout North America, South America and Europe. I have employed and mentored mining engineers and geologists continuously since 2003.
5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.
6. I made a personal inspection of the Aguablanca Project from January 30 to February 2, 2024.
7. I am responsible for Sections 1 through 12, 14, 18, 19, 20 and 23 through 27 of the Technical Report.
8. I am independent of the Issuer as independence is described in Section 1.5 of NI 43-101.
9. Prior to being retained by the Issuer, I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and Form 43-101F1, and this Technical Report was prepared in compliance with NI 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated: May 23, 2024

(signed/sealed) *Scott Wilson*
Scott E. Wilson, CPG, SME-RM

AUTHOR CERTIFICATE

Ray “Zeke” Blakeley

I, Ray “Zeke” Blakeley, QP, SME-RM, of Butte, Montana, as the author of the technical report entitled “Technical Report and Preliminary Feasibility Study for the Aguablanca Nickel-Copper Mineral Deposit, Extremadura Region, Spain” (the “Technical Report”) with an effective date of March 24, 2024 prepared for Denarius Metals Corp. (the “Issuer”), do hereby certify:

1. I am currently employed as a Principal by Minetech, LLC, 29 Denali Lane, Butte, Montana USA 59701.
2. I graduated with a Bachelor of Science degree in Mine Engineering from Montana College of Mineral Science and Technology, Butte, Montana 1994.
3. I am a Qualified Person and a Registered Member (#4029863) of the Society for Mining, Metallurgy and Exploration, Inc.
4. I have been employed as a mining engineer continuously for a total of 28 years. My experience included underground and surface mine planning and design, production scheduling, feasibility studies, cost estimating, strategic planning, mine expansion planning, ground control management, mobile equipment maintenance, economic studies, reclamation, construction management and mine closure throughout North America. I have mentored and employed mining engineers continuously since 1997.
5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.
6. I made a personal inspection of the Aguablanca Project January 30 through February 7, 2024.
7. I am responsible for Sections 15, 16 and co-authored 18, 21, 22, 25, and 26 of the Technical Report.
8. I am independent of the Issuer as independence is described in Section 1.5 of NI 43-101.
9. Prior to being retained by the Issuer, I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and Form 43-101F1, and this Technical Report was prepared in compliance with NI 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated: May 23, 2024

(signed/sealed) *Ray “Zeke” Blakeley*
Ray “Zeke” Blakeley, QP, SME-RM

AUTHOR CERTIFICATE

Rick Jordan

I, Rick S. Jordan, SME-RM QP, of Butte, Montana, as the author of the technical report entitled "Technical Report and Preliminary Feasibility Study for the Aguablanca Nickel-Copper Mineral Deposit, Extremadura Region, Spain" (the "Technical Report") with an effective date of July 31, 2023 prepared for Denarius Metals Corp. (the "Issuer"), do hereby certify:

1. I am currently employed as Process Engineer by Minetech, LLC, 29 Denali Lane, Butte, Montana USA 59701
2. I graduated with a Bachelor of Science degree in Mineral Processing Engineering, Montana College of Mineral Science and Technology, 1990.
3. I am a Registered Member (#408591) of the Society for Mining, Metallurgy and Exploration, Inc.
4. I have been employed as both a metallurgist and a process engineer continuously for a total of 35 years. My experience included metallurgical test work, process flow sheet design, mine process operation, operating statistical analyses, capital cost and operating cost estimating, due diligence, operations performance optimization. project development, and authorship of numerous technical reports and preliminary economic assessments of various projects globally
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I made a personal inspection of the Aguablanca Project January 30 through February 7, 2024.
7. I am responsible for Sections 13, 17 and co-authored 18, 21, 22, 25, and 26 of the Technical Report.
8. I am independent of the Issuer as independence is described in Section 1.5 of NI 43-101.
9. Prior to being retained by the Issuer, I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and Form 43-101F1, and this Technical Report was prepared in compliance with NI 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated: May 23, 2024

(signed/sealed) *Rick S. Jordan*
Rick S. Jordan, SME-RM QP

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1 SUMMARY

This report entitled "Technical Report and Preliminary Feasibility Study for the Aguablanca Nickel-Copper Mineral Deposit, Extremadura Region, Spain" (the "Technical Report"), summarizes the re-start of mining and processing operations for the Aguablanca Project ("Aguablanca" or the "Project" or the "Property") located near the community of El Real de la Jara, Spain. This report is prepared for Denarius Metals Corp. ("Denarius" or the "Company"), which operates in Spain through its 100%-owned subsidiary company, Alto Minerals S.L.U. ("Alto" or "Alto Minerals"). Alto owns 50% of Rio Narcea Recursos, S.A. ("RNR"), owner of the Aguablanca Project.

This preliminary feasibility study ("PFS") summarizes the restart of the Aguablanca mineral deposit by underground mining. The Project will operate under a currently approved Environmental Impact Statement ("EIS"). All necessary permits required for the operating mine are in place and approved. Mined ore will be processed by three-stage crushing followed by conventional grinding and product slurry conditioning. Processing of the conditioned slurry product will be followed by industry typical bulk sulfide flotation to produce a bulk sulfide concentrate for the recovery of nickel, copper, cobalt, gold, silver and platinum group metals ("PGM"). The flotation concentrate will be thickened, filtered and readied for shipment. Flotation tailings will be contained within the existing tails storage facility ("TSF").

Highlights of the Technical Report, the PFS parameters and costs, are listed in Table 1-1 and Table 1-2. Table 1-3 lists the Mineral Resource estimate ("MRE") for Aguablanca. Mineral Resources are reported according to the CIM Definition Standards of May 10, 2014 ("CIM"). The guidance and definitions of CIM are incorporated by reference in National Instrument 43-101 - *Standards of Disclosure for Mineral Projects within Canada* of the Canadian Securities Administrators ("NI 43-101"). Mineral Resources are geologically constrained and defined at economic cutoff grades that demonstrate reasonable prospects of eventual economic extraction. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources will be converted into Mineral Reserves. Table 1-4 lists the Mineral Reserves for the Project. Mineral Reserves are based on economic analysis of measured and indicated mineralization only. Inferred mineralization is treated as containing no mineralization for the PFS analysis.

1.1 PRE-FEASIBILITY STUDY

The PFS for Aguablanca covers a six-year mine life, processing 4.8 million tonnes of ore from an underground mining operation. The life-of-mine ("LOM") of the project delivers a net revenue of US$480.3 million with an after-tax cash flow of US$83.1 million at a 5% discount rate. The project has an IRR of 213% with a payback period of 1.2 years.

Highlights include:

- Refurbish and restart existing 5,000 tonne per day processing plant in 2024
- Dewatering the open pit and underground development
- Develop production from the underground to produce nickel-copper concentrates in 2025
- Six years of production producing 43.2 million pounds of payable nickel and 34.6 million pounds of payable copper from 406,359 tonnes of nickel-copper concentrate
- LOM all-in-sustaining-cost of US$4.04 per pound of payable nickel on a by-product basis

Table 1-1 Key Economic Parameters of the PFS

Assumption / Results	100% Basis
Total tonnes processed from underground mining over the LOM	4,807,000
Average LOM process rate	2,403 tpd
Projected mine life	6 years
Average Nickel Grade / Recovery	0.66% | 82.8%
Average Copper Grade / Recovery	0.58% | 93.6%
Average Gold Grade / Recovery	0.16g/t | 75.0%
Average Platinum Grade / Recovery	0.33g/t | 75.0%
Average Palladium Grade / Recovery	0.28g/t | 75.0%
Total Payable Production	
Nickel	43,204 Klbs | 19,597 t
Copper	34,612 Klbs | 15,700 t
Gold	7,205 oz
Platinum	15,092 oz

Assumption / Results	100% Basis
Palladium	13,144 oz
Expected long-term nickel/ copper prices (US$/lb)	$7.30 | $3.50
Expected long-term gold/ platinum/ palladium prices (US$/oz)	$2,000 | $900 | $1,200
LOM net revenue, after refining and treatment charges (US$ millions)	$480.3
LOM capital costs, including contingency (US$ millions)	$36.2
LOM operating costs, including contingency (US$ millions) (Table 1-2)	$303.2
LOM cash cost per lb of nickel (Table 1-2)	$3.20
LOM AISC per lb of nickel (Table 1-2)	$4.04
After-tax undiscounted LOM Project Cash Flow (US$ millions)	$105.7
After-Tax NPV (5% discount) (US$ millions)	$83.1
After-Tax IRR	213%
Payback Period	1.2 Years

1.1.1 CAPITAL EXPENDITURE

The Project is currently in a position to quickly restart mining and processing operations. The 5,000 tpd processing plant has been maintained in good condition over the years since it was last operated by Lundin Mining Corporation ("Lundin" or "Lunding Mining"). Total forecast capital expenditures in 2024 of US$6.1 million will include approximately US$2.7 million to restart the processing plant, US$1.3 million for surface mobile equipment and underground infrastructure and US$1.6 million to dewater the existing open pit to gain access to existing underground workings. Dewatering is expected to commence in the fourth quarter of 2024. Capital expenditures in 2024 also include approximately US$0.5 million associated with the commencement of underground development.

From the start of production in 2025 through 2030, capital expenditures are projected to total US$30.1 million, of which the majority represents an ongoing mine development program amounting to US$22.5 million and an ongoing exploration and delineation drilling program totaling US$4.2 million. The remaining US$3.4 million of capital expenditures over this period include a cemented rock fill plant, tailings storage facility capacity increase, surface mobile equipment and underground infrastructure.

1.1.2 OPERATING EXPENDITURE

A summary of the key operating costs, cash costs and AISC are presented below.

Table 1-2 Operating, Cash and AISC Costs

Operating Costs	LOM (US$M)	Per Lb Nickel (US$)
Mining	167.9	3.89
Processing	104.2	2.41
Site administration and social programs	30.8	0.71
Transport	0.3	0.01
Total operating costs	**303.2**	**7.02**
Less: by-product credits for copper, gold, platinum, palladium	(164.9)	**(3.82)**
Total cash costs (**)	**138.2**	**3.20**
Capital and exploration	**36.2**	**0.84**
All-in sustaining costs (**)	**174.3**	**4.04**

1.2 PROPERTY DESCRIPTION AND LOCATION

The Aguablanca Mine is located approximately 100 km north of Seville, Spain within the Monesterio municipal boundary south of Badajoz Province (Extremadura Region), and close to the Huelva and Sevilla (Andalucia Region) provincial boundaries. The Project lies at Latitude 37°57´40´´ and Longitude 6°11´7´´.

In December 2023, through a joint venture agreement, Alto acquired a 50% interest in RNR, a private company in Spain, which has the rights to exploit the Project. The Aguablanca Project consists of in situ, underground-exploitable mineral reserves, a non-operational open pit mine, an on-site processing plant and additionally, all the associated facilities which are required for an operational mining project.

1.3 GEOLOGY AND MINERALIZATION

The Aguablanca Stock is located in the northeast of the Santa Olalla Plutonic Complex which has yielded absolute ages of 338.6±0.8 Ma and 344±1.1 Ma. The intrusion consists of medium grained gabbro and gabbro-norites. Mineralization occurs in igneous rocks formed by partial melting of the mantle. When basalt magma is emplaced within the crust, it cools slowly at depth and forms gabbro which is the host rock of the Aguablanca mineral deposit.

A clear transition from semi-massive mineralized breccia in the central part of the Aguablanca mineralization to disseminated mineralization along both sides of the central breccia, suggests a vertical, “feeder like” emplacement of mineralization. The deepest portion of the main body remains open to depth.

Mineralization consists of magmatic sulphides within the gabbro and gabbro-norite units. Sulphide mineralization is predominantly represented by pyrrhotite, pentlandite, chalcopyrite and pyrite with minor marcasite and covellite. Magnetite is often present. Lesser amounts of platinum group minerals (PGM) and gold are also associated with the sulphide minerals.

Mineralization is generally hosted within a chaotic magmatic breccia, rich in fragments, in the matrix of which can be found semi-massive sulphides. Mineralization often grades laterally into more continuous disseminated sulphides and irregular patchy mineralization with variable textures. The mineralized body consists of two sub-parallel zones of mineralization, elongated in an east west direction and dipping sub-vertically. The larger South or Main Zone can be traced for 400 metres along an N115º orientation dipping between 75º and 80º to the north. This zone is over 100 metres thick and extends to a depth of more than 600 metres. The mineralization has a plunge of N65º to N80ºE. The North Zone contacts with the surrounding metamorphic country rocks and strikes N115º and is approximately 125metres long in the strike direction, 50 metres thick and extends to a depth of more than 300 metres.

1.4 EXPLORATION

Currently, there is no active exploration underway at the Project. Exploration essentially ended prior to 2015. In 2022, RNR drilled two exploration holes from the surface totaling 1,093 m. The holes were intended to intersect the north and south mineral bodies and then to intercept deeper targets. Results from the current drilling have been included in the Mineral Resource estimate for the Project.

1.5 DEVELOPMENT AND OPERATIONS

1.5.1 MINING

Development and exploitation activities in the underground mine will be carried out by a local mine contractor, alleviating the need for a significant upfront investment in underground mining equipment. The approximately 2,403 tpd production profile is sourced from up-hole sublevel extraction and traditional long-hole open stoping employing cement rock fill. The top half of the mineralization is scheduled to be extracted on 25-meter lifts day-lighting to the bottom of the pit. When four horizons have been exhausted, surface backfill will be hauled from the existing waste dump to backfill the pit to the 181-meter elevation as required by the mine operating permits and the environmental impact study (“EIS”).

1.5.2 PROCESSING

The processing plant will produce a nickel-copper concentrate from the material sourced from the Aguablanca underground mine and will operate four days per week at an average feed rate of 199 tph. This schedule will result in the processing of 877,200 tonnes per year, approximately 50% of the plant’s total capacity. Concentrate production over the LOM at the Aguablanca Project is estimated to total 406,359 tonnes with average grades of 6.4% nickel, 6.4% copper, 1.42 grams per tonne (“g/t”) gold, 2.89 g/t platinum and 2.52 g/t palladium. Denarius is currently carrying out an international tender process to identify a long-term offtake arrangement suitable for the sale of these concentrates. The PFS assumes that concentrates will be delivered free on board (“FOB”) to the port of Huelva in Southern Spain. The payable quantities of nickel (75%), copper (60%), gold (40%), platinum (40%) and palladium (40%) included in the PFS are based on early indicative terms received through this process. Actual terms may vary once a long-term offtake arrangement is finalized.

1.6 MINERAL RESOURCE ESTIMATES

The MRE is based on 496 diamond drillholes containing 25,025 assay intervals. Drilling includes two exploration holes which were drilled in 2022. Outlier grades were capped prior to compositing to 24,250 two-meter intervals. Nickel, copper and cobalt mineralization was estimated using ordinary kriging techniques based on detailed variography analysis of the mineral deposit. Gold, platinum and palladium mineralization was interpolated using inverse distance estimation techniques. Three-dimensional geology models were constructed to identify the mineralized domains of the mineral deposit. Mineralization is constrained geologically to the mineralized domains to accurately reflect in situ mineralization. The mineral resource estimate was completed using Vulcan scientific software in a 3D block model, with blocks ranging from 4x4x4 meters down to 2x2x2 meters which is a size reflective of the selective mining unit envisioned for underground mining of the deposit. These blocks demonstrate the reasonable prospects for eventual economic extraction. All other material, smaller than the Selective Mining Unit ("SMU"), was reported as non-mineralized material.

Table 1-3 below lists the current MRE for the Project at cut-off grade of 0.35% Nickel (Ni). Mineral Resources are not Mineral Reserves and do not have demonstrate economic viability. The stated Mineral Resources are inclusive of Mineral Reserves.

Table 1-3: Aguablanca Mineral Resource Estimate. Effective Date March 24, 2024

Resource Category	Cutoff (Ni%)	Tonnage (K tonnes)	Grade							Contained Metal						
			Ni (%)	Cu (%)	Co (%)	Pd (ppm)	Pt (ppm)	Au (ppm)	NiEq (%)	Ni (Klbs)	Cu (Klbs)	Co (Klbs)	Pd (Oz)	Pt (Oz)	Au (Oz)	NiEq (Klb)
Measured	0.35%	4,048	0.66	0.60	0.02	0.29	0.34	0.17	0.95	58,836	53,512	1,473	38,033	43,919	21,954	84,493
Indicated	0.35%	1,273	0.64	0.52	0.02	0.27	0.31	0.14	0.89	17,986	14,462	503	11,060	12,492	5,760	24,919
Measured + Indicated		**5,321**	**0.65**	**0.58**	**0.02**	**0.29**	**0.33**	**0.16**	**0.93**	**76,822**	**67,974**	**1,976**	**49,094**	**56,411**	**27,715**	**109,412**
Inferred	0.35%	4	0.67	0.61	0.02	0.31	0.37	0.17	0.96	66	60	2	45	54	24	95

1. Scott Wilson, CPG, President of RDA is responsible for this Mineral Resource Estimate and is an "independent Qualified Person as such term is defined by NI 43-101.
2. Reasonable prospects of eventual economic extraction were assessed by enclosing the mineralized material in the block model estimate in a 3D wireframe shape that was constructed based upon geological interpretations as well as adherence to a minimum mining unit with geometry appropriate for underground mining.
3. The cutoff grade of 0.35% Ni considered mining costs of:
    a. Metal selling prices Ni at US$7.30/lb and Cu selling prices of US$3.50/lb,
    b. Recoveries of Ni 82.8% and Cu 93.6% and
    c. Costs including mining, processing, general and administrative (G&A), and off-site realization (TCRC).
4. Nickel Equivalent is estimated as ((3.50/7.30) * Cu grade) + Ni Grade.
5. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
6. Mineral Resources are inclusive of Mineral Reserves.
7. Figures may not add up due to rounding.

1.7 MINERAL RESERVE ESTIMATES

Mineral Reserve Estimates for the Project were determined by the aid of stope optimization software. Additionally, Mineral Reserves are constrained to an underground mining extraction sequence that has been developed with 3D mining software. The mine plan in the PFS is based on Measured and Indicated mineralization only, as summarized in Table 1-4, which have been estimated for a combination of sub-level extraction and long-hole open stoping underground mining methods. The MRE reflected in Table 1-3 above is inclusive of the Mineral Reserve estimate, which represents approximately 89% of the tonnes in the Measured and Indicated MRE.

Table 1-4 Aguablanca Mineral Reserve Estimate Effective Date March 24, 2024

Reserve Category	Cutoff (Ni%)	Tonnage (K tonnes)	Grade							Contained Metal						
			Ni (%)	Cu (%)	Co (%)	Pd (ppm)	Pt (ppm)	Au (ppm)	NiEq (%)	Ni (Klbs)	Cu (Klbs)	Co (Klbs)	Pd (Oz)	Pt (Oz)	Au (Oz)	NiEq (Klb)
Proven	0.35%	3,650	0.67	0.61	0.02	0.29	0.34	0.17	0.97	54,051	49,281	1,343	34,454	39,798	19,835	77,678
Probable	0.35%	1,062	0.67	0.53	0.02	0.28	0.31	0.14	0.92	15,582	12,452	429	9,419	10,578	4,875	21,553
Proven + Probable		**4,713**	**0.67**	**0.59**	**0.02**	**0.29**	**0.33**	**0.16**	**0.96**	**69,633**	**61,733**	**1,772**	**43,874**	**50,375**	**24,709**	**99,231**

Notes:

1. CIM Definition Standards were followed for Mineral Reserves.
2. Mineral Reserves are not additive to Mineral Resources.
3. Mineral Reserves are based on the March 24, 2024 MRE.
4. Totals may not add up due to rounding.
5. Mineral Reserves are reported using US$7.30/lb Ni, US$3.50/lb Cu, US$12/lb Co, US$2,000/oz Au, US$900/oz Pt and US$1,200/oz Pd
6. The cutoff grade of 0.35% Ni considered mining costs of:
    a. Metal selling prices Ni at US$7.30/lb and Cu selling prices of US$3.50/lb,
    b. Recoveries of Ni 82.8% and Cu 93.6% and
    c. Costs including mining, processing, general and administrative (G&A), and off-site realization (TCRC).

7. Mineral Reserves are constrained to a mine design.
8. Units are metric tonnes, metric grams, troy ounces and imperial pounds. Contained metal are estimates of in situ material and do not account for dilution or processing losses.

1.8 CONCLUSIONS

The Aguablanca MRE has been re-estimated independently and confirms the past mineral resource estimates for the Project. Nearly 89% of the Measured and Indicted Mineral Resources have been converted to Proven and Probable reserves. The stated MRE provides a suitable basis for the existing processing plant to recommence the production of nickel and copper concentrates. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty the Mineral Resources will be converted to Mineral Reserves.

The underground mine will be contractor operated. Two mining methods will be implemented. Up-hole Sublevel Extraction ("USE") will be employed for the upper portion of the deposit. This will essentially extract mineralization from the 75 level up to the bottom of the pit, caving the pit bottom and exposing a void. The void will be back-filled to the current pit bottom of 181 msl as required by the current EIS. Below the 75 level, Long Hole Open Stoping ("LHOS") will be the primary mining method. Stopes will be supported by cement backfill, with the cement being produced onsite.

Aguablanca process plant design calls for a dry feed throughput rate of 195 tph or approximately 5,000 dry tpd. However, the underground mine will produce on average only approximately 2,400 tpd. The milling schedule will be modified accordingly. The process plant operations cycle will include seven weeks with the plant operating Monday through Thursday. During the eighth week, the plant will operate only on Monday and Tuesday, thus allowing for two extra days during the eighth weeks to complete large maintenance projects. It is estimated that plant startups will take one hour while shutdowns will require two hours. Thus, the plant will operate 93 hours for seven weeks and 45 hours during the eighth week.

Since the crushing plant design throughput is 450 tph, the crusher will not operate for as many hours each week as the rest of the plant. Because of this, crusher maintenance projects can be completed while the rest of the plant is in operation.

While the plant is operating, staffing will be similar to 2015 staffing including two reagent and plant employees as well as a commercial supervisor and commercial operator to be responsible for concentrate storage and loading. During periods when the plant is scheduled to be down, the plant will be staffed by one supervisor and one operator on day shifts and one supervisor and one operator on night shifts. These individuals will handle fire watch, monitor any equipment still operating, and complete cleanup projects. The sample preparation, assay, and metallurgical laboratories will be staffed with sufficient manpower to complete mine and plant sample analyses and testing.

The Project has all of the required environmental and operating permits in place. The Project will operate according to the currently approved Environmental Impact Statement ("EIS").

The economic performance of the Project was evaluated with a cash flow based economic model using project costs and revenues as the financial basis. Costs are in constant 2024 US dollars, no escalation of cost has been assumed. Operating costs are generated based on production and unit rates. The Project is expected to yield an after-tax undiscounted LOM net cash flow of US$105.7 million, and an NPV of US$83.1 million at a discount rate of 5% per year. The results for the Project economic analysis are summarized in Table 1-1.

1.9 RECOMMENDATIONS

Recommendations for recommissioning the processing plant are as follows:

1. Non-destructive thickness testing should be completed on all tanks, especially where sulfide material has been in contact with the bases of the tanks;
2. A structural steel survey should be completed, especially where sulfide material has been in contact with lower portions of vertical structural members;
3. Non-destructive testing should be completed on SAG and ball mill trunnions;
4. SAG and ball mill gear alignment needs to be measured; and
5. A group including Denarius management personnel and those familiar with the Aguablanca operation should meet and prioritize a list of projects to be completed and categorized into four groups as follows:
    a. **Absolutely** must be completed and **cannot** be completed once the plant is operational;
    b. **Absolutely** must be completed but **can be** completed after the plant is operational;
    c. Not absolutely necessary but can only be completed prior to plant startup; and

d. Not absolutely necessary and can be completed after the plant is operational.

Implementation of 5a and 5c above may significantly lower the risk of cost overruns during the recommissioning of the processing plant.

An exploration drilling program is recommended. This program would be focused on expansion to depth of the mineral resources as well as expanding the resource footprint of the satellite mineralization of the deposit. A budget is presented below for a 20,000 m drilling program. Results from the drilling would be used to update the MRE for Aguablanca.

Geotechnical and rock mechanics evaluations should be conducted to conclusively support the USE mining which will breach the bottom of the existing open pit.

Budget Item	Description	Cost (US$ 1,000's)
Exploration Drilling	Drilling program to expand the mineral resources beneath the north and main mineralization as well as expansion of the satellite mineralization footprint	US$3,100
	Assaying	US$150
	Technical Services	US$210
Geology	Resource Model Updates	US$125
Geotechnical and rock mechanics assessments	Field and engineering to support the USE mining method	US$300
Subtotal		**US$3,885**
Contingency	Monte Carlo Simulation suggests there is a 20% probability the program will exceed US$3,885,000. A contingency of 8% has been added to the work program budget	US$311
Total		**US$4,196**

The QP is not recommending successive phases.

2 INTRODUCTION

2.1 TERMS OF REFERENCE

Denarius retained the services of Scott Wilson, President of Resource Development Associates Inc. ("RDA"), to complete an independent NI 43-101 Technical Report and Preliminary Feasibility Study ("PFS") for the Aguablanca Project located near the community of El Real de la Jara, Spain. RDA's associates for this study were:

- Scott E. Wilson, CPG, President of Resource Development Associated Inc. Mr. Wilson is acting as the overall QP for the NI 43-101 Technical Report
- Zeke Blakeley, P.E. of Minetech, LLC. Mr. Blakeley performed mining engineering, mine designs, mine scheduling and economic analyses for the Project.
- Rick Jordan, Registered Member SME of Minetech, LLC. Mr. Jordan provided input regarding mineral processing and metallurgical studies, recovery methods and mill restart activities.

Mr. Scott E. Wilson, (CPG #10965, SME 4025107RM), an independent qualified person under the terms of NI 43-101, conducted a site visit of the Property on January 30 through February 2, 2024. The site visit was to review the Property processing facilities, to meet the local Project staff, and to gather information related to tasks required for RDA to develop a PFS on the Project.

Mr. Zeke Blakeley, (QP, SME #4029863 RM), an independent qualified person under the terms of NI 43-101, conducted a site visit of the Property on January 30 through February 7, 2024. The site visit was to review the Property mining layout, to meet the local Project staff, and to gather information related to tasks required for RDA to develop a PFS on the Project.

Mr. Rick Jordan, (SME 408591RM), an independent qualified person under the terms of NI 43-101, conducted a site visit of the Property on January 30 through February 7, 2024. The site visit was to review the Property processing facilities, to meet the local Project staff, and to gather information related to tasks required for RDA to develop a PFS on the Project.

The Authors have worked closely with the Company to follow the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, November 29, 2019 and the CIM Mineral Exploration Best Practice Guidelines, November 23, 2018 with respect to the implementation and execution of the collection of scientific data for the Property.

This Technical Report was prepared by the Authors, at the request of Mr. Serafino Iacono, President and CEO of Denarius Metals, a publicly listed company with its corporate office at 401 Bay Street, Suite 2400, PO Box 15, Toronto, ON M5H 2Y4.

All dollar amounts in this document are United States dollars unless otherwise noted.

2.2 SOURCES OF INFORMATION

This Technical Report is based on existing Company Environmental Impact Study ("EIS"), technical reports, maps, published reports, company letters, memoranda, public disclosure and public information as listed in the References at the conclusion of this Technical Report. Budgetary capital equipment quotes were solicited from a number of suppliers for major equipment. Supplies and material costs primarily are from other similar projects and estimates for which Minetech has been recently associated. Labor costs are benchmarked against other known Spanish mining operations.

Mineral resource estimates included information such as geology maps, drilling databases, underground sampling databases, geology logging codes, underground development surveys, reviews of core photography and production results from the mining operation.

2.3 EFFECTIVE DATE

The effective date of this Technical Report is March 24, 2024.

2.4 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.

Table 2-1 Table of Abbreviations

Unit	Abbreviation
Centimeter	cm
comma separated values	csv
Degree	°
Degrees Celsius	°C
Drawing Exchange Format	dxf
From Origin of Business	FOB
Gram	g
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m2)	ha
Inverse Distance to the Power	IDP
Kilogram	kg
Kilometre	km
Less than	<
Life of mine	LOM
Meter	m
Micrometer (micron)	^m
Millimeter	mm
Million pounds	Mlbs
Million Tonnes	Mt
Ordinary Kriging	OK
Percent	%
Parts per million	ppm
Parts per billion	ppb
Quality assurance and quality control	QAQC

3 RELIANCE ON OTHER EXPERTS

With respect to land issues, leases and information, the Author of this Technical Report has relied upon the Title Opinion of VRIVM Legal, Seville, Spain, dated May 15, 2024 as well as written and verbal communication with Denarius in the preparation of Section 4. VRIVM Legal are experts with respect to land title and mineral tenure. VRIVM Legal are not qualified persons as defined by NI 43-101.

No other experts were relied upon in the preparation of this Technical Report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 PROPERTY LOCATION

The Aguablanca Mine is located approximately 100 km north of Seville within the Monesterio municipal boundary in the south of Badajoz Province (Extremadura Region), and close to the Huelva and Sevilla (Andalucia Region) provincial boundaries, in the south of Spain. It lies at Latitude 37°57'40" and Longitude 6°11'7" (Figure 4-1).

The closest villages are Real de la Jara, 2 km south-east, and Santa Olalla de la Cala, 10 km southwest. The village of Santa Olalla del Cala is on the N630 national route, between Sevilla in the south and Merida in the north. Monesterio, with a population of approximately 10,000, is located approximately 30 km north.

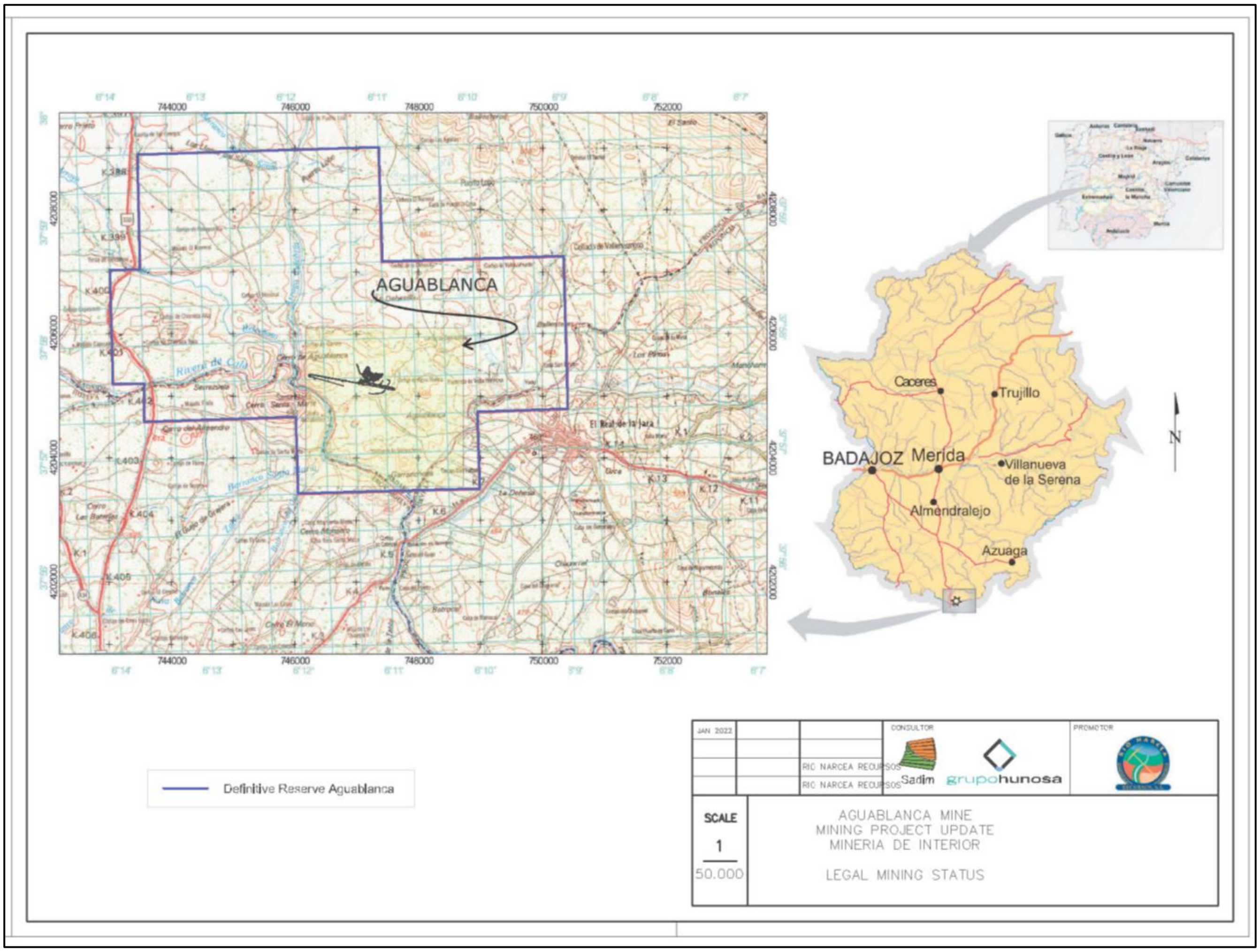

Figure 4-1 Location Map of the Aguablanca Showing the Definitive Reserve

RNR holds various mineral licenses and exploration permits covering an area greater than 350 hectares (ha). The mining rights are covered under the *Reserva Definitiva a favor del Estado* ("Definitive Reserve") with the name La Monaguera which covers 95 contiguous claims with a surface area of 2,861.94 ha. A Definitive Reserve denotes a state mineral or "strategic" reserve and therefore the responsible government agencies for its control are the central government, not the regional government. The boundaries are valid for 30 years extendable for another 30. The Definitive Reserve is shown in Figure 4-2.

The Aguablanca Project consists of an idle open pit mine and an on-site processing plant and all necessary facilities to recommence production (Figure 4-3 and Figure 4-4). Mining will take place via underground mining methods.

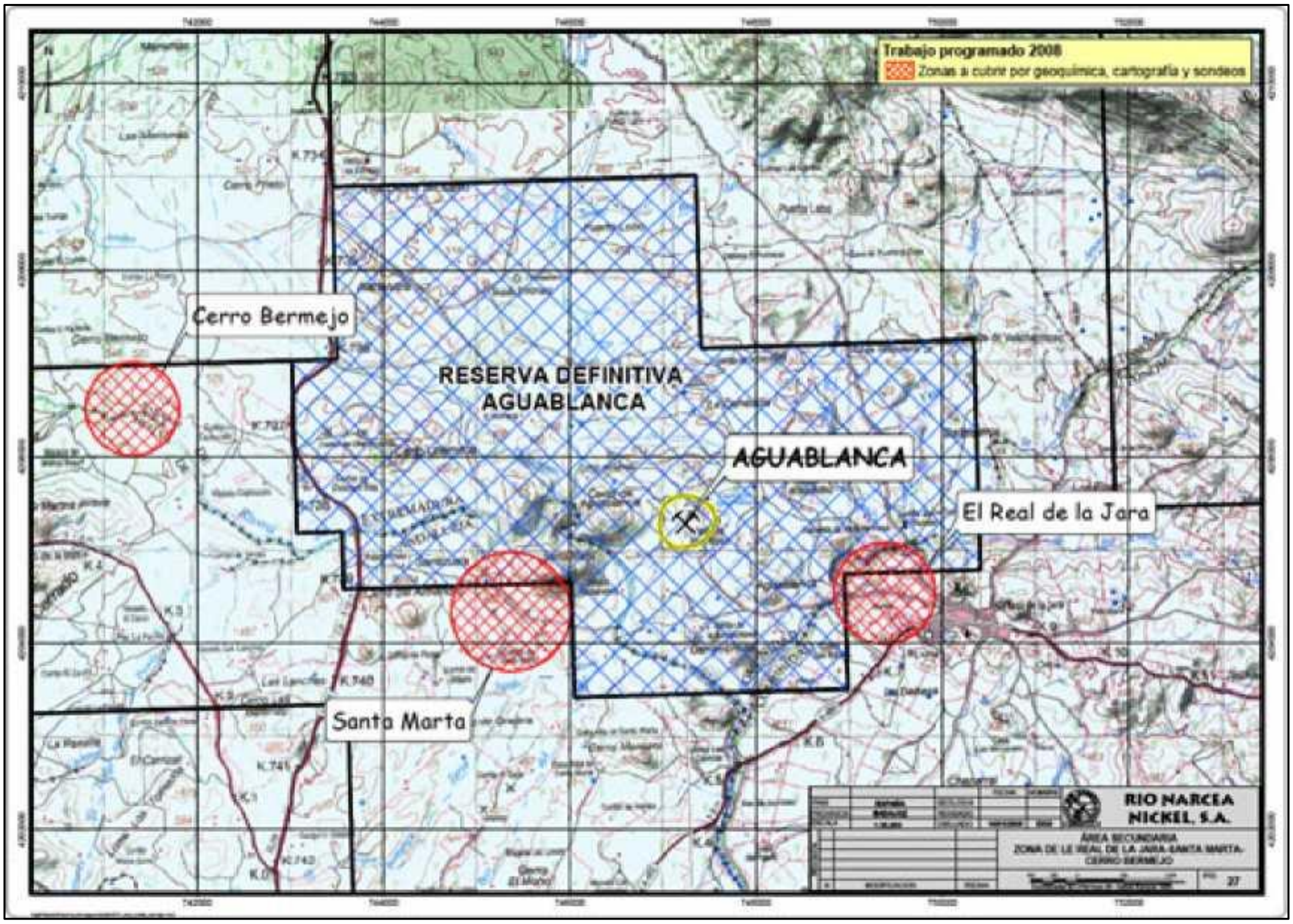

Figure 4-2 Aguablanca Licenses

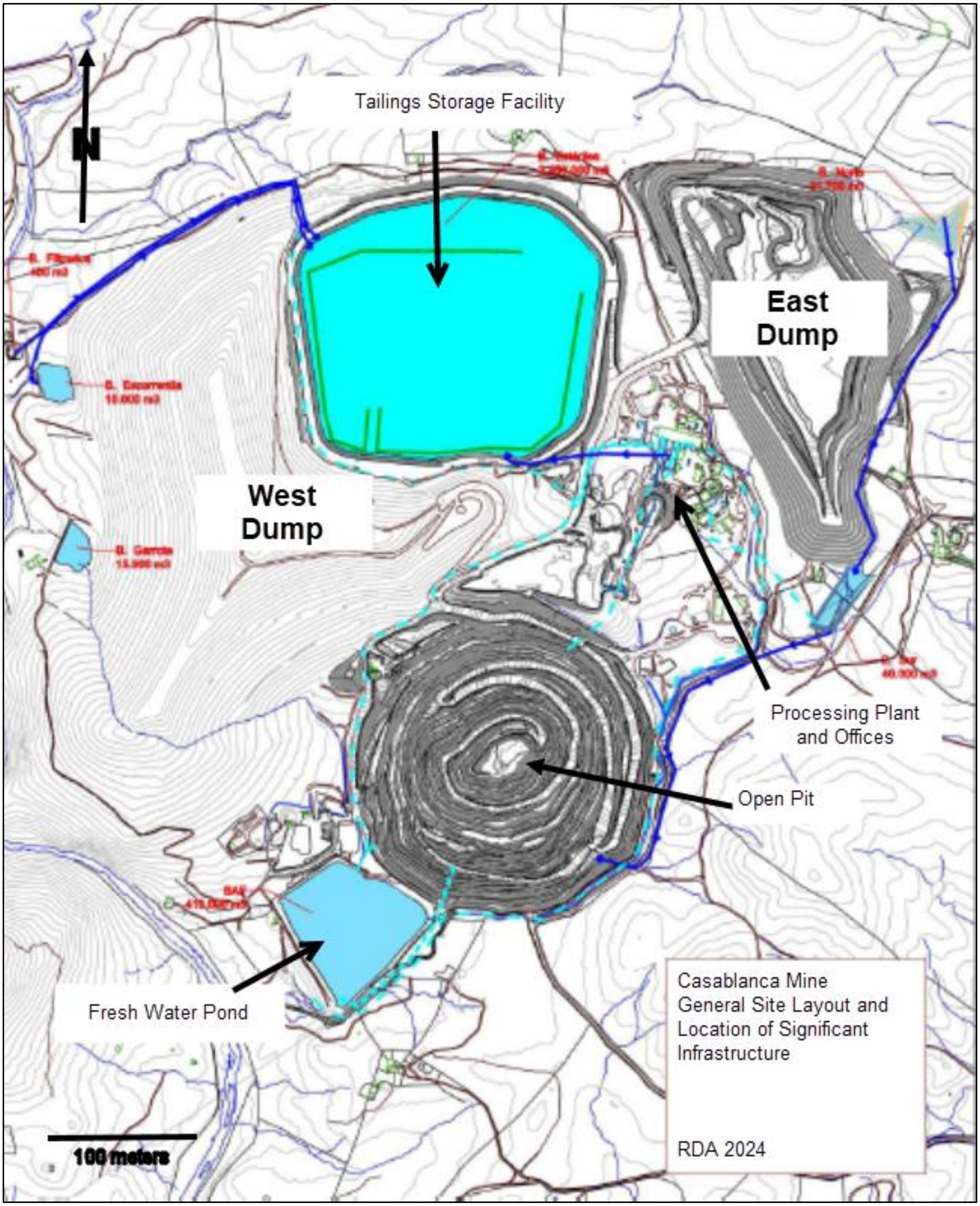

Figure 4-3 Aguablanca Mine Site Layout

Figure 4-4 Aerial Image of the Aguablanca Mine

4.2 MINING RIGHTS FOR AGUABLANCA – THE "DEFINITIVE RESERVE"

Mining rights were granted through a Definitive Reserve in favor of the State for the exploitation of mineral resources of iron, copper, gold, silver, lead, zinc, tin, wolfram, bismuth, platinum, molybdenum, chromium, nickel, cobalt, phosphates, palladium, osmium, rhodium and iridium, as well as the rest of the minerals associated with the platinum group, the area known as "Agua Blanca", inscription number 207 (La Monaguera), inscription number 380 (La Monaguera II) and inscription number 272 (La Remonta), located in the provinces of Badajoz, Huelva and Seville, for a period of 30 years. The Definitive Reserve is extendable.

In Spain, the Definitive Reserve in favor of the State is one of the conveyances of rights for the exploitation of minerals, by virtue of which the State sets aside areas of national territory in which the exploitation of one or more mineral deposits and other geological resources may be of special interest for economic and social development or for national defense (Article 8.1.c) of Law 22/1973 of 21 July 1973 on Mining (hereinafter, "LMi"). For its part, the State may decide to exploit the mineral resources directly, to call a public tender with private entities, or to enter into a Consortium with private companies, as in the present case.

4.3 TITLE TO AGUABLANCA

Royal Decree 1032/2003 of 25 July 2003 declared the Definitive Reserve in favor of the State, being the area known as 'Agua Blanca', entry number 207 (La Monaguera), entry number 380 (La Monaguera II) and entry number 272 (La Remonta), located in the provinces of Badajoz, Huelva and Seville, and expressly ceded to RNR the right to exploit the mineral resources of iron, copper, gold, silver, lead, zinc, tin, wolfram, bismuth, platinum, molybdenum, chromium, nickel, cobalt, phosphates, palladium, osmium, rhodium and iridium, as well as the rest of the minerals associated with the platinum group in the Reserve (Official State Gazette, number 187, 6 August 2003) (hereinafter, the "Definitive Reserve").

The term of the Definitive Reserve is established by virtue of the provisions of article 62.1 LMi, in relation to article 8.3 LMi, for a period of 30 years (until July 25, 2033, extendable by consecutive 30-year terms, up to a maximum of 75 or 90 years, depending on the interpretation of the LMi.

4.4 MINING AGREEMENTS WITH THE SPANISH STATE

On July 25, 2001, a Consortium Agreement was signed between the Spanish State and RNR in which, prior to the declaration of the Definitive Reserve, RNR was recognized as a member and Operator of the Consortium with the Spanish State for the development of mining exploration and exploitation in the provisional reserves "La Monaguera", "La Monaguera II" and "La Remonta", as provided for in Article 11.3.c) LMi. According to the Consortium Agreement, the shares in the exploration and exploitation phases of the mineral resources would be 50% for the Spanish State and 50% for RNR; however, in relation to the Definitive Reserve, during the exploitation phase RNR would acquire 50% of the Spanish State's share by paying a "Net Smelter Return" on smelter sales (hereinafter, "NSR") as consideration.

On September 24, 2003 a contract was signed between the Secretary of State for Energy, Industrial Development and RNR for the Operation of a zone in priority area 2 - Agua Blanca - Bodonal, to be known as the Agua Blanca Definitive Reserve (the "Operations Contract"), which, among other aspects, formalized the transfer of the operation of the Definitive Reserve to RNR for a period of 30 years, extendable, in exchange for payment of the aforementioned NSR, as well as regulating other aspects relating to the operation. According to the Operations Contract, RNR would pay a percentage of NSR sales according to the scale described in Table 4-1 below.

Table 4-1 NSR Royalty payable to the Spanish State

Average annual nickel price (US $/t)		
Minimum	**Maximum**	**Percentage**
0	4,945	0.5%
4,945	6,235	1.0%
6,235	7,740	1.5%
> 7,740		2.0%

According to the Definitive Reserve and with all agreements with the Spanish State, RNR is currently the entity that holds the rights to exploit the resources of Aguablanca.

4.5 ENVIRONMENTAL IMPACT STATEMENTS AND MINING PERMITS

By a Resolution of June 2, 2003, of the General Secretariat for the Environment, formulating the Environmental Impact Statement on the "Mining Project for the Aguablanca deposit", municipal district of Monesterio, Badajoz, located in 95 squares belonging to the provisional State reserves called La Monaguera and La Remonta, promoted by RNR (Official State Gazette number 144 of 17 June 2003) (hereinafter, the "Original EIS"), RNR was granted the environmental impact assessment for the exploitation of the Definitive Reserve. This resolution allowed the commencement of open pit mining at Aguablanca according to the mining rights which were granted in the Definitive Reserve.

Mining continued at Aguablanca until January 29, 2015 when RNR notified the Mining Authority that they were going to transition to underground mining of the Project. RNR had determined that open pit mining was no longer feasible.

July 21, 2015, the Minister of Industry, Energy and Tourism announced the suspension of underground mining. The order was to remain in effect until RNR furnished an updated environmental impact statement for underground mining.

By Resolution of 21 June 2017, of the Secretary of State for the Environment, an environmental impact statement was issued for the Aguablanca underground mining project (Monesterio-Badajoz) (Official State Gazette number 185, of 4 August 2017) (hereinafter, the "EIS"), with an initial validity of 4 years for the start of underground mining operations, starting from its publication in the Official State Gazette. The EIS establishes the environmental conditions for the approval of the project to exploit the Definitive Reserve by means of underground mining that would allow the restart of mining.

The EIS expressly provides that the conditions of the Original EIS, which do not conflict with the EIS, shall remain in force in relation to the pre-existing mining features and operations that are to remain unchanged.

Among the environmental conditions of the EIS is the need to obtain a groundwater concession for the groundwater flowing into the underground workings, which, as will be seen below, makes the effective restart of exploitation conditional on the granting of this concession (hereinafter, the "Water Concession")

By Resolution of 7 October 2021, the Directorate-General for Quality and Assessment granted an extension to the validity of the Environmental Impact Statement for the "Exploitation of the Aguablanca Mine (Monesterio, Badajoz) for a further two years, starting from the publication of the EIS in the Official State Gazette, i.e. until 4 August 2023, when work on the project needed to commence or the EIS would expire.

By Resolution of 20 July 2023, the Directorate General for Energy Policy and Mines approved "the Project for the Exploitation of the interior of the Definitive Reserve in favor of the State called 'Aguablanca' and authorized the updating of the associated Reclamation Plan" (hereinafter, the "Restarting Authorization").

The conditions of the Restarting Authorization include RNR's obligation to have a guarantee for a total amount of EUR 12,726,997.98 deposited with the competent Mining Authority to guarantee compliance with the Reclamation Plan, prior to restarting mining operations.

In a letter from RNR, dated 3 August 2023, the Mining and Environmental Authorities were informed of RNR's compliance with the conditions prior to the commencement of the works contemplated in the Restarting Authorization, including the deposit of the mining reclamation guarantees, and of the commencement of these works on 4 August 2023, within the period of validity of the EIS.

In this regard, the Restarting Authorization approves the resumption of work in a sequential manner in four phases. The mining work in the Definitive Reserve is currently in the initial phase of "Preparatory work", which ends with the drainage of the open-pit and underground working, after obtaining the Water Concession, in order to be able to begin the following phase II "Exploitation of the Upper Body (up to 75 meters above sea level)".

4.6 REGIONAL UNIFIED ENVIRONMENTAL AUTHORIZATION FOR THE PROCESSING PLANT

By Resolution of 5 March 2013, of the Directorate-General for the Environment of the Regional Ministry of Agriculture, Rural Development, Environment and Energy of the Regional Government of Extremadura, a Unified Environmental Authorization was granted for the mining operation of the Aguablanca deposit, promoted by RNR, in the municipality of Monesterio (Official Journal of Extremadura No 69 of 11 April 2013) (hereinafter the "UEA").

In response to RNR's prior consultation, dated 17 November 2015, the Environmental Protection Service of the Directorate General for the Environment of the Regional Ministry for the Environment and Rural, Agricultural and Territorial Policies of the Regional Government of Extremadura informed RNR that the change from open-pit to underground mining at the Aguablanca mine does not require modification of the granted UEA.

Finally, by Resolution, dated 21 December 2022, of the Directorate General for Sustainability of the Regional Ministry for Ecological Transition and Sustainability of the Regional Government of Extremadura, by which the procedure for the declaration of the loss of validity of the Unified Environmental Authorization held by RNR, AU 11/251 is closed, the validity of the current UEA was confirmed.

4.7 MUNICIPAL ACTIVITY LICENSE

In accordance with the Decree, dated 28 July 2023, of the Mayor's Office of Monesterio Town Council, regarding the ratification of the validity of the activity license for the extraction of nickel and industrial treatment plant in the deposit located in the area of Aguablanca de Monesterio, granted by Resolution of the Mayor's Office no. 403, dated 18 July 2005, the validity of the current municipal activity license for the development of the mining operation was ratified.

4.8 WATER CONCESSION

As mentioned in Section 4.5, one of the environmental conditions of the EIS is the need to obtain the Water Concession for the use of groundwater for the groundwater flows arising in the underground mine, which makes the effective resumption of exploitation of the Definitive Reserve conditional on the granting of the Water Concession.

Likewise, the Restarting Authorization approves the resumption of work in a sequential manner in four phases. Mining in the Definitive Reserve is currently in the initial phase of "Preparatory Work", which ends with the drainage of the open pit and underground workings, after obtaining the Water Concession. Meanwhile, the effective start of the exploitation of the mineral resources cannot start.

By letter dated 15 June 2022, RNR applied to the Guadalquivir River Basin Authority (hereinafter the "Water Authority ") for the granting of the Water Concession.

By means of Announcement 3309/2022, in the file A-2833/2022, of the Water Authority, for the application of the Water Concession by RNR, the process of Competence of Projects was initiated for three months from its publication in the Official Gazette of the Province of Badajoz, number 144, of 29 July 2022.

Subsequently, by means of Announcement 3906/2023, in the file A-2833/2022, of the Water Authority, of application for a Water Concession by RNR, the public information process of the Water Concession took place for 20 working days from its publication in the Official Gazette of the Province of Badajoz, number 138, of 20 July 2023.

According to the information received from RNR, the granting of the Water Concession is still being processed by the Water Authority, and no information is available on the status of the following procedures that may be pending:

- Field reconnaissance act
- Report from the Water Authority in relation to the documentation of the dossier
- Hearing of interested parties
- Report of the Abogacía del Estado (State Attorney's Office)
- Communication to RNR of the proposed concession conditions for acceptance
- Granting of Water Concession
- Publication of the Water Concession

4.9 DENARIUS MINERAL TENURE

On November 29, 2023, the Company entered into a definitive agreement (the "RNR Agreement") with the shareholders of RNR (collectively, the "RNR Shareholder Group") to acquire a 50% interest in RNR. Under the RNR Agreement, Alto acquired 50% of all of the issued and outstanding shares of RNR in an arm's length transaction with the RNR Shareholder Group for cash consideration totaling EUR 25 million (equivalent to approximately US$27 million), of which EUR 2.5 million was paid on signing of the RNR Agreement in 2023 and the balance was to be paid in instalments of EUR 5.0 million on March 31, 2024 and approximately EUR 5.8 million at the end of each of June, September and December 2024. Subsequent to December 31, 2023, pursuant to an amendment to the RNR Agreement dated March 19, 2024, the instalment due March 31, 2024 was modified with EUR 2.5 million to be paid on April 2, 2024 and the balance of EUR 2.5 million to be paid on May 24, 2024. The Company advises that it is currently negotiating a further revision to these payment dates and terms.

The Company agreed to pay a finder's fee of EUR 0.2 million to an unrelated third party, of which EUR 0.1 million was paid on April 2, 2024 and the balance will be paid on May 24, 2024.

In addition, the Company, through Alto, and the RNR Shareholder Group have entered into a Shareholders' Agreement (the "JV Agreement") pursuant to which Alto has appointed three members of the RNR board of directors and the RNR Shareholder Group has appointed the remaining three members. Pursuant to the JV Agreement, Alto will be the operator of the Aguablanca Project.

4.10 QP COMMENT

To the extent known, there are no other royalties, back in rights, payments or other agreements and encumbrances to which the Property is subject. To the extent known, there are no other environmental liabilities to which the Property is subject. To the extent known, all permits that must be acquired to conduct the proposed work on the Property have been obtained. To the extent known, there are no significant factors and risks that may affect access, title, or the right or ability to perform work on the Property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Aguablanca Mine is accessed via the paved, four lane N630 national highway from Seville. The highway runs approximately 100 km north from Seville towards the town of Santa Olalla del Cala. Exit the N630 and head east on the paved A-5301 for approximately 8.5 km to the town of El Real de la Jara. Then take the paved Camino de Aguablanca for approximately 3 km directly to the mine.

The Aguablanca region has a Mediterranean climate. Temperatures in the summer months (May to October), range from 10°C to 33°C.Temperatures in the cool winter months (November to April) range from 3°C to 18°C. Rainfall and light snow prevail in this period.

The Aguablanca Mine is at an elevation of 450 m to 500 m above sea level and the geography comprises low hills with moderate relief. The area is covered with trees and bushes forming the classical type of Mediterranean forest of the mountains of central and southern Spain. The area is vegetated with meadows comprising grass, oak trees and small olive trees. The area is sparsely populated with the population concentrated in small villages throughout the region.

The Project is reasonably close to several villages and readily accessible from major urban centers. The Project is well serviced by paved highways to Seville, Huelva and to several surrounding villages, which represent potential sources of labor, accommodation, and general services. Seville (population 700,000) is the administrative center of the Autonomous Community of Andalucía. High-speed train service links the regional towns of Cordoba, Seville and Huelva with the capital Madrid. There are many international flights that connect the provincial cities of Seville and Malaga with Madrid and other major cities in Europe and North America.

Aguablanca has adequate surface rights to operate the mine and to conduct surface exploration activities.

The Project has ample power distribution, sufficient sources of water, mining personnel, existing tailings storage areas, waste disposal areas, and a functional processing site.

6 HISTORY

6.1 OWNERSHIP

Exploration for nickel and copper deposits has been carried out in the area since the mid nineteen eighties. In 1986, two areas known as La Monaguera and La Remonta, which covered a total area of over 620,000 ha in the Provinces of Huelva and Badajos, were declared a State Reserve for Mineral Investigation in which investigation rights were awarded to Presur, the State Corporation which later entered into a joint venture agreement with Rio Tinto Minera (RTM). In 1993, the Presur/Rio Tinto Minera Joint Venture discovered the Aguablanca target through geochemical stream sediment and soil sampling and in 1994 discovered the Aguablanca Ni-Cu deposit.

During 1996, Freeport McMoRan Copper and Gold Company acquired Rio Tinto's Spanish assets and later transferred their interest in the deposit to the newly formed subsidiary, Atlantic Copper S.A.

By March 1997, the State Reserves were categorized as either a "Priority Area", in which Atlantic and Presur each had a 50% interest with the remainder categorized as "Secondary Area" in which Presur had a 100% interest. Under the terms of the Joint Venture agreement Atlantic had the option to include areas within the "Secondary Area" in the "Priority Area", thereby giving Atlantic the right to a 50% interest.

In March 2001, Rio Narcea Recursos (RNR), a wholly owned Spanish subsidiary of Rio Narcea Gold Mines Ltd (RNGM) signed an option agreement with Atlantic Copper S.A. and the Spanish Government and in mid-2001, after due diligence work, exercised its option to acquire 100% interest in the Aguablanca Project by purchasing 50% interest in the Monaguera Mineral Reserve from Atlantic Copper S.A.

Commercial production started in January 2005 and the first nickel-copper concentrate was sold in May 2005. In conjunction with the commencement of open pit operations, a 2,700 m long ramp to access the mineralization below the Aguablanca open pit was developed and completed in 2005. This facilitated commencement of an underground exploration drilling program to test the continuation of mineralization at depth. This was completed in 2008.

The Aguablanca mine was acquired by Lundin Mining in July 2007, through its purchase of RNGM.

In December 2016 RNR was acquired by Valoriza Minera from Lundin Mining.

On November 29, 2023, Denarius, through its wholly owned subsidiary Alto Minerals, acquired a 50% interest in RNR, giving it the rights to exploit Aguablanca.

6.2 HISTORICAL MINERAL RESOURCE ESTIMATES

6.2.1 ORE RESERVES ENGINEERING MINERAL RESOURCE ESTIMATE

In August 2000, a Mineral Resource was estimated by Alan C. Noble of O.R.E. Reserves Engineering, Colorado USA (ORE). This estimate was based on a three-dimensional geological block model within an envelope of 0.1% Ni and was estimated using a 0.2% Ni cut-off based on a 4*4*4 block model using the results of drilling completed by RTM + RNR. Block grades for Ni, Cu, Au, Co, Pt + Pd were interpolated using IDP methods.

As an internal check, ORE utilized three other methods of estimating Mineral Resources and reported that the tonnage and grade estimates varied by about 8%.

The historical Measured and Indicated Resources of ORE estimate were reported as 19.7 million tonnes at an average grade of 0.66% Ni, 0.48% Cu, 0.50 g/t PGE (combined), 0.13 g/t Au and 0.02% Co. ORE also estimated an Inferred Resource of 6 million tonnes at an average grade of 0.53% Ni, 0.46% Cu, 0.40 g/t PGE (combined), 0.13 g/t Au and 0.02% Co.

Though the ORE mineral resources were reported as measured, indicated and inferred, it is unclear that the categorization was completed in conformity to NI 43-101.

6.3 AGUABLANCA PROCESS PLANT PRODUCTION HISTORY

The mineral processing plant within the Aguablanca mining operation was designed by Fluor Corporation to recover nickel and copper values from open pit ore via the froth flotation process. The plant was built during 2003 and 2004 and went into production in 2005 and operated until December of 2010 when it was shut down due to open pit instability. In order to maintain process plant design daily throughput levels in the 5,000 tpd range during 2009 and 2010, fresh open pit ore was supplemented with partially oxidized stockpile material. Due to a landslide, operation was stopped during 2011 and part of 2012 to carry out reinforcement works in the southwest wall of the mine. The plant resumed operations in August 2012 after the open pit stability issues were rectified and continued operating until being shut down at the end of the open pit life in December 2015. The Property production summary is shown in Table 6-1.

Table 6-1 Processing Production History

	2005	2006	2007	2008	2009	2010	2011	2012	2013	2014	2015	Total
Ore Milles, ktonnes	997	1,468	1,669	1,825	1,913	1,435	-	577	1,606	1,660	1,292	14,442
Ni Head Grade, %	0.76%	0.60%	0.52%	0.56%	0.55%	0.54%	-	0.51%	0.57%	0.63%	0.74%	0.59%
Cu Head Grade, %	0.55%	0.50%	0.41%	0.42%	0.41%	0.41%	-	0.42%	0.41%	0.47%	0.54%	0.45%
Ni Recovery, %	71.0%	72.6%	76.2%	80.2%	77.0%	81.4%	-	81.4%	82.8%	82.5%	75.7%	78.0%
Cu Recovery, %	89.1%	90.1%	91.6%	93.1%	90.1%	93.0%	-	92.2%	93.9%	93.9%	89.5%	91.6%
Concentrate tonnes	95,363	97,182	91,146	115,340	119,035	92,193	-	35,361	111,439	132,319	109,713	999,091
Ni Con Grade, %	5.64%	6.58%	7.27%	7.05%	6.75%	6.83%	-	6.78%	6.80%	6.52%	6.57%	6.67%
Cu Con Grade, %	5.13%	6.81%	6.89%	6.13%	5.87%	5.95%	-	6.39%	5.60%	5.58%	5.67%	5.95%
Ni Production, tonnes	5,381	6,398	6,630	8,136	8,029	6,296	-	2,398	7,574	8,631	7,213	66,687
Cu Production, tonnes	4,888	6,616	6,281	7,071	6,989	5,484	-	2,260	6,242	7,390	6,221	59,441

Note the nickel and copper recovery increases between 2007 and 2008. The crushing plant was originally built with only a jaw crusher. During 2007, a cone crusher was added to the crushing circuit to produce a finer final crusher product which contributed to increased plant throughput and recovery. In addition, the nickel flotation scavenger circuit was expanded during 2010 to increase retention time. According to a competent person with extensive experience at the Aguablanca process plant, the scavenger circuit expansion lead to a 2% nickel recovery increase.

The plant was shut down at the end of 2015 according to equipment manufacturers' recommendations and has been well maintained since being shut down. Work required to restart the plant is expected to be minimal.

The Aguablanca tailing storage facility and plant site are presented in Figure 6-1.

Figure 6-1 Aguablanca tailing storage facility and process plant

7 GEOLOGICAL SETTING AND MINERALIZATION

The following summary section is excerpted from the excellent published work of Rubén Pińa, *The Ni-Cu-(PGE) Aguablanca Ore Deposit (SW Spain)*. The publication is very well referenced from hundreds of reports, studies and other documents. It is highly recommended to obtain this report for a thorough understanding and description of the geology of Aguablanca. The only changes to the text were to apply nomenclature in compliance with NI 43-101 and CIM Best Practices Guidelines.

7.1 REGIONAL GEOLOGY

7.1.1 THE OSSA-MORENA ZONE

The Aguablanca Ni–Cu sulfide mineralization occurs within the Aguablanca stock, a small (approximately 3 km2 in area) subcircular mafic intrusion located in the northern part of the Santa Olalla Igneous Complex (SOIC). The SOIC is located in the southern limb of the Olivenza-Monesterio antiform, a major WNW-ESE trending Variscan structure, occupying a central position within the Ossa-Morena Zone (OMZ) (Figure 7-1). This zone comprises one of the SW divisions of the Iberian Massif, which corresponds to the westernmost outcrops of the Variscan orogen in Europe. A detailed geological review of the Iberian Massif is given by Quesada (1991) and references therein.

The OMZ is located between the Central Iberian Zone (CIZ), to the north, and the South Portuguese Zone (SPZ), to the south (Figure 7-1a). The boundary between the OMZ and the CIZ is marked by the Badajoz-Córdoba shear zone, a tectonic unit interpreted as a Cadomian suture reactivated as an intraplate shear zone during the Variscan orogeny or a real Variscan suture. The contact with the SPZ, to the south, is marked by the Beja-Acebuches amphibolites and the Pulo do Lobo accretionary prism. These units have been interpreted as remnants of the Rheic Ocean accreted to the southern margin of the OMZ during the oblique Variscan convergence. These remnants delineate the Variscan suture between Gondwana and Laurussia during the formation of Pangaea.

The structural evolution of the OMZ during the Variscan orogeny (from Middle Devonian to Early Permian) was mainly governed by transpressional tectonics. This transpressional regime resulted in the formation of a dominant thick-skinned transcurrent architecture delineated by oblique thrust nappes, folds, and strike-slip fault associations. Internal deformation of each horst is variable and includes several folding and oblique thrust generations. The Olivenza-Monesterio antiform is a major Variscan structure in the OMZ. It was generated by basement-involved ductile thrusting which gave rise to a basement antiformal stack, while the Paleozoic cover detached from the basement and initially formed a typically thin skinned, SW-verging imbricate thrust fan and regional associated recumbent folds to the south. A second folding event, also SW-vergent but characterized by steep axial planes, affected the already deformed thin-skinned imbricate thrust fan where some syn-orogenic basins were formed. Transpression is indicated by existence of a major sinistral component during the entire evolution in the main NW-SE thrust structures, anticlockwise fold transection and associated (NE-SW trending) synthetic and (NNW-SSE trending) antithetic Riedel faults. The sigmoidal map pattern that characterizes the OMZ represents the cumulative result of all these elements.

7.1.2 MAGMATISM

The complex tectonic evolution of the OMZ was accompanied by intense magmatism which can be divided into three main stages well represented in the Olivenza-Monesterio antiform (Figure 7-1b): (1) subduction-related magmatism during the Neoproterozoic Cadomian orogeny, (2) Cambrian-Ordovician magmatism related to rifting leading to the opening of oceanic crust (Rheic ocean), and (3) Carboniferous Variscan collision- and extension-related magmatism. The main magmatism related to the Cadomian orogeny corresponds to calc-alkaline volcanic and plutonic rocks (mainly, amphibolites and diorite-granites), showing a typical subduction arc signature and dated in the range c. 587-532 Ma. The Cambrian-Ordovician rifting event was accompanied by bimodal igneous activity of tholeiitic and alkaline affinity in the range c. 530-470 Ma. The last main magmatic event recorded in the OMZ took place during the Variscan orogeny. It was characterized by intermediate to felsic calc-alkaline compositions ranging from metaluminous tonalite and granodiorite to peraluminous granite and leucogranite, with volumetrically minor gabbros. Numerous sub circular plutons were emplaced throughout the Olivenza-Monesterio antiform in the range c. 350-330 Ma (Figure 7-1b): Valencia del Ventoso, Bazana, Brovales, Valuengo and Burguillos del Cerro. Separated from this group of plutons, 50 km to the SE, is the Santa Olalla Igneous Complex that includes the Aguablanca stock, the small mafic igneous body that hosts the Aguablanca Ni-Cu sulfide mineralization.

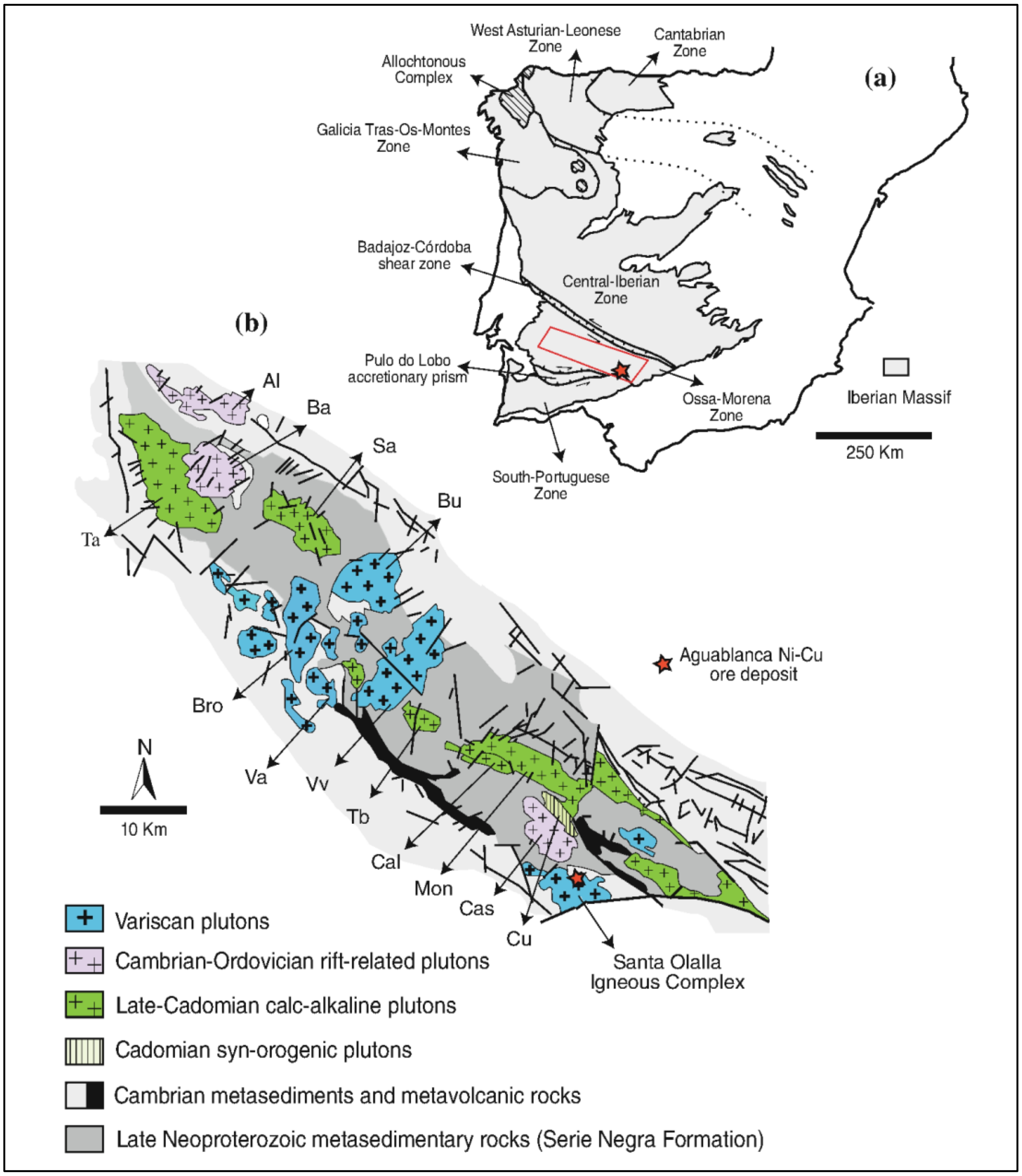

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Figure 7-1 Santa Olalla Igneous Complex – Ossa Moreno Zone

7-1a: Sketch of the Iberian Peninsula showing the main zones of the Iberian Massif and the location of the Olivenza-Monesterio antiform that hosts the Aguablanca ore deposit; **7-1b:** Simplified geological map of the Olivenza-Monesterio antiform showing outcrop areas of the main igneous bodies and the location of the Aguablanca Ni–Cu ore deposit in the north boundary of the Santa Olalla Igneous Complex. Cu: Culebrín, Cas: Castillo, Mon: Monesterio, Cal: Calera de León, Tb: Tablada, Vv: Valencia del Ventoso, Va: Valuengo, Bro: Brovales, Ta: Táliga, Al: Almendral, Ba: Barcarrota, Sa: Salvatierra, Bu: Burguillos del Cerro

7.1.3 THE IBERIAN REFLECTIVE BODY

A deep seismic reflection profile (IBERSEIS) running 303 km long across the northern South Portuguese Zone, Ossa-Morena Zone and southern Central Iberian Zone revealed the existence of a 140 km-long and up to 5 km-thick high amplitude reflectivity body, called the Iberian Reflective Body, at about 10-15 km depth throughout the Ossa-Morena Zone (Figure 7-2). From its relationships with major structures, the Iberian Reflective Body has been interpreted to represent a large layered mafic-ultramafic intrusion formed during the activity of a syn-orogenic mantle plume in Early Carboniferous times (~340-350 Ma. This mantle plume was likely active during a short-lived intra-orogenic extensional event intermediate between two main transpressional tectonic regimes associated with a subduction-related magmatic arc (Figure 7-3). Alternatively, it has been postulated for the Iberian Reflective Body an origin related to regional compression decollement, which became a zone of major crustal decoupling resulting in the injection of juvenile magmas. Almost simultaneously to the discovery of the Iberian Reflective Body was found a high-conductivity layer at a depth of 15-20 km below the whole Ossa-Morena Zone spatially correlated with the Iberian Reflective Body. It was concluded that the mafic-ultramafic intrusions could not by themselves explain the high conductivity of the layer, because igneous rocks typically exhibit low conductivity. Alternatively, it is suggested the presence of conductive graphite-rich shale screens, likely belonging to the Precambrian Serie Negra Formation, intercalated among multiple sheet-like mafic intrusions. More recently, the existence of this mafic-ultramafic magmatism in the middle crust of the Ossa Morena Zone has been also supported by the geophysical studies, such that the emplacement of voluminous mafic-ultramafic magmas in the middle crust of the Ossa-Morena Zone in Early Carboniferous times is well established. It is also suggested that dismembered parts of this mafic-ultramafic rocks crop out in the Aracena located adjacent to the south Portuguese Zone.

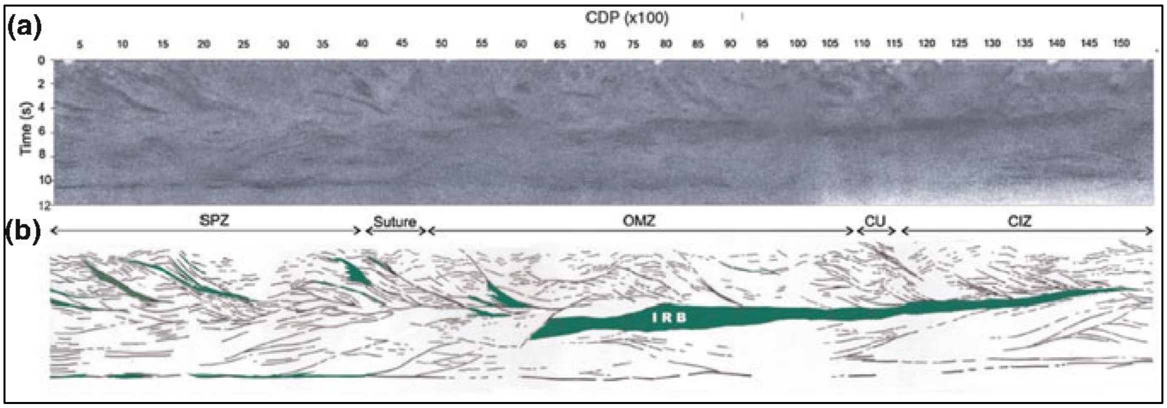

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Figure 7-2 Deep seismic reflection profile

7-2a Time migrated stack image of the IBERSEIS deep seismic reflection profile. **7-2b** Line drawing interpretation of the seismic reflection profile showing the location of the main tectonic units: SPZ, South Portuguese Zone; OMZ, Ossa-Morena Zone; CIZ, Central Iberian Zone; Suture, accretionary complex between SPZ and OMZ; CU, accretionary wedge between OMZ and CIZ. The green body indicates the location and geometry of the Iberian Reflective Body (IRB) (Reprinted from Simancas et al. 2006, with permission from The Geological Society of London)

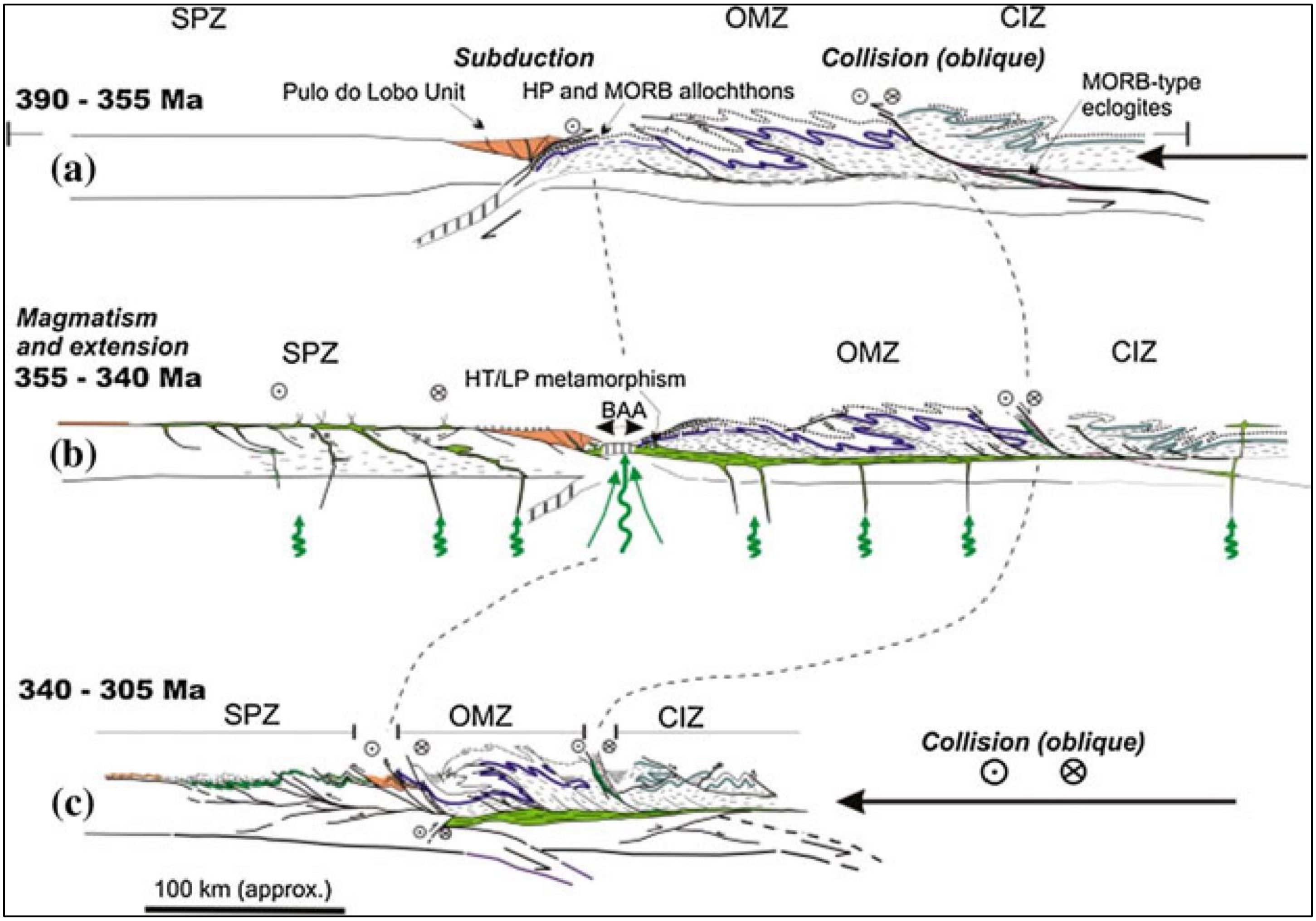

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Figure 7-3: Schematic cartoon of the geological evolution of the SW Iberia during the Variscan orogeny (Middle Devonian—Late Carboniferous) showing the emplacement of the Iberian Reflective Body (IRB, green) during an intra-orogenic extensional event intermediate between two main collisional events.

Several studies have highlighted the regional implications of the emplacement of this huge volume of magmas in the Ossa Morena Zone. It is suggested that gabbros and diorites intruded in the OMZ during the Variscan orogeny (at ca. 350 Ma) probably came from the Iberian Reflective Body magmatic chamber. Similarly, it is suggested that the Iberian Reflective Body magmatism promoted melting, assimilation and mixing processes between mafic and felsic magmas that ultimately resulted in the formation of hybrid magmas later emplaced into the OMZ upper crust. Many have pointed to the emplacement of the Iberian Reflective Body as the main heat source for the high-temperature, low-pressure metamorphism developed in the Evora-Aracena-Lora del Rio metamorphic belt of the OMZ. Other studies genetically link the Iberian Reflective Body magmatism with the metallogeny in the OMZ. For example, the source of magmas and fluids responsible of the Ni-Cu sulfide mineralization in Aguablanca and several occurrences of IOCG mineralization, respectively, in the Ossa Morena Zone, was probably the Iberian Reflective Body magmatism. It is also suggested that the Iberian Reflective Body below the south part of the Central Iberian Zone could be involved in the genesis of Ni-Cu sulfide mineralization in gabbros and norites located in the northern part of Extremadura region, far away from the Aguablanca deposit.

7.1.4 THE SANTA OLALLA IGNEOUS COMPLEX

The SOIC is a subrounded (upto7 km long) calc-alkaline plutonic group structurally located in a wedge bounded by two main faults: the Cherneca Fault to the north, and the Zufre Fault to the south (Figure 7-4). The SOIC is formed by two main plutons, the Santa Olalla intrusion and the Aguablanca mafic stock. The Santa Olalla intrusion, the largest pluton of the complex, shows a reverse compositional zoning with amphibole-biotite quartz-diorite in its north side grading to tonalite in the center and monzogranite toward the southern limit. Tonalite is the most abundant igneous rock type and consists of an equigranular, medium to coarse-grained leucocratic rock (Figure 7-5a) composed of zoned plagioclase, quartz, biotite and amphibole locally containing relicts of pyroxene. The intrusion contains variably digested gabbro and leucogranite enclaves (Figure 7-5a, b), synplutonic dykes and a wide variety of disequilibrium and reaction microstructures (Bateman et al. 1992).

The Aguablanca mafic stock, the host of the Ni-Cu sulfide ores, occurs in the northern part of the SOIC, adjacent to the Cherneca ductile shear zone (Figure 7-6). The Aguablanca stock consists of a small (~3 km2) subcircular intrusive body of dark medium-grained gabbroic rocks (Figure 7-5c). In detail, the Aguablanca rocks comprise cumulate-textured hornblende-bearing gabbronorite and minor gabbro and norite, grading to the south into diorite and quartz-diorite. These rocks commonly host leucocratic enclaves (Figure 7-5d) and, at the southern parts of the stock close to the contact with the Santa Olalla tonalite, gabbro shows mingling textures with felsic hybrid rocks (Figure 7-5e). Most rocks of the Aguablanca intrusion contain no Fe-Ni-Cu sulfides, the sulfide mineralization being restricted to the magmatic breccia, although small sulfide patches can be locally found where the igneous rocks envelope partially-digested xenoliths of country sedimentary rocks with no visible sulfides (Figure 7-5f).

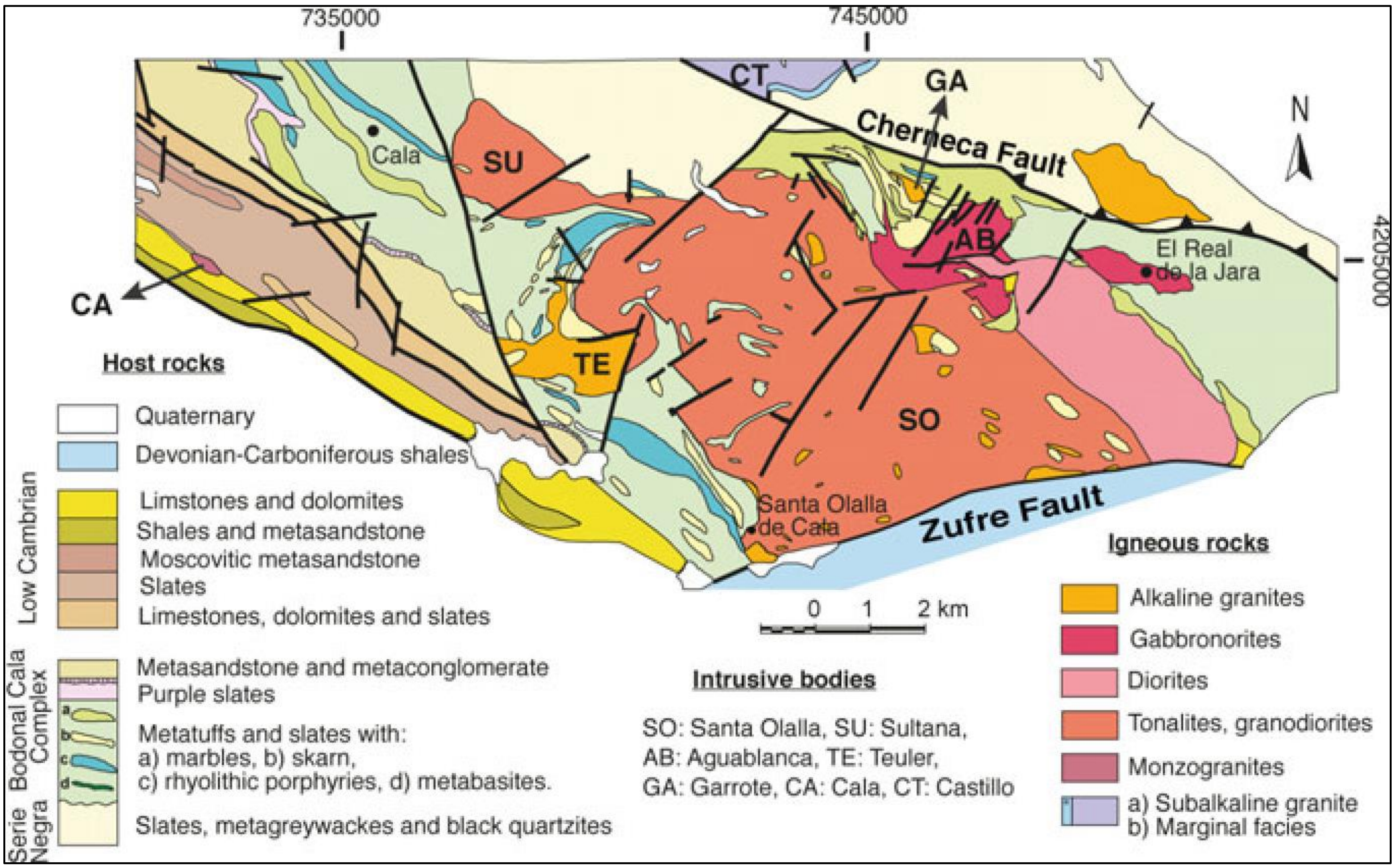

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Figure 7-4: Geological map of the Santa Olalla Igneous Complex showing the location of intrusive bodies and the main fault structures, Zufre and Cherneca. Abbreviations: AB, Aguablanca, SO, Santa Olalla; SU, Sultana; TE, Teuler; GA, Garrote; CA, Cala; CT, Castillo

The petrography and mineral chemistry of the Aguablanca igneous rocks have been described in detail. Hornblende-bearing gabbronorite is a medium to coarse-grained meso- and orthocumulate containing variable amounts of orthopyroxene (27-48 modal %, Mg# 0.85-0.78), plagioclase (23-47 modal %, An79-51) and clinopyroxene (4-11 modal %, Mg# 0.89-0.83), and interstitial green-brown hornblende (10-21 modal %, Mg# 0.86-0.70), phlogopite (<5 modal %, Mg# 0.79-0.73) and minor quartz (<1 modal %) (Figure 7-7 a-d). Primary green-brown hornblende and phlogopite often include poikilitically pyroxene and plagioclase crystals (Figure 7-7d). Locally, gabbronorites show subhorizontal layering at decimetre scale consisting of variations in the plagioclase modal contents. At the south, quartz diorite occurs as variably thick masses (up to 150 m) between gabbronorite with transitional contacts. The quartz diorite is composed of cumulus plagioclase (56-63 modal %, An45-37), clinopyroxene (<5 modal %, Mg# 0.77-0.73) and orthopyroxene (<7 modal %, Mg# 0.67-0.62), and intercumulus phlogopite (7-18 modal %, Mg# 0.63-0.60), quartz (5-15 modal %) and amphibole (<7 modal %) (Figure 7-7e, f). In all rock-types, primary mineralogical assemblages are variably altered to a secondary assemblage comprising actinolite, chlorite, bastite, talc, carbonates, serpentine, albite, sericite, and epidote-zoisite group minerals (Figure 7-7g, h).

Around the SOIC, other minor granitic bodies occur: the Sultana, Teuler and Garrote intrusions (Figure 7-1). Sultana, located at the NW of SOIC, consists of a small mafic apophysis composed of hornblende-biotite tonalite and quartzdiorite. The Garrote intrusion is a small (<1 km2) alkaline hornblende-bearing syenitic granite located close to the northern boundary of the Aguablanca stock. Finally, the Teuler intrusion is a fine-grained biotite-rich monzogranite located at the W of SOIC that has generated a magnesian skarn with stratiform magnetite mineralization. The SOIC includes some minor intrusion-related Cu-Au veins and a large calcic skarn adjacent to a minor apophysis to the SW.

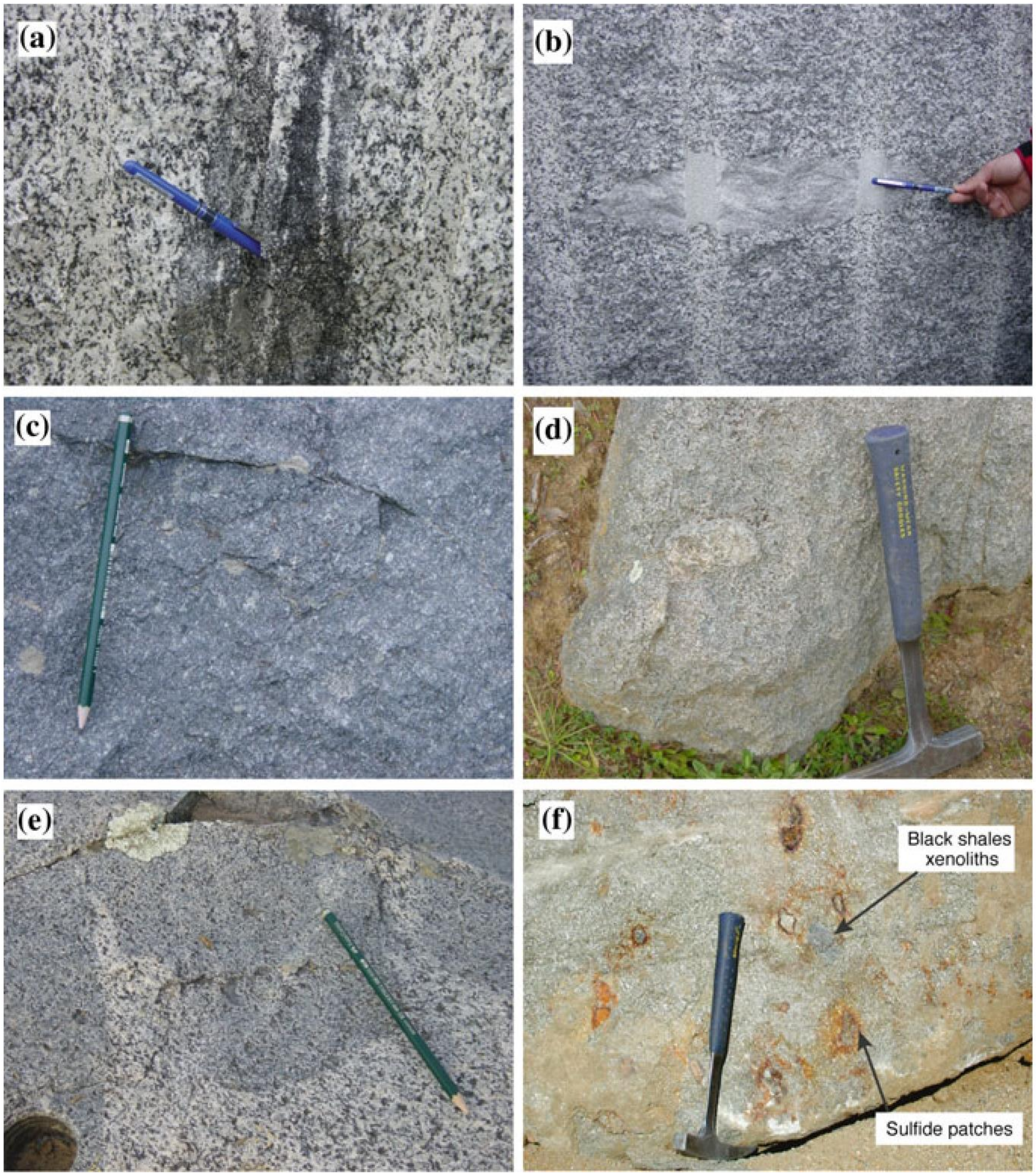

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Figure 7-5: Tonalite rocks from the Santa Olalla intrusion enveloping mafic (a) and leucratic (b) enclaves. (c) Dark gabbroic rocks from the Aguablanca stock. (d) Leucocratic enclave in Aguablanca mafic rocks. (e) Mingling texture between leucocratic and mafic igneous rocks in the bed of the Ribera de Cala River, close to the contact between Aguablanca and Santa Olalla intrusions. (f) Sulfide patches hosted by Aguablanca gabbroic rocks associated with partially digested xenoliths of black shales.

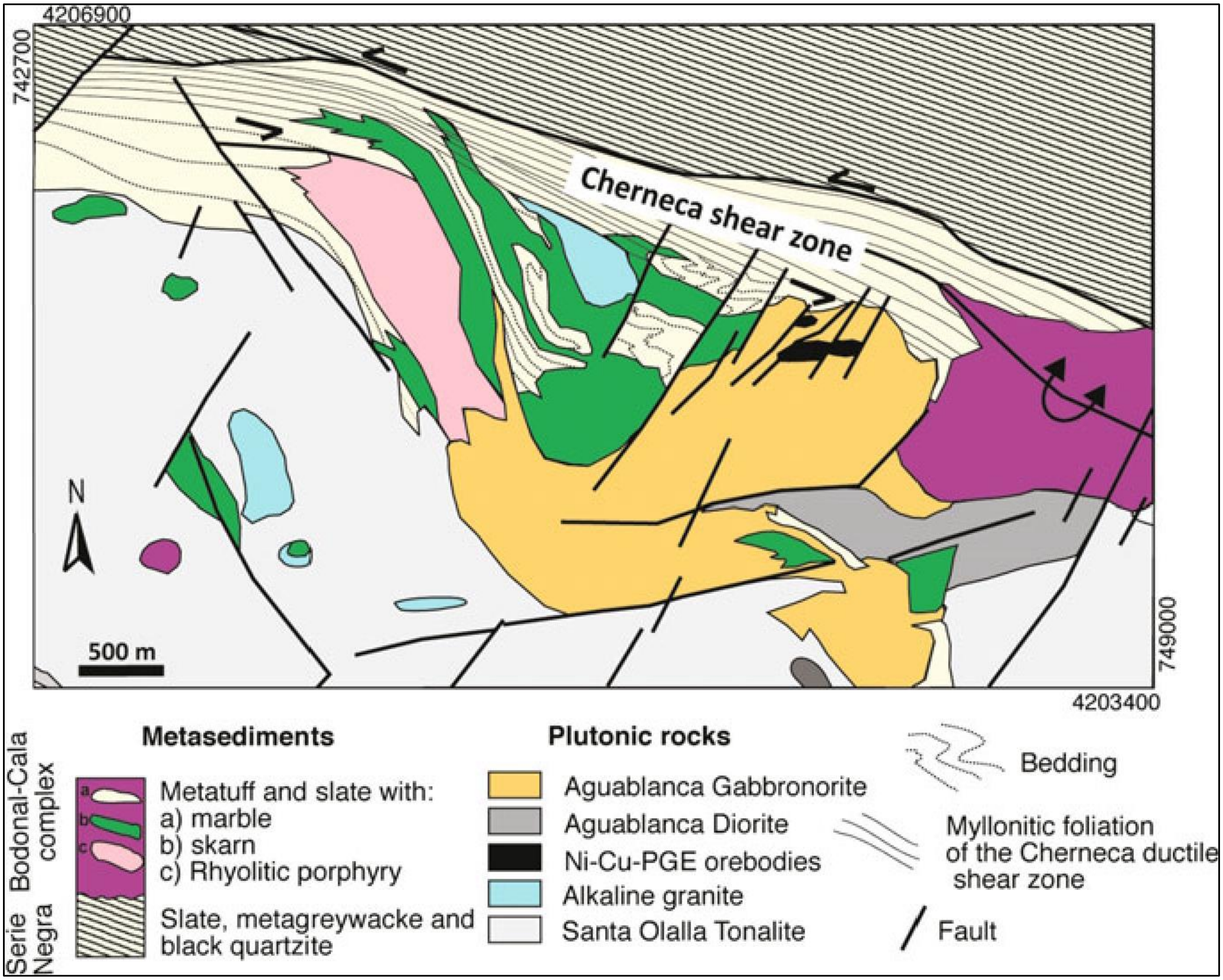

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Figure 7-6: Geological map of the Santa Olalla Igneous Complex showing the location of intrusive bodies and the main fault structures, Zufre and Cherneca. Abbreviations: AB, Aguablanca, SO, Santa Olalla; SU, Sultana; TE, Teuler; GA, Garrote; CA, Cala; CT, Castillo

7.2 LOCAL GEOLOGY

7.2.1 HOST ROCKS

The SOIC intruded in two different stratigraphic units regionally metamorphosed to greenschist facies (Figure 7-1). In the NW margin, the SOIC intruded metasedimentary rocks composed of alternating pyrite-rich black slates and metagraywackes with minor intercalations of metavolcanic rocks and black quartzites (Figure 7-8a). These rocks correspond to the Tentudia succession, the upper part of the Late Neoproterozoic Serie Negra Formation, probably the most representative formation of the OMZ, reaching up to 15-25 km depth, which crops out along the central area of the Olivenza-Monesterio antiform. The Serie Negra Formation consists of different graphite- and pyrite-bearing rock-types including metacherts, quartz phyllilites, mica schist, metagreywackes and parageneises. An established maximum age of deposition for this formation is 564±9 Ma. Sulfur content in these rocks is variable but can reach up to 3000 ppm. The S-rich black slates of the Serie Negra Formation are considered as the favorite contaminant providing the S necessary to the Aguablanca magma to reach the sulfur saturation and, hence, form the sulfide mineralization.

At the E and W, the SOIC intruded a volcano-sedimentary sequence formed by tuffs and porphyritic rhyolites (Figure 7-8b) of calc-alkaline affinity and variable-thick carbonate rocks at the top. This succession corresponds to the Cambrian Bodonal Cala Complex (530 ± 3 Ma, U-Pb ID-TIMS on zircons) that uncomformably overlies the Serie Negra Formation.

The contact between the Aguablanca stock and Bodonal Cala rocks is well exposed in the open-pit (Figure 7-8c). It is a well-defined, sharp intrusive contact (Figure 7-8d, e). Along the northern contact with the Bodonal Cala carbonate rocks, the intrusion produced a well-developed exoskarn (~2 km wide) characterized by garnetite, marble and calc-silicate rocks (Figure 7-8f. The skarn mineralogy comprises scapolite, epidote, garnet, actinolite and clinopyroxene, with minor amounts of pyrite, pyrrhotite, sphalerite, chalcopyrite and galena. The marbles were affected by penetrative ductile deformation related to the sinis-tral transpressional kinematics of the Cherneca shear zone. Decimetric-sized xenoliths of skarn, marble and calc-silicate rocks are locally found in the intrusive rocks near to the contact. Casquet (1980) estimated temperatures up to 750 °C (i.e., hypersthene hornfels facies) within the thermal aureole, near to the contact with the Aguablanca stock and inferred a depth of emplacement of 1.7-3.5 km from metamorphic mineral equilibria. The Santa Olalla intrusion shows numerous roof pendants of host country rocks scattered within igneous rocks, mostly skarnified limestone, interpreted as having come from the subhorizontal upper contact.

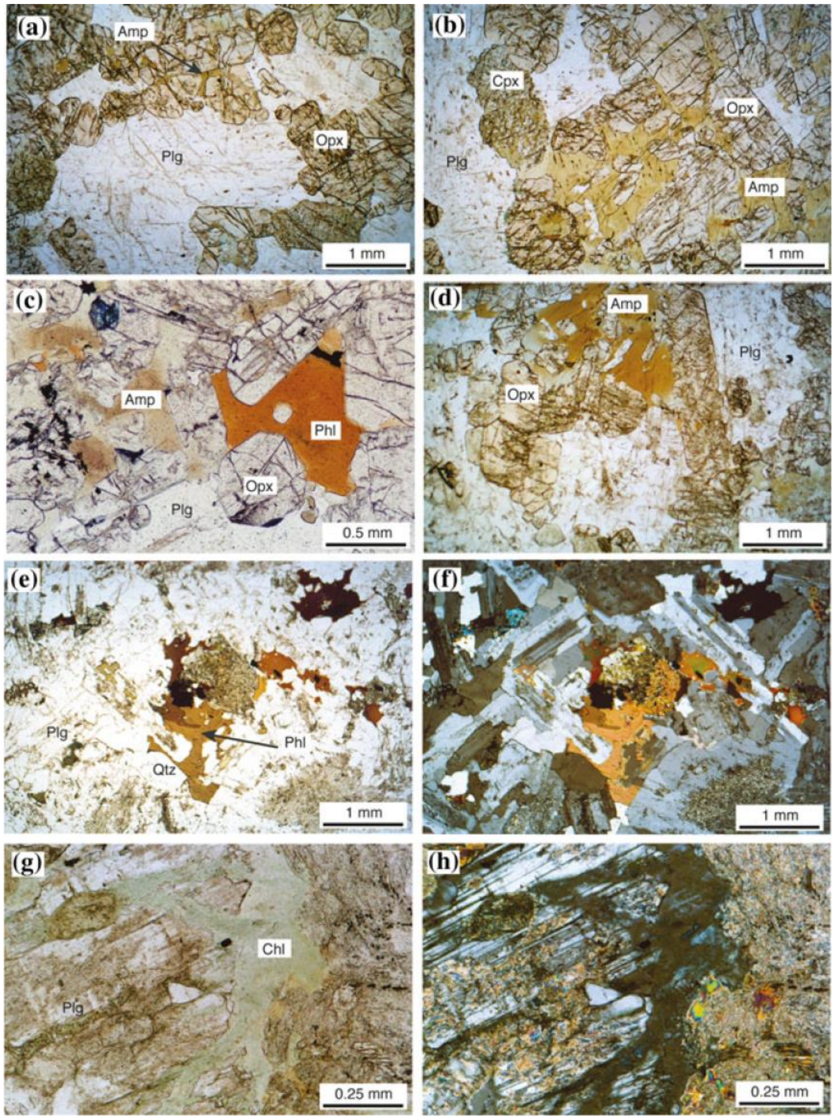

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Figure 7-7: Representative transmitted light photomicrographs of Aguablanca mafic rocks. a-d Hornblende-bearing gabbronorite showing euhedral to subhedral pyroxene and plagioclase crystals with interstitial amphibole and phlogopite grains. e, f Quartzdiorite with plagioclase, quartz and interstitial phlogopite (plane-polarized and cross-polarized light, respectively). g, h Partially-sericitized plagioclase replaced by chlorite along grain boundary in gabbronorite rock (plane-polarized and cross-polarized light, respectively)

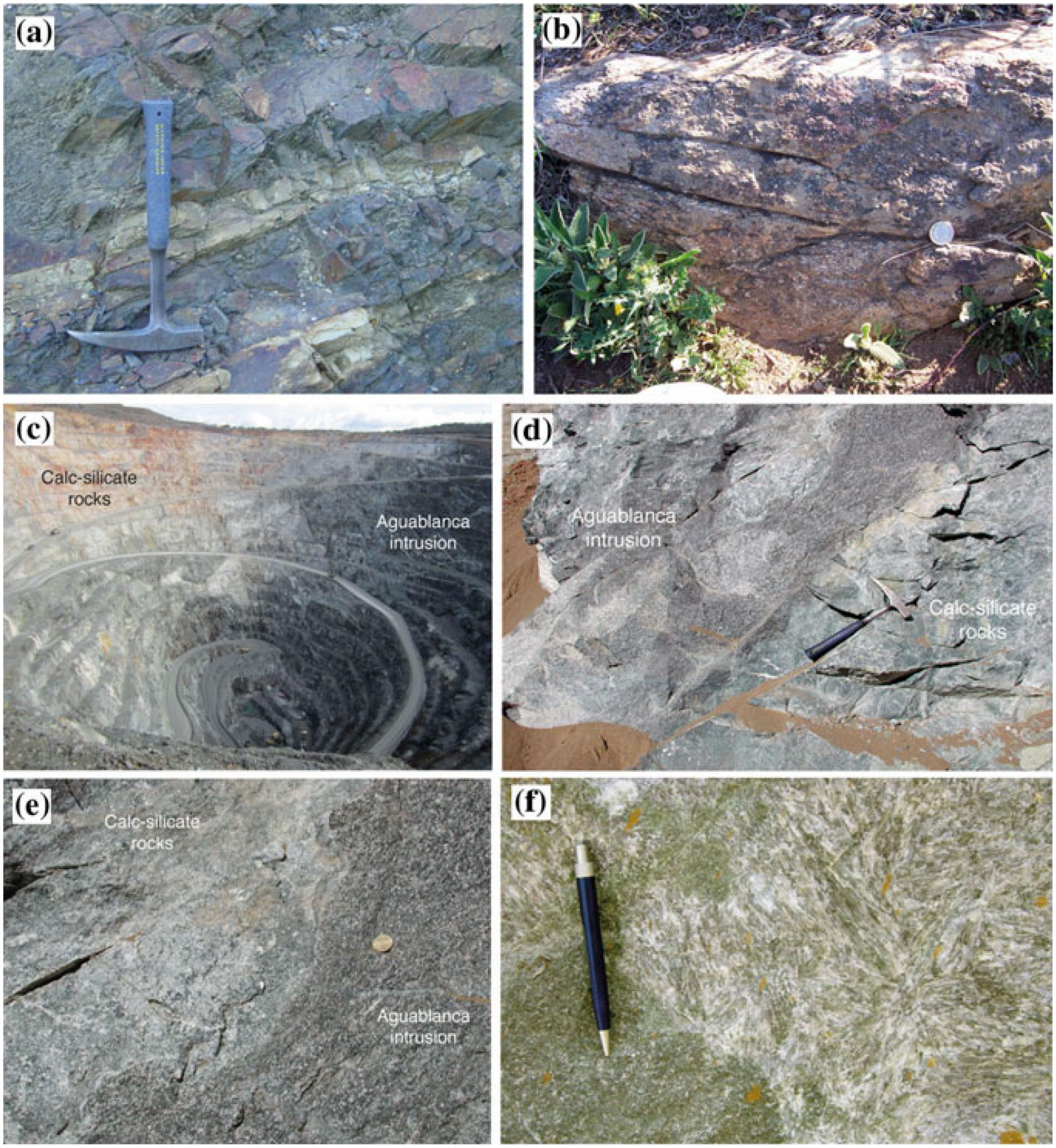

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Figure 7-8: a Intercalations of black slates and metagreywackes from the Tentuda succession of the Late Neoproterozoic Serie Negra Formation. b Volcano-sedimentary rocks from the Cambrian Bodonal Cala Complex. c Open-pit of Aguablanca showing the contact between the calc-silicate rocks to the north with the ore-bearing Aguablanca intrusion to the south. d, e Detail of the intrusive contact between the calc-silicate rocks and the Aguablanca intrusion. f Skarn developed in the contact with the Aguablanca rocks composed of variable amounts of actinolite, epidote, garnet and scapolite

7.2.2 GEOCHEMISTRY

The SOIC is characterized by a high-K calc-alkaline affinity, similar to other nearby intrusions such as Burguillos and Brovales. Rocks from the Santa Olalla intrusion and Aguablanca stock exhibit a well-defined trend of increasing in SiO_2,Al_2O_3, alkali and trace incompatible elements such as Rb and Ba with decreasing MgO (Figure 7-9). FeOt and Cr_2O_3 significantly decrease with decreasing MgO. Igneous facies of the Santa Olalla intrusion (namely, tonalite, diorite and monzogranite) have high SiO_2 (56-68 wt%), Al_2O_3, K_2O (>4.6 wt%), Ba, Sr and REE contents, and low MgO and Cr (<280 ppm) abundances and are hence more differentiated than the gabbronorites. In the case of the Aguablanca mafic stock, gabbronorite and norite rocks have more primitive compositions than diorites located in the southern part of the stock. In terms of major elements, there is no difference between mineralized and barren gabbronorites. Both gabbronorites contain relatively low SiO_2 contents (48-57 wt%), K_2O (<0.7 wt%), Ba, Rb, Th and Nb, and high MgO (12.2-16.2 wt%) and Cr (738-1200 ppm) abundances.

Trace element whole rock geochemistry of the sulfide-bearing gabbronorite and sulfide-free rocks (including gabbronorite and diorite) from the Aguablanca stock is quite similar (Figure 7-10). In all rock-types, primitive mantle-normalized trace incompatible element patterns are characterized by enrichment in large-ion lithophile elements (LILE), Rb, Ba, Th and U, relative to high field strength elements (HFSEs), Nb, Ta, Zr and Hf, strong negative Nb and Ta and positive Eu and Sr anomalies (probably, reflecting plagioclase accumulation). Rocks are enriched in LREE relative to HREE with $(La/Lu)_N$ ranging from 3.8 to 5.4 in gabbronorite. The abundance of trace incompatible elements is typically higher in rocks from the Santa Olalla intrusion than in those from Aguablanca stock (Figure 7-10).

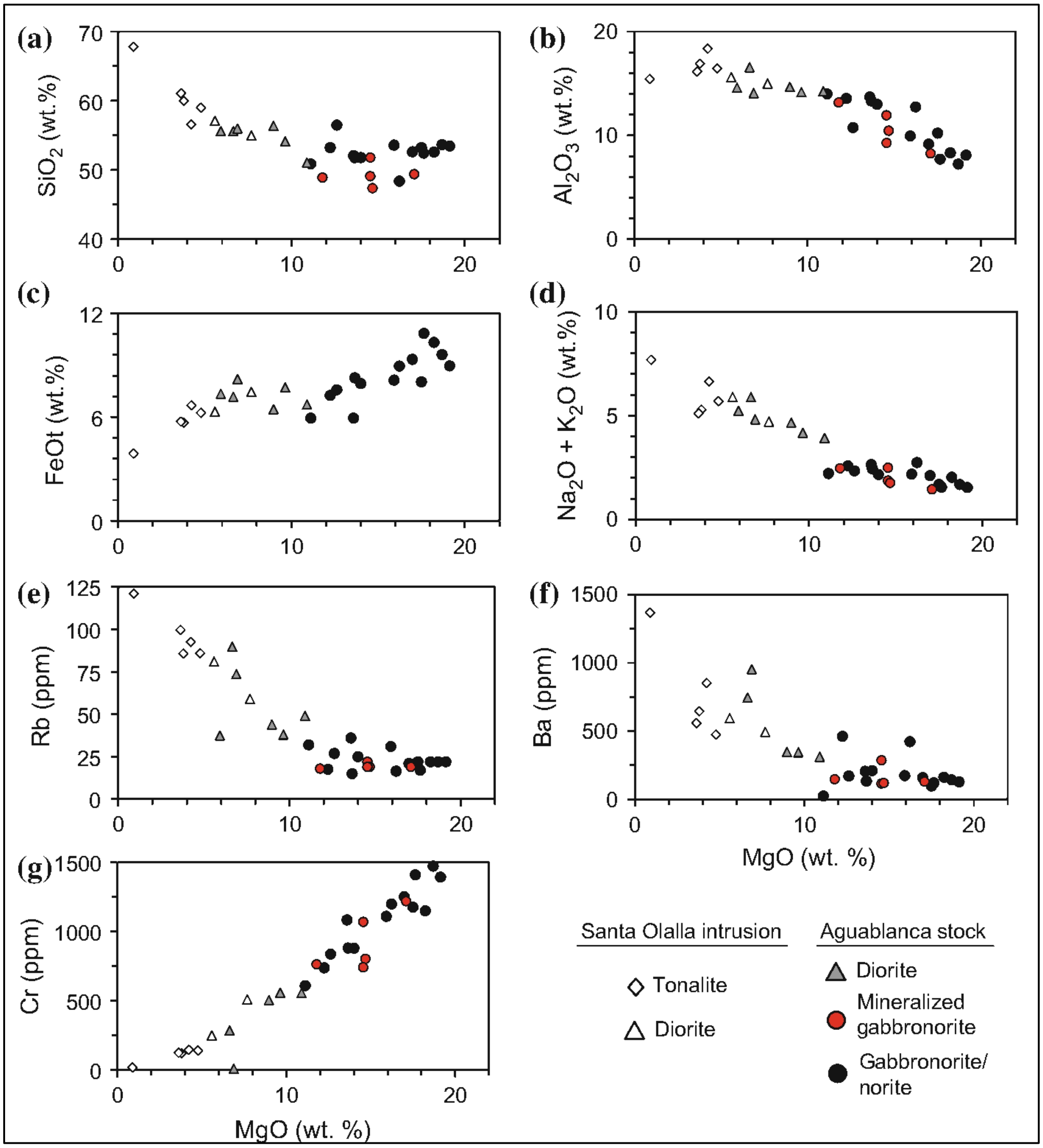

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Figure 7-9:Binary variation diagrams of major oxides (SiO2, AI2O3, FeOt, and Na2O + K2O, (a-d) and minor elements (Rb, Ba and Cr, e-g) versus MgO for rocks from the Santa Olalla and Aguablanca intrusions

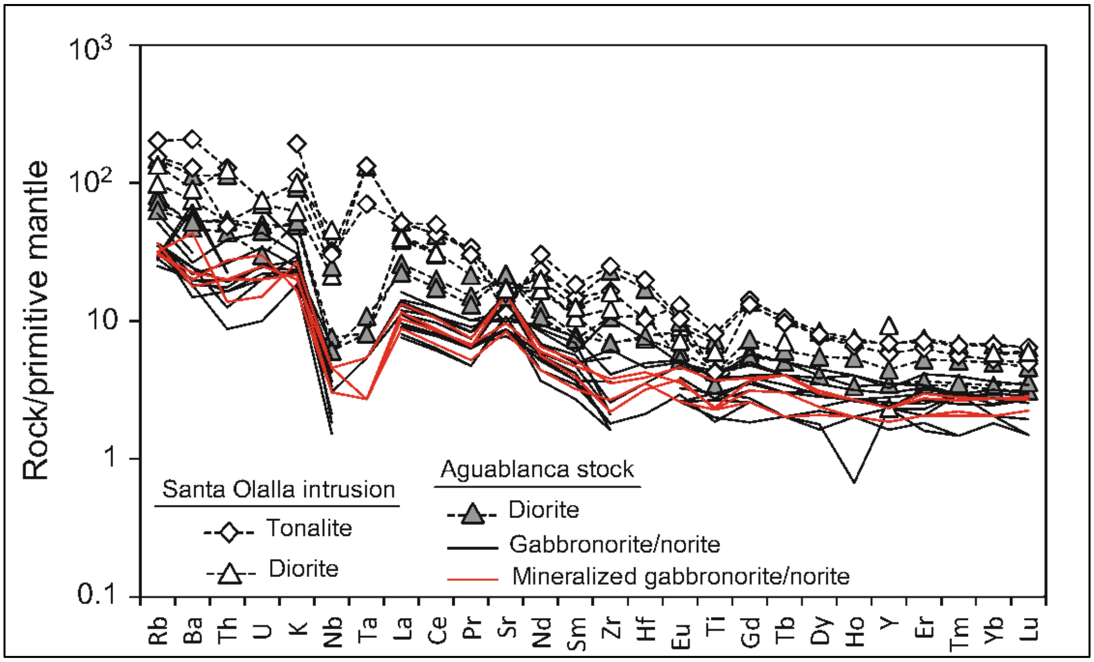

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Figure 7-10: Whole-rock primitive mantle-normalized trace element patterns of rocks from the Santa Olalla and Aguablanca intrusions

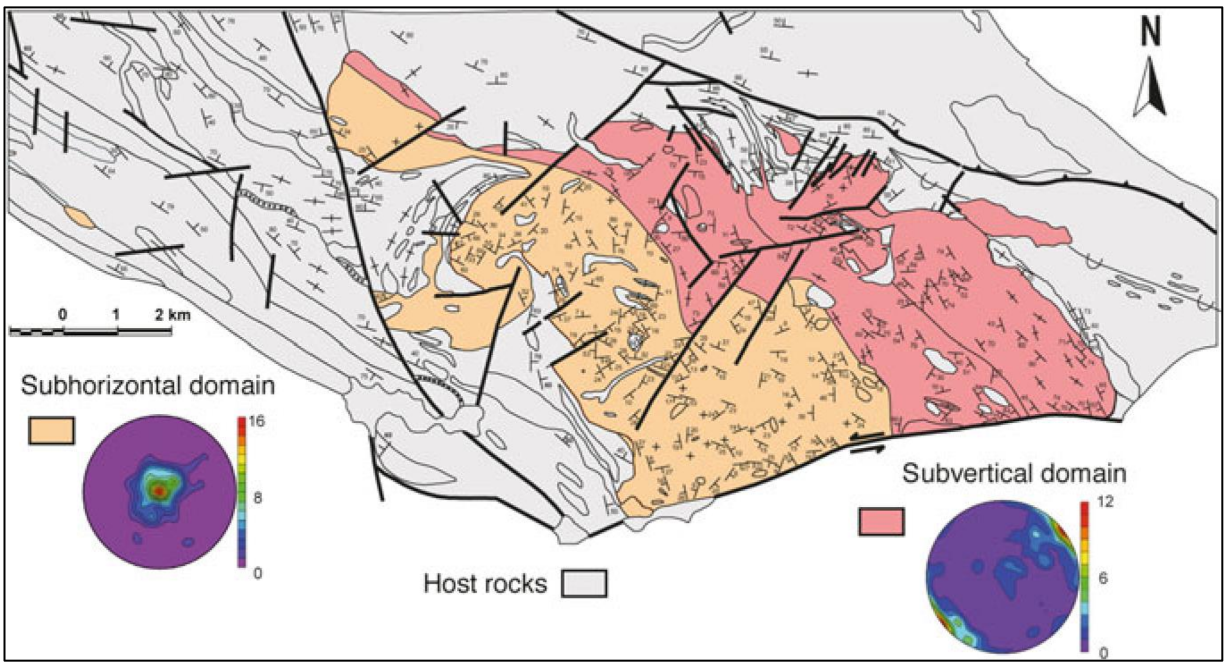
7.2.3 TIMING AND GEOCHRONOLOGY

Based on geochemical and petrological similarities with other nearby Variscan igneous complexes located in the Olivenza-Monesterio antiform such as Burguillos del Cerro, Brovales and Valencia del Ventoso, the SOIC is interpreted to be a product of the Variscan calc-alkaline magmatism developed in the Ossa-Morena Zone. The timing and genetic relationships between the Aguablanca stock and the Santa Olalla intrusion have been the subject of a number of studies. One study considered that Aguablanca gabbro was crosscut by the Santa Olalla tonalite, suggesting that Aguablanca predated the Santa Olalla intrusion. By contrast, another study considered both intrusions as different igneous facies of a same post-tectonic intrusive body emplaced during Early Carboniferous. However, these works proposed different processes to explain the compositional variability. One proposal is that the SOIC is an I-type igneous body whose reverse compositional zoning records a typical calc-alkaline evolution by fractional crystallization. By contrast, another study suggested that the different igneous facies are not the result of single fractional crystallization and alternatively proposed that the tonalite facies are the result of mixing between S-type cordierite-rich magmas with basaltic magmas that previously fractionated giving rise to the Aguablanca mafic rocks. Several later studies based on the isotope Sr-Nd compositions and whole rock geochemistry of the SOIC proposed an evolution of the parental magmas by assimilation-fractional crystallization (AFC). Accordingly, the initial fractionation of pyroxene and olivine from parental basaltic magmas and the early assimilation of S-bearing metasediments resulted in the formation of early cumulate rocks and sulfide melts. Subsequently, the fractionated magma enriched in crustal components evolved by fractional crystallization of plagioclase and amphibole giving rise to the different igneous facies of Santa Olalla intrusion (i.e., diorite, tonalite and monzogranite).

From the discovery of the Ni-Cu sulfide mineralization, the age determination of the Aguablanca stock became a priority objective due to its important implications on exploration. Some have considered the Aguablanca intrusion as mafic facies of the Santa Olalla intrusion and obtained by Rb-Sr whole rock geochronology an errorchrone of 359 ± 18 Ma for the SOIC. One explorationist obtained by the method Pb-Pb Kober an age of 332±3 Ma for the Santa Olalla tonalite. Almost simultaneously, others carried out isotopic dating of the Aguablanca stock by two different geochronological methods. One study obtained an age of 338 ± 3 Ma by 40Ar-39Ar on primary phlogopite in gabbronorites. Meanwhile, another study dated a gabbronorite mingled with a felsic hybrid rock in a zone exposed in the bed of the Ribera de Cala River (Figure 7-2e), near the contact with the Santa Olalla intrusion, yielding an age of 341 ± 1.5 Ma (U-Pb ID-TIMS on zircons). Field, petrological and textural evidence suggest that the age of this gabbronorite represents the age of the Aguablanca stock, so the age of this gabbronorite was interpreted as the age of the Aguablanca stock. All samples yielded ages clustering around 340 ± 3 Ma (U-Pb ID-TIMS on zircons: Santa Olalla tonalite, 341.5 ± 3 Ma; Garrote granite, 339±3 Ma; Teuler granite, 338 ± 2 Ma; and Sultana tonalite, 341 ± 3 Ma) indicating that all these intrusions formed part of the same magmatic event. Later was obtained a slightly older age for the Aguablanca gabbro of344± 2.1 Ma (SHRIMP U-Pb on zircons), and others obtained a Re-Os isochron with an age of 383 ± 59 Ma for disseminated sulfides. In summary, field, geochrono-logical, structural, geochemical and isotope evidence indicate that Aguablanca and Santa Olalla intrusions formed part of a same magmatic event developed in Early Carboniferous time and that the Aguablanca mineralization took place during the Variscan orogeny.

7.2.4 STRUCTURE AND EMPLACEMENT OF THE SOIC

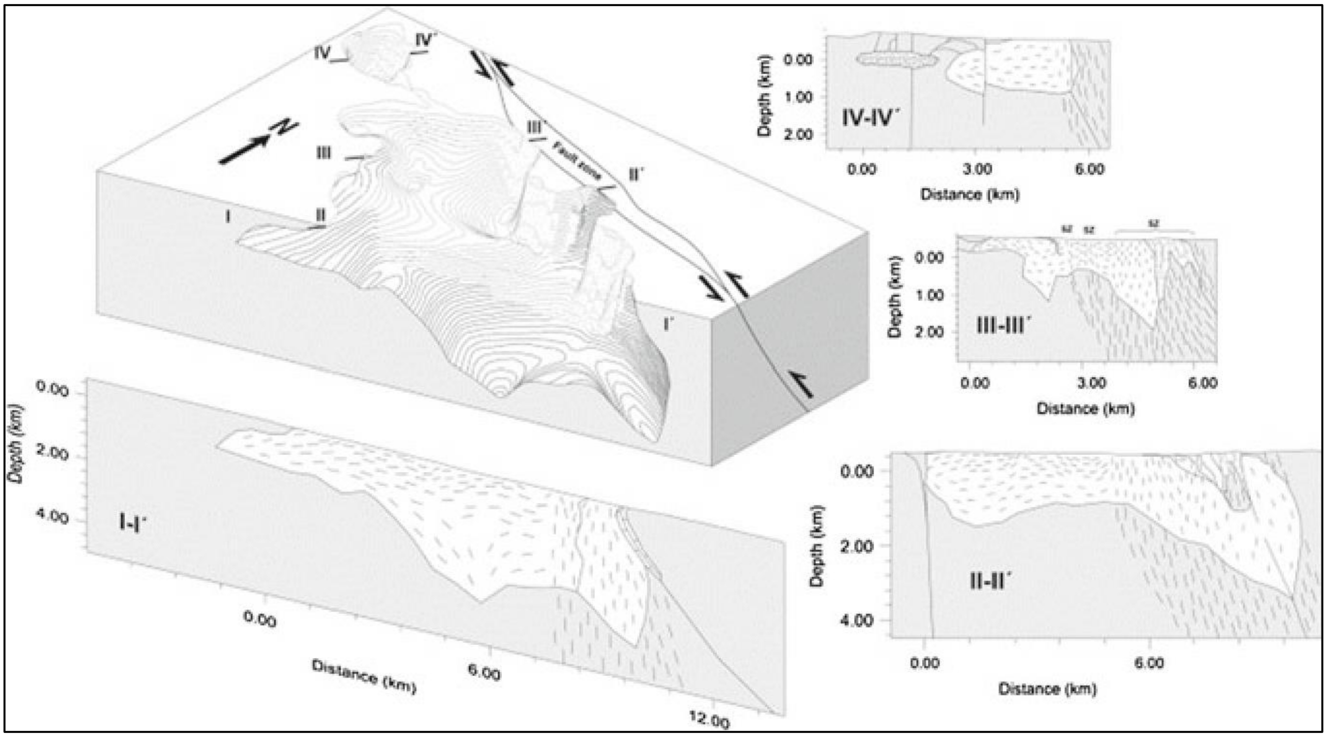
The SOIC is structurally located between the Cherneca Fault to the north, and the Zufre Fault to the south (Figure 7-6). The Cherneca Fault is a SW-verging structure trending parallel to the general Variscan direction in this zone (N120°) with a reverse and sinistral kinematics, whereas the Zufre Fault is a late N80° sinistral strike-slip fault that cuts off the igneous complex to the south. The SOIC was intensely deformed during its magmatic emplacement and crystallization but does not show any evidence of significant subsolid deformation. With the aim of knowing in detail the structure and geometry in depth of the SOIC, a study completed a magmatic foliation map along the entire igneous complex (Figure 7-11) and an exhaustive collection of gravity data that led to a 3D reconstruction of the complex (Figure 7-12). Two different structural domains were defined: (a) NE domain with foliations showing dominantly NW-SE strikes and vertical or high angle dips that is coincident with a thickening of the complex adjacent to the Cherneca Fault (hereafter, subvertical domain), and (b) a thinner SW domain with subhorizontal sheet geometry and predominantly subhorizontal foliations parallel to the upper and lower intrusive contacts (hereafter, subhorizontal domain). Between both domains, a transition zone was also observed, characterized by subvertical foliations superimposed on the subhorizontal foliations. The Zufre fault, located at the south of the SOIC, cuts the magmatic foliations of tonalite providing its post-intrusive age.



(Pińa 2019)

Figure 7-11: Magmatic foliation map of the Santa Olalla Igneous Complex showing two well-defined different structural domains: NE domain with NW-SE vertical foliations, and SW domain with subhorizontal foliations. Lower-hemisphere equal-area projections of the magmatic foliations from each structural domain are shown

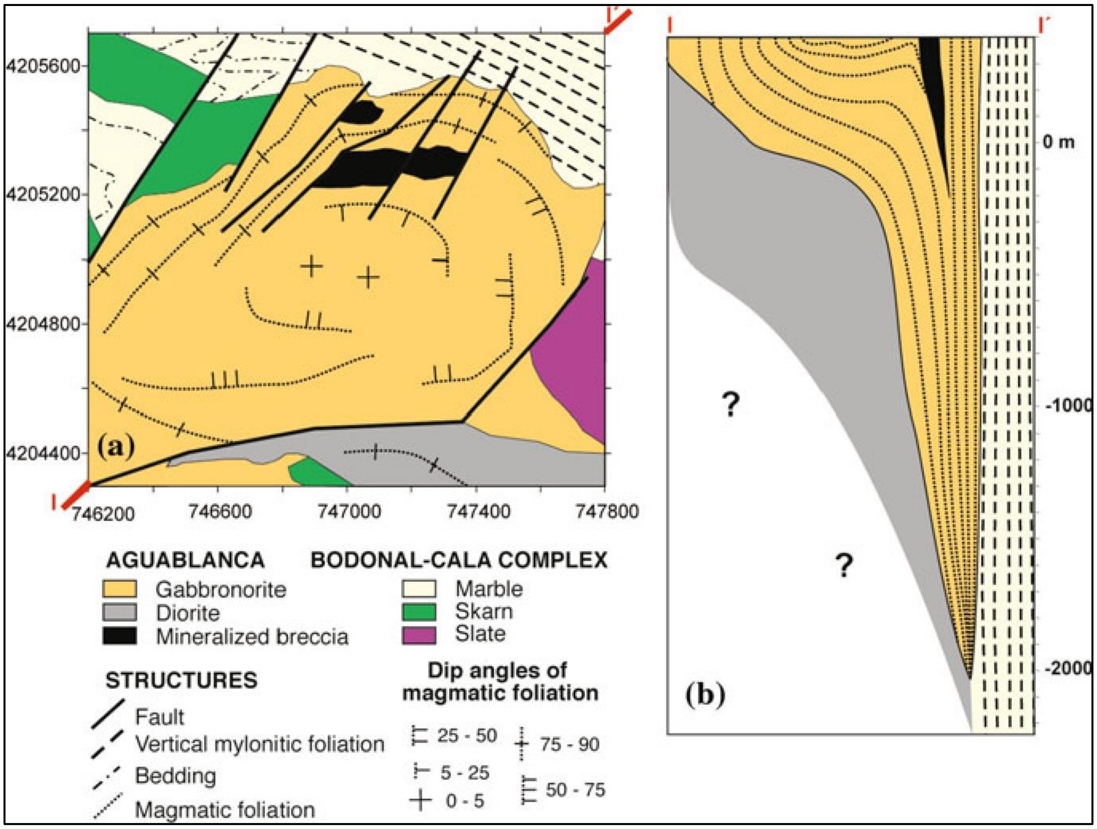
The NE subvertical domain is concordant with the trajectories of the host Bodonal-Cala volcano-sedimentary complex and shows a clear parallelism with the adjacent Cherneca fault. The foliation pattern in the NE domain defines rhomboidal geometries with two main trajectories similar to S-C microstructures formed under non-coaxial shear: one with a N130° strike and 70-90° south dip, and another with a N155° strike and vertical dips. It is suggested that the magmatic foliations of the subvertical domain were caused by the shear associated to the Cherneca fault strain field, so that the sinistral strike-slip movement of the fault would have been the responsible of the rhomboidal foliation patterns. Assuming that the deepest parts of intrusions are most likely to be the entrance ways for magmas, the NE contact of the SOIC was interpreted to be a feeder zone, and the Cherneca fault probably the conduit used for magma ascent. By the way, in the SW subhorizontal domain, the tonalite is thinner and exhibits a subhorizontal sheet geometry that is not parallel to that of the host rocks exhibiting a subvertical structure. These subhorizontal foliations and geometries may be the result of the stress tensor of the Variscan collision being accommodated by depressing the floor and lifting the roof, giving rise to lopolithic geometry at the SW.



(Pińa 2019)

Figure 7-12: 3D structure of the Santa Olalla Igneous Complex obtained from the gravity modelling carried by Romeo et al. (2006a). The 3D image has been reconstructed by the extrapolation of the gravity profiles I-I', II-II', 111-111', and IV-IV'

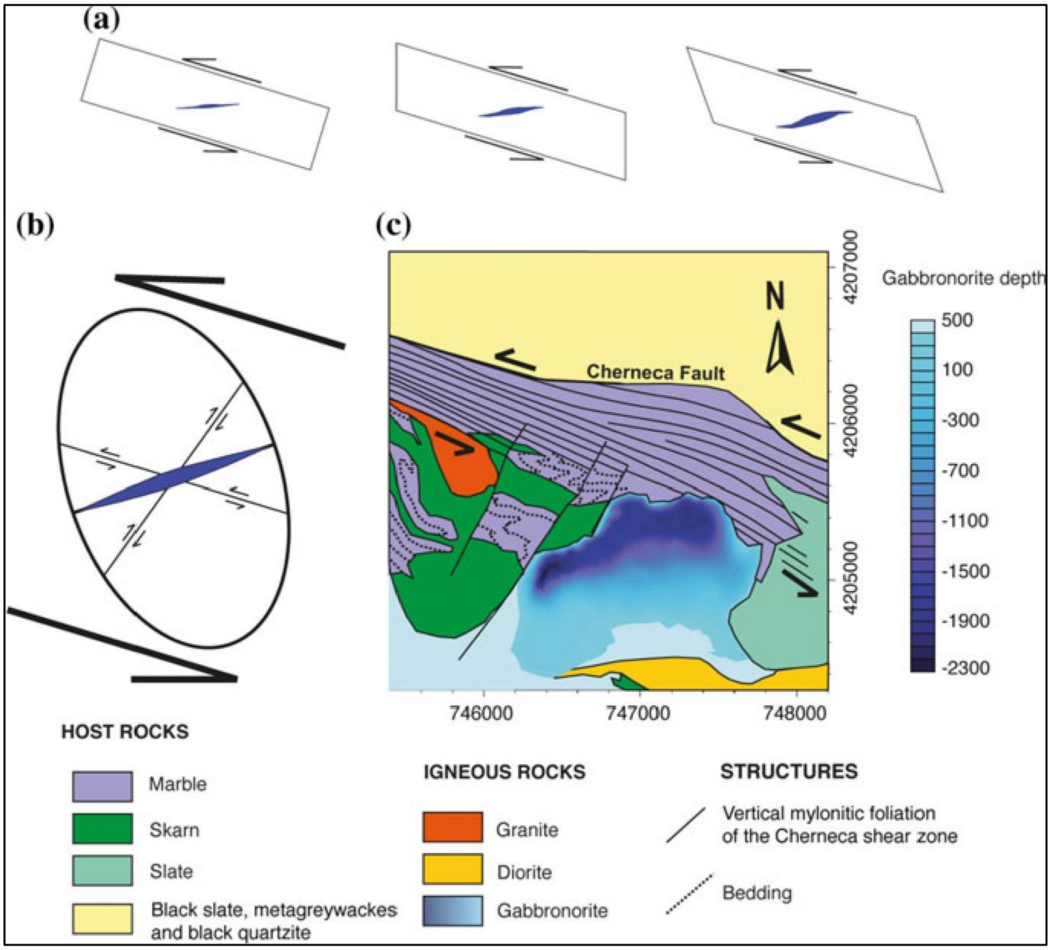
The Aguablanca stock belongs to the subvertical domain of the SOIC. The magmatic foliations defined by the preferred orientation of planar crystals of plagioclase (1-4 mm long) are mostly vertical in parallel to the pluton boundaries near to the N, NE and NW contact with the Bodonal-Cala rocks and predominantly subhorizontal in the central part (Figure 7-13a). In the south, magmatic foliations are linked to the magmatic structure of the surrounding Santa Olalla intrusion with N150° strike and high dip angle towards the NE. The intensity of magmatic foliations is higher toward the NE contact of the Aguablanca intrusion with the Cherneca shear zone. 3D gravity modelling has revealed that the Aguablanca stock has inverted drop geometry and that the root for the intrusion, located in the northern margin of the stock adjacent to the Cherneca Fault, has a vertical wedge shape (Figure 7-13b). This led geologists to conclude that the Cherneca Fault was probably the magma feeder structure for the Aguablanca intrusion. Crosscutting relationships between Aguablanca stock and the Cherneca Fault (e.g., intrusive rocks cutting the mylonitic foliation in the NE contact; exoskarn produced by Aguablanca deformed by the Cherneca deformation) suggest that the emplacement of Aguablanca took place after the beginning of the deformation associated with the Cherneca Fault. The long axis of the root for Aguablanca intrusion (N65°E) is not parallel to the Cherneca Fault (N115°E) but also seems to be coincident with the orientations expected for tension cracks developed in a sinistral ductile strike-slip shear zone with the strike corresponding to the Cherneca Fault (Figure 7-14). This led geologists to propose that the Aguablanca stock, and the Ni-Cu mineralized breccia pipes, may be emplaced along successive opening hundred-metre-scale tensional cracks formed within the strain field of the Cherneca Fault. Once the magma reached its current erosion level ascending along tensional cracks, it expanded towards SW adopting subhorizontal foliations and its inverted drop geometry (Figure 7-13b).



(Pińa 2019)

Figure 7-13: a Schematic map of the magmatic foliations of the Aguablanca stock. b Cross-section I-I' based on the 3D gravity modelling of Romeo et al. (2008) showing the inverted drop geometry of the Aguablanca stock and the vertical wedge shape of its root adjacent to the Cherneca Fault

In summary, the emplacement and structural evolution for the SOIC can be seen in three stages: (1) Magma ascent by the trace of the Cherneca fault during its syntectonic sinistral movement; (2) Once magma reached its present level of emplacement, it propagated toward SW as a horizontal sheet-like intrusion; and (3) After emplacement, the sinistral motion of the Cherneca fault provoked a tectonic stress field in the NE half of the complex giving rise to the subvertical foliation domain.



(Pińa 2019)

Figure 7-14: Emplacement model of the Aguablanca stock along open tensional cracks developed during the sinistral displacement of the Cherneca ductile shear zone. a Formation of tensional cracks in a sinistral shear zone. b Strain ellipse deduced for the Cherneca shear zone indicating the expected orientation of tensional cracks. c Geological map of the Aguablanca stock and the Cherneca shear zone indicating the orientation of the root of the Aguablanca stock (darker area) parallel to the expected orientation of tensional cracks in the Cherneca shear

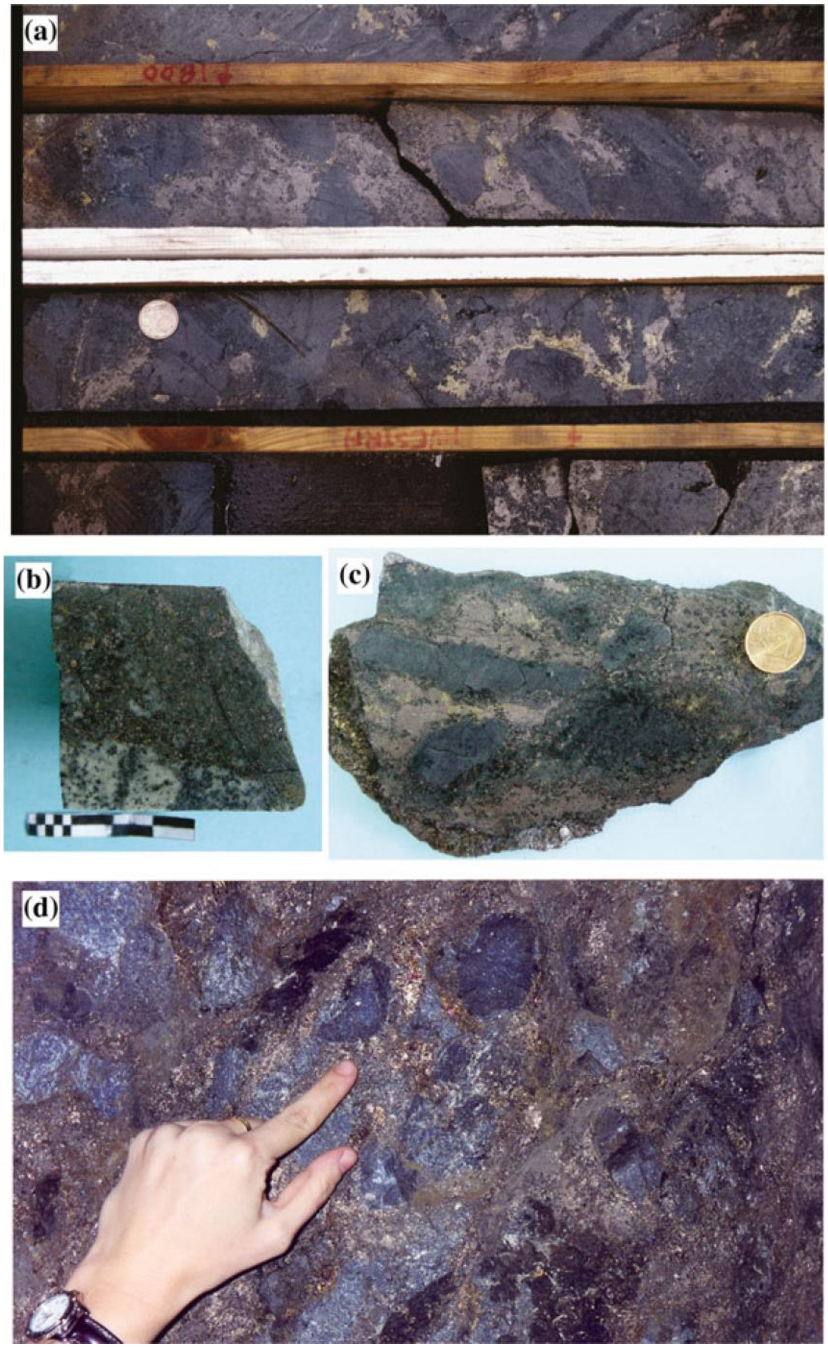
7.3 PROPERTY GEOLOGY

7.3.1 MINERALIZED BRECCIA

The Aguablanca Ni-Cu sulfide mineralization occurs in form of a poorly exposed subvertical mineralized breccia in the northern part of the Aguablanca stock, near to the sedimentary host rocks. The mineralized breccia consists of unmineralized (or very weakly mineralized) mafic-ultramafic fragments embedded in a variably mineralized matrix (Figure 7-15). The breccia forms a subvertical funnel about 250-300 m wide (N-S), about 600 m long (E-W) and a dip of 70°-80°N. From 600 m depth, the dip of the breccia shallows to 60°S (Figure 7-16). At this point, the mineralized breccia is apparently truncated by a subvertical ultramafic body. The downward continuity of the breccia is unknown because drilling and geophysical surveying has been indefinitely stopped on closure of the mine at the beginning of 2016.

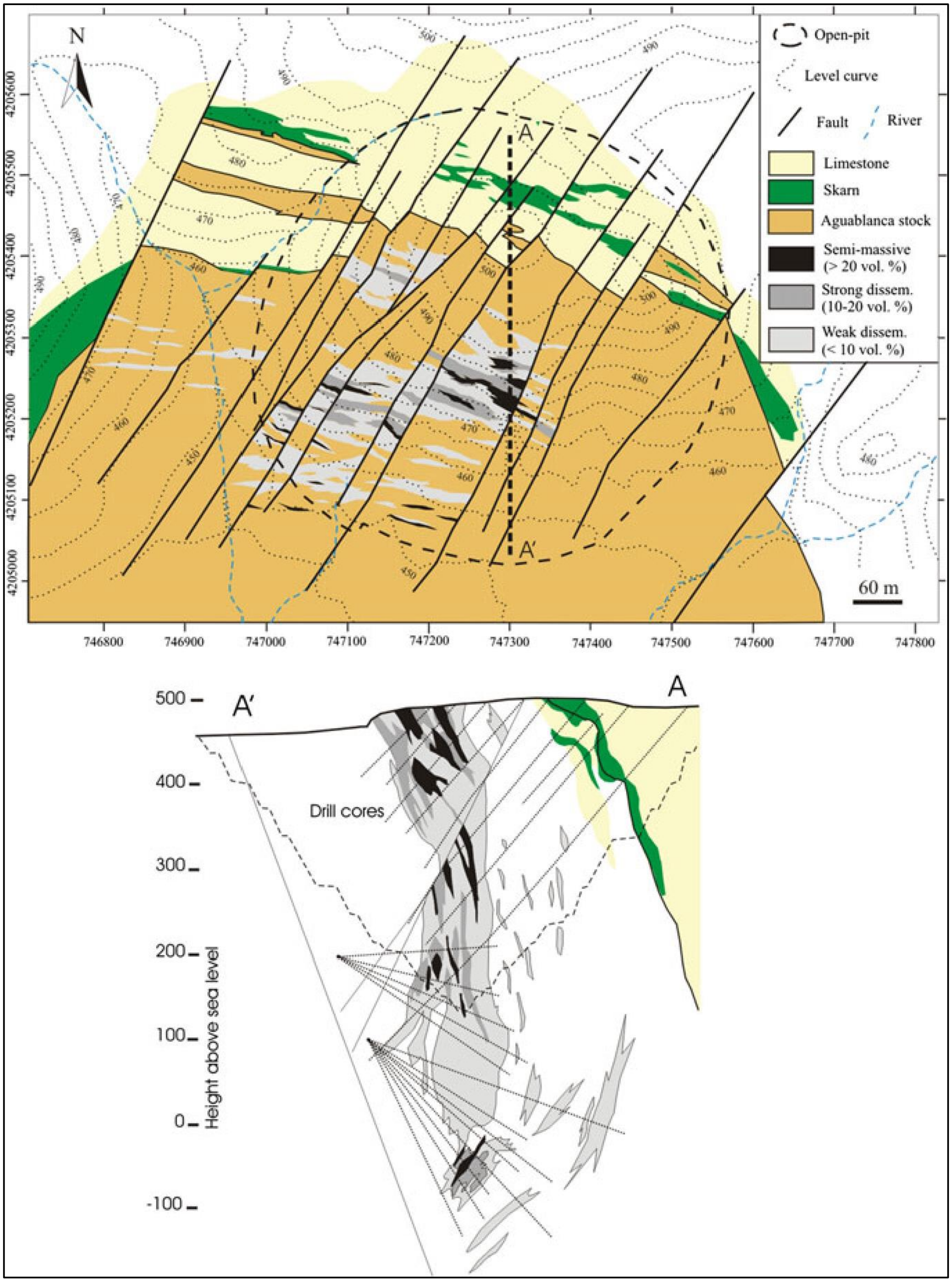
Within the mineralized breccia, the sulfide mineralization is concentrated in two northward dipping and E-W-trending subvertical bodies characterized by Ni grades higher than 0.08 wt%. Because these orebodies have been outlined from Ni grades, they do not really represent geological or mineralogical contacts, but also uniquely represent the strongly mineralized parts within breccia. The north orebody is the smallest one, occurs very close to the host rocks and reaches a depth of 160 m, whereas the south orebody extends to more than 650 m deep. Both orebodies are locally truncated by NE-oriented strike-slip faults. Out of these orebodies, the sulfide content is significantly lower but locally some zones contain more than 0.08 wt% Ni grades.

The breccia comprises a variably mineralized matrix hosting unmineralized mafic-ultramafic igneous fragments. Two main types of ore are distinguished based on sulfide abundance: semi-massive and disseminated ores. The semi-massive ore consists of up to 85 modal % sulfides, commonly between 20 and 70 modal %. These sulfides poikilitically enclose euhedral to subhedral crystals of olivine, pyroxene and/or plagioclase (Figure 7-17a, b). The disseminated ore is volumetrically more abundant than the semi-massive ore and consists of sulfides (<20 modal %) situated interstitially between silicate minerals in a gabbronorite rock (Figure 7-17c, d). The mineralized breccia is concentrically zoned; in general, the semi-massive ore occurs in the core of the breccia surrounded by the disseminated ore (Figure 7-16), which grades outwards laterally without systematic changes in the silicate mineralogy to sulfide-free gabbronorite. Minor chalcopyrite-rich veins occur crosscutting both semi-massive and disseminated ore throughout the deposit (Figure 7-17e, f).



(Pińa 2019)

Figure 7-15: Photographs of drill cores (a, b), hand sample (c), and field taken in the exploration gallery (d) illustrating the ore-bearing breccia consisting of unmineralized mafic-ultramafic igneous and metasedimentary fragments hosted by variably mineralized rocks



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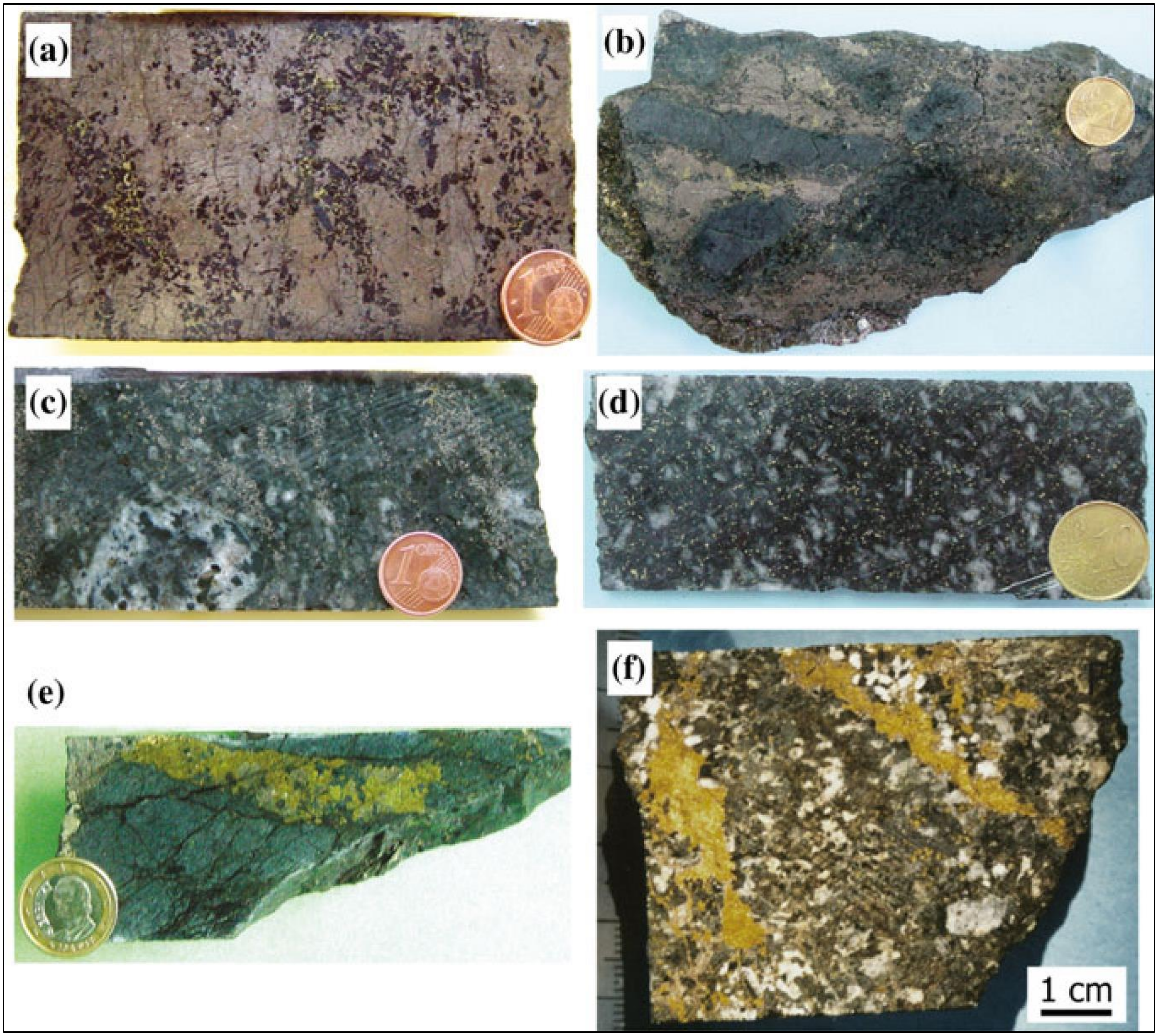
(Pińa 2019)

Figure 7-16:Simplified geological map (a) and cross-section (b) of the Aguablanca ore-bearing breccia at depth 450 as interpreted by drill cores showing the disposition of ore-types



(Pińa 2019)

Figure 7-17: Types of mineralization in the Aguablanca Ni-Cu deposit: semi-massive (a, b), disseminated (c, d) and chalcopyrite veined (e, f) mineralization

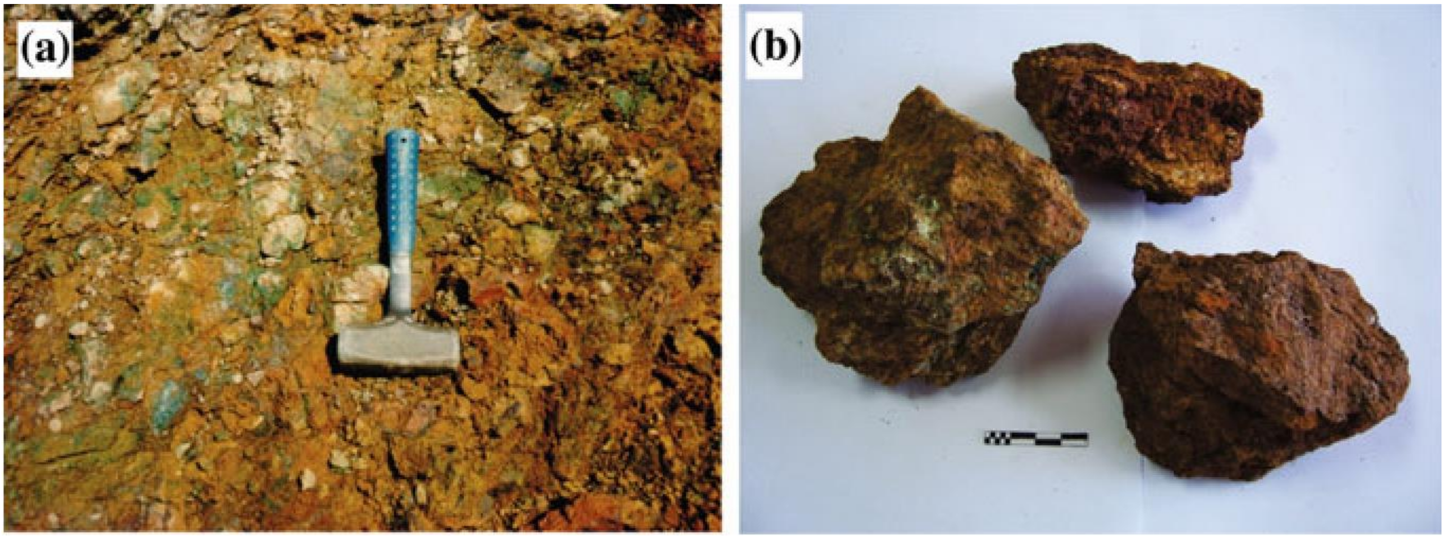
Unmineralized mafic-ultramafic and country-rock (mainly, calc-silicate hornfels) fragments are preferentially concentrated in the semi-massive ore, but they also occur in the disseminated ore and even in sulfide-free gabbronorite (Figure 7-18). Mafic-ultramafic igneous fragments comprise different cumulate-textured rock types: peridotite (dunite, harzburgite and werhlite), pyroxenite (orthopyroxenite and clinopyroxenite), gabbro (gabbro, hornblende gabbro and gabbronorite) and anorthosite. Gabbro s.s. (clinopyroxene + plagioclase) is by far the most abundant fragment type. In general, country-rock fragments have sharper contacts with the mineralized matrix than mafic-ultramafic fragments. Both fragment-types have commonly subangular to rounded shapes, and a size ranging from few centimetres to 10 cm. Synchrotron-based microbeam XRF mapping has revealed that mafic-ultramafic fragments surrounded by sulfides show some degree of disaggregation along the original grain boundaries, whereas the country-rock fragments show only minor reaction rims. Locally, pyroxene and plagioclase laths of the gabbronorite matrix appear to have nucleated on the fragments and tabular crystals of plagioclase show planar alignment parallel to the fragment boundaries suggesting some orientation during flow of the sulfide magma. Fragments are commonly unmineralized, although some host very minor disseminations interstitial to primary silicates or in association with secondary minerals such as epidote or actinolite. Minor chalcopyrite veinlets cut both fragments and host rocks. Based on the cumulate textures, the wide range of rock types from ultramafic to mafic, and a range of Mg-numbers in the primary ferromagnesian silicates, the mafic-ultramafic igneous fragments have been interpreted to be derived from previously crystallized rocks representing different stages of cumulate formation in an underlying differentiating magma chamber.

(Pińa 2019)

Figure 7-18: Unmineralized mafic fragment hosted by Aguablanca gabbronorite rock

7.3.2 GOSSAN

In the shallow parts of the Aguablanca mineralized breccia there is a supergene oxidation profile ~10 m thick (Figure 7-19). This gossan has been divided in an “upper unit” dominated by massive goethite, and a less oxidized “lower unit” where primary sulfides are relatively well preserved. The gossan mineralogy comprises goethite, hematite, limonite and minor garnierite, relicts of base metal sulfides, inherited silicates, and a complex assemblage of alteration minerals including actinolite, talc, chlorite and quartz. The study revealed that gossan has relatively high PGE contents, 1723 ppb total PGE in the upper gossan and 3418 ppb in the lower gossan, similar to the PGE abundances in the underlying unweathered sulfide mineralization. Geologists described a great variety of PGE-bearing minerals in the lower unit of gossan in close textural relationship with goethite and silicates: Pt- and Pd-bearing oxides, relicts of PGM documented in the unweathered sulfide mineralization, PGE-bearing Fe (± Ni-Cu)-oxides (up to 1.1 wt% Pt and 5.4 wt% Pd), PGE-hydroxides, and PGE-bearing goethite and hematite. According to these geologistss, these phases are the result of in situ alteration and dispersion of PGE during gossan formation.



(Pińa 2019)

Figure 7-19: Field (a) and hand sample (b) photographs of the Aguablanca gossan

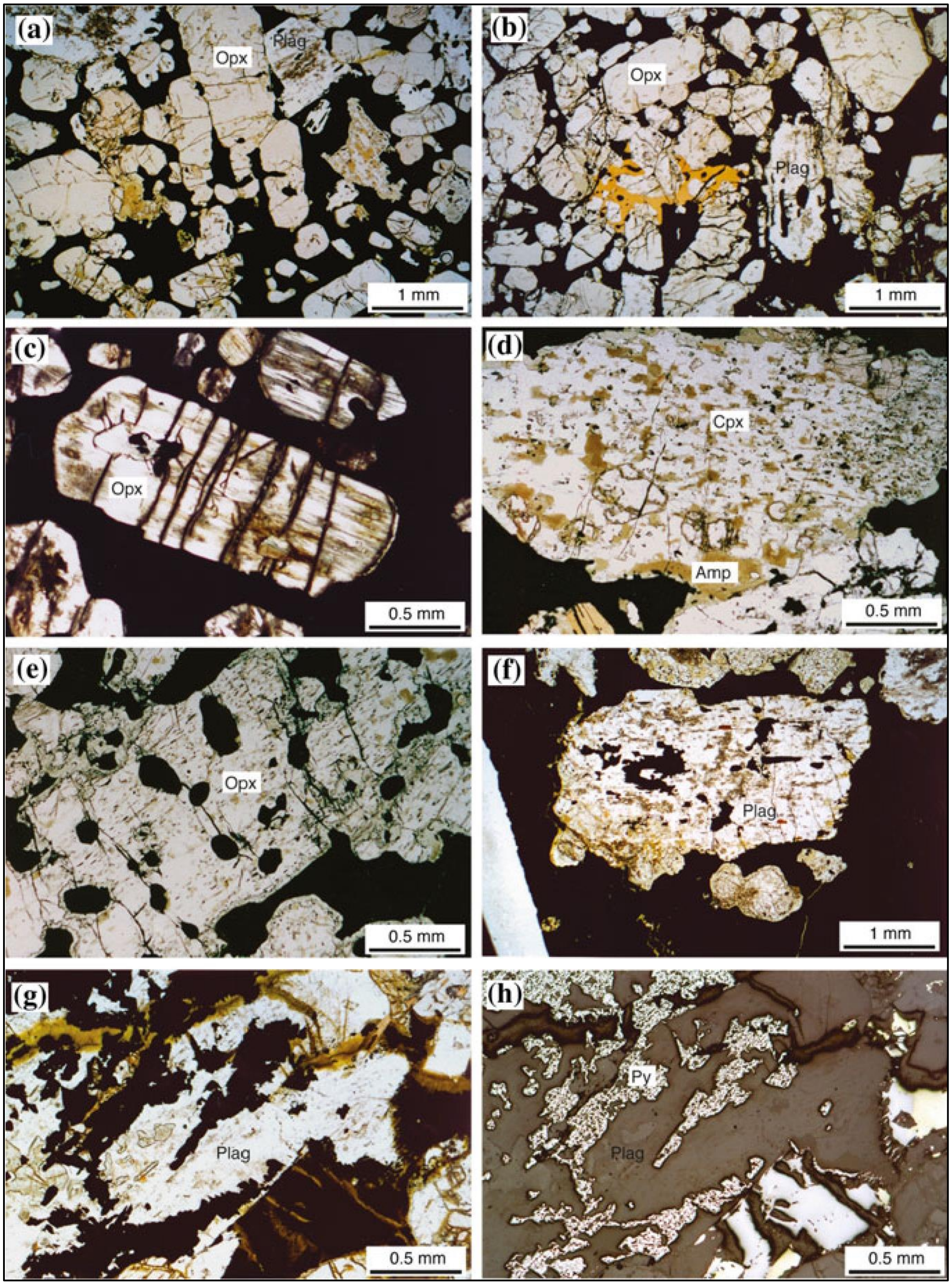
7.4 MINERALIZATION

The matrix to the breccias takes two main forms: (a) semi-massive mineralization, and (b) disseminated mineralization. In addition, chalcopyrite veinlets (chalcopyrite-veined ore) occur crosscutting both fragments and mineralized matrix.

7.4.1 SEMI-MASSIVE MINERALIZATION

Semi-massive mineralization preferentially occupies the inner zones of the breccia, which are surrounded by disseminated mineralization. Texturally, semi-massive mineralization is characterized by euhedral to subhedral (up to 0.5 cm across) silicates enclosed by an assemblage of sulfides composed of pyrrhotite and pentlandite with minor amounts of chalcopyrite and pyrite. The modal proportion of sulfides typically ranges from 45 to 70% but locally reaches modal proportions as high as 85%. Silicates comprise variable proportions of orthopyroxene (<48%), plagioclase (<30%), clinopyroxene (<27%) and minor amounts of hornblende (<9%), phlogopite (<3%) and olivine (<2%) (Figure 7-20a, b). This primary silicate assemblage is variably replaced to sericite, serpentine, bastite, talc, actinolite, chlorite, calcite and clay minerals.

Orthopyroxene occurs in form of euhedral crystals (0.3-4 mm, typically <3 mm) variably altered to bastite along fractures and cleavage planes (Figure 7-20c), talc, actinolite and calcite. In places, it shows thin clinopyroxene exsolution lamellae along (100) planes. Clinopyroxene forms subhedral grains (0.5-5 mm, typically <3.5 mm) commonly replaced by actinolite and phlogopite (Figure 7-20d). Some clinopyroxene grains show complex oscillatory zoning with respect to the Cr content, consisting of Cr-poor cores, concentric Cr-rich zones, and Cr-poor outermost rims. Some clinopyroxene crystals contain orthopyroxene inclusions, suggesting that orthopyroxene crystallized early, and small subrounded sulfide inclusions (Figure 7-20e) pointing the existence of an immiscible sulfide liquid before and/or during its crystallization. Tabular plagioclase crystals are irregularly altered to sericite and, at a lesser extent, chlorite. At the most strongly mineralized zones, plagioclase hosts inclusions of magmatic sulfides (mainly, pyrrhotite and chalcopyrite, Figure 7-20f) and is partially replaced by pyrite (Figure 7-20g, h). Olivine is a minor cumulus silicate, forming subrounded grains (<0.6 mm) totally replaced by serpentine. Phlogopite, hornblende and minor quartz comprise interstitial phases located between pyroxene, plagioclase and sulfides. Hornblende and phlogopite usually have sharp contacts with plagioclase but irregular with pyroxenes, suggesting formation by reaction between pyroxene and silicate liquid.

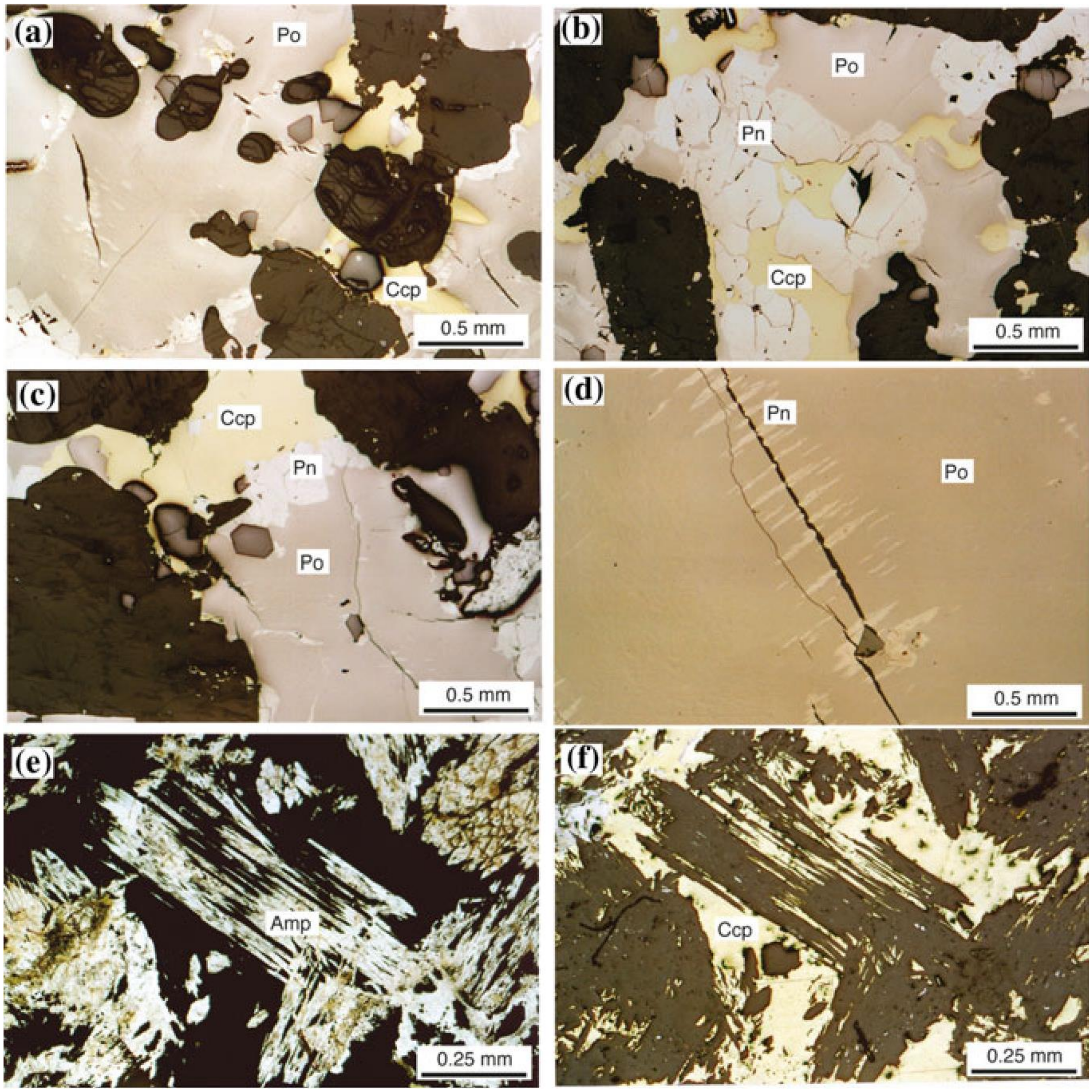


(Pińa 2019)

Figure 7-20: Representative transmitted-light photomicrographs of the semi-massive ore. a, b Sulfides enveloping early-formed pyroxene and plagioclase crystals. c Euhedral crystal of orthopyroxene partially altered to bastite within sulfides. d Clinopyroxene patchy replaced to actinolite. e Orthopyroxene hosting subrounded inclusions of magmatic sulfides. f Euhedral plagioclase enclosing sulfides. g, h Plagioclase partially replaced by secondary pyrite (g transmitted-light; h reflected-light). Abbreviations: Opx orthopyroxene, Plg plagioclase, Amp amphibole, Py pyrite

Economic Minerals

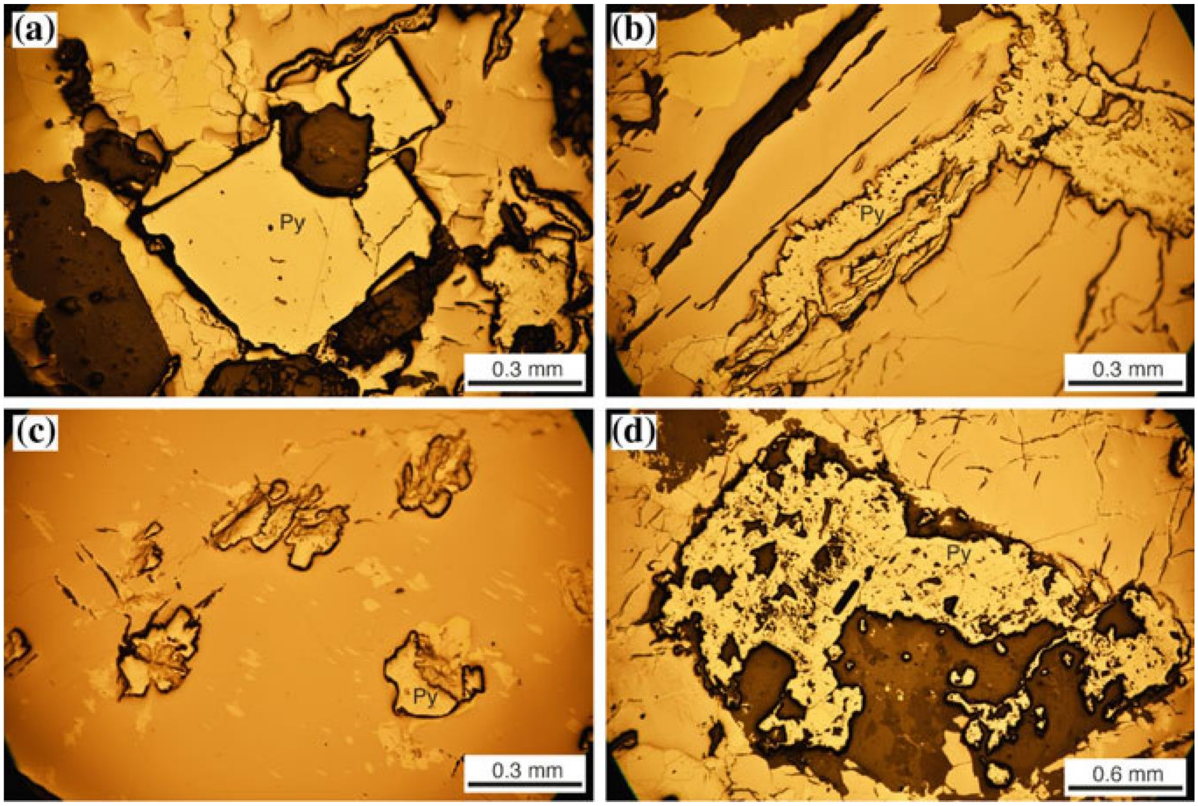
The mineral assemblage is typical of magmatic Ni-Cu-Fe sulfide mineralization, consisting of pyrrhotite ($Fe_{1-x}S$), pentlandite [$(Fe, Ni)_9S_8$], chalcopyrite ($FeCuS_2$) and minor pyrite (FeS_2). Magnetite (Fe_3O_4), minor amounts of violarite ($FeNi_2S_4$) and marcasite (FeS_2) replacing pentlandite and pyrrhotite, respectively, and a complex assemblage of platinum-group minerals (PGM), mostly Pd-Pt-Ni bismutho-tellurides, complete the assemblage. Pyrrhotite is, by far, the most abundant sulfide in the semi-massive ore, occurring as large (1-2 mm) anhedral twinned crystals in proportions higher than 55 modal % (Figure 7-21a-d). Pentlandite (10-35 modal %) occurs as polycrystalline, chain-like aggregates surrounding pyrrhotite (Figure 7-21b, c), formed by relative high-temperature grain boundary exsolution, and as small exsolution flames along grain boundaries and fractures within pyrrhotite (Figure 7-21d). Chalcopyrite (<10 modal %) forms anhedral grains of variable size randomly distributed between pyrrhotite and pentlandite. Magnetite (up to 2 modal %) occurs as individual euhedral crystals within sulfides (mostly, pyrrhotite) and in the contact between sulfides and silicates (Figure 7-21a-c). The euhedral morphology of magnetite suggests early crystallization. In the most altered samples, chalcopyrite and, at a lesser extent, pyrrhotite, are located along cleavage planes of actinolite and chlorite (Figure 7-21e, f), suggesting some sulfide remobilization during alteration.



(Pińa 2019)

Figure 7-21: Reflected-light photomicrographs of sulfide textures in the semi-massive ore of the Aguablanca Ni-Cu ore deposit. a-c Typical sulfide assemblage formed by pyrrhotite, pentlandite and chalcopyrite with minor amounts of magnetite. d Pentlandite flames within pyrrhotite. e, f Chalcopyrite occurring along cleavage planes of actinolite (e, transmitted light). Abbreviations: Po pyrrhotite, Pn pentlandite, Ccp chalcopyrite, Amp amphibole

Pyrite is a common sulfide phase in the Aguablanca semi-massive mineralization. Typically, its modal abundance oscillates between 5 and 15 modal %, and it can reach up to 20-25 modal % in areas with intense fracturing and alteration. Similarly, the deepest parts of the mineralized breccia seem to have even higher modal abundance of pyrite, with contents up to 40-50 modal %. In this deep part of deposit, pyrite occurs in form of large euhedral grains with octahedral morphologies. In general, four main textural types of pyrite are recognized in Aguablanca (Figure 7-22): (a) large idiomorphic-subidiomorphic crystals (1-5 mm in size) within pyrrhotite; (b) ribbon-like crystals (0.5-3 mm long) hosted by pyrrhotite; (c) small subidiomorphic single crystals or aggregates (up to 600 ^m) associated with pyrrhotite and chalcopyrite; and (d) irregular crystals partially or totally replacing plagioclase. Ortega et al. (2004) also identified pyrite filling late fractures that crosscut earlier aggregates of ribbon-like pyrite.

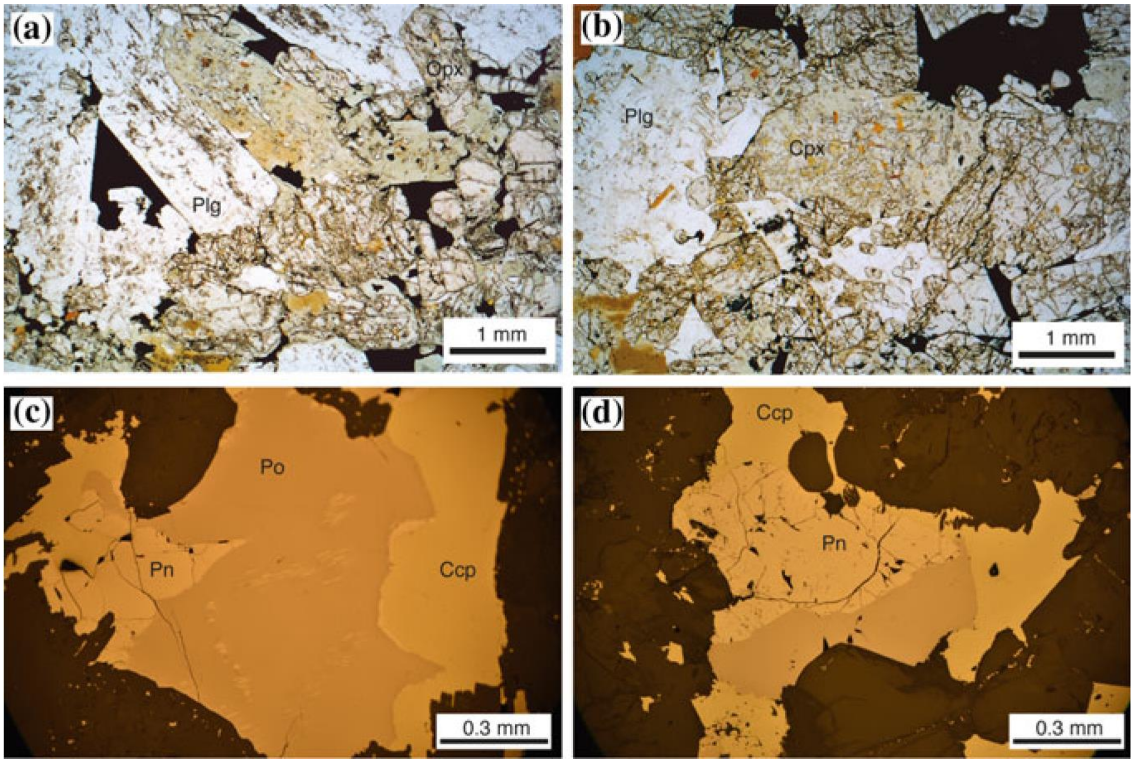


(Pińa 2019)

Figure 7-22: Reflected-light optical photomicrographs showing the different textural types of pyrite recognized in semi-massive ore from the Aguablanca Ni-Cu sulfide ore

7.4.2 DISSEMINATED MINERALIZATION

In the disseminated mineralization, sulfides comprising less than 20 vol.% of the rock occur interstitially between primary silicate minerals with sharp and well-defined contacts (Figure 7-23a, b). Gabbronorite is, by far, the most common lithology hosting disseminated ore, with norite, gabbro and amphibole-rich pyroxenite as other minor igneous rocks. These rocks show cumulate igneous textures formed by variable proportions of cumulus crystals of orthopyroxene (19-56 vol.%), clinopyroxene (<22 vol.%) and plagioclase (<52 vol.%), and intercumulus amphibole (<46 vol.%), phlogopite (<14 vol.%) and minor quartz. Actinolite, sericite, epidote, chlorite, carbonates, talc, serpentine and clay minerals are common secondary phases. Despite locally intense alteration, magmatic textures are usually well preserved and primary silicates are recognized from their morphologies. Chemically, pyroxene and plagioclase of mineralized gabbronorite are quite similar to these silicates from the unmineralized Aguablanca intrusion as is mainly indicated by the similarity in the Mg and An numbers, respectively. However, the compositions of pyroxene and plagioclase are slightly more primitive in semi-massive mineralization relative to disseminated mineral-bearing gabbronorite.



(Pińa 2019)

Figure 7-23: Representative transmitted-light (a, b) and reflective light (c, d) photomicrographs of disseminated sulfides. a, b Gabbronorite hosting disseminated sulfides with sharp and well-defined grain boundaries with sulfides. c, d Disseminated sulfides composed of variable amounts of pyrrhotite, pentlandite and chalcopyrite. Abbreviations: Opx orthopyroxene, Cpx clinopyroxene, Plg plagioclase, Po pyrrhotite, Pn pentlandite, Ccp chalcopyrite, Amp amphibole

Economic Minerals

Sulfides have typical interstitial disseminated textures in the form of variably sized (from few millimetres to 1-2 cm) polymetallic aggregates between silicates with well-defined boundaries (Figure 7-23c, d). The sulfide assemblage is quite similar to that of the semi-massive ore although there are differences in the modal proportions. The most important is that the disseminated ore contains greater proportions of chalcopyrite than the semi-massive ore. Pyrrhotite (22-68 vol.% total ore minerals) is equally the most abundant sulfide but in the disseminated ore chalcopyrite (12-58 vol.%) predominates over pentlandite (up to 20 vol.%) with chalcopyrite/pentlandite ratios ranging typically from 0.06 to 0.64. Another significant difference is that pyrite and magnetite are much less frequent in the disseminated mineralization. Pyrite is generally absent and, when present, occurs in contents < 5 vol.%.

7.4.3 CHALCOPYRITE VEINS

Chalcopyrite veins represent a very minor mineral-type in Aguablanca, comprising less than 5% of the mineralization present in the deposit. Chalcopyrite veins occur crosscutting both semi-massive and disseminated matrix and mafic-ultramafic fragments. Widths range from a few millimetres to about 2 cm and they extend up to several centimetres in length. They are made up of massive chalcopyrite, minor amounts of pyrrhotite and pentlandite, and traces of argentopentlandite located within chalcopyrite. Pentlandite has more Co (>2 wt%) and less Fe (<27.3 wt%) and pyrrhotite more Ni and Co (~1 and 0.1 wt%, respectively) than in semi-massive and disseminated ores.

8 DEPOSIT TYPES

The following summary section is excerpted from the excellent published work of Rubén Pińa, *The Ni-Cu-(PGE) Aguablanca Ore Deposit (SW Spain)*. The publication is very well referenced from hundreds of reports, studies and other documents. It is highly recommended to obtain this report for a thorough understanding and description of the geology of Aguablanca. The only changes to the text were to apply nomenclature in compliance with NI 43-101 and CIM Best Practices Guidelines.

The structural history of the Aguablanca stock and the cross-section geometry based on the 3D gravity modelling led geologists to conclude that the Aguablanca stock has an inverted drop geometry with the root of the intrusion located in its northern margin adjacent to the Cherneca ductile shear zone (Figure 7-13). The long axis of this root in the 3D gravity model is not parallel but is also oblique (~50°) with respect to the Cherneca shear zone (Figure 7-14). This oblique orientation fits well with that expected for tensional fractures developed within a transpressional sinistral strike-slip shear structure corresponding to the Cherneca shear zone (Figure 7-14). It is proposed that the Aguablanca stock was emplaced due to successive opening events of these tensional fractures related to the Cherneca shear zone.

During 2006 and 2010 geologists envisaged the emplacement of the Aguablanca mineral deposits through multiple injections of over pressured magma mainly controlled by the gradual opening of tensional fractures. According to this model, overpressure would have been the responsible of brecciation and transport upward of the dense sulfide- and fragment-charged silicate magmas from a deep level to a shallower site through low-density crustal rocks. The unmineralized igneous rocks of the Aguablanca stock (mostly, gabbrodiorites) would represent early injections of fractionated silicate melt from the uppermost parts of the deep magma chamber, whereas semi-massive mineralization would represent the latest injections containing a mixture of sulfide melt, partially consolidate mafic-ultramafic fragments and remaining silicate melt. The emplacement model envisaged is described as following. The first stage is the injection of sulfide-free differentiated silicate melts carrying minor mafic-ultramafic fragments. These melts would have flowed toward the south, taking on the inverted drop geometry of the Aguablanca stock and would have evolved by fractional crystallization giving rise to sulfide-free gabbronorite, norite and gabbrodiorite. Later, a new injection of silicate magma coming from the same magmatic source but containing droplets of unfractionated sulfide melt would have generated the disseminated mineral-bearing gabbronorites and norites, likely accreting inward from the sidewalls of the Aguablanca rocks. Geologists suggested that the barren igneous rocks and the disseminated mineral-bearing rocks may represent the product of turbulent and variable mixing between fragments and sulfide and silicate melts. Finally, the emplacement of the semi-massive mineral deposit would have taken place in the latest stages of emplacement of the intrusion. In this last injection, the sulfide-rich silicate melt charged with mafic-ultramafic fragments from deep, partially consolidated rocks, would have given rise to the semi-massive mineralization in the inner parts of the mineralized breccia.

An alternative model has been recently proposed in 2018 based on the textures observed in the Aguablanca breccias, where sulfide melt is seen to flood the pore space between melting and disaggregating silicate rock fragments by a process analogous to that responsible for similar sulfide matrix breccias at Voisey's Bay. In this model, the disseminated mineralization represents an early stage of emplacement developed as sulfide-bearing cumulates within a funnel shaped widening of the Aguablanca subvertical feeder. The semi-massive mineral-bearing breccia can have formed by a number of mechanisms. A possibility is that the breccia represents a downward-directed gravity flow of sulfide melt, silicate phenocrysts and igneous and country rock fragments developed during a late drain-back stage as the magma flux through the intrusion network waned. Alternatively, an upward-migrating "sludge" of silicate melt, crystals and rock fragments may also have collapsed due to a great increase of the transported load to the point where the suspension was no longer buoyant relative to the country rocks, forming a "log jam" that choked the flow into the magma chamber. The sulfide melt may have deposited initially in gravitationally unstable pools higher in intrusion network of which the Aguablanca stock is a part, and have then percolated downward through the previously emplaced, partially molten, silicate-matrix breccia. Superheat introduced into the breccia by sulfide liquid percolation led to melting and displacement of the original interstitial silicate melt, leaving behind refractory rock fragments, pyroxene crystals and an interstitial network of sulfide melt.

9 EXPLORATION

9.1 PRESUR/RTM (1983-2001)

Aguablanca mineralization was discovered in 1993 by Presur-Atlantic Copper S.A. (a joint venture between the Spanish State company Presur and Atlantic Copper S.A., formerly Rio Tinto Minera S.A.) during a regional exploration program that revealed Ni geochemical anomalies related to a gossan developed on altered gabbros (Lunar et al. 1997; Ortega et al. 2000). From this discovery, the company carried out detailed geological and geophysical work, including a program of short (less than 5 m) vertical percussion drillholes and long (up to 550 m) 45° dip diamond core holes, to trace the extension of the oxidized cap and to identify sulfide mineralization at depth. The shape and extension of orebodies were then delineated from a more exhaustive drilling program with more than 33,000 m of core.

Geochemistry

A soil sampling campaign of 26 lines in a north-south direction and samples every 40 m was completed with mesh closures. Analysis was for for Ni and Cu. An additional soil campaign of 30 radial lines with approximate separations of 50 meters and sample every 10 was completed with analysis for Ni, Cu, Zn, Au, Ag and Co.

Geophysics

A terrestrial magnetometry survey was conducted with a layout of 26 N-S lines, separated by 200 m and sampled every 5 m, collecting 2122 data points. An airborne magnetometry and radiometry campaign over the Spanish Pinto Belt which covered Aguablanca. Third, a time-domain electromagnetic geophysics (“TDEM”) campaign was completed.

9.2 RIO NARCEA RESOURCES / LUNDIN MINING (2001-2016)

Briones Gravimetric Study

In 2006 (Briones 2006) a gravimetric study was evaluated for the Aguablanca deposit. A Bouguer anomaly map was created and a 3-D model was interpreted which established the geometry of the pluton and of the mineralization. The results were relied upon to develop drill programs for the project. In the study area, the presence of a gravimetric anomaly with a maximum amplitude of 3.5 mGal was determined. The geometry of the maximum is easily correlated with the Aguablanca stock, whose basic composition gives it a density that contrasts substantially with the surrounding rocks.

There is the presence of Ni-Cu-(PGM) mineralized bodies, which are associated with secondary maxima surrounding the main pluton. The third lithology associated with the stock that contributes to the anomaly are the skarn bodies that appear in contact with marble units.

Subsequent to 3D modeling, the gabbronorites of Aguablanca were reconstructed. The model included the mineralized bodies with the geometries that had been recognized by boreholes, as well as the most important skarn body located to the NW of the gabbronorites. The model indicates that the gabbronorites take root in their northern part at 1850 m below msl, acquiring in depth a planar morphology that is arranged parallel to the northern edge of the stock. This seems to suggest that the rocks have ascended in favor of a band of ductile shears, oriented NW-SE, well developed in the marbles and which, in turn, parallels the brittle Cherneca fault.

A final recommendation for prospecting suggests it is logical to look for them along this shear band parallel to the Cherneca fault, since this seems to be the conduit through which the magmas ascend. There is a possibility to discover other similar mineralized bodies using the results gathered from the study.

9.3 AUTHOR COMMENTS

The area of interest for all the above-mentioned programs focused on the immediate area of the Aguablanca pluton which covers the area of the existing open pit. The programs identified an economic occurrence of Ni and Cu which eventually became the Aguablanca Mine which operated until 2015. The exploration programs were useful in identifying the mineral deposit and were useful in discovering the various smaller mineralization occurrences that surround the main mineral body.

10 DRILLING

Since 2001, RNR/Lundin has drilled more than 260 diamond drill holes for some 64,000 m of surface and underground drilling. The drilling program was completed in 2008. RNR also completed a 2,700 m underground decline in 2005.

The current drill hole database includes over 32,000 m of diamond drilling, carried out by RTM from 1994 and 64,000 m of surface and underground drilling completed since 2001 by RNR and Lundin as detailed in Section 6.2. The drill hole database supplied to RDA contains a total of 362 drill holes for a total of 96,515 m.

AGU prefixes denote drilling undertaken by Rio Narcea. Drilling numbers without alpha numeric prefixes identify legacy holes drilled by RTM. The legacy drilling used single shot down hole surveying methods, measuring only the dip of the holes and not the azimuth. The holes were re-logged and re-assayed by Lundin and plotted on sections. Twenty drill holes which were considered to have unacceptable down hole survey readings were removed from the drilling database. These holes are listed in Table 10-1.

Table 10-1 Holes removed from the database

6739	6758
6536	6758R
6735	AGU-46
6646	AGU-47R
6741	6715
6754	6630
Adit	6750
6740	6718
6763	6729
6739	AGU-46
6536	AGU-47R
6735	6715
6646	6630
6741	6750
6754	6718
Adit	6729
6740	6763

Resource definition and infill drilling at the Aguablanca Mine used two underground drill rigs and one surface drill rig around the periphery of the pit. The current Aguablanca drill hole database contains a total of 362 drill holes for a total of 96,515 m. Drilling completed is 64,000 m of surface and underground drilling since 2001. The drilling coverage for the Aguablanca Project used in the 2009 Resource Estimate is shown in Figure 11-1.

The resource definition drill fleet used a Sandvik DE 140 underground diamond drill rig, an Omran wireline underground diamond drill rig and an ACKER MP8 surface diamond drill rig. Both the underground rigs were capable of drilling at least 400 m deep HQ (77.8 mm) and NQ (60.3 mm) diameter core.

The core diameter depended on the planned length of the hole (i.e. smaller diameter for longer holes), the information being targeted (i.e. larger diameter for geotechnical holes), and the likely geological units to be encountered (i.e. larger diameter if the hole is likely to go through a major fault zone).

The author recommends the drilling method of wireline with diamond core triple-tube is appropriate for the deposit. The drill hole orientations were appropriate to intersect the mineralized zones of the deposit.

Surveying of some historical holes was carried out by single shot methods as detailed in Section 10.0 which is considered to be a poor quality down hole survey method as only the dip was measured.

Down hole survey was carried out at 3 m intervals using Maxibore surveying equipment. The tool collects the dip and azimuth of the hole and this data was input into the drilling database. Magnetic rocks will not affect the performance of this survey tool.

Validation of the Maxibore tool was carried out by site personnel by running the tool through a 40 m length poly-pipe (simulating a drill hole) located on surface which has been surveyed using a total station. The results of the Maxibore and the total station survey are compared. This test was carried out every few months. The results and the methods are considered to be appropriate for the style of deposit.

Additional drilling and sampling were carried out by open hole percussion rigs for the purpose of grade control drilling. These holes were drilled to the depth of one bench (8 m), plus 0.5 m of sub drill. The sample is collected by spearing through the various parts of the cuttings around the drill collar using a stainless-steel tube until a sample of approximately 10 to 15 kg in the mineral zone and 4 kg on the mineral/waste boundary is achieved. These samples were sent to the on-site laboratory for sample preparation and chemical analysis.

11 SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 CORE COLLECTION

Drill core was recovered in 3 m runs from the drill rig and placed in purpose built, wooden core trays. Wooden depth markers were placed in the boxes at the end of each 'run' or at regular depth intervals. The core boxes were appropriately numbered and labelled with indelible ink and incorporate the hole number, box number, metres and depth (Figure 11-1). Core was transported to the site core shed by utility vehicles. The labels were observed to be correct and appropriately reflective of core intervals. It was also observed that core blocks were present and correct in the holes which were inspected.

Core handling procedures and core box labelling are appropriate.

Figure 11-1: Labelling of core boxes for Aguablanca resource drilling

11.2 CORE STORAGE

The core is stored in a covered core shed located at the mine site (Figure 11-2). The core was systematically photographed using a rack and a standard labelling system. This is shown in Figure 11-3.

Figure 11-2: The Aguablanca Core Shed

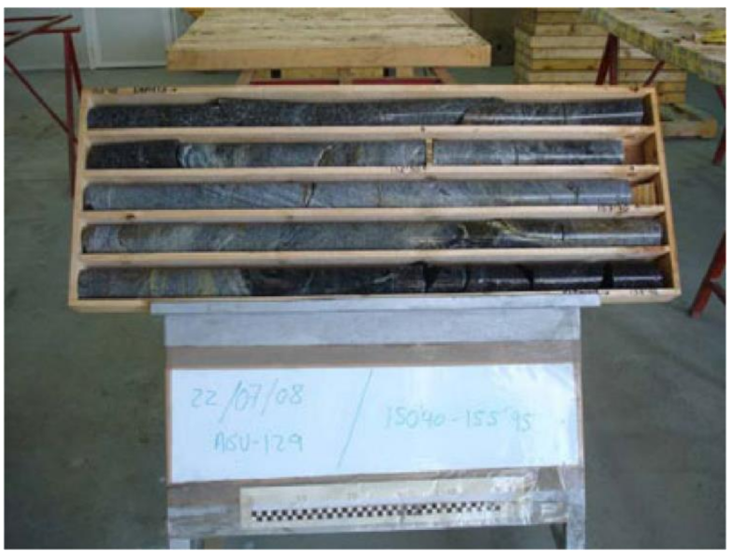


Figure 11-3: Example of core photography

11.3 SAMPLING

The drill core was cut in half with a core saw perpendicular to the predominant foliation of the rock (Figure 11-4) and sampled at approximately 2 m intervals, always sampling the same side of the core. Sampling intervals were broken by lithology. The maximum sampling interval is 2.2 m and the minimum sampling interval is 0.5 m.

The core samples were placed in a thick plastic bag with a sample identification tag placed inside the bag and sent to a site laboratory.

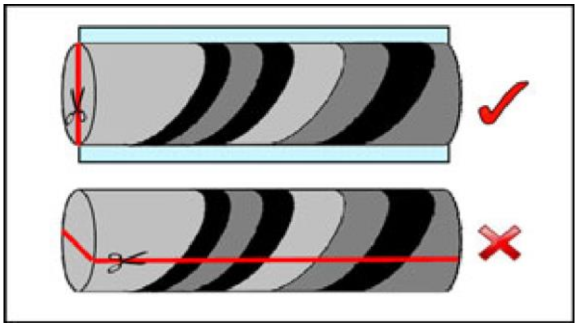


Figure 11-4: Correct core sawing orientation being used at Aguablanca

Potential loss of fines material from the core during sawing was minimal and did not impact on the sampling of mineralization due to the competency of the core. The sampling intervals and methodologies were suitable for this style of mineral deposit.

11.4 LOGGING

Drilled core was received at the Aguablanca core shed for logging and sampling. The core was simultaneously logged for geology and geotechnical parameters and marked out for sampling. Geologists marked the sampling contacts onto the core box with a suitable marker. Logging information was recorded on paper and later transposed to Microsoft Excel spreadsheets.

Core logging considered 23 codes for lithology and 4 codes for mineralogy. The geological unit, major faults, fragments types, alteration type, and estimated sulphide percentage were also logged. The list of codes for logging is shown in Table 11-1 and an example of a completed logging sheet is shown in Figure 11-5.

Table 11-1 Aguablanca Logging Codes

Rock Type	Code	Description
Igneous rocks	GM	Melanocratic gabbro: low plagioclase content
	GP	Porphyritic gabbro (gabbro-norite): porphyritic crystals of euhedral plagioclase in a
		pyroxenite matrix
	GPL	Plagioclase gabbro: abundant plagioclase.
	GD	Doleritic gabbro: abundant plagioclase connected with doleritic textures
	GI	Irregular gabbro: variable texture variable, zones of poor crystallization and/or contaminates
	GC	Contact gabbro: chilled texture chilled, fine grained, generally brecciated and/or banded
	Gsm	Gabbro with abundant alteration to serpentinite-magnetite-talc. Associated with skarn
	Gb	Magmatic Breccia with enclosed fragments. Gabbroic Matrix gabbroic with or without
Breccias		sulphides
	Gsb	Magmatic Breccia with enclosed fragments. Semi massive sulphide matrix
Others	PX	Pyroxenite
	DI	Dioritic rocks
	GR	Granite-Granodiorite
	M	Banded marble
	SK	Pyroxenite skarn
Metamorphics	ES	Endoskarn
	H	Hornfels
	VS	Volcano-sedimentary rocks
	PZ	Slates with volcanic intercalations
	OX	Gossan
Others	ZF	Fault zone. Very fractured gabbro
	REC	Fold

	NR	**Not recovered**		
Mineralization	Code	Description	% Sulfide	% approx. Ni
Sterile	NS	No sulphides	<1	<0.05
Disseminated	S	Disseminated intergranular, weak to moderate po- pn>=cpy>>py	5-10	<0.7
	SS	Disseminated intergranular, strong po-pn>=cpy>>py	10-20	>0.7
Breccia	SB	Fragments in a semi massive sulphide matrix (po-pn>>cpy)	20-60	>1.0
Patchy	SP	Regular patches (net-texture) or irregular (associated with a GI y Gb) po-pn-py-cpy	variable	variable
Core recovery is calculated by measuring actual core lengths and comparing it to the down hole depths listed as the start of each core run				

Core recovery for the current drilling was generally around 80 to 95% in mineralized and non-mineralized zones. Core logging procedures are appropriate for the geological setting and mineralization. Core recoveries above 85% are appropriate for Mineral Resource estimation.

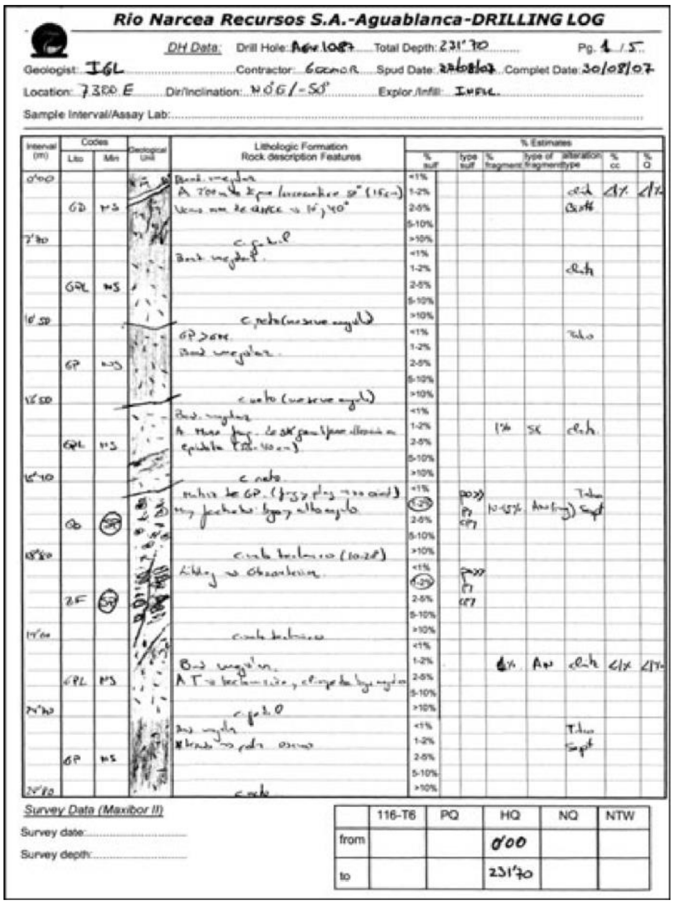


Figure 11-5: Typical Aguablanca drilling log

11.5 DENSITIES

The density values for the various rock types of the deposit were determined by the water displacement method.

A sample interval of approximately 20 cm was selected by the field geologist. The sample was removed from the core box prior to splitting. The samples are weighed in air before drying, in air after drying and finally in water. The dry density is then calculated using the formula:

Dry density = P2 / (P1 - P3)
where P1 = weight in air (wet)

P2 = weight in air (dry)
P3 = weight in water

The samples were taken from appropriate rock types, from a range of lithologies, and from a variety of drill holes. A density database called DensityStats_deAlan_fullset.xls contains nearly 7,000 density measurements.

The order in which measurements were taken to measure bulk density is unusual and could lead to a risk of bias. Usually P3 would be measured first, then P1, then P2. In such a sequence, the degree of water saturation inherent in the P3 and P1 measurements can reasonably be assumed to be the same and therefore the dry density measurement will be correct.

The problem with the Aguablanca sequence is that the sample may absorb more water when P3 is measured than was present when P1 was measured. Absorbed water will increase P3 and hence decrease (P1-P3) and thereby increase dry density. For highly absorbent rocks, there is a risk of overestimating the dry bulk density with the stated method. The author does not believe this has a material impact on the estimation of densities but should be investigated.

The data set was checked for outliers and the mean density value for each of the rock types and found the data to be appropriate and the mean values reasonable.

11.6 SAMPLE PREPARATION

Sample preparation was conducted at the site laboratory run by RNR staff which had been operating for eight years. The laboratory was audited on 23rd July 2008 for the sampling and preparation procedures used. The sample preparation flow sheet is shown in Figure 11-6.

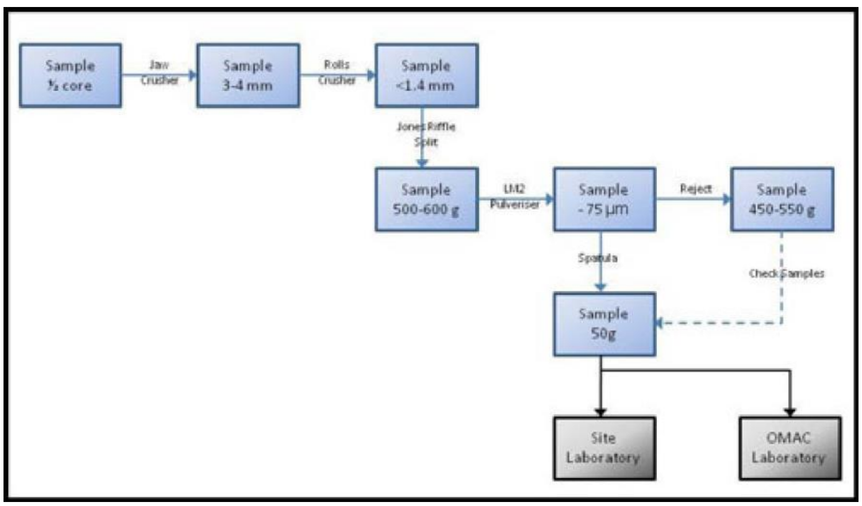


Figure 11-6: Sample preparation flow sheet as used for Aguablanca resource definition samples

11.6.1 SAMPLE RECEPTION

The samples arrived at the sampling shed with a sample submission sheets which contained a list of the samples sent by the geologists. Each sample had a tag with an individual barcode indicating the hole identification and interval (Figure 11-7).

The samples were dried in an oven for 12 hours at no greater than 105°C. The samples were weighed after drying on a Viper SW 35 scale which had a precision of 2 decimal places. The sample reception system used electronic barcodes to track the progress of the samples throughout the preparation flow sheet.

Figure 11-7: Bar code system used at Aguablanca

11.6.2 CRUSHING

The samples were crushed with a jaw crusher, reducing the particle size to approximately 3 to 4 mm. The entire sample was then passed through a rolls crusher to reduce the particle size to less than 1.4 mm. The crushers were cleaned by air and a quartz wash was passed through the crushers after every sample. Size fraction tests were carried out using a 1.4 mm sieve once per month.

The condition of the crushing equipment was considered adequate and appeared well maintained for the purpose.

11.6.3 FIRST STAGE SPLITTING

The entire crushed sample was split in a 50:50 Jones riffle splitter until a 500 to 600 g sample was achieved. The slots had even widths and the samples were poured over all the slots evenly. The splitter was then cleaned by compressed air. Loss of material due to dust appeared to be minimal.

The condition of the splitting equipment was deemed acceptable.

11.6.4 PULVERIZATION

The split samples were pulverized to -75 micron using an LM 2 pulverizer for approximately 2 minutes. The pulverizer appeared to be in good condition. The pulverizer was cleaned by compressed air and washed with a quartz rinse after every sample. Size fraction tests were conducted once a month.

11.6.5 SECOND STAGE SPLITTING

The pulverized samples were split by passing a thin spatula through the samples once they had been placed on a smooth work bench. The spatula was run though the full width of the samples several times until a 50 g sample was achieved as shown in Figure 11-8.

This splitting method is not ideal given the density of the mineralization particles as there is the chance that the sample will not be representative due to gravity separation. This can happen if the spatula does not pass through the entire width of the sample or if the sample spills over the top of the spatula during the scooping action. It is not considered likely that this has a material impact on the representativity of the assays but the practice could have been improved by employing a small laboratory scale Jones riffle splitter or a rotary sampling device.

Figure 11-8: Splitting of pulverized samples using a spatula

11.6.6 STORAGE

The sample rejects are stored on site in the core shed indefinitely inside heavy-duty plastic bags which have been placed inside labelled plastic bins. The bins are stored in an orderly and retrievable fashion. The general housekeeping at the site laboratory was well organized and the samples prepared in orderly fashion. Appropriate dust extraction was installed in the laboratory to prevent contamination.

11.7 QUALITY ASSURANCE AND QUALITY CONTROL PROCEDURES

Previous operators of the Aguablanca mine did not implement a rigorous quality assurance and quality control regime. Previous technical reports detailed some of the issues found with the database, particularly the poor reproducibility of the standards. Rio Narcea used internal standards which apparently were not sent for round-robin analysis nor were they certified. The 2009 technical report concluded that at least two of the four standards used showed inadequate results and should be replaced with certified reference standards.

11.7.1 STANDARDS

Four standards were prepared by Rio Narcea named A, B, C and D, manufactured on site by Rio Narcea personnel. The standards were manufactured by collecting 55 to 60 kg of material sourced from the 1.4 mm reject fraction of exploration drilling assays. The material was mixed for 15 to 20 minutes by shovel before the entire sample was pulverized to -80 mesh. The samples were then split, separating 4 kg from the main sample. This 4 kg sample was further split and 11 samples were sent to six external laboratories for analysis. An arithmetic mean and standard deviation of the results was calculated in Excel spreadsheets.

The list of the standards used with their expected grades and standard deviations is shown in Table 11-2.

Table 11-2: Aguablanca reference standards

Standard	Description	Expected grade (ppm)			Standard deviation (ppm)
		Ni	Cu	Ni	Cu
A	Waste	197	58	25	35
B	Low grade	3,492	3,924	104	173
C	Medium grade	6,711	6,711	242	207
D	High grade	13,768	8,414	667	361

11.7.2 LAB DUPLICATES

The main source of quality control data are lab duplicates from OMAC. There is no indication on whether these were coarse or fine duplicates. Certified results for 219 lab duplicates were found. Original and duplicate results for copper, nickel, cobalt, gold, platinum, and palladium are shown in Figure 11-9. There is good correlation for copper, nickel, cobalt, and palladium. Gold and platinum have poorer correlations. Duplicate results for gold show poor correlations for both higher and lower grade results, indicating that there is poor reproducibility for the assay method.

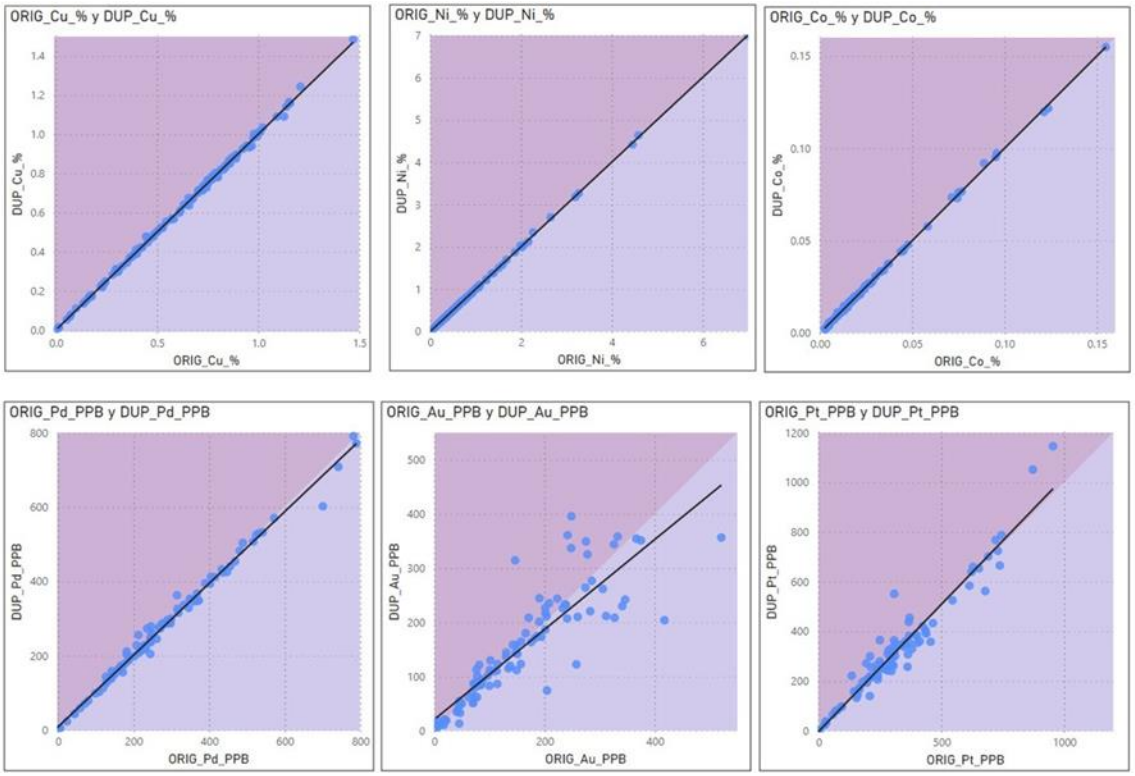


Figure 11-9 Lab duplicate results from OMAC

11.7.3 INTER-LABORATORY REPEATS

Inter-laboratory repeat assays of mineralized intervals from the drilling are selected by the site geologist and a 500 g split of material pulverized to -80 mesh is prepared by the on-site laboratory and sent to the OMAC Laboratory (now ALS Loughrea) in Ireland for assay by a four-acid digest with reading by ICP. OMAC Laboratories Limited was owned by the Alex Stewart Group and was accredited to ISO/IEC 17025.

Repeats were submitted at a rate of 1 in every 20 samples.

11.8 ASSAYING

Pulverized samples were sent to the internal assay laboratory located on site. A second split of pulverized material of mineralized intervals, was chosen by the logging geologist and sent to the OMAC Laboratory in Ireland for repeat analysis.

Assaying was carried out by AAS in the site laboratory with some checks carried out using an XRF machine. XRF was also used to assay arsenic and sulphur in exploration samples.

Calibration standards were used by the laboratory at a rate of one in every 10 assay results. The laboratory used seven standards for Ni, seven for Cu and four for Co. The results were recorded but there was no evidence that these were being charted or analyzed.

The assay results were entered into Excel worksheets manually by the laboratory staff and sent back to the resource geology department.

11.9 OPINION OF ADEQUACY

In the opinion of the QP, the methods employed for sampling preparation, security, analytical procedures, and QA/QC protocols were less than adequate by today's standards. However, through 2015 this was an operating mine with model-to-production reconciliations that were within norms. Because of this, the author considers the assay database adequate for mineral resource estimation.

12 DATA VERIFICATION

12.1 INTRODUCTION

The author reviewed the geological logs, available sampling and assaying Quality Assurance (QA) procedures and the Quality Control (QC) results from the 2023 Aguablanca drilling program during the current personal inspection. The author has relied on the following described database validation conducted by Denarius from December 2023 up to the effective date of this technical report. The author kept in contact with Denarius' technicians throughout the database validation. No limitations were placed on the QP during the validation process.

12.2 DRILLING DATABASE

The complete drill hole database was supplied to in five text files: Assays.csv, Collar.csv, Survey.csv, Lithology.csv, and Mineralization.csv. The files were evaluated for internal consistency of database tables, incorrect characters, data mismatches and cross checks of the individual tables against each other. Minor errors were found in the database and these were corrected by prior to the mineral resource being estimated. The drill hole database appears to be of an industry standard and the author recommends the database is suitable for a mineral resource estimate.

Denarius completed a database validation on May 9, 2024. Results of the drilling dataset are summarized below:

i. The assay database does not include sample numbers.
ii. Each drill hole has a file with printed information and the hand-drawn logs. The assay sheets are Excel printouts, and do not include sample numbers. We have found printed worksheets from three different drill holes with sample numbers. The sample number nomenclature is as follows:
    a. AGUXXXX-YYYYYS. The XXXX is the drill hole number and the YYYYY is the meterage at the end of each sample. For example, a sample from 483.00 to 484.25m in drill hole AGU1117 would be numbered AGU111748425S. Sample numbers are therefore alphanumerical and not consecutive.
iii. Duplicate samples were sent to OMAC labs in Ireland (now ALS Ireland) and to a local lab called ITMA.
iv. OMAC results consistently showed higher values in Ni, Cu, and Co. The difference is less than 10% for Ni and Cu, but increases to over 40% for Co. Where available, the OMAC is preferred over the in-house laboratory.
v. The in-house lab at Aguablanca has an XRF and an AA spectrometer. It reported results for Ni, Cu, MgO, S, Fe (Total), Fe (sulphide), Co and As. No records of Fire Assay gold or PGEs was found.
vi. OMAC assayed for gold and PGEs on top of the elements assayed in house. Where duplicate samples were not collected, the gold and PGE values are reported as -9 or 0. It is unclear how these values were incorporated into the resource.
vii. OMAC certificates might be available from ALS and should be requested.
viii. No signed assay certificates ITMA were found.
ix. Assay certificates from the internal lab are handwritten and signed. Nickel, copper, and cobalt values are reported in parts per tens of thousands (pptt or ‱). In other words, the values need to be multiplied by 100 to obtain ppm or divided by 100 to obtain percentages. This unit is highly unusual, and no document has been found explaining why it was chosen. None of the certificates state clearly the unit used.
x. The data dump uses ppm for Ni, Cu and Co grades. It is assumed that an operator multiplied the values received from the internal laboratory by 100. For instance, if the lab reported 5.03pptt, the database records 503ppm.
xi. The internal laboratory had an apparent lower detection limit of 10ppm (or 0.1 ‱). However, the database reports all values below 50 ppm as either 50ppm or -9ppm. In cobalt´s case, this lower detection limit appears to have been poorly enforced as values up to 100ppm were reported as containing 50ppm Co.
xii. Assuming our interpretation of the sampling nomenclature is correct, the values in the data dump match the values in the individual drill hole file and the signed certificates. However, further validation is recommended.
xiii. MAC values are reported with up to 6 decimal places of accuracy. This is likely due to a conversion from % to ppm, but it needs corroboration once we have access to the certificates.
xiv. Future validation should include collect ¼ core from selected drill holes to send to certified laboratory to check. Core is in good shape and well stored. Sample numbers are in the core boxes, but the depth markers are.
xv. There are assay certificates from the internal laboratory with sample numbers referencing drill holes that are not included in the database. For example, we have found a certificate to what appears to be drill hole AGU1233, which is not in the database.

Following this initial review, Denarius personnel spent several weeks obtaining written records from the internal Aguablanca laboratory, and digital certificates from OMAC, now part of the ALS group of assay laboratories. Some certificates were also obtained from ITMA, a local Spanish laboratory. Out of 153 drill holes from which selected samples were sent to OMAC for check analysis and gold and PGE assays, eleven certificates have not been found, and material errors were found on individual results from approximately a third of all certificates. Table 12-1shows a summary of the drill hole database, and the assay certificates, both paper and digital, found to date.

Table 12-1 Summary of Updated Drilling Database

	Validated	Not Validated	Total
Drillholes	187	323	509
Samples	11,275	14,277	25,551

Most of the drill holes and samples that could not be validated were from historical drill holes completed prior to Lundin´s involvement in Aguablanca. However, these holes are included in the current mineral resource estimate.

A review by the Denarius field crew found multiple inconsistencies in the database compared to the hand-written internal laboratory certificates, mostly to do with lack of lower detection limits, lack of certainty when comparing original sample numbers with duplicate samples sent to umpire labs, material errors when transcribing handwritten geochemical data into spreadsheets, and others. The complete scanned and digital assay certificates and the data dump was sent to a reputable database specialist to generate a new validated dataset. Validated grades are those where original assay certificates are available and have been checked one by one against the database.

Table 12-2 and Figure 12-1 shows a summary of the new validated database.

Table 12-2 Summary of newly updated drilling database

	AG_PPM		CO_%		CU_%		NI_%	
	NOT VALIDATED	VALIDATED	NOT VALIDATED	VALIDATED	NOT VALIDATED	VALIDATED	NOT VALIDATED	VALIDATED
Samples	5398	953	9008	11117	14263	11273	14275	11273
Mode	-0.1	-5	0.002	0.005	0.01	0.005	0.01	0.005
Median	0.3	-5	0.004	0.005	0.04	0.035	0.06	0.052
Mean	0.842	-1.391	0.009	0.008	0.207	0.187	0.286	0.223
Minimum	-31.1	-5	-44	-0.01	-17	-0.01	-44	-0.01
Maximum	200	26.534	0.173	0.38	3.99	3.204	6.5	7.888

	AU_PPB		PT_PPB		PD_PPB	
	NOT VALIDATED	VALIDATED	NOT VALIDATED	VALIDATED	NOT VALIDATED	VALIDATED
Samples	8396	1321	8246	1167	8266	1167
Mode	15	25	15	15	15	15
Median	15	110	15	287.58	15	245.599
Mean	58.61	136.662	101.276	305.806	95.561	268.787
Minimum	-2	-0.05	-31.1	-2	-31.1	-2
Maximum	4100	867	2960	1604.439	2053	1763.503

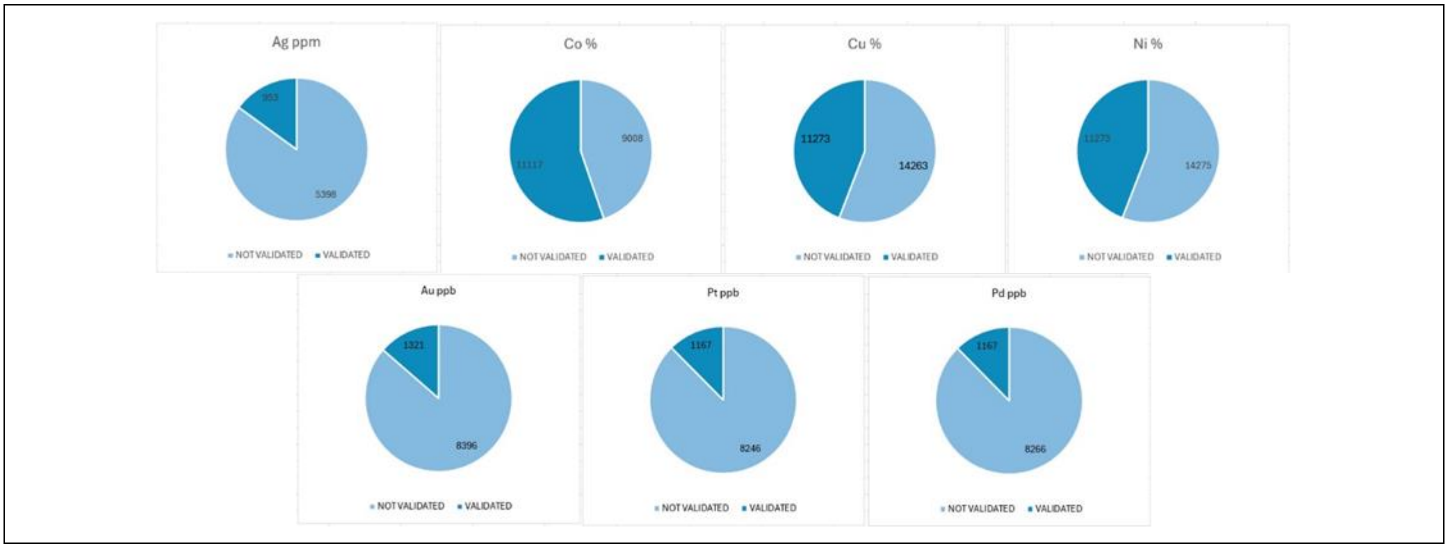


Figure 12-1 Distribution of validated and non-validated drilling samples

The inconsistencies found in the internal laboratory results are being addressed. The review of the assay certificates from OMAC also showed several inconsistencies. Most of the errors were material errors when transcribing the results into the sampling spreadsheet, a process that was likely done manually. For example, 37 samples do not have valid sample numbers in the database, with the relevant cells left empty.

Aguablanca has a history of producing nickel and copper concentrates with no known large model-to-production variances. Additionally, the above-mentioned database validation demonstrated that the nickel and copper grades in the database are valid. It is the opinion of the QP that the data is adequate for the purposes used in the Technical Report.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 MINERALOGY

As described in Section 7, the Aguablanca Deposit is a magmatic sulfide deposit hosted in gabbro and gabbro-norite rocks. The deposit is rich in nickel (Ni) and copper (Cu) with anomalous cobalt (Co), platinum (Pt), palladium (Pd) and gold (Au).

13.2 EXISTING PROCESS PLANT

Due to historic existing plant performance data and because the underground ore body is an extension of the open pit ore body, no mineral processing test work was deemed necessary to forecast underground ore amenability with the process already in place. Therefore, equations were developed using historic nickel and copper recoveries as a function of ore grade and plant ore feed rate based on past process performance.

Because of the 2007 crusher system upgrade, data prior to 2008 were not used when determining future recovery equations. Additionally, 2009 and 2010 data also were not included because stockpile material, likely at least partially oxidized, was blended with open pit ore during those years to maintain optimum plant capacity. Oxidized material does not respond to flotation as effectively as fresh ore so blending stockpile material with fresh open pit ore was possibly detrimental to plant performance.

While the process plant was restarted in August of 2012, August and September production data were also excluded from recovery equation determination since typical startup problems that impact plant performance were likely experienced. Finally, data from the final six months of 2015 operations were not used because stockpile material was again processed and some reagent and grinding media additions were decreased as consumables inventories were depleted.

13.3 RECOVERY EQUATIONS

Although the nickel scavenger circuit expansion was not completed until 2010, May through December 2008 data was used when determining future recoveries. January through April 2008 monthly process performance data was not available. Process plant feed rate during May through December 2008 averaged 237 tph while feed rates during 2012, 2013, 2014, and 2015 averaged 181, 201, 206, and 209 tph, respectively. Because some future months' process plant feed rates will exceed the 2012 through 2015 feed rates, 2008 data was used to represent recoveries at higher feed rates.

The influence of higher 2008 plant feed rate on recovery cannot be ignored because over 18% of underground ore processing plant feed rates would exceed the recovery equation determination plant feed rate if 2008 data were not included. Nickel recoveries for May through December 2008 averaged 80.5%. Because the Aguablanca competent person stated that nickel recovery increased by 2% after the nickel scavenger circuit was expanded, 2% was added to 2008 recovery for recovery equation determination.

Figure 13-1 shows actual nickel recoveries as a function of nickel head grade. Figure 13-2 illustrates actual nickel recoveries as a function of plant feed rate. The linear regression curve and regression equation for each graph is included on the graph. Note that recovery increases with head grade but decreases with increased feed rate. While increased head grade increases the likelihood of mineral contact with flotation bubbles, increased throughput decreases mineral liberation and flotation cell retention time.

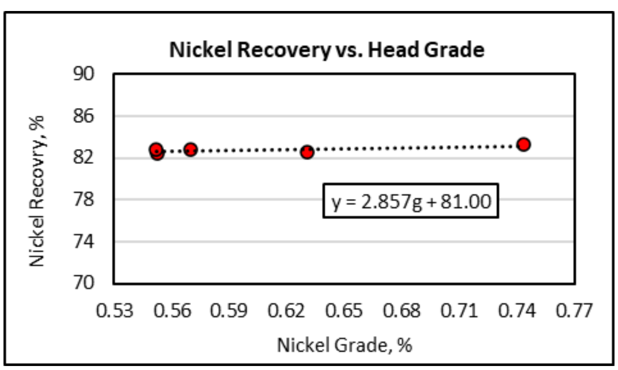


Figure 13-1 Actual nickel recovery vs. head grade

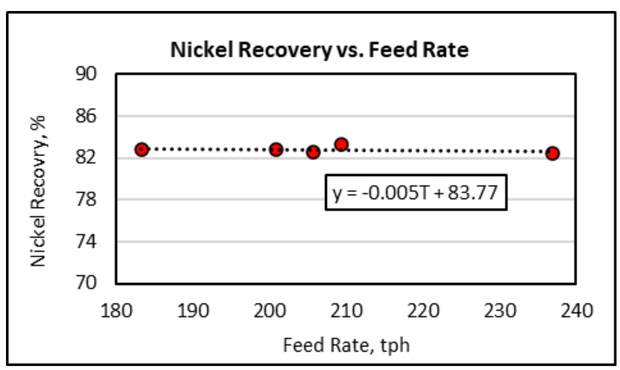


Figure 13-2 Actual nickel recovery vs. feed rate

Because flotation recovery is impacted by both grade and throughput, the linear regression curves were combined to create a single recovery equation. The equation developed to representative nickel recovery from underground ore is"

$$R_{Ni} = 0.651 \times (2.857g_{Ni} + 81.00) + 0.349 \times (-0.005T + 83.77)$$

Where: R_{Ni} = nickel recovery, g_{Ni} = nickel head grade and, T = plant feed rate in tph.

The multipliers 0.651 and 0.349 represent the influence that grade and feed rate had on recovery over the grade and feed rate ranges.

Presented in Table 13-1 are the actual plant data represented in Figures 13-1 and 13-2. Also included in Table 13-1 are calculated nickel recoveries based on the nickel recovery equation.

Table 13-1 Plant nickel data and calculated recoveries

Year	Ni Grade, %	Feed Rate, tph	Actual Recovery, %	Calculated Recovery, %
2008	0.552	237	82.5[1]	82.6
2012	0.552	183	82.7	82.7
2013	0.570	201	82.8	82.7
2014	0.630	206	82.5	82.8
2015	0.743	209	83.2	83.0

1. Added 2% to compensate for nickel scavenger expansion

A copper recovery equation was developed using the same method as for nickel recoveries and using data over the same periods. Actual copper recoveries as a function of copper feed grade and mill feed rate are shown in Figures 13-3 and 13-4, respectively

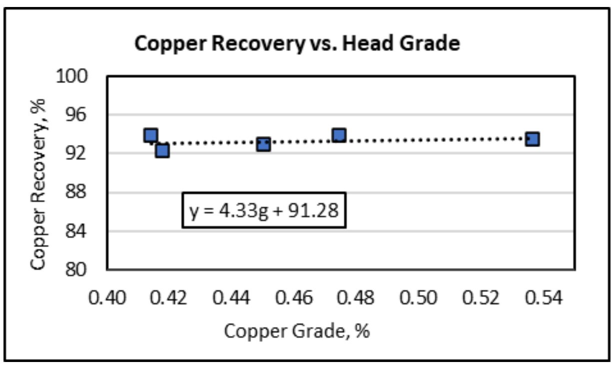


Figure 13-3 Actual copper recovery vs. head grade

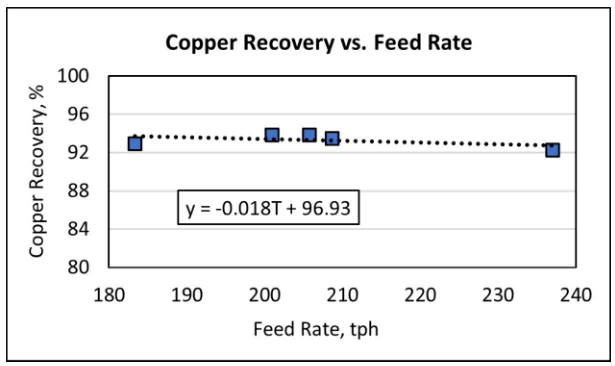


Figure 13-4 Actual copper recovery vs. feed rate

The equation developed to representative copper recovery from underground ore is:

$$RCu = 0.437 \times (4.334gCu + 91.28) + 0.563 \times (-0.018T + 96.93)$$

Where: RCu = copper recovery, gCu = copper head grade, and T = plant feed rate in tph.

Table 13-2 shows data represented in Figures 13-3 and 13-4. Also included in Table 13.2 are calculated copper recoveries based on the copper recovery equation.

Table 13-2 Plant copper data and calculated recoveries

Year	Cu Grade, %	Feed Rate, tph	Actual Recovery, %	Calculated Recovery, %
2008	0.418	237	92.2	92.9
2012	0.450	183	92.9	93.5
2013	0.414	201	93.8	93.2
2014	0.474	206	93.9	93.3
2015	0.537	209	93.5	93.4

These equations were developed based on actual nickel and copper head grades and feed rate ranges. The equations may not be inaccurate for head grades and feed rates outside of those ranges. Planned nickel grades (based on the mine plan within this study) fall within the equation calculation grade range in all cases except for November, 2025, when the plan head grade will be 0.760%. Plan nickel head grades are compared to the equation head grade range in Figure 13-5. The Plan points shown in Figure 13-5 are monthly for 2025, quarterly for 2026 through 2028, and yearly for 2029 and 2030

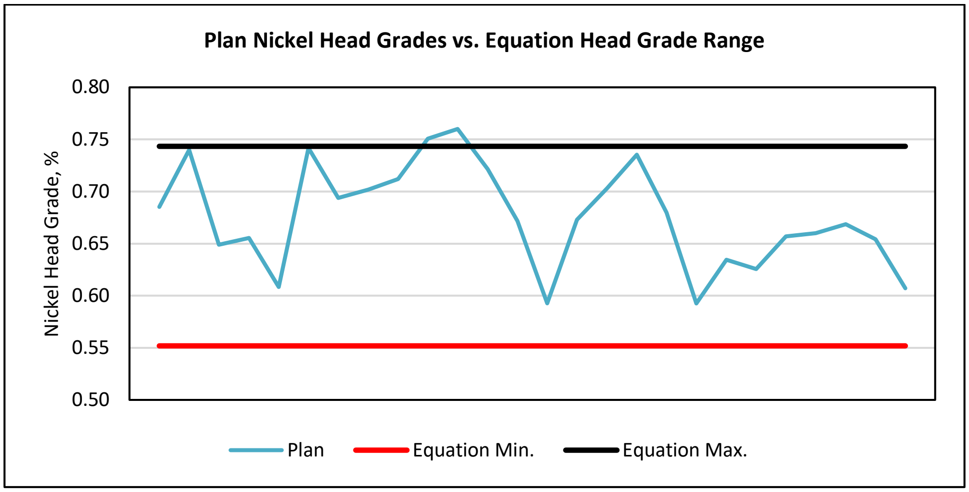


Figure 13-5 Plan nickel head grades compared to recovery equation grades

Copper head grades fall within the copper recovery equation grade range during only three of the twenty-four plan periods. Recoveries from ore having grades outside of the equation grade range may not follow the recovery equation trend. Plan copper head grades are compared to the equation head grade range in Figure 13-6. Again, the Plan points shown in Figure 13-5are monthly for 2025, quarterly for 2026 through 2028, and yearly for 2029 and 2030.

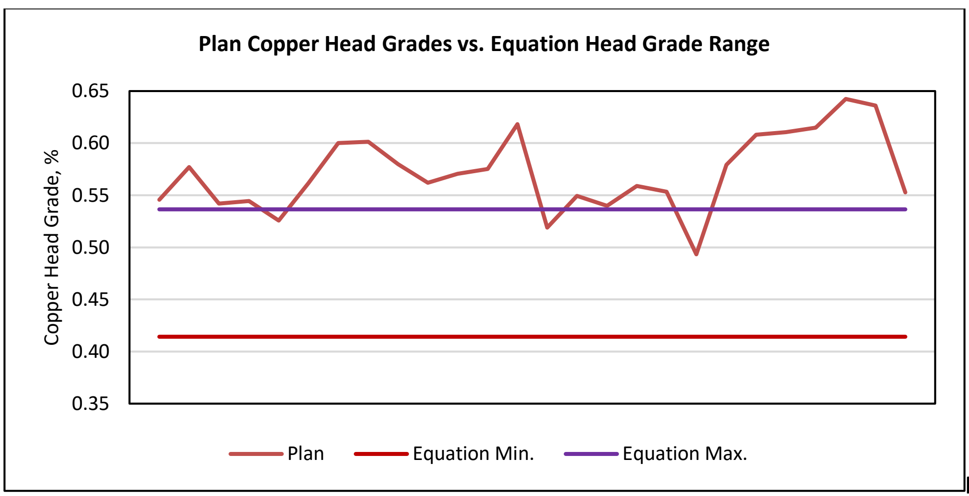


Figure 13-6

Figure 13-6 Plan copper head grades compared to recovery equation grades

Shown in Figure 13-7 is a comparison between Plan feed rates to the equation determination feed rate range. Here Plan data is shown monthly for the entire Plan life with sixty of the seventy Plan operating months having feed rates falling within the equation determination feed rate range. Plan feed rate does not exceed the equation determination maximum feed rate at any time.

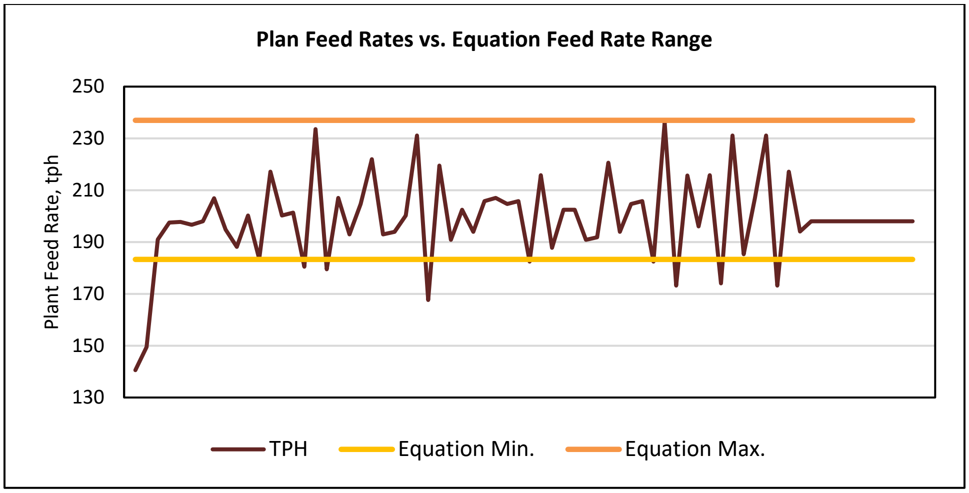


Figure 13-7 Plan feed rates compared to recovery equation feed rates

Plan monthly mill feed tonnages were based on the assumption that all ore mined during each month will be processed during the same month. Realistically, depending on space available for short-term stockpile storage, feed rates will likely be adjusted to maintain a more consistent plant throughput than what is shown in Figure 13-7.

An equation was also developed to forecast concentrate weight as a percentage of plant feed weight, again dependent on ore grades and plant feed rate. For the concentrate weight equation, the sum of nickel and copper grades was used as the grade factor. Concentrate weight percentages of feed weights as a function of head grade and plant feed rate are shown in Figures 13-8 and 13-9, respectively.

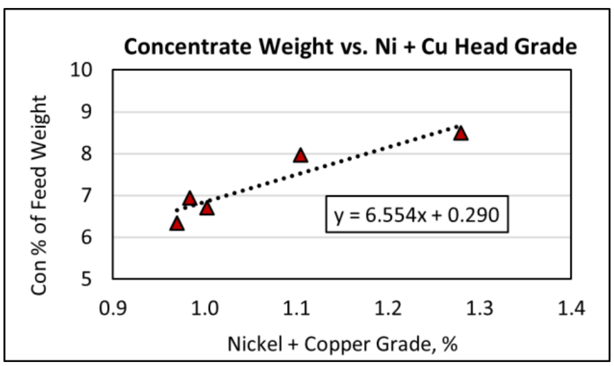


Figure 13-8 Actual concentrate % vs. Ni+Cu head grade

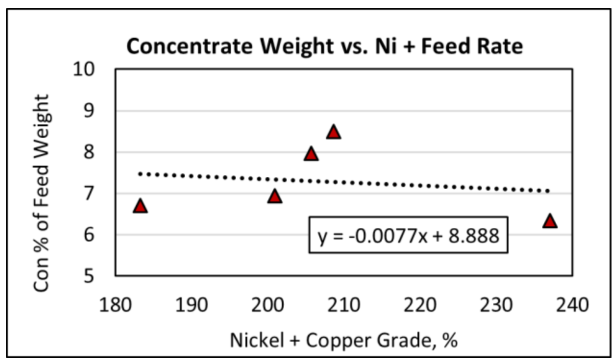


Figure 13-9 actual Concentrate % vs. plant feed rate

The equation developed to concentrate weight percentage of plant feed weight from underground ore is:

> Rwt = 0.883 x (6.554gNi+Cu + 0.290) + 0.117 x (-0.008T + 8.89)
>
> Where: Rwt = the % of feed weight reporting to concentrate, gNi+Cu = the sum of nickel and copper head grades, and T = plant feed rate in tph.

The regression curve in the Figure 13-9 is not a close fit with the data points. However, the Feed Rate factors of the concentrate weight equation only represent 11.7% of the equation outcome. Table 13-3 shows data represented in Figures 13-8 and 13-9 and also includes calculated weight recovery percentages based on the combined grade and feed rate equations.

Table 13-3 Actual and calculated feed weight % to concentrate

Year	Ni + Cu Grade, %	Feed Rate, tph	Actual Feed Weigh % Reporting to Concentrate	Calculated Feed Weigh % Reporting to Concentrate
2008	0.970	237	6.34	6.70
2012	1.002	183	6.71	6.93
2013	0.984	201	6.94	6.81
2014	1.105	206	7.97	7.50
2015	1.280	209	8.49	8.51

14 MINERAL RESOURCE ESTIMATES

The global mineral resource estimate was prepared based on a resource model constructed using Vulcan Geomodeller®. The Aguablanca mineral resource was estimated using a combination of Ordinary Kriging (OK) and Inverse Distance Weighting (IDW) interpolation techniques.

Two sources of volumetric determination were used for the resource model. One was a three dimensional (3D) structural model of the various Aguablanca deposit breccia pipes. These models have been refined throughout the life of the project yielding a very accurate representation of the sulfide bodies at Aguablanca. A second volume model was an implicit model determined through indictor kriging techniques. This implicit model was used to differentiate and validate mineral grade domains within the 3D Aguablanca models.

Mineralization for the deposit consists of Ni, Cu, Co, Au, Pd and Pt. Ni, Cu and Co were estimate into the block model using OK. Au, Pd and Pt were estimated using Inverse Distance Cubed (ID3) interpolation techniques. Rock densities are variable based on the nickel grade of each model block. Hard boundaries and domains were used to ensure mineralization was not estimated past any hard boundaries in the mineral deposit. Mineral resource estimates for Aguablanca closely match the final mineral resource estimated of Lundin Mining of 2015 at the close of mining operations.

14.1 DATA USED

The total Aguablanca drilling and sampling datasets are shown in Table 14-1. Drilling through the history of the mine was completed by Rio Tinto and Rio Narcea. The dataset contains records for 523 diamond drillholes. Of these, two deep exploration holes were drilled in 2022. The remaining holes were drilled from project inception through 2015. For data validation purposes, Denarius completed a data entry analysis, comparing XXX assay certificates with the dataset. The analysis showed a less than 1% error data entry which is considered well within the standards of accuracy for use in mineral resource estimates. The errors have been addressed in the current dataset.

Topography is based on the current Digital Elevation Model from 2015 aerial photography. This topography surface takes into account the current pit surface and is used t estimate mineral resource remaining in the ground.

Densities used in the resource are based on 7,280 individual measurements from core samples. The slope of regression was calculated for Ni grade vs density and then calculated on a block-by-block basis based on the slope of regression.

Table 14-1 Drilling and Sampling

Company	Drilling type	Number of Holes	Meters Drilled
Rio Tinto	Diamond	195	32,771
Rio Narcea	Diamond	326	81,346
Total		521	114,117

14.2 DATA ANALYSIS

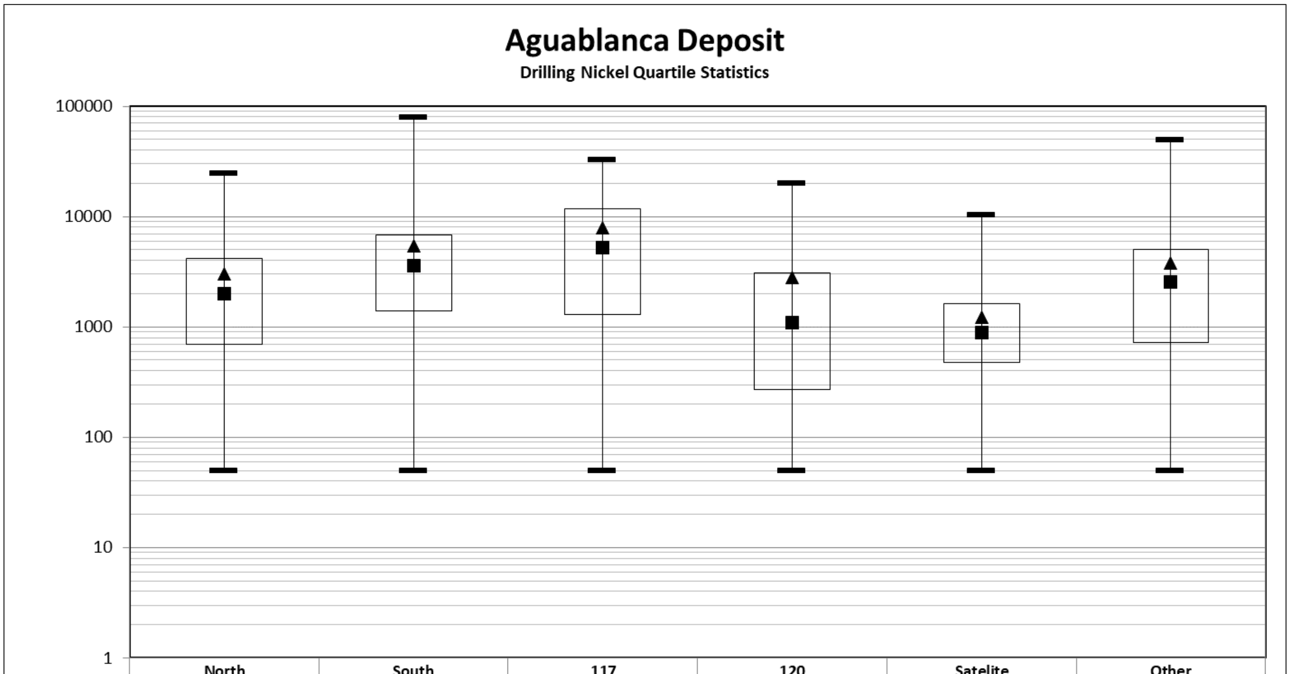
A statistical summary of Ni, Cu, Co, Au, Pd and Pt is shown in Table 14-2. These elements are of major interest and drive the mining, metallurgical and economic considerations for the Aguablanca mineral deposit. Even though the Au and PGM grades are low, these add significant value to the deposit on a byproduct credit basis.

Table 14-2 Metal Summary Statistics

Element	Unit	N	Mean	Maximum	Std. Dev.	C.V.
Ni	ppm	26,016	2,583	78,880	4,884	1.89
Cu	ppm	25,996	1,997	39,900	3,049	1.53
Co	ppm	20,741	91	7,100	145	1.59

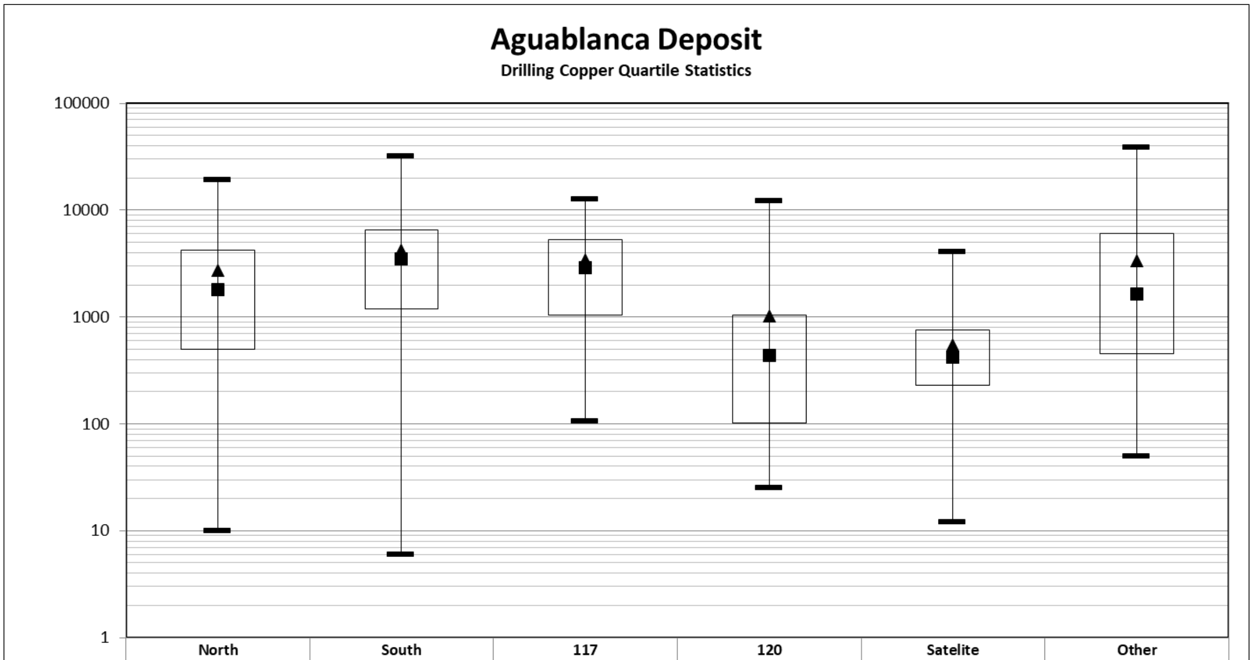
Element	Unit	N	Mean	Maximum	Std. Dev.	C.V.
Au	ppb	7,956	89	6,494	132	1.49
Pd	ppb	8,064	164	2,053	178	1.09
Pt	ppb	8,065	178	2,960	195	1.09

Each of the assay intervals were logged for lithology, alteration and mineralization. Mineralization displays varying average grades for each of the six mineral bodies identified at the Project (Figure 14-1 and Figure 14-2). Samples assay values have low coefficients of variation (CV) for Ni and Cu. There are very few high-grade outliers which could skew the mean average grade above the third quartile. Evaluation of the graphs suggests that capping is not required for the dominant mineralization in the Aguablanca deposit.



	North	South	117	120	Satelite	Other
Q1	700	1400	1304	272	481	723
− Min	50	50	50	50	50	50
■ Median	1990	3579	5260	1103	895	2549
▲ Mean	2999	5435	7915	2820	1220	3807
− Max	24400	78880	32705	19828	10399	49238
Q3	4200	6800	11733	3080	1611	5050
NSamples	1043	10164	42	100	489	156
Std. Dev.	3321	6503	9024	4068	1124	5568
C.V.	1.110	1.190	1.140	1.440	0.920	1.460

Figure 14-1 Uncapped nickel grade distribution by mineral body



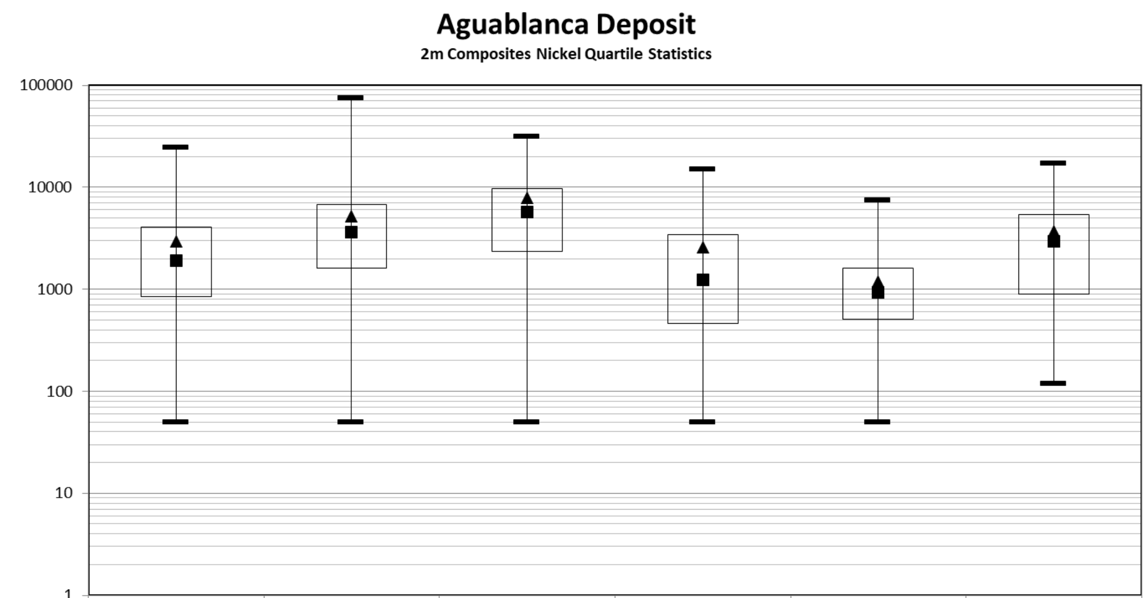
	North	South	117	120	Satelite	Other
Q1	500	1200	1038	102	228	455
− Min	10	6	106	25	12	50
■ Median	1800	3500	2904	439	417	1628
▲ Mean	2752	4224	3441	1029	549	3344
− Max	19050	32045	12651	12148	4078	38900
Q3	4222	6500	5342	1035	757	6100
NSamples	1043	10157	42	100	488	156
Std. Dev.	2793	3548	2875	1699	488	4358
C.V.	1.010	0.840	0.830	1.650	0.880	1.300

Figure 14-2 Uncapped copper grade distribution by mineral body

14.3 COMPOSITING

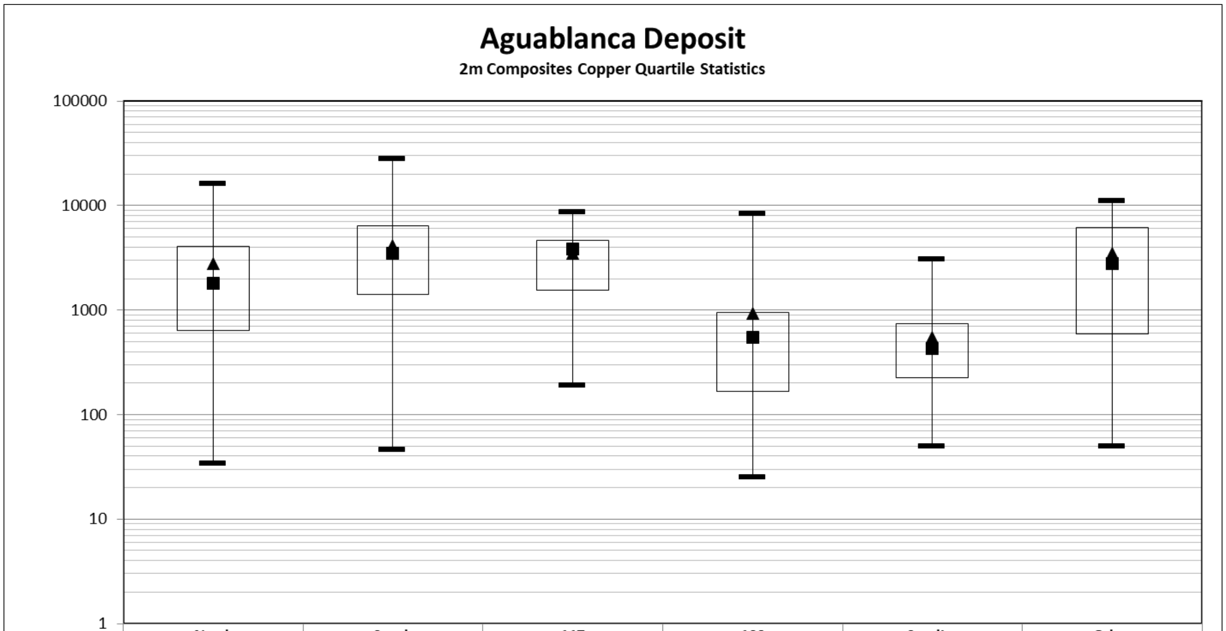
Compositing reduces the impact of short assay intervals and helps to better estimate the average grade of the deposit. Compositing incorporates a certain amount of dilution into the raw assay data prior to estimation. The mining operation envisioned for the Project, underground, will be at a scale larger than the assays sampled for the deposit. The selective mining unit for the Project is expected to be 2 m, therefore, the assays for the database have been composited to 2 m. Composites are length weighted down hole composites of the six minerals of interest; Ni, Cu, Co, Au, Pd and Pt.

Figure 14-3 and Figure 14-4 detail the final composite statistics, by mineral deposit, that have been used for the mineral resource estimate. CV's are within acceptable ranges, high grade outliers are accounted for and average NI and Cu grades are within acceptable ranges. The manipulation from assays to composites has been carried out with industry accepted practices and the author recommends that the final composite database can be used for mineral resource estimation of the Aguablanca mineral deposit.



	North	South	117	120	Satelite	Other
Q1	850	1600	2343	463	507	891
− Min	50	50	50	50	50	118
■ Median	1900	3660	5665	1241	933	2943
▲ Mean	2959	5230	7950	2562	1188	3673
− Max	24400	74322	31334	15073	7504	17196
Q3	4040	6835	9767	3434	1614	5364
NSamples	656	7364	23	71	383	107
Std. Dev.	3196	5616	7710	3228	975	3367
C.V.	1.080	1.070	0.960	1.250	0.820	0.910

Figure 14-3 Aguablanca Ni composite statistics



	North	South	117	120	Satelite	Other
Q1	640	1400	1545	168	224	590
− Min	34	46	191	25	50	50
■ Median	1800	3490	3845	550	432	2810
▲ Mean	2760	4179	3501	938	544	3506
− Max	16127	28143	8629	8331	3042	10989
Q3	4100	6400	4644	954	739	6180
NSamples	658	7347	23	71	378	107
Std. Dev.	2744	3303	2192	1321	438	3066
C.V.	0.990	0.790	0.620	1.410	0.800	0.870

Figure 14-4 Aguablanca Cu composite statistics

14.4 DECLUSTERING

Cell declustering was evaluated to ensure that gold and PGM grades were not biased by a large sample set of grades that may have been localized to any one area. Figure 14-5, Figure 14-6 and Figure 14-7 demonstrate the results. Gold shows an increase in grades following the minimum at the 150 m cell size. PGM grades do not. Therefore, declustering weights were used for the estimation of gold but not platinum or palladium.

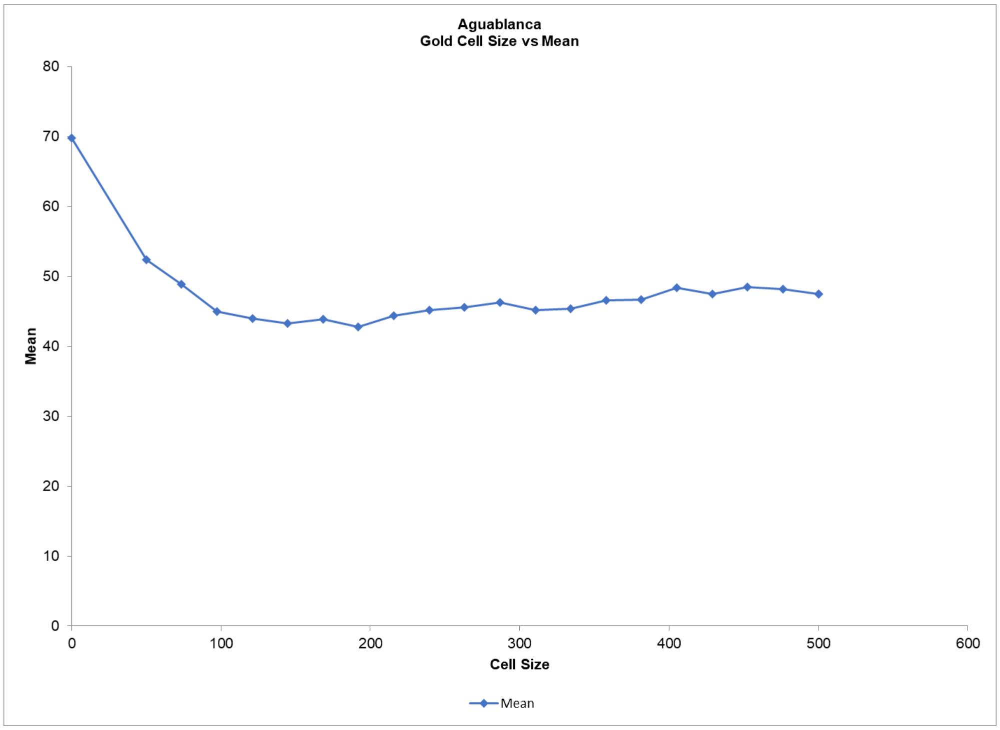


Figure 14-5 Gold cell declustering graph

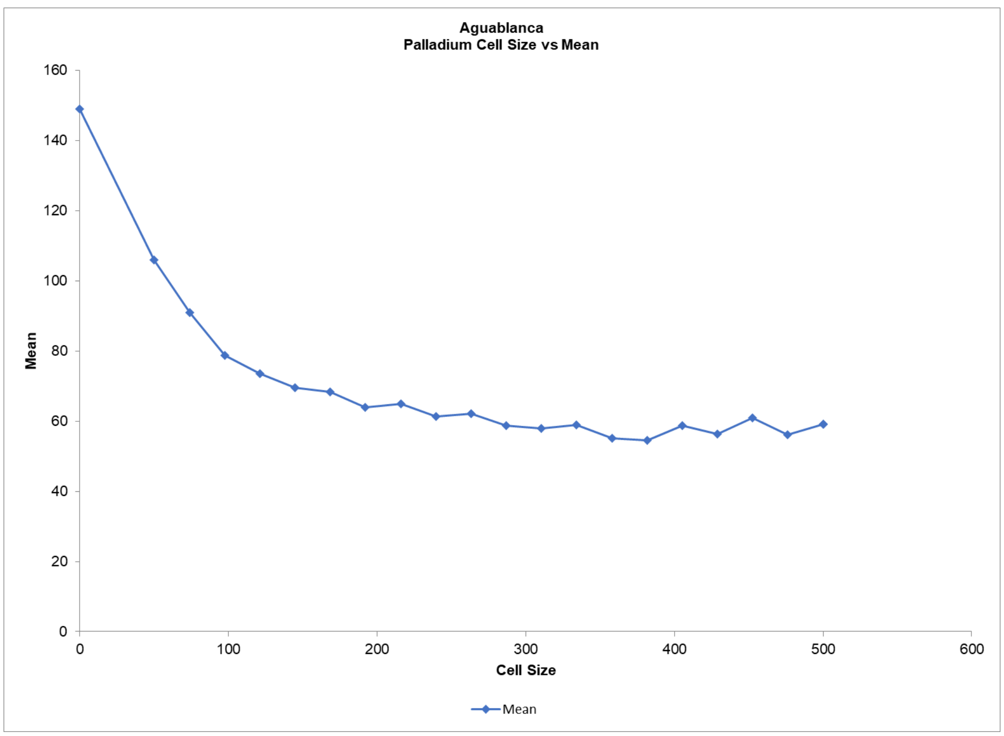


Figure 14-6 Palladium cell declustering graph

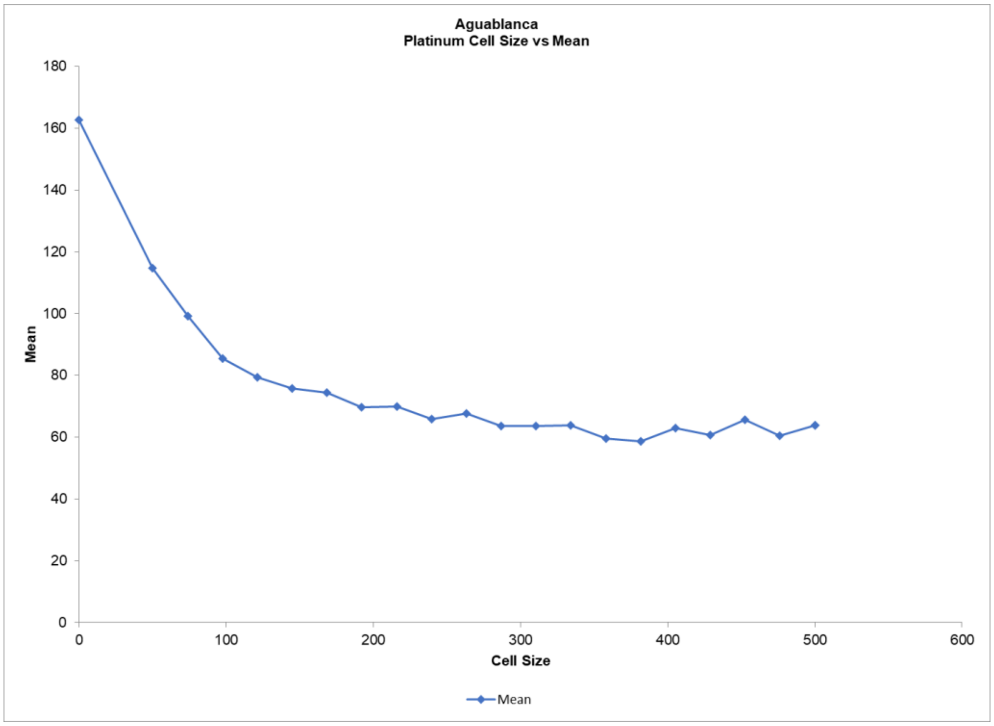


Figure 14-7 Platinum cell declustering graph

14.5 VARIOGRAPHY

Variograms were prepared and evaluated for Ni and Cu. Variograms determine the directions of least variance, which is clearly vertical at Aguablanca. Additionally, variograms aid in the determination of the anisotropy of mineralization. This is helpful for estimating mineralization for kriging plans and inverse distance estimation plans. Variograms were calculated with Sage2000 software.

Nickel Variogram, anisotropy fit parameters and search ellipsoid

Figure 14-8 demonstrates the range down the dip (-60) of the Aguablanca mineralized breccia for nickel. The ellipsoid fit criteria for nickel are shown in Figure 14-9. These parameters were modeling in Vulcan software. For comparison to Figure 14-9, Figure 14-10 shows the Vulcan search ellipsoid entries used for the estimation of nickel mineral resources for the Project.

Copper variogram, anisotropy fit parameters and search ellipsoid

Figure 14-11 demonstrates the range down the dip (-75) of the Aguablanca mineralized breccia for copper. The ellipsoid fit criteria for nickel are shown in Figure 14-9. These parameters were modeled in Vulcan software. Figure 14-10 shows the search ellipsoid used for the estimation of nickel mineral resources for the Project.

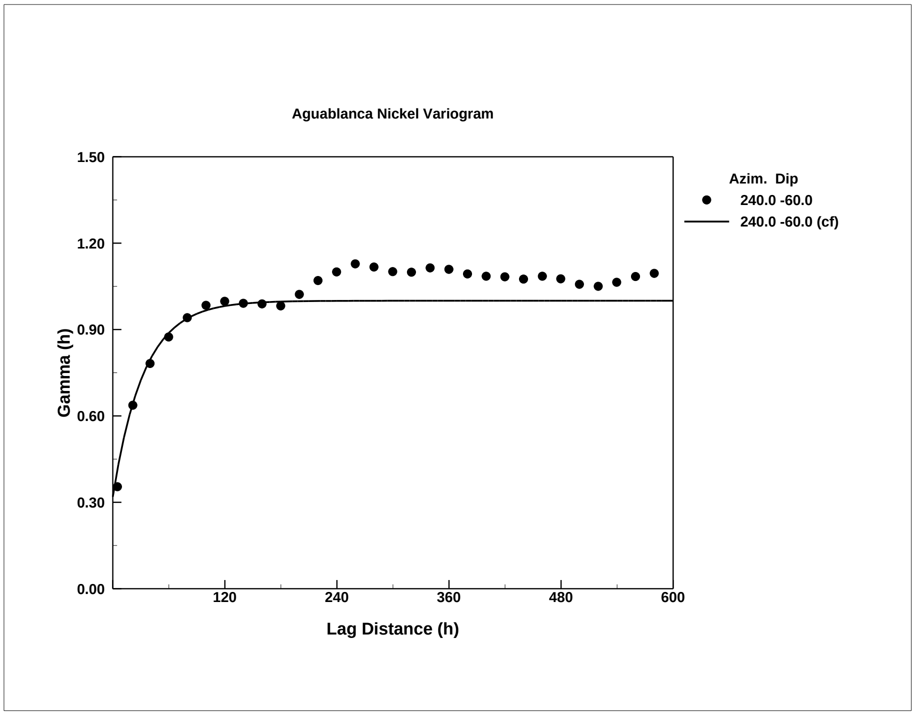


Figure 14-8 Nickel variogram

Aguablanca Nickel Variogram

Vulcan Rotation Conventions

Nugget ==> 0.319
C1 ==> 0.681

First Structure -- Exponential with Practical Range

LH Rotation about the Z axis ==> -22
RH Rotation about the X' axis ==> 1
LH Rotation about the Y' axis ==> 34
Range along the Z' axis ==> 175.2 Azimuth ==> 247 Dip ==> 56
Range along the Y' axis ==> 71.6 Azimuth ==> 338 Dip ==> 1
Range along the X' axis ==> 97.2 Azimuth ==> 69 Dip ==> 34

Figure 14-9 Sage2000 nickel anisotropy search parameters

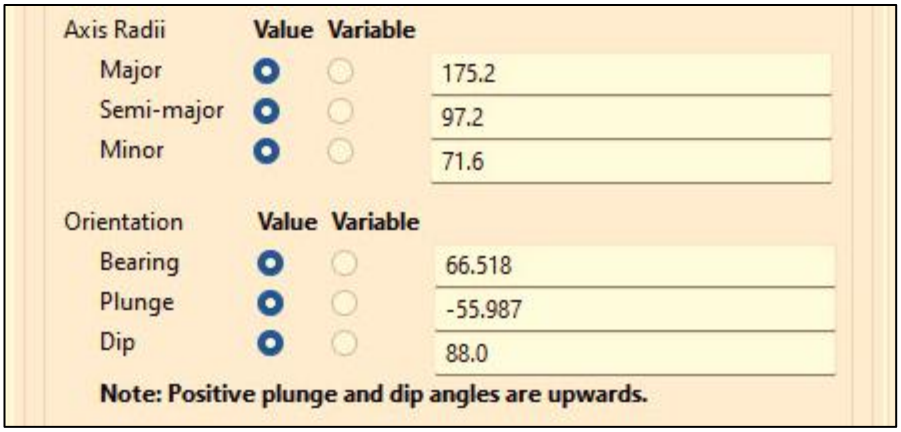


Figure 14-10 Vulcan Anisotropy Search Parameters based on Sage2000 output

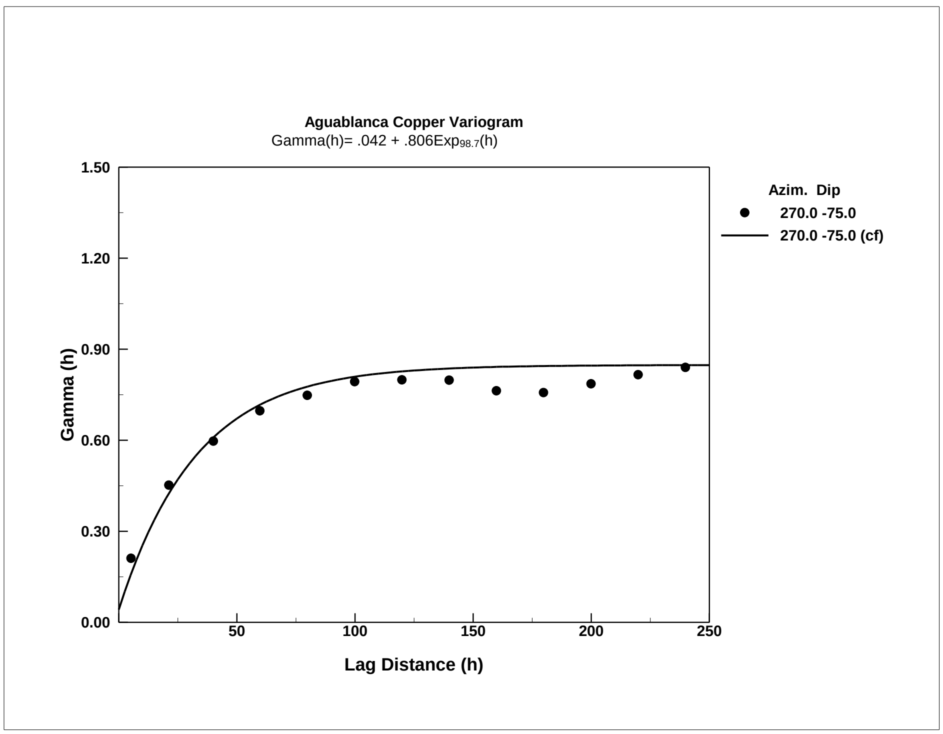


Figure 14-11 Aguablanca Copper Variogram

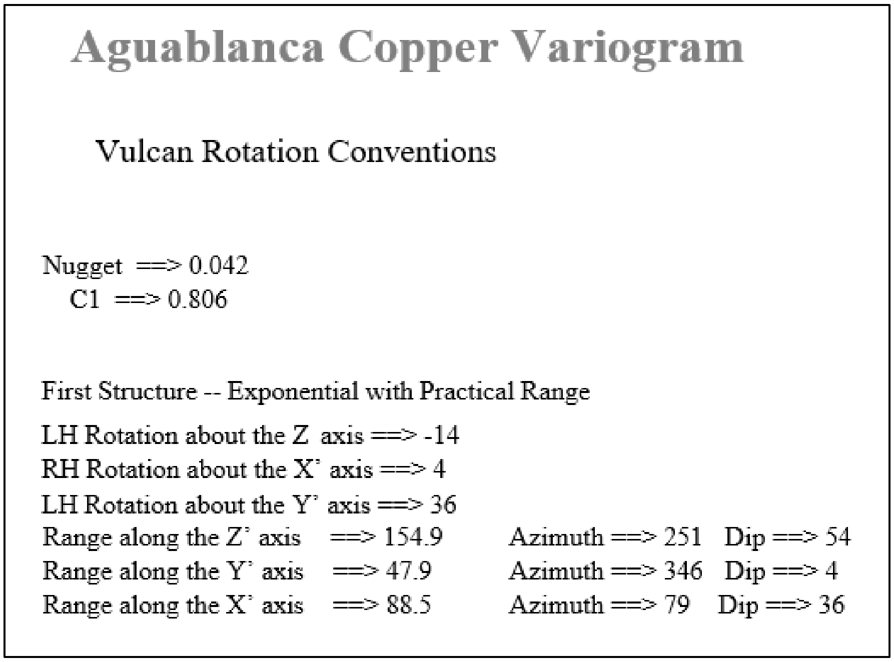


Figure 14-12 Sage2000 copper anisotropy search parameters

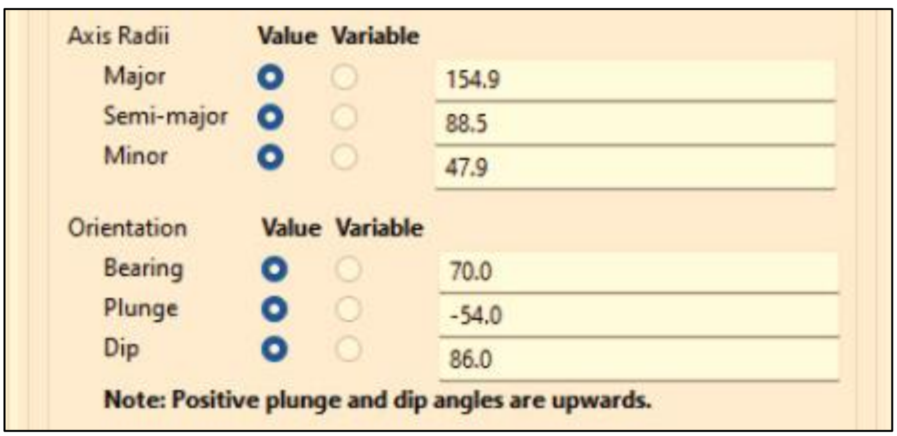


Figure 14-13 Vulcan Anisotropy Search Parameters based on Sage2000 output

14.6 BLOCK MODEL

Vulcan software was used to produce the block model for the Project. The block model used the interpreted mineral domains from 2015. These being the main north and south mineral bodies as well as the 117, 120 and other satellite mains. These wireframes are used to limit the mineralization to known, interpreted mineral bodies at Aguablanca.

The block model was estimated into 4 m cubed parent blocks. Parent blocks were sub-blocked into 2 m divisions along each axis. Sub-blocking was used to accurately reflect the defined mineralization and lithological models, with a sub-block size of 2 m x 2 m x 2 m used to reflect the wireframes and the minimum selective mining unit for underground mining.

Model extents are listed in Table 14-3.

Table 14-3: Block Model Origin, Extents, and Block Sizes

	Easting (X)	Northing (Y)	Elevation (Z)
Model Origin	6,600 m	4,700 m	-755 m
Model Extents	1,400 m	1,200 m	1,400 m
Parent Block Dimensions	4 m	4 m	4 m
Sub-Cell Size	2 m	2 m	2 m

14.7 GRADE ESTIMATION AND INTERPOLATION PLAN

Grade estimation is based on the parent block dimensions of 4 m x 4 m x 4m. Sub-block dimensions of 2 m x 2 m x 2m are reflective of the selective mining (SMU) for the Project. Parent block grades are stored in the sub-block regions. The block size reflects the QP's opinion on a representative size variation for any underground mining. Ni, Cu and Co grade estimation has been completed using Ordinary Kriging methodology. Au, Pd and Pt grades were estimated using Inverse Distance Cubed methods.

Mineralization was categorized as higher intensity mineralization versus lower intensity mineralization based on a cutoff of 1,000 ppm Ni and 1,000 ppm Cu. Using the cutoff criteria, mineralization greater than 1,000 ppm was flagged with a 1 and mineralization below 1,000 ppm was flagged as 0. Indicator variograms were calculated. Subsequently, the 1's and 0's were estimated into a temporary block model using ordinary kriging. The results stored in the block model are estimates between 0 and 1. These results represent the probability that a block will have a grade of 1, i.e., 1,000 ppm or higher. These results were used to define high grade domains within the deposit; to be segregated in mineral resource estimates for Aguablanca.

Composite grade selection required that only high-grade samples be used in high grade domains. Only low-grade samples were used in low grade domains. All domains and mineralized bodies are treated as hard boundaries.

Grade estimation search ellipses follow the orientation of the mineralized breccia and structures within the deposit. Statistical characteristics such as search region, variance measures, and number of samples used in an estimate, were also computed, and stored in each individual block for descriptive evaluations and post processing. Estimation parameters are listed below for each domain in Table 14-4. Kriging estimates were performed in four separate passes for Ni, Cu and Co. Co used the Ni parameters. Single pass ID3 was utilized for Au, Pd and Pt.

Table 14-4: Search Parameters by Mineralized Domain

Domain	Ellipsoid Ranges			Nugget	Sill Differential	Number of Samples		Drillhole Limit
	Maximum (m)	Intermediate (m)	Minimum (m)			Minimum	Maximum	Max Samples per Hole
All – Nickel	175	97	72	0.319	0.681	4	20	3
All – Copper	155	88	48	0.042	0.806	4	20	3
All – Cobalt	175	97	72	0.319	0.681	4	20	3
All – Gold	175	97	72	Inverse Distance Cubed		4	20	3
All – Gold	175	97	72	Inverse Distance Cubed		4	20	3
All – Gold	175	97	72	Inverse Distance Cubed		4	20	3

14.8 MODEL VALIDATION

All estimates of the domains were validated using visual comparison of grade to nearby sampling, statistical population comparisons, and swath plots comparing estimates to drilling.

14.8.1 VISUAL COMPARISON

Estimation results were verified by visual comparison of samples and estimated blocks. An example is shown below in Figure 14-14 for nickel grades in principle deposit. Note that cool colors for sample grades are paired with areas of cool colors in the block model. The same is true for warmer colors, with a reasonable level of gradation between the two relatives to the level of sampling. This visual check was performed for each six mineral domains individually, to ensure reasonable estimates with as few artifacts as possible. Artifacts are defined as areas in the model where mathematically the estimation calculation produces improbable estimated mineralization.

The visual comparison of the samples to the estimated grades shows no obvious bias with a fair reflection of both the high and low-grade samples. It is the QP’s opinion that a reasonable correlation between the block estimates and sampling data can be observed.

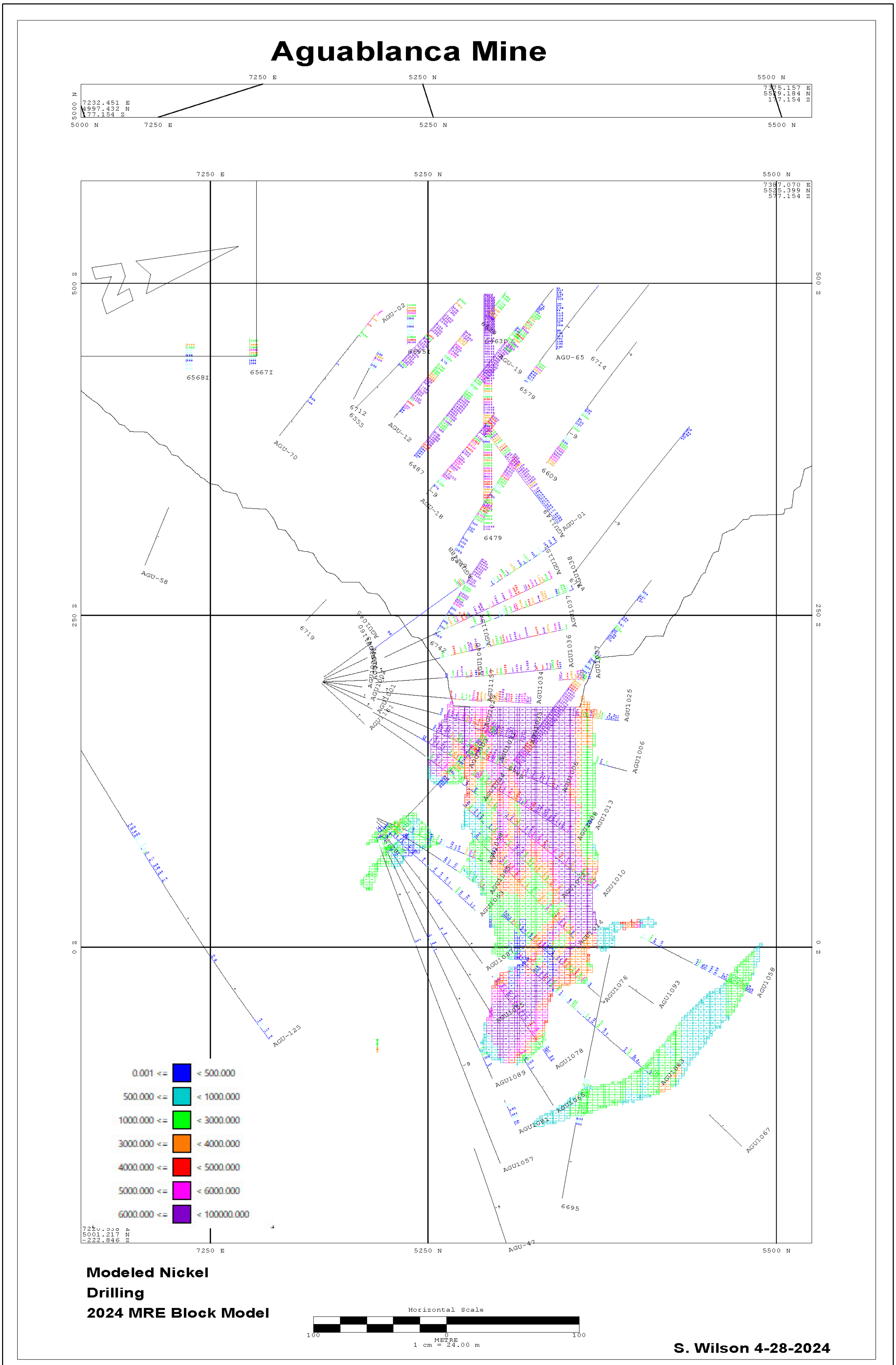


Figure 14-14: Cross Section through the main Aguablanca deposit demonstrating consistency between drilling assay results and the interpolate mineralization for the deposit.

14.8.2 COMPARATIVE STATISTICS

A comparative statistical analysis was completed to determine if there was a positive bias in the estimation of metal grades. There are several different ways to assess this but for Aguablanca an analysis of the composite grades versus the modeled grade was evaluated. Additionally, the coefficients of variance were assessed for each metal. Any time the modelled grade is less than the composite grades indicated that the mineral estimate has properly smoothed the grade estimate for metal. If the percent change would have been positive this would have indicated a bias in the mineral estimate. This would indicate that more work would need to be done before the reported grades would be sufficient for a mineral resource estimate. In the case of Aguablanca, there are no indications of erroneous mineral estimates for any of the estimated metal grades (Table 14-5).

Table 14-5: Percent Difference Composites vs IDW vs NN for Gold and Silver

		Composite Grade	Model Grade	% Change
Nickel	Mean	4,025	4,022	0%
	C.V.	1.18	0.89	-24%
Copper	Mean	3,730	3,716	0%
	C.V.	0.92	0.81	-12%
Cobalt	Mean	112	112	0%
	C.V.	1.03	0.71	-31%
Gold	Mean	142	141	0%
	C.V.	0.77	0.72	-8%
Palladium	Mean	251	251	0%
	C.V.	0.67	0.61	-8%
Platinum	Mean	290	289	0%
	C.V.	0.66	0.62	-6%

14.9 RESOURCE CLASSIFICATION

Mineral Resource classification is typically a subjective concept. Industry best practices suggest that classification should consider the confidence in the geological continuity of mineralization, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Resource classification for Aguablanca was determined by evaluating kriging variance and distances to the nearest drill hole (Figure 14-15).

1. Measured Mineral Resources - Mineralization within 20 m of a sample, estimated with at least two drillholes, and kriging variance less than 0.4 is classified as measured.
2. Indicated Mineral Resources – Mineralization within a range of 20 m to 75 m of a sample and estimated with at least two drillholes and kriging variance less than 0.75 is classified as indicated.
3. Inferred Mineral Resources – All other mineralization is classified as inferred.

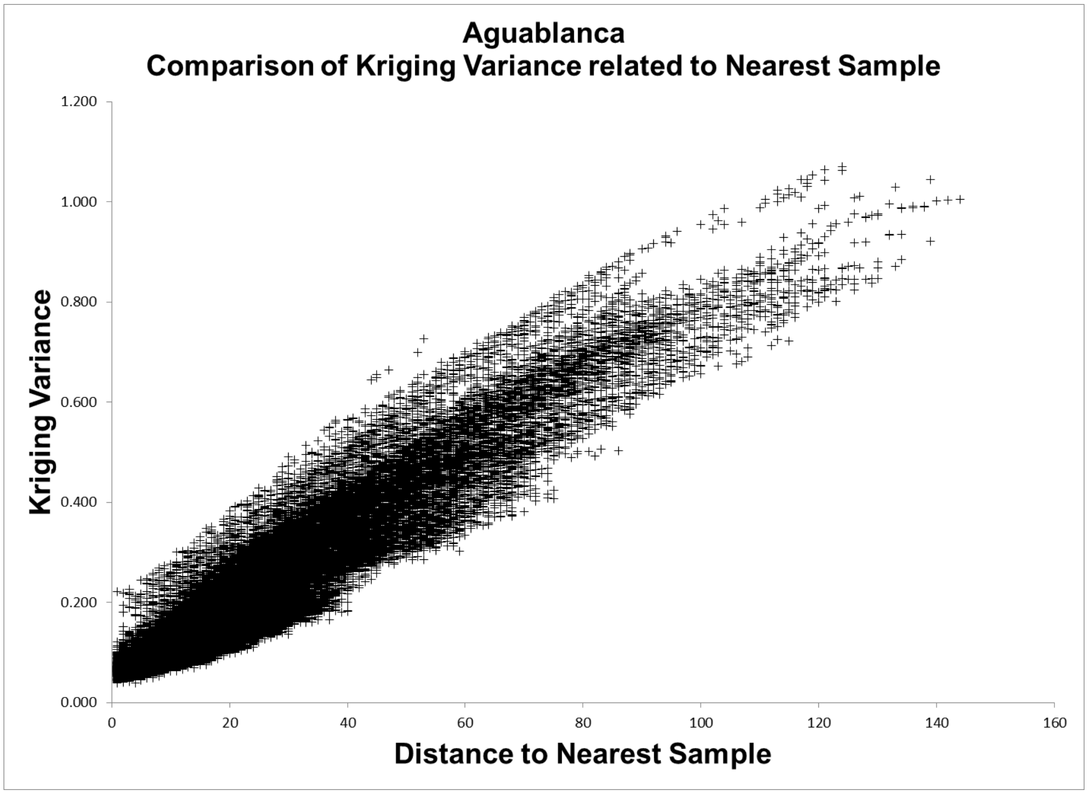


Figure 14-15: Kriging variance versus distance to nearest sample

14.10 MINERAL RESOURCE STATEMENT

CIM 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 implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade (COG), taking into account extraction scenarios and processing recoveries. To meet this requirement, the QP considers for the purpose of this exercise that the Project is amenable for underground mining.

Mineral resources are considered to have potential for economic extraction for the mineral resource based on a single Ni COG of 0.35%. To determine the potential for economic extraction, the following key assumptions of costing and a metallurgical recovery were used. The Aguablanca Mineral Resource estimate is summarized in Table 14-6.

Based on the review of consensus market forecasts, the following metal prices, recovery assumptions, mining and processing costs were considered for the COG;

- Nickel – US$7.30/lb
- Copper – US$3.50/lb
- Nickel Recovery – 82.8%
- Copper Recovery – 93.6%
- Mining, Processing, G&A – US$62.00/t

Based on the analysis above a nickel equivalent grade (NiEq) has been determined on a block-by-block basis. NiEq considers both nickel and copper with no other metals and is based on the assumed recovery and metal prices. The Qualified Person for the mineral resource estimate is Scott Wilson AIPG CPG 10965.

The QP is not aware of any additional environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other factors that could affect the mineral resource estimate.

Table 14-6: Aguablanca Mineral Resource Estimate. Effective Date March 24, 2024

Resource Category	Cutoff (Ni%)	Tonnage (K tonnes)	Grade							Contained Metal						
			Ni (%)	Cu (%)	Co (%)	Pd (ppm)	Pt (ppm)	Au (ppm)	NiEq (%)	Ni (Klbs)	Cu (Klbs)	Co (Klbs)	Pd (Oz)	Pt (Oz)	Au (Oz)	NiEq (Klb)
Measured	0.35%	4,048	0.66	0.60	0.02	0.29	0.34	0.17	0.95	58,836	53,512	1,473	38,033	43,919	21,954	84,493
Indicated	0.35%	1,273	0.64	0.52	0.02	0.27	0.31	0.14	0.89	17,986	14,462	503	11,060	12,492	5,760	24,919
Measured + Indicated		**5,321**	**0.65**	**0.58**	**0.02**	**0.29**	**0.33**	**0.16**	**0.93**	**76,822**	**67,974**	**1,976**	**49,094**	**56,411**	**27,715**	**109,412**
Inferred	0.35%	4	0.67	0.61	0.02	0.31	0.37	0.17	0.96	66	60	2	45	54	24	95

1. Scott Wilson, CPG, President of RDA is responsible for this Mineral Resource estimate and is an "independent Qualified Person" as such term is defined by NI 43-101.
2. Reasonable prospects of eventual economic extraction were assessed by enclosing the mineralized material in the block model estimate in a 3D wireframe shape that was constructed based upon geological interpretations as well as adherence to a minimum mining unit with geometry appropriate for underground mining.
3. The cutoff grade of 0.35% Ni considered mining costs of:
    a. Metal selling prices Ni at US$7.30/lb and Cu selling prices of US$3.50/lb,
    b. Recoveries of Ni 82.8% and Cu 93.6%, and
    c. Costs including mining, processing, general and administrative (G&A), and off-site realization (TCRC).
4. Nickel Equivalent is estimated as ((3.50/7.30) * Cu grade) + Ni Grade.
5. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
6. Mineral Resources are inclusive of Mineral Reserves.
7. Figures may not add up due to rounding

15 MINERAL RESERVES

Mineral Reserve Estimates for the Project were determined by the aid of stope optimization software. Additionally, mineral reserves are constrained to an underground mining extraction sequence that has been developed with 3Dminnig software. The mine plan in the PFS is based on Measured and Indicated mineralization only, as summarized in Table 15-1, which have been estimated for a combination of sub-level extraction and long-hole open stoping underground mining methods. Measured resources has been converted to proven reserves. Indicated resources have been converted to probable reserves. The MRE reflected in Table 14-6 above is inclusive of the Mineral Reserve estimate, which represents approximately 89% of the tonnes in the Measured and Indicated MRE.

Table 15-1 Aguablanca Mineral Reserve Estimate Effective Date March 24, 2024

Reserve Category	Cutoff (Ni%)	Tonnage (K tonnes)	Grade							Contained Metal						
			Ni (%)	Cu (%)	Co (%)	Pd (ppm)	Pt (ppm)	Au (ppm)	NiEq (%)	Ni (Klbs)	Cu (Klbs)	Co (Klbs)	Pd (Oz)	Pt (Oz)	Au (Oz)	NiEq (Klb)
Proven	0.35%	3,650	0.67	0.61	0.02	0.29	0.34	0.17	0.97	54,051	49,281	1,343	34,454	39,798	19,835	77,678
Probable	0.35%	1,062	0.67	0.53	0.02	0.28	0.31	0.14	0.92	15,582	12,452	429	9,419	10,578	4,875	21,553
Proven + Probable		**4,713**	**0.67**	**0.59**	**0.02**	**0.29**	**0.33**	**0.16**	**0.96**	**69,633**	**61,733**	**1,772**	**43,874**	**50,375**	**24,709**	**99,231**

Notes:

1. CIM Definition Standards were followed for Mineral Reserves
2. Mineral Reserves are not additive to Mineral Resources
3. Mineral Reserves are based on the March 24, 2024 MRE
4. Totals may not add up due to rounding
5. Mineral Reserves are reported using US$7.30/lb Ni, US$3.50/lb Cu, US$12/lb Co, US$2,000/oz Au, US$900/oz Pt and US$1,200/oz Pd
6. The cutoff grade of 0.35% Ni considered mining costs of:
   a. Metal selling prices Ni at US$7.30/lb and Cu selling prices of US$3.50/lb, and
   b. Recoveries of Ni 82.8% and Cu 93.6%
   c. Costs including mining, processing, general and administrative (G&A), and off-site realization (TCRC).
7. Mineral Reserves are constrained to a mine design
8. Units are metric tonnes, metric grams, troy ounces and imperial pounds. Contained metal are estimates of in situ material and do not account for dilution or processing losses.

15.1 CUTOFF GRADE DETERMINATION

A preliminary Cut-off Grade (COG) was estimated based on the criteria of geometry of the mineralization including total mass, horizontal/vertical extents, historical geotechnical performance, and existing infrastructure. The results of the previously stated criteria validates that the mining extraction is amenable to a bulk mining scenario. Long Hole Open Stoping (LHOS) and a modified Sublevel are considered the primary methods for extraction. Based on the industry benchmarks for mining and processing costs, commodity pricing, external dilution, and anticipated mill recoveries a Cut-off Grade (COG) was calculated. A Zero-Base estimate was completed to determine mining unit costs for bulk extraction and compared against the industry benchmarks. The cost estimate methodologies were compared and determined compatible. To determine the economic COG utilizing the two primary elements, Nickel and Copper, the following methodology was applied:

$$M\&P\ cost = Nieq\mathrm{P} * (1 - \mathrm{D}) * \mathrm{R} * \mathrm{G}$$

Where:

M&P cost = Mining & Processing Cost, $/t
$Nieq\mathrm{P}$ *= Nickel Equivalent Price, $/lb*
D = Unplanned Dilution (Overbreak), %
R = Anticipated Mill Recovery, %
G = Cut-off Grade, %

The Nickel Equivalent Price utilized the nickel and copper content in the Mineral Resource Estimate and commodity pricing:

$$Nieq\mathrm{P} = (\mathrm{NiG} * \#\ per\ tonne * NiP) + (CuG * \#\ per\ tonne * CuP)$$

Where:

NiG = Mineral Resource Nickel Grade, %
NiP = Projected Nickel Price, $USD/#
CuG = Mineral Resource Copper Grade, %
CuP = Projected Copper Price, $USD/#

Solving the equation for NieqP:

$$Nieq\mathrm{P} = (0.65\% * 2204 * \$7.30) + (0.58\% * 2204 * \$3.50)$$

$$Nieq\mathrm{P} = (\$104.58 + \$44.74) = \$149.32$$

Substituting the values to determine the economic COG G:

$$G = M\&P\ cost/((Nieq\mathrm{P} * (1 - \mathrm{D}) * \mathrm{R}))$$

$$G = \frac{\$48.00}{(\$149.32 * (1 - 2\%) * 90\%)} = 0.35\%$$

15.2 MINE SHAPES AND DEVELOPMENT

A grade shell was constructed by cutting the economic COG against the block model identifying all mineralization above the 0.35% Nickel equivalent grade. In addition, this shell confirmed the continuity and massive nature of the mineralization to feasibly apply bulk mining methods. Based on the open pit limits and geotechnical wall integrity, it was determined that an Up-hole Sublevel Extraction (USE) method for the top 100 meters of the mineralization was applicable. Mining transitions to utilizing LHOS with cemented rock fill below the USE area. The Vulcan™ Optimizer was selected to run the stope shapes, including stope development based on nominal dimensions, geotechnical limits, and economic COG. The results of the Optimizer run provided the minable reserve to which a schedule and financial model was completed.

Waste development is designed to maximize vertical advance providing significant stope access and minimize schedule/cost impacts. However, some Optimized stopes were eliminated due to the uneconomic waste development required to effectively access them.

Mineral Reserve estimates could be materially affected by highwall instability issues related to the USE open mining stope. When complete, the USE stoping will expose an unsupported 100-meter vertical wall below the existing highwall. Inherent geotechnical stresses will be imposed which may cause an unstable operating environment due to spalling of material into the void. Efforts to mitigate the risk may include changing from the UDS mining method to 100% LHOS mining of the Project.

16 MINING METHODS

16.1 MINERALIZED CONFIGURATION

The Aguablanca underground reserve is the down dip extension of a previous Open Pit Operation. The Resource Block Model completed identified six (6) distinct mining zones. Five steeply dipping structures identified as the South Zone, North Zone, Satellite Zone, 117 Zone, and Other Satellites. In addition, one flat lying mineralized structures have been identified as the 120 Zone. Mine Stope Optimization (MSO) has been applied to the block model to identify economic minable stope blocks. As can be seen in Figure 16-1, Minable Reserves occurs occur primarily in the two near vertical structures (South Zone and North Zone), and one flat lying structure (120 Zone) below the 181 mRL pit elevation; as can be seen in Figure 16-1. Utilizing the existing underground excavation, a mine plan has been developed to extract these stope blocks.

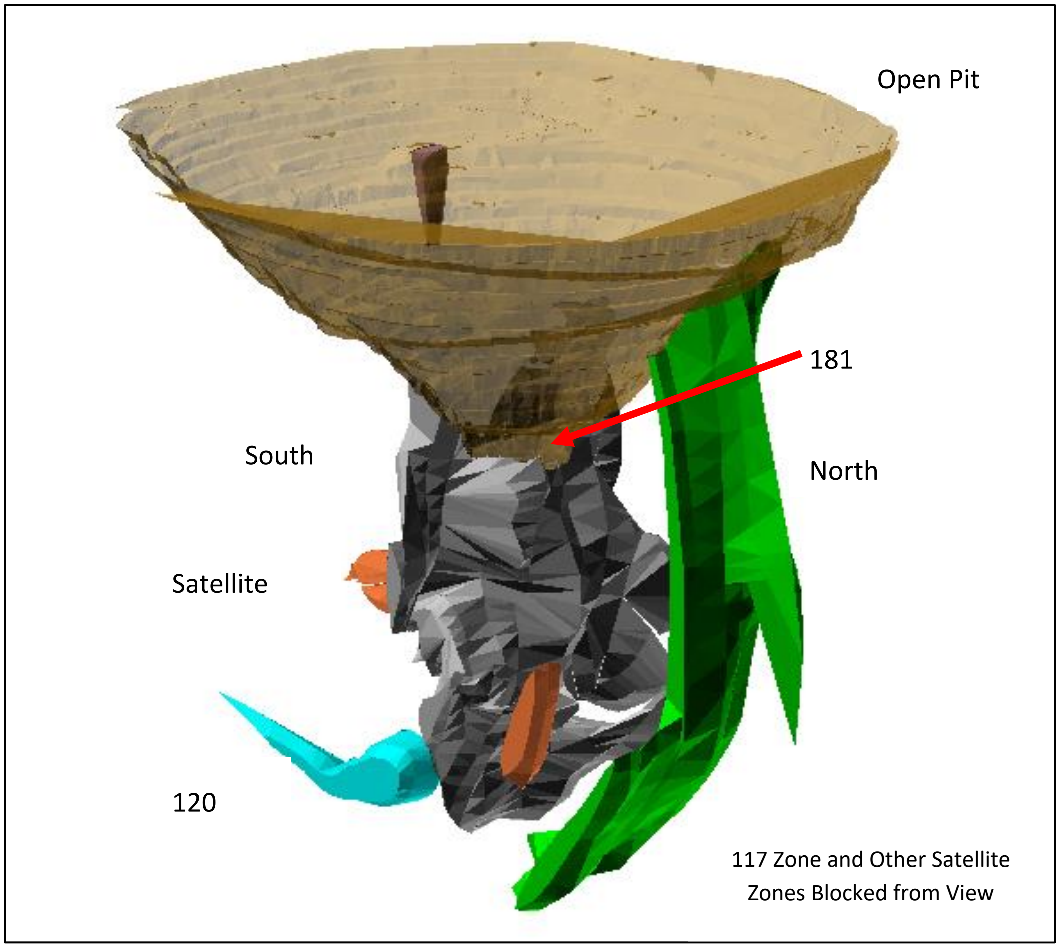


Figure 16-1 Aguablanca Mineralization Depiction

16.2 MINING METHOD

Open pit mining to the 181mRL was completed in April 2015, the final configuration of the open pit is shown in Figure 16-2. This PFS focuses on the continuation of underground mining from 181mRL elevation to the 85mRL elevation utilizing Up-hole Sublevel Extraction (USE) and Long Hole Open Stoping (LHOS) mining techniques. The mine design includes utilizing USE from the 181mRL to the 80mRL on 25-meter lifts for a total of four levels. Backfill will not be employed until the entire 100-meters of ore is exploited, following complete extraction a surface haul from the existing north dump to the bottom of the pit will commence to backfill to the 181mRL elevation. Long Hole Open Stoping (LHOS) levels are designed from the 80mRL to the 85mRL utilizing a primary/secondary sequencing employing cemented rock fill (CRF) to create a backfill cap to allow for underhand mining to continue. When applicable based on CRF placement, the next LHOS level below will begin mining creating an inverted pyramid advancing down dip. Figure 16-3 depict a cross section of the designed mine advancement and nominal stope dimensions.

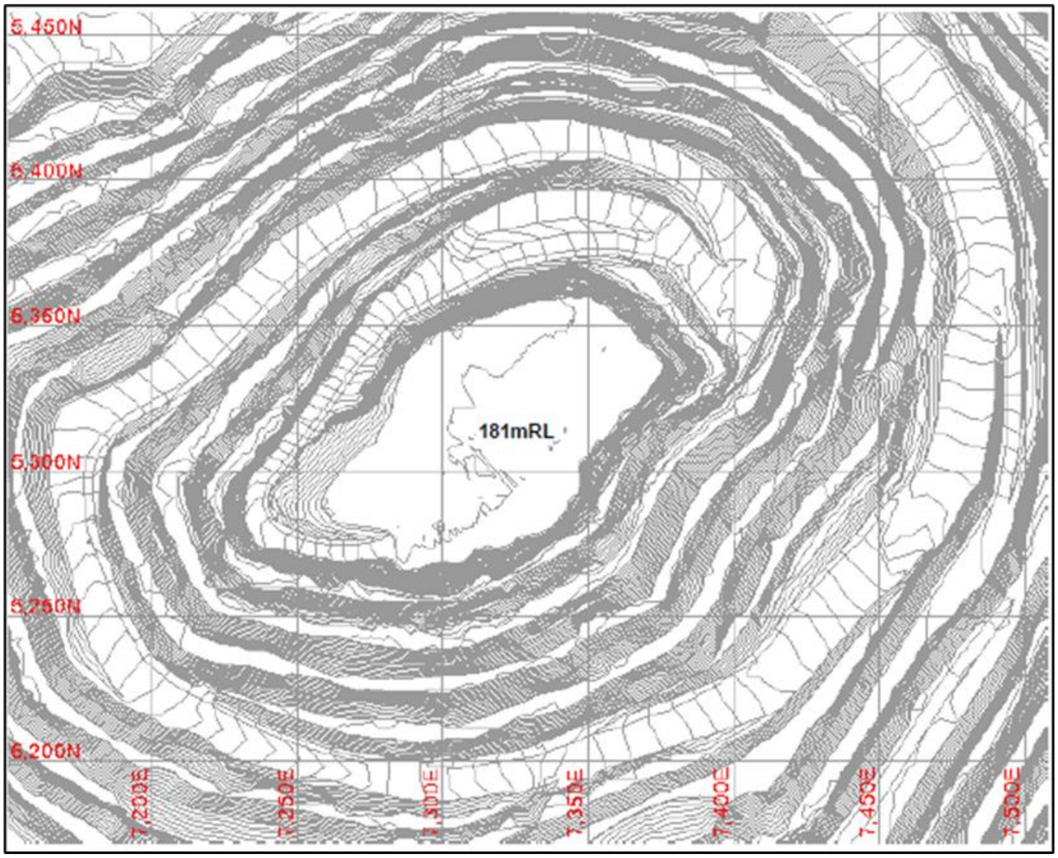


Figure 16-2 Final Open Pit Configuration

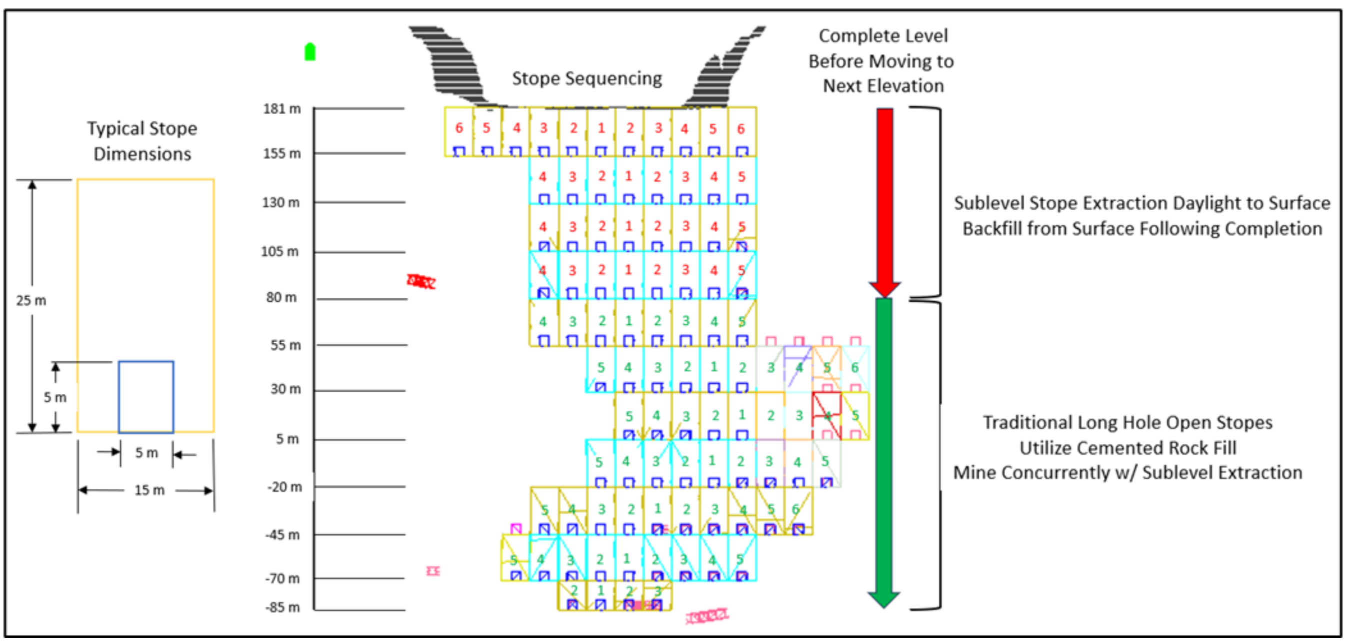


Figure 16-3 Stope Sequencing and Nominal Dimensions

Prior to mine closure in 2015, approximately 3,400 meters of development drifting and vertical ventilation circuits had been completed, displayed in Figure 16-4. The current as-built excavations are scheduled to be rehabilitated prior to mining commencing. One Sublevel Stope from the 160 mRL elevation to the bottom of the pit was mined, the slot raise is displayed in Figure 16-5.

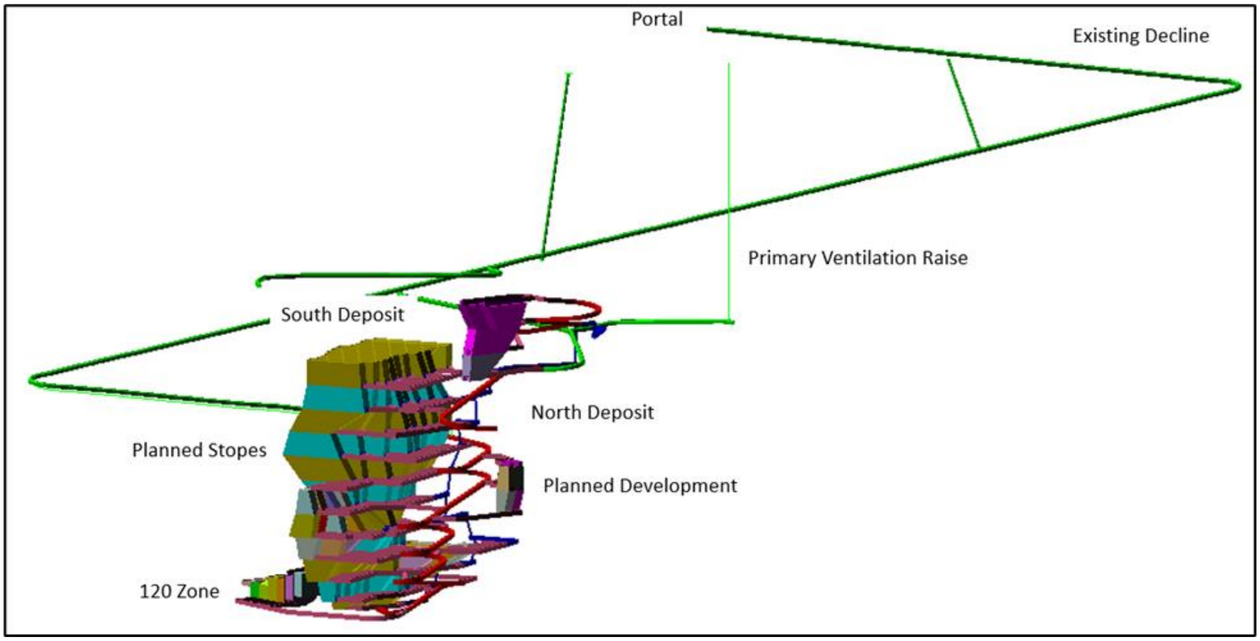


Figure 16-4 Designed Development & Existing Underground Development

Figure 16-5 160-181mRL Slot Raise

16.2.1 UPHOLE SUBLEVEL EXTRACTION (USE)

The mine plan utilizes USE from the 181mRL to the 80mRL on 25-meter sublevel intervals. Each level is commenced with a footwall drive (FWD) located on the North-West edge of the orebody. From the FWD a series of parallel drives are developed to the South-East extent of the orebody on 15-meter centers as depicted in Figure 16-6. A slot raise is developed on the first stope of the level to allow for blast expansion. To maintain a safe environment the plan calls for mining in a retreating sequence from the south-east extent to the north-west extent. When the stope is complete, the void is left unfilled and mining transitions to the adjacent stope. The sequence is repeated until the level is exhausted; the stope level below then becomes the primary stoping area as seen in Figure 16-7.

The ore is mucked from the stope utilizing remote controlled Load Haul Dumps (LHD) and transported from the production levels to the portal by rubber-tired haul trucks to the surface.

USE is a highly productive mining method allowing for 1,000 tonnes/day when several stopes are in production.

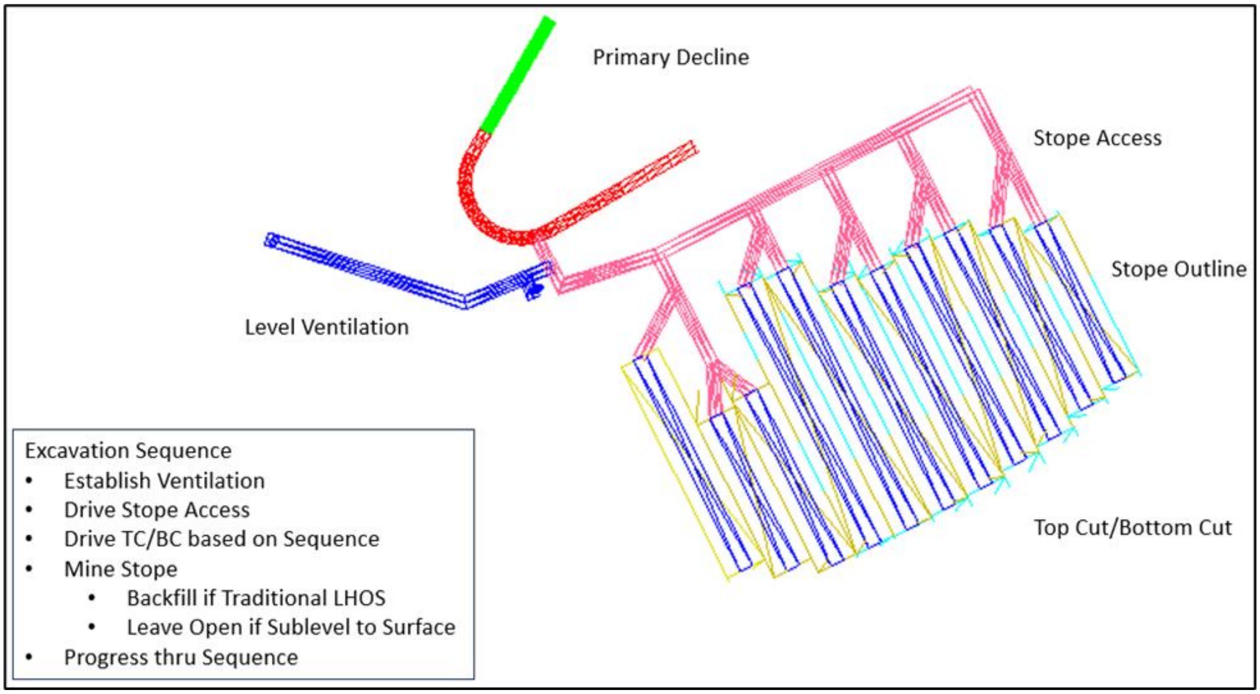


Figure 16-6 Typical Stope Access Level

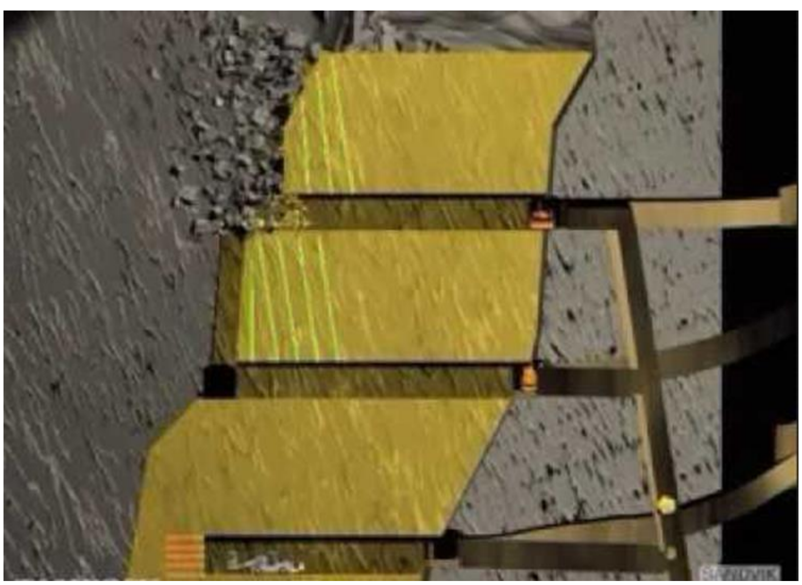


Figure 16-7 Section View of Up Hole Stope Extraction

16.2.2 LONG HOLE OPEN STOPING (LHOS)

The Aguablanca mine plan employs LHOS from the 80mRL to the 85mRL levels. The method includes developing 5m x 5m top and bottom cuts for drilling and mucking access. The nominal dimension of the stopes is 15m x Length of the ore x 25m (WxLxH). A series of stopes are arranged to allow for primary and secondary sequencing utilizing Cemented Rock Fill. LHOS drilling is a series of vertical or inclined holes that are drilled from the top of the stope to the bottom utilizing a long hole drill. The holes are drilled in a pattern of rings with multiple holes per ring creating the spacing. These rings are spaced at regular intervals across the stope establishing the burden, as seen in Figure 16-8. Blasting is performed in a subset of rings; broken ore is removed using a remote Load Haul Dump (LHD) to load the trucks. LHOS allows for greater flexibility in the mining process, as the drill holes can be adjusted as the mining progresses allows for more efficient extraction of the ore.

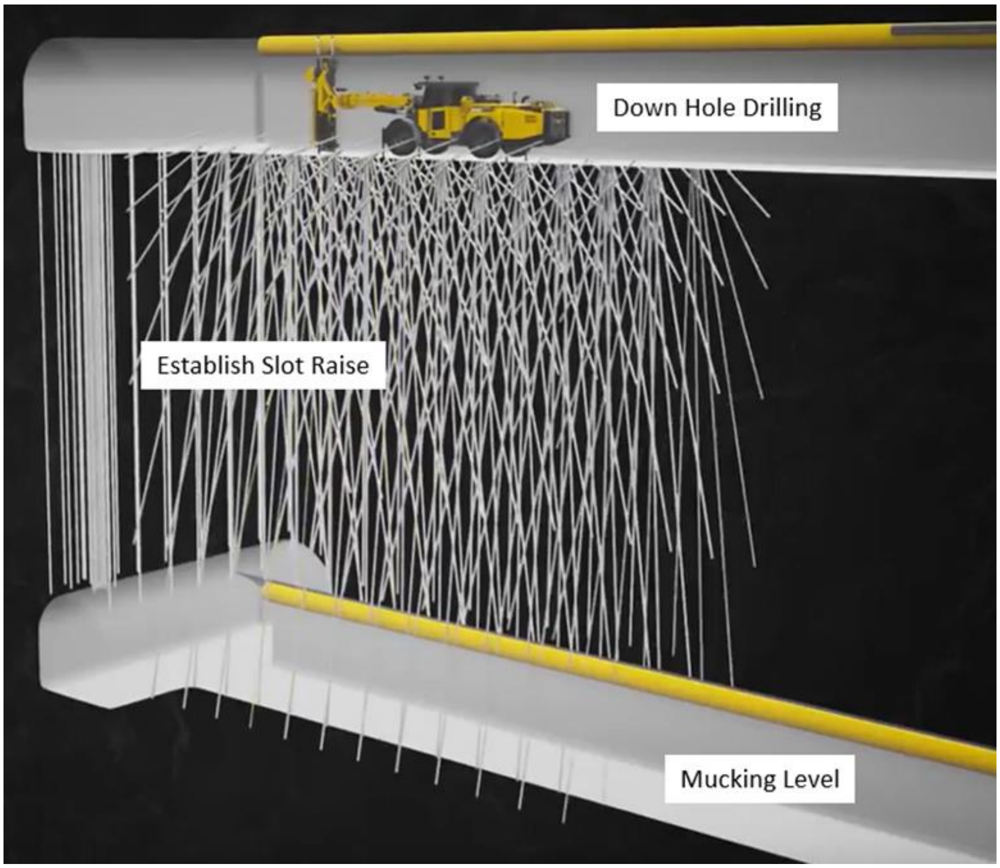


Figure 16-8 Cross Section View of Longhole Open Stoping

LHOS requires less development work than other underground mining methods, such as cut-and-fill or block caving. LHOS method generates less waste rock, which results in less environmental impact and may help to reduce the overall cost of waste rock disposal. Additionally, by requiring less development, it helps to minimize the overall environmental footprint of a mining operation.

The LHOS mine production schedule is based on 1,000 tonnes/day with 3 stopes available. The development of the Top/Bottom cuts are scheduled at 3 meters/day or 240 tonnes/day.

16.3 GEOTECHNICAL FINDINGS AND ASSUMPTIONS

Work completed by Lundin stated that geotechnical conditions during the transition from Open Pit mining to Underground in 2015 produced the following observations and predictions. Slope movements continue; however, velocities were steady when examined during the May 2015 review visit. As noted above a break-through slot raise had been developed between the pit floor and the 160 level. Drilling for the raise development had been carried out from the pit floor, and extension of the slot was underway. While the walls of the stope are essentially stable, potential exists for the rockfalls from the walls due to the influence of inherently complex geology. In an adverse situation it is conceivable that such falls could compromise stope wall stability. Fall rocks can be ejected considerable distances after impact with the sill at an opening; therefore, it is recommended to employ remote LHD's and no personnel should be permitted beyond the brow.

During the 160 Level Raise extension, a step increase of 2-3 mm of movement occurred in the lower south-eastern wall immediately following the blast, and continued to increase to 8mm beyond the pre-blast positions over a 12-hour period. Displacements regressed over the ensuing 12 hours almost back to pre-blast velocities. The displacement curves suggest that blast-induced wall rebound may have occurred, implying the blast imposes an outward force onto the wall, energy dissipates and the wall rebounds; however, there is nothing to stop the wall moving past its pre-blast position.

Drill and blast for underground USE stope mining will develop stope walls that will not be able to be scaled. In addition, there will be no opportunity for installation of reinforcement into or application of surface support to the outer USE walls. The proposal to develop dedicated stope reinforcement drives outside the orebody certainly has technical merit; however, this is impractical. The cost of drive development and the impact on overall mine development advance are exorbitant demands to place on economics. Increasing resources to allow overall mine development to proceed unhindered would also add further to mine costs and increase demands on the ventilation system.

A potential solution exists to reinforce the USE walls from planned drives on each sub-level. In this approach moderately long cable bolts would be installed into the USE outer walls in closely spaced arrays. Figure 16-9demonstrates the concept.

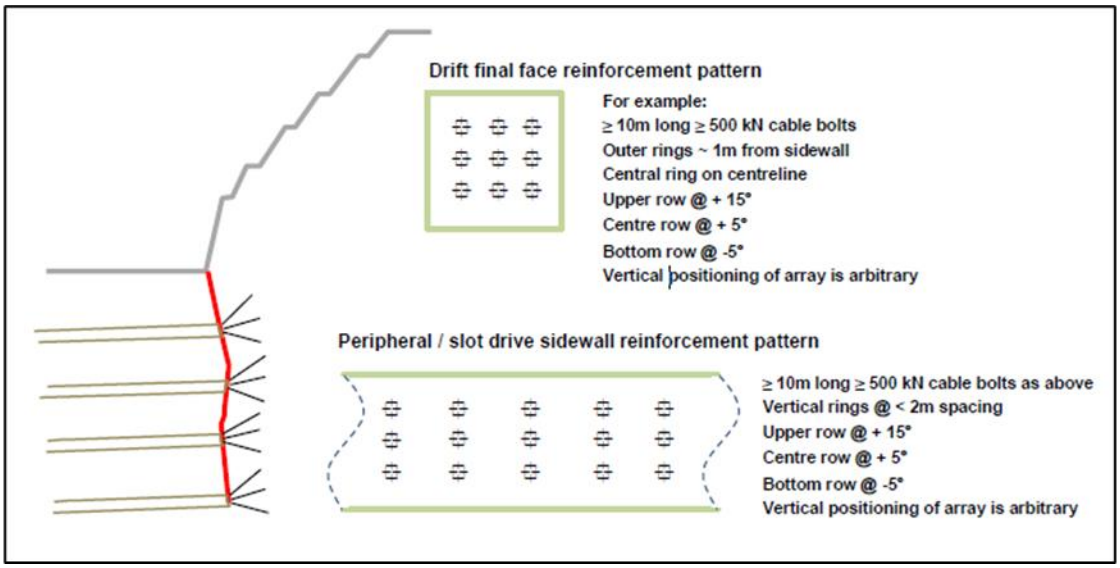


Figure 16-9 USE Panel Reinforcement Concept

Structural conditions are such that despite the above recommended reinforcement, large blocks from the walls of the USE panel could move into the void. Loss of some individual blocks may not be detrimental to wall stability, but it is possible that movement of other blocks could initiate progressive deterioration. Evidence of block failure would be evident via underground observation of ore flow and from displacement data from radar and monitoring.

The mine plan is flexible. If geotechnical stabilities are compromised the production profile would transition from USE to straight LHOS which provides additional support from the CRF. A portion of the ore may be left depending on the severity of the failure.

In the Author's opinion, the described rock stability of the excavated drift is sufficient as a preliminary design basis, but confirming rock mechanics and geotechnical assumptions should be high priority to verify the mining approach and details of the PFS layout and design.

16.3.1 GROUND SUPPORT ASSUMPTIONS

The rock stability in the existing Aguablanca excavations has been reported as excellent and, it is assumed to be representative of the conditions that will occur with the advancement of development and production. If ground conditions deteriorate artificial ground support consisting of 2.6 m friction rock stabilizers (rock bolts) on a 1.2 m pattern in the back, spot bolting with 3.2 m Dywidags, and welded wire mesh with steel straps is recommended on the specified portion of the development and stope drifts. In addition, on rare occasions it may be necessary to employ shotcrete to support the compromised areas.

16.4 BACKFILL

Due to the two mining methods, two distinct backfill media will be employed. Following the completion of the USE stoping horizons, a surface backhaul will commence from the east waste dump to the bottom of the pit. The unconsolidated backfill will be dumped into the void until a pit bottom elevation of 181mRL is achieved.

Cemented Rock Fill will be utilized with the LHOS mining method. Mix designs will vary depending on aggregate water content application requirements of the CRF. A typical design uses a 15 cm minus aggregate, sourced and sized from the north waste dump with 7% cement content; fly ash is not initially considered.

One CRF mix plant is currently envisioned to meet Aguablanca CRF requirements. The total integration of the aggregate/cement/supplies-in and ore-out haulage system has not been completed as the profiles are being developed. Costs and tonnages are included although final alignment has not been determined.

The envisioned CRF plant design uses weigh hoppers to premeasure components. This allows the weigh hoppers to be filling as a mix is underway, and with rapid recharging of the mixer which reduces batch times. A high shear mixer is envisioned to premix cement and water which can reduce cement consumption for a given mix strength. Weigh hoppers also tend to be more precise and robust that than weigh-belt and flow meter systems, and result in better batch consistency.

The CRF is scheduled to be transported to the desired location via rubber-tired haul trucks. A jammer is required to fill the top of the stope to ensure the CRF is tight against the back.

16.5 MINING LAYOUT

The mining layout is illustrated in Figure 16-10 and Figure 16-11 providing isometric views of the mining and development layout of the mining reserves. NSR values determined economic parameters and the block model dictated the nominal dimensions and orientation of the excavations. These parameters were applied to the MSO which produced the individual mining blocks containing volume and grade.

The development layout in the Vulcan model is produced by strings (lines), a primitive attribute was applied to theses strings producing a volume. Main development drifts, ventilation crosscuts and raises, and ore cross cuts can be observed in the illustrations.

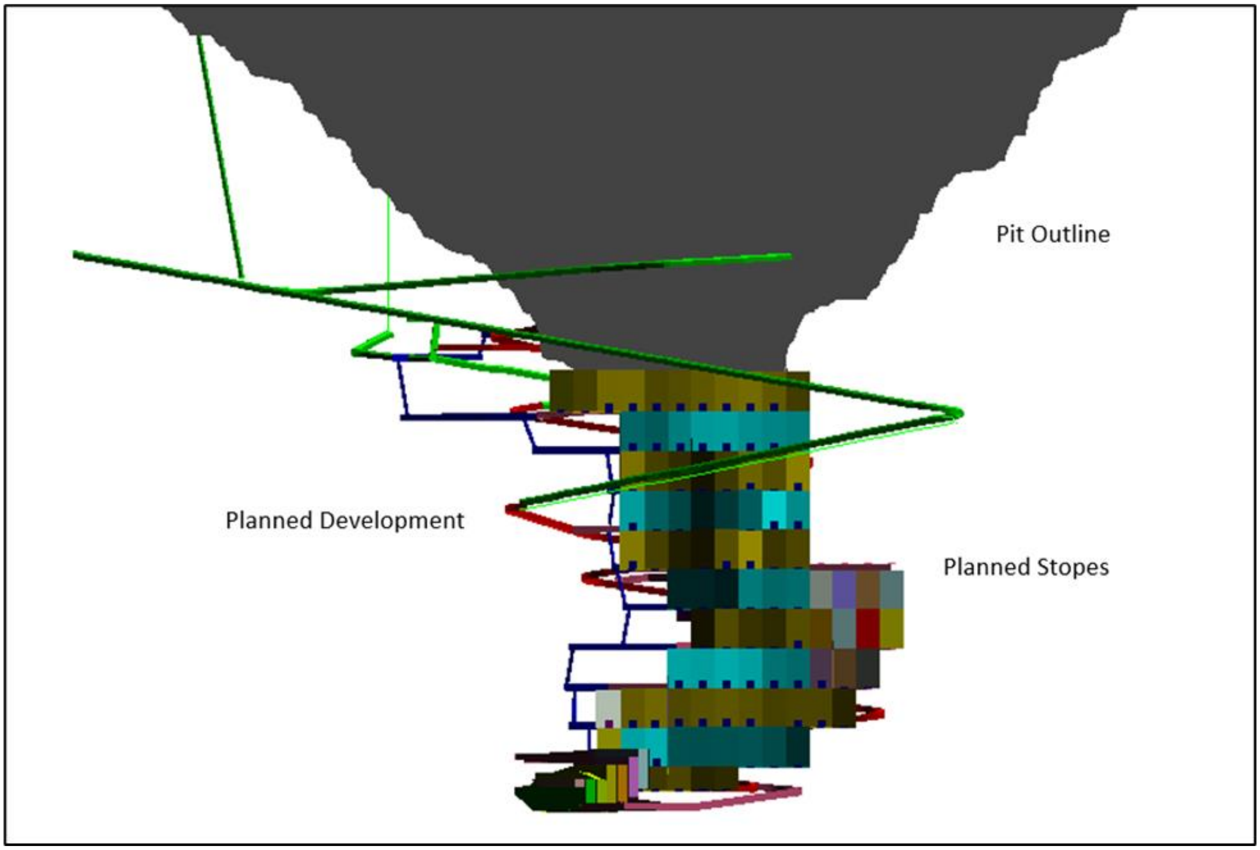


Figure 16-10 Isometric view looking northwest

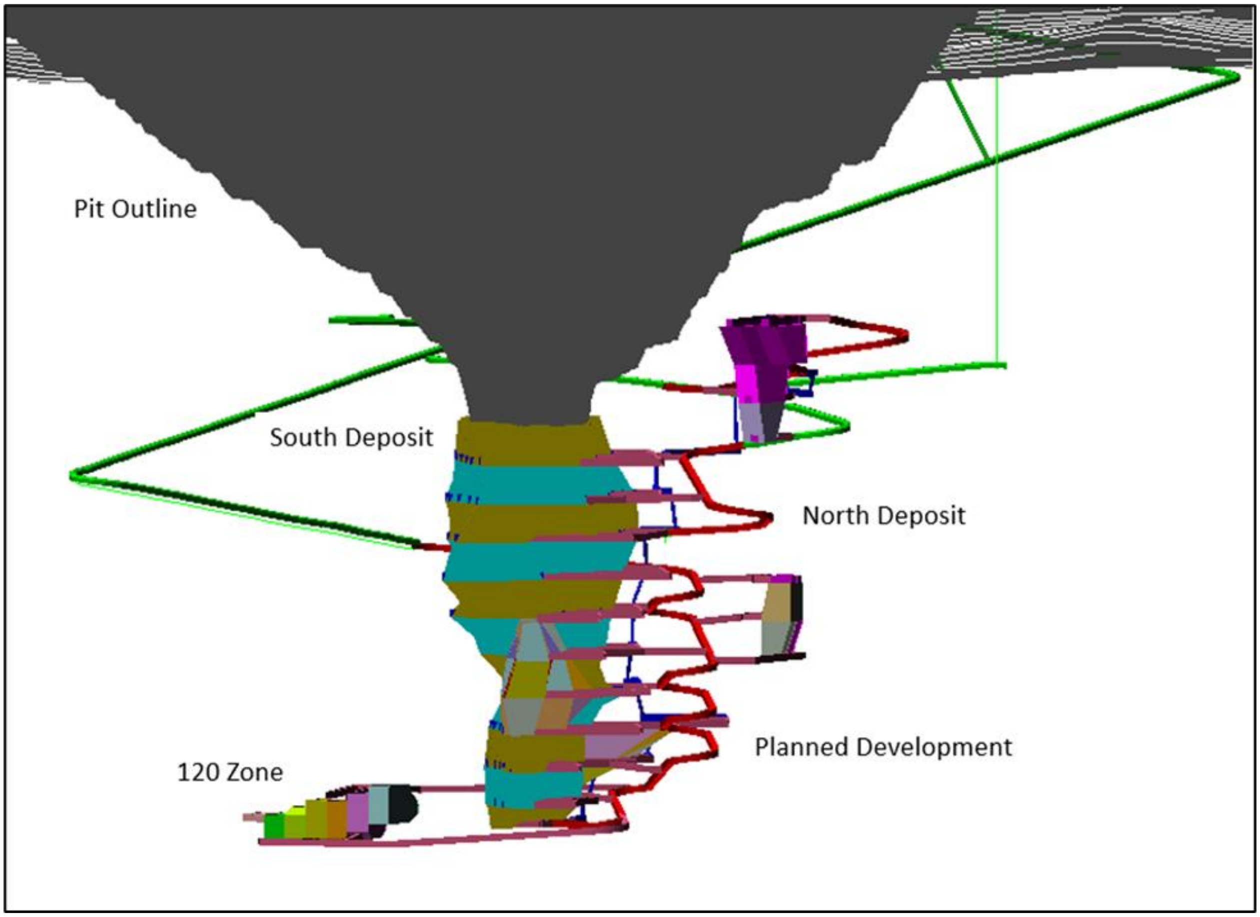


Figure 16-11 Isometric view looking east

The mine is accessed from surface via the existing portal and decline. Ventilation air would be drawn into the mine from the primary surface intake fan to a series of internal ventilation raises. The ventilation raises will be excavated vertically along the stope blocks and would be connected between the main ventilation levels (intake and exhaust). Ventilation raises (2.5 m x 2.5 m) would be constructed at each location with a short ventilation access drift connecting to the stope. The ventilation raises will provide conduits for electrical power, compressed air, make up water into the stope blocks, and a secondary egress ladder network would also be constructed. Exhaust is planned to travel through the primary decline exiting the mine at the portal.

Rock mechanics analysis of the layout should be conducted to project rock stability impacts of the planned geometry. No analysis of ramp locations in the hanging wall has been conducted, and interactions between historical workings need to be simulated. Stand-off distances of ramps and ventilation raises need to be evaluated to assure stability of the raise and infrastructure as the mining proceeds vertically.

16.6 MINING PRODUCTION

The MSO inventory of stope shapes and development strings were scheduled utilizing Microsoft Excel. The resulting schedule produced the development and production schedule including the metal profile. An external dilution factor of 2% at zero grade is included into the production schedule to account for over-break.

Mine scheduling was controlled by the development primary ramps and ventilation circuits which provide fresh air to the working areas. For individual stoping blocks, a ramp would be developed between the lower and upper extents of the contiguous stope areas, which would be connected by a vertical raise. The ventilation raises would also provide secondary egress from the stope working area and provide the pathways for electrical, compressed air, and make-up water services required for mining. Sustaining development to the various stope production areas was scheduled to maintain a sufficient buffer of stopes ahead of required ore production.

Development and production schedules were driven by the peak mill throughput of 877,200 tonnes per year. The mine operating schedule is based on 24 hour, 7-day per week, 365 days per year.

The development and production rates per underground operating day used for the scheduling were:

- Ramps and access drifts: 3 meters/day (single heading); maximum advance 9 meters/day
- USE Production Target: 1,000 tonnes/day, minimum stopes 2
- LHOS Production Target: 1,000 tonnes/day, minimum stopes 3

The development for the mine initially focuses on creating the access and ventilation circuit for the first USE stoping level. Development priorities then focus on advancing to the 80mRL and 55mRL levels to drive the top and bottom cuts for the LHOS section.

The production profile is limited by the permit requirement of only producing 877,200 tonnes/annum. Therefore, initial ore production is targeted at 500 tonnes/day gradually ramping to 2,356 tonnes/day. The physical units in the underground production schedule are listed in Table 16-1 and Table 16-2, for the detailed mine output, the Production and Development Schedule by year, respectively.

Table 16-1 Aguablanca production schedule by year

Denarius Metals Corporation Aguablanca	LOM Year 1-8	2024	2025	2026	2027	2028	2029	2030	2031
Diluted Minable Resource									
Preliminary Minable Resource									
Scheduled Sublevel Development									
Ore, tonnes	269,880	-	44,789	51,408	45,450	50,530	47,873	29,830	-
Ni, %	0.642	-	0.668	0.666	0.612	0.673	0.639	0.557	-
Cu, %	0.592	-	0.542	0.553	0.565	0.648	0.654	0.584	-
Co, %	0.016	-	0.017	0.017	0.015	0.016	0.016	0.015	-
Pd, g/t	0.283	-	0.242	0.256	0.302	0.315	0.302	0.278	-
Pt, g/t	0.327	-	0.271	0.290	0.334	0.374	0.359	0.338	-
Au, g/t	0.164	-	0.124	0.141	0.172	0.186	0.19	0.17	-
Scheduled Panel Production									
Ore, tonnes	4,537,302	-	557,475	825,793	831,750	826,669	829,327	666,288	-
Ni, %	0.663	-	0.713	0.660	0.663	0.652	0.655	0.609	-
Cu, %	0.582	-	0.573	0.557	0.545	0.617	0.635	0.551	-
Co, %	0.017	-	0.018	0.017	0.017	0.016	0.016	0.017	-
Pd, g/t	0.282	-	0.240	0.273	0.281	0.309	0.305	0.277	-
Pt, g/t	0.322	-	0.271	0.308	0.315	0.354	0.360	0.328	-
Au, g/t	0.157	-	0.127	0.144	0.151	0.180	0.19	0.16	-
Scheduled Diluted Mine Production									
Ore, tonnes	4,807,182	-	602,264	877,200	877,200	877,199	877,201	696,118	-
Ni, %	0.656	-	0.709	0.660	0.660	0.653	0.654	0.607	-
Cu, %	0.582	-	0.571	0.556	0.546	0.619	0.636	0.553	-
Co, %	0.017	-	0.018	0.017	0.017	0.016	0.016	0.017	-
Pd, g/t	0.283	-	0.240	0.272	0.282	0.310	0.304	0.277	-
Pt, g/t	0.325	-	0.271	0.307	0.316	0.355	0.360	0.328	-
Au, g/t	0.160	-	0.127	0.144	0.152	0.180	0.19	0.16	-

Table 16-2 Aguablanca Underground Capital and Expensed Development by Year

Denarius Metals Corporation Aguablanca			**LOM Year 1-8**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**
Waste Development	**Waste Density**										
Capital Development	W	x									
Horizontal											
Primary Declines, m	4.5	x	2,140	80	480	400	400	400	380	-	-
Primary Declines, tonnes			*148,071*	*5,535*	*33,212*	*27,677*	*27,677*	*27,677*	*26,293*	-	-
Primary Cross Cuts, m	4.5	x	6,364	50	*1,800*	1,200	1,200	1,200	600	314	-
Victor Cross Cuts, tonnes			*440,338*	*3,460*	*124,546*	*83,030*	*83,030*	*83,030*	*41,515*	*21,726*	-
Primary Ventilation Cross Cuts, m	4.5	x	687	60	181	178	92	115	61	-	-
Primary Ventilation Cross Cuts, tonnes			*47,535*	*4,152*	*12,524*	*12,316*	*6,366*	*7,957*	*4,221*	-	-
Underground Work Stations (shop), m	7.0	x	-	-	-	-	-	-	-	-	-
Underground Work Stations (shop), tonnes			-	-	-	-	-	-	-	-	-
Miscelaneous (muck bays, sumps, vent X-cuts, power bays, etc), m			171	6	38	32	32	32	30	-	-
Miscelaneous (muck bays, sumps, vent X-cuts, power bays, etc), tonnes			11,846	443	2,657	2,214	2,214	2,214	2,103	-	-
Primary Drift Rehabilitation,m			3,400	2,000	1,400	-	-	-	-	-	-
Horizontal Development, m			9,362	196	2,499	1,810	1,724	1,747	1,071	314	-
Horizontal Development, tonnes			*647,789*	*13,589*	*172,938*	*125,238*	*119,287*	*120,878*	*74,132*	*21,726*	-
Exploration Development, m	4.5	x	-	-	-	-	-	-	-	-	-
Exploration Development Waste, tonnes			-	-	-	-	-	-	-	-	-
Exploration Drilling, m			*10,000*		*2,000*	*4,000*	*1,000*	*1,500*	*1,000*	*500*	-
Delineation Drilling, m			*7,500*	-	2,400	1,600	*1,200*	1,200	*600*	*500*	-
Total Capital Horizontal Advance, m			9,362	196	2,499	1,810	1,724	1,747	1,071	314	-
Total Capital Horizontal Waste, tonnes			*647,789*	*13,589*	*172,938*	*125,238*	*119,287*	*120,878*	*74,132*	*21,726*	-
Expensed Development											
Horizontal											
Stope Cross Cuts, m	4.5	x	-	-	-	-	-	-	-	-	-
Stope Cross Cuts, tonnes			-	-	-	-	-	-	-	-	-
Cut and Fill Attack Ramps, m	4.5	x	-	-	-	-	-	-	-	-	-
Cut and Fill Attack Ramps, tonnes			-	-	-	-	-	-	-	-	-
Total Expensed Horizontal Advance, m			-	-	-	-	-	-	-	-	-
Total Expensed Horizontal Waste, tonnes			-	-	-	-	-	-	-	-	-
Vertical											
Drop Raise, m	2.5	diameter	241	-	103	69	30	39	-	-	-
Drop Raise Waste, tonnes			3,786	-	1,618	1,084	471	613	-	-	-
Total Expensed Vertical Advance, m			241	-	103	69	30	39	-	-	-
Total Expensed Vertical Waste, tonnes			3,786	-	1,618	1,084	471	613	-	-	-
Total Advance, m			9,603	196	2,602	1,879	1,754	1,786	1,071	314	-
Total Waste, tonnes			651,575	13,589	174,556	126,321	119,758	121,491	74,132	21,726	-

16.7 MINE MOBILE EQUIPMENT

Denarius is currently soliciting contractors to perform all mine activities; therefore, the current mine plan is created on a zero-based analysis. The future contractor will commit the necessary resources to meet the production profile. Therefore, underground manpower and equipment determination is not required and relies entirely on the yet-to-be-named contractor.

Aside from the surface support equipment, Aguablanca will be responsible (or third party) for all primary material haulage on the surface including from the portal to the process facility. The truck specification and quantity determination must be completed when the haulage profile is established.

16.8 UNDERGROUND MINE MANPOWER

The mine underground operating workforce is provided by the contractor. This includes mine contractor administration, operating labor, and direct maintenance. Aguablanca General and Administrative workforce are described in Section 21.2.1.

16.9 UNDERGROUND MINE VENTILATION

The Aguablanca mine layout is based on vertical and horizontal stoping of different zones which are located relatively close to the topographic or pit surface. The existing mine excavation includes a primary intake and exhaust conduits, which have been incorporated into the mine ventilation plan. The general ventilation system concept is the same for development of all the zones and relies on the intake and exhaust controlled by fans and bulkheads. The establishment of individual level ventilation circuits would follow the steps of:

1. Develop primary access, haulage ramp, resource crosscuts, and ventilation bays;
2. Construct a ventilation raise system (2.5 m x 2.5 m) between level crosscuts at a nominal setback distance from the stopes allowing direct intake air to the working faces;
3. Install fans in bulkheads on the ventilation crosscut to draw air from the surface;
4. Develop next raise on subsequent vent crosscuts and tie in with raise to pull fresh air to the next level; and
5. Fresh air reports to the working areas and exhausts out primary ramp.

As mining develops in the stope areas, several raise fan installations would be working in parallel resulting in the total ventilation intake substantially exceeding the requirements of the operating equipment and personnel. Figure 16-12 illustrates the typical ventilation concept. The ventilation raises would be cast down with the individual raise fans controlling the split of air flow, and serial flow mixing as the air moves towards the exhaust level. The series of ventilation raises also act as a secondary egress.

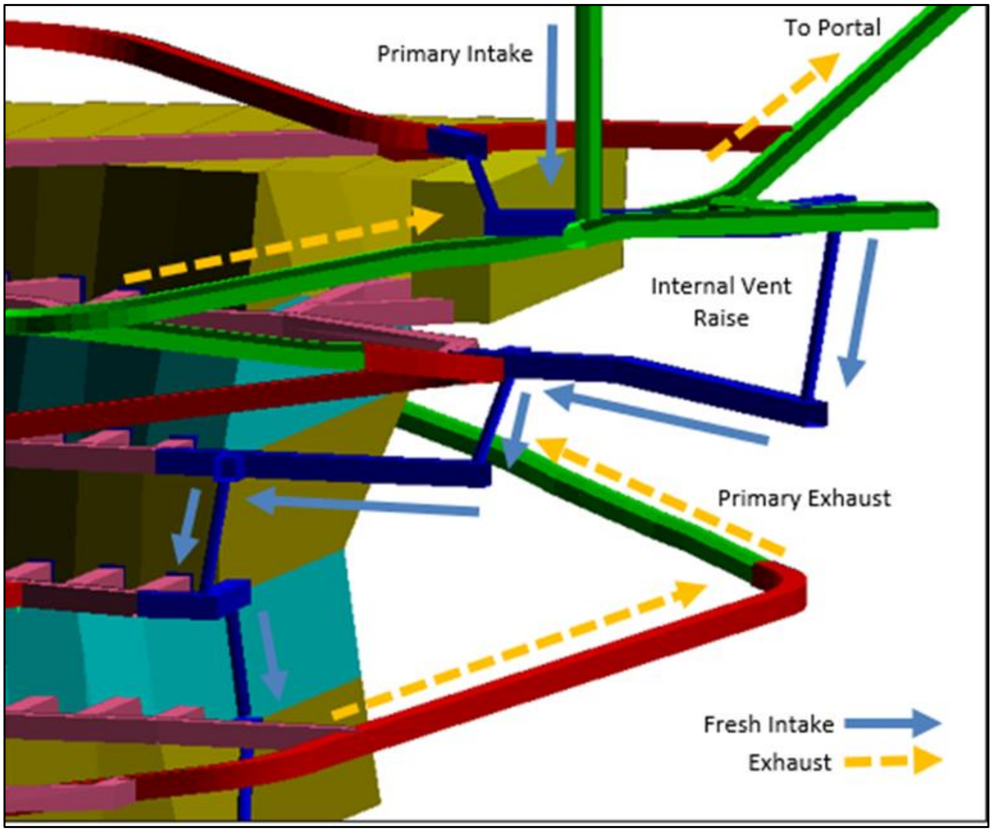


Figure 16-12 Typical Conceptual Ventilation System

16.10 UNDERGROUND INFRASTRUCTURE

The Aguablanca mine is scheduled to produce 71,667 tonnes per month at full production with direct feed to the process facility. Dry mining conditions are assumed based historic reports. However, excessive meteoric rainfall creates geotechnical and operational concerns. An extensive mine pumping system is currently in place and will be utilized when necessary.

Underground electrical power is sourced via the existing local overhead power lines distribution. The underground power cable will be installed during the excavation of the of the primary ramp to each working area that requires power Mine Load Centers (MLC) and switchgear will be strategically placed next to the working areas. As mining progresses, the power MLC will be advanced corresponding to development and production. Voltage loss shall occur with long runs; therefore, when applicable, boreholes will be drilled and power cable installed to minimize run length.

Make up water and mine consumable water will be sourced from the site water containment facility.

Infrastructure utility costs are included in the Zero-Based estimate.

17 RECOVERY METHODS

17.1 PROCESS PLANT OPERATING SCHEDULE

Although the Aguablanca process plant design called for a dry feed throughput rate of 195 tph, plant feed rates were often higher with process plant throughput often exceeding 5,000 dry tpd. However, the underground mine will produce on average only approximately 2,400 tpd. The milling schedule will be modified accordingly.

The process plant operations cycle will include seven weeks with the plant operating Monday thought Thursday. During the eighth week, the plant will operate only on Monday and Tuesday, thus allowing for two extra days during the eighth week to complete large maintenance projects. It is estimated that plant startups will take one hour while shutdowns will require two hours. Thus, the plant will operate 93 hours for seven weeks and 45 hours during the eighth week.

Since crushing plant design throughput is 450 tph, the crusher will not operate for as many hours each week as the rest of the plant. Because of this, crusher maintenance projects can be completed while the rest of the plant is in operation.

While the plant is operating, staffing will be similar to 2015 staffing including two reagent and plant employees as well as a commercial supervisor and commercial operator to be responsible for concentrate storage and loading. During periods when the plant is scheduled to be down, the plant will be staffed by one supervisor and one operator on day shifts and one supervisor and one operator on night shifts. These individuals will handle fire watch, monitor any equipment still operating, and complete cleanup projects. The sample preparation, assay, and metallurgical laboratories will be staffed with sufficient manpower to complete mine and plant sample analyses and testing.

The 2015 process plant workforce was made up of 111 employees. The process plant will operate just slightly over 50% of the time while processing underground ore and will employ a total of 66 people. Proposed underground ore processing manpower is compared to 2015 manpower in Table 17-1 and the process department organizational structure is shown in Figure 17-1.

Table 17-1 2015 vs. Proposed Manpower

Position	2015 Headcount	Proposed Headcount
Mill Manager	1	1
Mill Supervisor	1	1
Mill Crew Supervisor	4	2
Mill Operators	32	16
Reserve Operators	10	5
Reagents & Plant Work	4	2
Weekend Supervisors	2	2
Weekend Operators	2	2
Commercial Supervisor	1	1
Commercial Operator	1	1
Mechanical General Supervisor	1	1
General Supervisor Assistant	1	1
Mechanical Supervisor	2	2
Mill Mechanics	22	12
Electrical Supervisor	1	1
Electricians	8	4
Lab Supervisor	1	1
Sample Prep Supervisor	1	1
Technicians	5	3
Metallurgical Technicians	2	1
Assayer Supervisor	3	3
Technicians	6	3
Totals	**111**	**66**

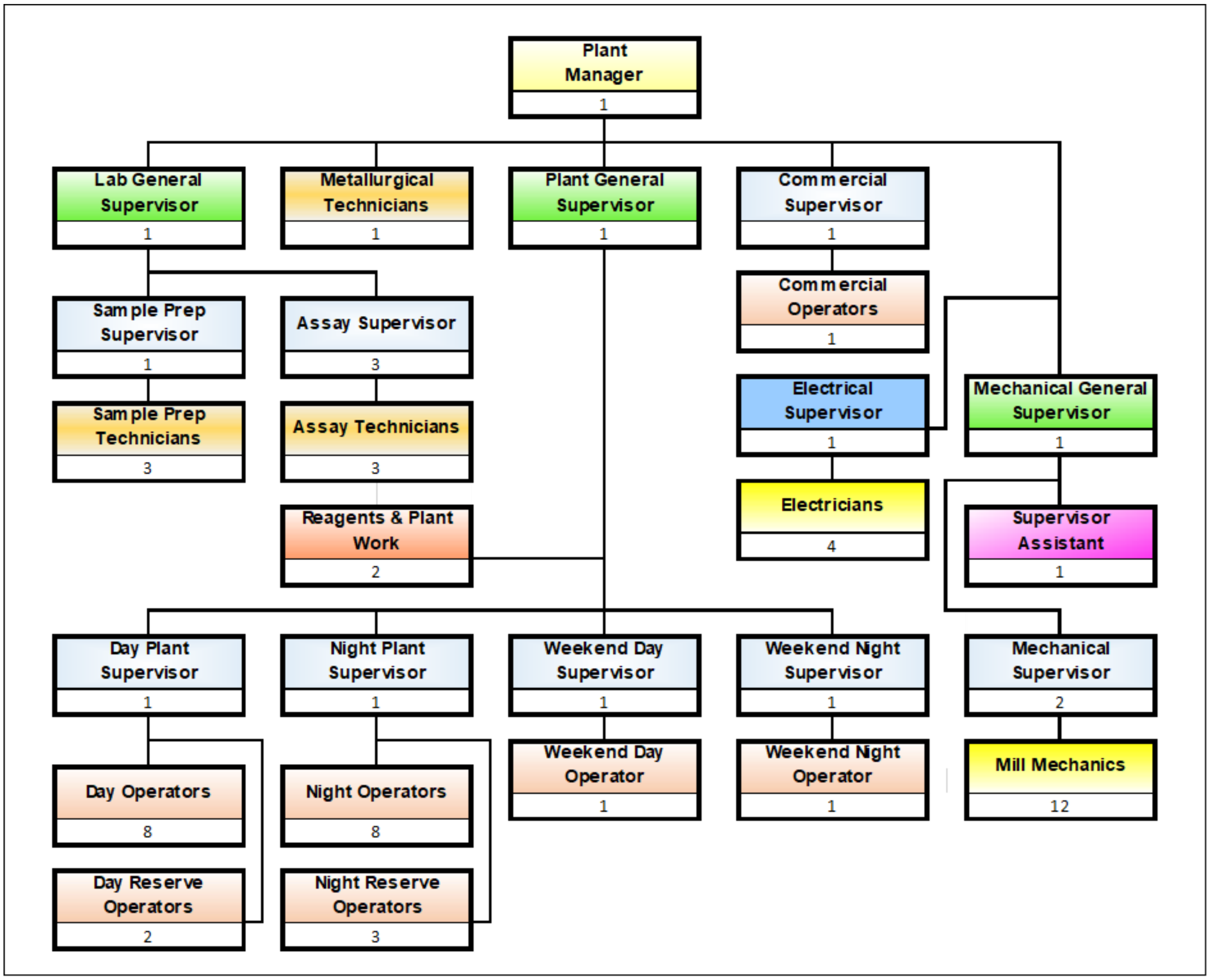


Figure 17-1 Process Department Organizational Structure

17.2 PROCESS PLANT OPERATIONS DESCRIPTION

Within the plant there is a division by areas according to the work completed at each circuit. The individual plant areas are defined below.

- Area 10: Primary Crushing
- Area 20: Stockpile Feed & Pebble Crushing
- Area 30: Grinding
- Area 40: Flotation
- Area 60: Concentrate Thickening and Filtering
- Area 70: Tailing Disposal
- Area 80: Services
- Area 90: Storage and distribution of reagents

The Aguablanca mineral processing plant flowsheet is presented in Figure 17-2.

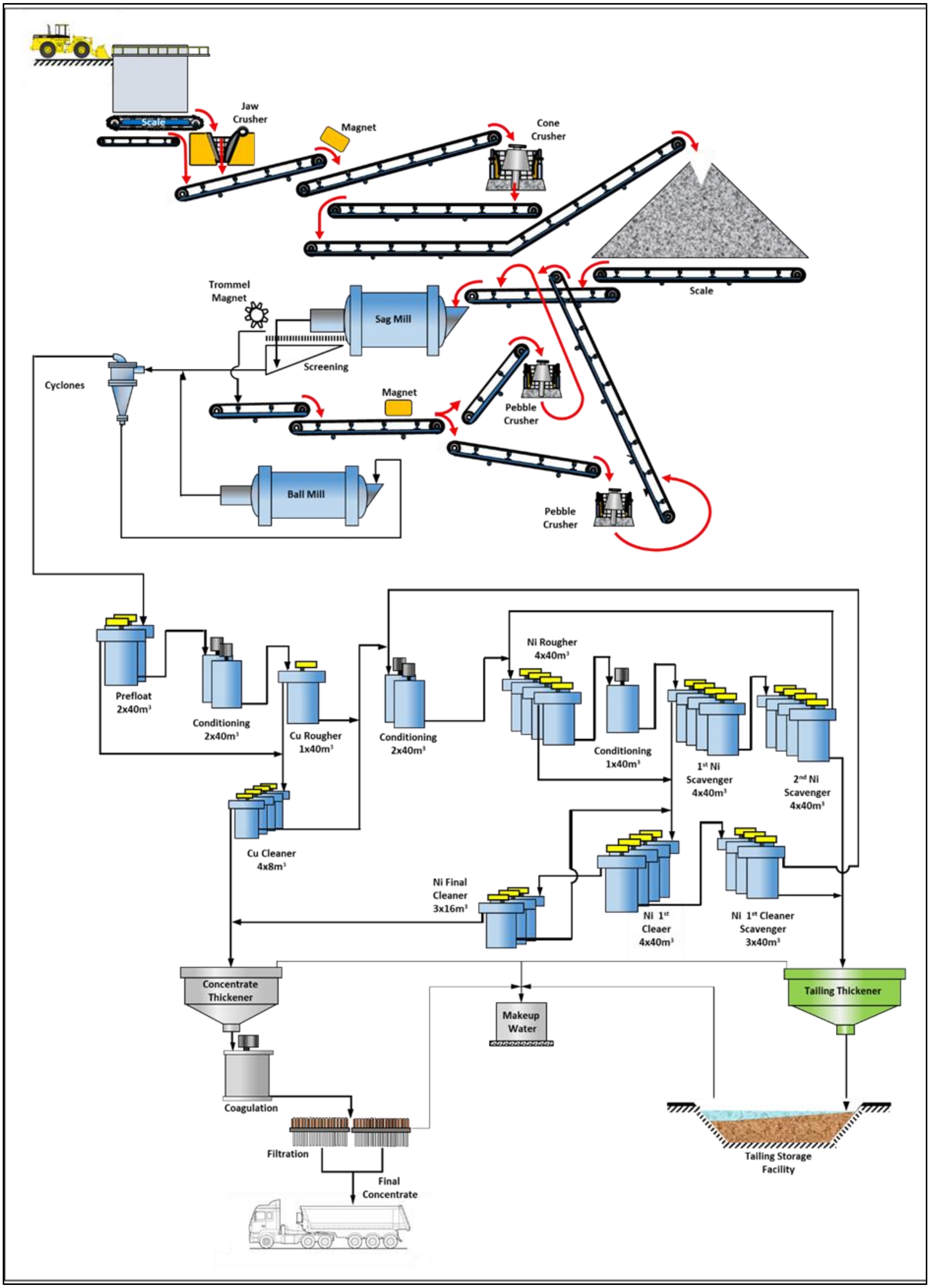


Figure 17-2 Aguablanca Mineral Processing Plant Flowsheet

17.3 AREA 10 – CRUSHING

The Aguablanca crushing plant consists of two stages both operated from a central control room. The primary stage is a C160 Nordberg 250 kW (335 Hp) jaw crusher. Run-of-mine rock is dumped either by haul trucks or front-end loader on a level above the primary crusher into a crusher feed bin. The rock flows by gravity onto a variable speed pan feeder that delivers rock to the jaw crusher which reduces rock up to 900 mm (35 inches) in diameter to 100 mm (4 inches). Below the pan feeder is a belt conveyor to catch fine material spilled from the apron feeder. This conveyor transports the spilled material to the jaw crusher discharge belt conveyor. A 900 mm x 900 mm grate prevents rocks too large for the jaw crusher from going into the crusher. Located adjacent to the jaw crusher is a hydraulic rock breaker used to break rocks too large for the rock crusher opening. The pan feeder discharge and a photo looking down into the jaw crusher are shown in Figure 17-3and Figure 17-4, respectively. A photo showing the primary crusher area is presented in Figure 17-5.

Figure 17-3 Pan Feeder Discharge (Lain photo)

Figure 17-4 Looking into jaw crusher

Figure 17-5 View of primary crushing area (Lain photo)

The jaw crusher product drops onto a belt conveyor which transfers the rock to a second belt conveyor that transports rock to the secondary crusher. The first belt is equipped with a weight scale. A large electric magnet is suspended over the conveyor discharge transfer point to collect tramp iron. A photo of the electric magnet suspended over the conveyor head pulley is shown in Figure 17-6.

Prior to 2007, the Aguablanca crushing circuit consisted only of the primary jaw crusher with the jaw crushing product being conveyed directly to the crushed ore stockpile. A 250 kW (335 Hp) Symons 5 ½-foot standard cone secondary crusher and associated belt conveyors were installed in 2007 to produce finer process plant feed. Cone crusher product target size is 50 mm (2 inch). Cone crusher discharge drops to a belt conveyor which transports the product to the mill feed stockpile. A photo of the secondary crusher structure is presented in Figure 17-7. The stockpile feed conveyor is shown in Figure 17-8. The crusher circuit flowsheet is presented in Figure 17-9.

Figure 17-6 Electric Magnet (Lain photo)

Figure 17-7 Secondary crusher feed conveyor

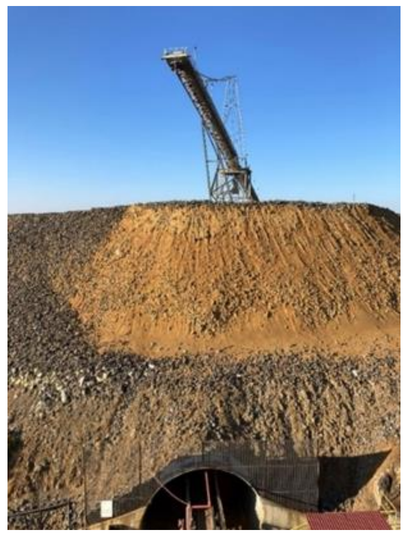


Figure 17-8 Stockpile Feed Conveyor

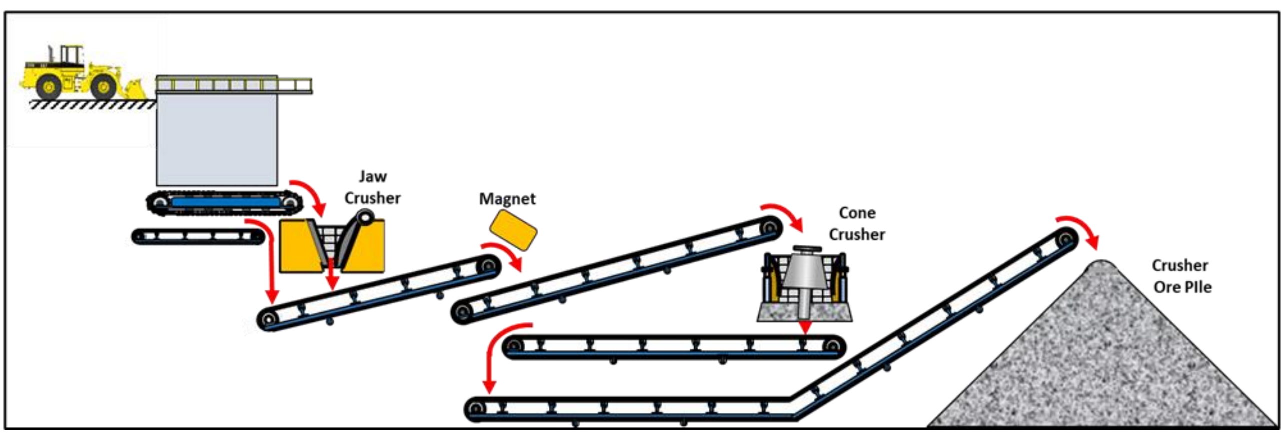


Figure 17-9 Aguablanca crushing circuit

17.4 AREA 20 – STOCKPILE FEED AND PEBBLE CRUSHING

Crushed ore is stored in a 50,000-tonne total and 12,000 tonne live capacity stockpile. The crushed ore flows by gravity through six 1.83 x 1.83-meter (6 x 6 foot) hoppers onto vibratory feeders which load the material onto a variable speed conveyor housed in a galley below the crushed ore stockpile. Four of the six feed hoppers are shown in Figure 17-10. A vibratory feeder is shown in Figure 17-11.

Figure 17-10 Stockpile feeders (Lain photo)

Figure 17-11 Stockpile feeders (Lain photo)

The conveyor below the stockpile is variable speed to control Semi-Autogenous Grind (SAG) mill feed rate. This conveyor is also equipped with a belt scale to monitor feed rate and total mill feet tonnes. The crushed ore pile conveyor discharges onto another conveyor that conveys the crushed rock to the SAG mill.

Located near the SAG mill feed conveyor are two pebble crushers. One, an HP200 132 kW (177 Hp) Metso cone crusher, was installed during original plant construction while the second crusher, a model CH 430 150 kW (200 Hp) Sandvik cone crusher, was installed in 2010. Both cone crushers are 900 mm (35 ½-inch).

Feed to these two crushers consists of oversize pebbles screened from the SAG mill discharge slurry. Plus 19 mm (3/4 inch) SAG mill discharge material drops to a conveyor that discharges to a second conveyor that transports the oversize product to a tower where a diverter directs the material to one of the two crushers. A magnetic separator is suspended over the second conveyor to remove reject SAG mill balls and other tramp iron. The magnetic separator is shown in Figure 17-12. A magnetic separator is also located in the pebble crusher feed tower. Metso and Sandvik crushers photos are presented in Figure 17-13 and Figure 17-14, respectively. The pebble crushers produce a 12 mm (1/2 inch) product. Discharge from the Metso and Sandvik crushers are returned to the SAG mill feed conveyor.

Figure 17-12 Magnetic Separator

Figure 17-13 Metso pebble crusher (Lain)

Figure 17-14 Sandvik pebble crusher

17.5 AREA 30 – GRINDING

The Aguablanca grinding circuit includes a SAG mill, SAG mill discharging screening, cyclones, and a ball mill. The grind and pebble crushing areas are operated from a central control room. The pebble crushing and grind circuit flowsheet is shown in Figure 17-15.

Crushed ore is ground in two stages. The primary stage is an Outokumpu EGL 6.45 meter in length x 5.5 meter diameter (21 foot x 18 foot) SAG mill powered by a 3,400 kW (4,560 Hp) 6,600 volt 3-phase induction motor. The SAG mill liners are steel. While design SAG mill feed rate is 195 tph, the addition of the cone crusher to the crushing circuit allowed for higher feed rates. Actual monthly mill feed rates between May 2008 and December 2015 when the plant was shut down ranged from 161 tph to 246 tph with an average feed rate of 212 tph. The mill charge is made up of 165 mm (6 ½ inch) cast steel grinding balls with a charge target at 28 to 30% of the mill volume. Mill speed is 14 rpm, 76.5% of critical speed. A photo of the SAG mill is shown in Figure 17-16.

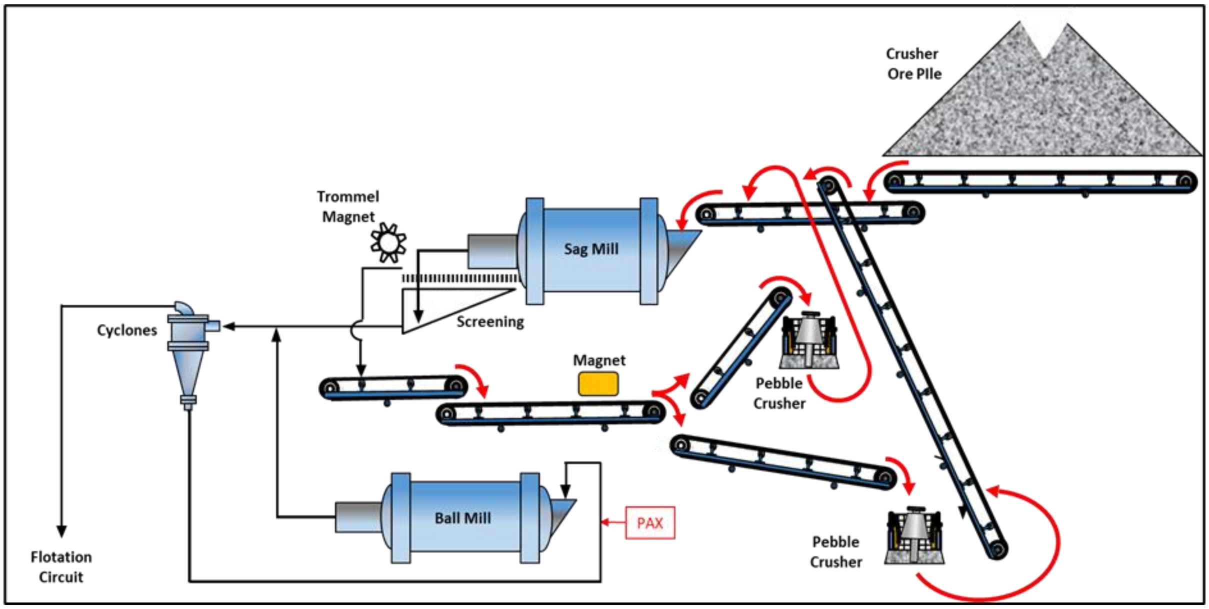


Figure 17-15 Grind circuit flowsheet

Figure 17-16 SAG mill (Lain photo)

The SAG mill runs in closed circuit with the pebble crushers with a circulating load of 35%. Water is added to the SAG mill to produce a slurry density is 65% solids by weight. SAG mill discharge particle size is 90% passing 12 mm (½ inch) and 80% passing 6 mm (¼ inch). SAG mill discharge slurry passes over a vibratory screen with the screen undersize reporting to cyclones. The oversize material is conveyed to the pebble crushers.

In addition to slurry, some grinding balls also exit the SAG mill. A rotary magnet positioned at the discharge end of the vibratory screening plant captures the fugitive balls and drops them into a collection bin. The balls are then added to the ball mill. The rotary magnet is shown in Figure 17-17.

Figure 17-17 SAG mill discharge rotary magnet

The cyclone bank is made up of five 500 mm (19 ½-inch) Warman model S15 500 mm hydrocyclones. Normally only two or three cyclones are operating with the remaining cyclones on standby. The cyclones are fitted with 100 to 110 mm (˜4 inch) apexes and 170 mm (6 ¾ inch) vortex finders.

Cyclone feed density is 60% solids by weight. Cyclone underflow slurry, with a density of 78% to 80% solids by weight, reports to the ball mill. Cyclone overflow at 30% solids by weight with a particle size target of 80% passing 80 microns flows to the pre-float flotation cell. A photo of the cyclone bank is shown in Figure 17-18

Figure 17-18 Cyclone bank

The second stage of the Aguablanca grinding circuit is an Outokumpu EGL 5 meter in length x 7.75 meter diameter (16 ½ foot x 25 ½ foot) rubber lined ball mill powered by a 3,400 kW (4,560 Hp) 6,600 volt 3-phase induction motor. The ball mill is charged with 63 mm (2 ½ inch) cast steel balls and the charge volume is maintained at between 28% and 30%. A photo of the ball mill is presented in Figure 17-19.

Figure 17-19 Aguablanca ball mill

The ball mill runs in closed circuit with the Warman hydrocyclones. Slurry discharge from the ball mill blends with SAG mill discharge and is pumped to the cyclone bank via either the Metso HR 300 CR 315 kW (422 Hp) electric motor-powered centrifugal pump or a Warman 12/10 AH centrifugal pump also powered by a 315kW electric motor. When one pump is running, the other is on standby. Ball mill circulating load is between 250% and 350%. Water is added to the cyclone underflow to maintain a ball mill slurry density of 75% solids by weight. Potassium amyl xanthate (PAX) collector is also added to the ball mill feed slurry.

17.6 AREA 40 – FLOTATION

The flotation plant includes a pre-float stage prior to copper rougher flotation and a nickel flotation circuit downstream from the copper flotation cells. The pre-float stage was originally used to separate naturally hydrophobic talc prior to copper flotation. That option no longer exists. The flotation circuit is operated from the same control room as is the grinding circuit. The copper flotation circuit incudes a rougher and cleaner stage while the nickel flotation circuit includes rougher, scavenger, and cleaner stages. Flotation slurry density target ranges from 30% to 33% solids by weight and flotation slurry pH is in the neutral range. The Aguablanca flotation plant flowsheet, complete with reagent addition points, is presented in Figure 17-20. Flotation reagents and their purpose are presented in Table 17-2.

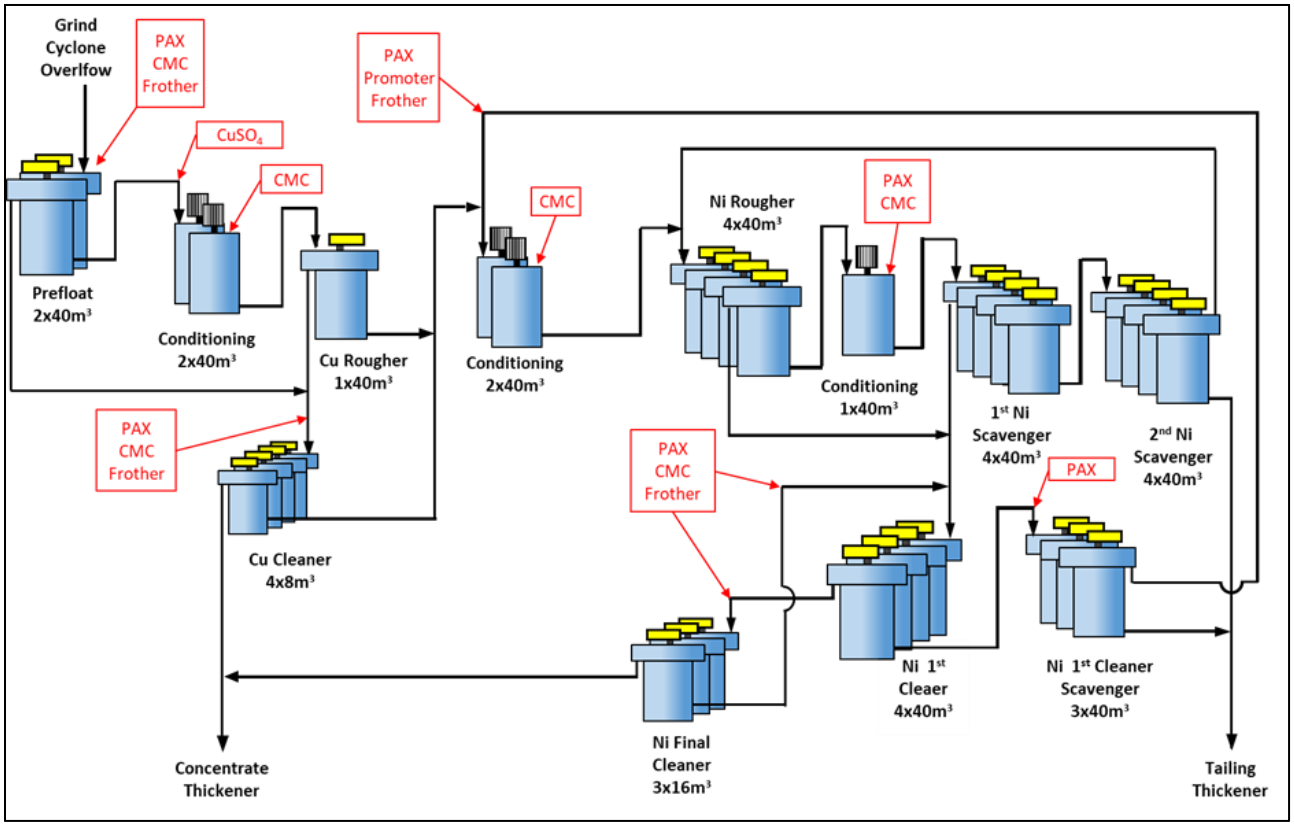


Figure 17-20 Flotation circuit

Table 17-2 Flotation reagents

Reagent	Supplier Name	Purpose	Consumption, kg/tonne
Glycol	FloMin F 650	Frother	0.066
Carboxymethyl Cellulose	Depramin 347	Depressant	0.756
Potassium Amyl Xanthate (PAX)	FloMin C 3430	Collector	0.125
Thiocarbamate	Aero3894A	Promoter	0.004
Copper Sulfate	$CuSO_4$	Activator	0.030

Cyclone overflow slurry reports to flotation circuit. An automated online sampler collects samples from twelve flotation circuit product streams which are fed to a Courier 5 On-Stream XRF Analyzer to provide real-time data allowing operators to monitor plant performance. The process flows shown in Figure 7-20 describe normal process flows. However, based on XRF Analyzer data, flotation product streams can be directed in different manners to maintain optimum product grades and metal recoveries. The on-line sampler and sample analyzer are shown in Figure 17-21 and Figure 17-22, respectively.

Figure 17-21 On line automated sampler

Figure 17-22 Sample analyzer

Cyclone overflow slurry flows to pre-float flotation consisting of two 40 cubic meter Dorr Oliver Eimco Smart Cell 40 flotation cells with agitators driven by 75 kW (100 Hp) electric motors. Reagents added to the pre-float cells include PAX, a sulfide mineral collector; carboxymethyl cellulose (CMC), a talc depressant; and glycol frother. Originally, the pre-cleaner stage was used to float naturally hydrophobic talc minerals with the concentrate flowing directly to the tailing thickener. This option no longer exists.

Except for the second nickel scavenger flotations cells, flotation circuit air is supplied by two Continental Industries Model 077A 05, 75 kW (100 Hp) blowers. The second scavenger flotation cells, installed in 2010, receive air from two Model 51A 06, 30 kW (40 Hp) blowers.

Based on XRF analyzer data, pre-float tank concentrates flows either to the copper concentrate thickener or to the copper cleaner circuit consisting of a bank of four 8 cubic meter Dorr Oliver Eimco Smart DO 300 UT flotation cleaner cells with agitators driven by 11 kW (14 ¾ Hp) electric motors. Collector, depressant, and frother are added at the copper cleaner circuit. Copper cleaner concentrate reports to the concentrate thickener. Copper cleaner circuit tailing can be pumped to the second copper rougher conditioning tank, the nickel rougher condition tanks, or the nickel scavenger conditioning tank.

Pre-float tailing slurry flows to two 40 cubic meter Dorr Oliver Eimco conditioning tanks operating in series. The conditioning tanks are fitted with 11 kW (14 ¾ Hp) MIXTEC mixers. Copper Sulfate ($CuSO_4$) is added to the first conditioning tank to activate sulfite minerals and CMC is added to the second mix tank to depress talc. Conditioning tank slurry flows to the copper rougher flotation stage, a single 40 cubic meter Dorr Oliver Eimco Smart Cell 40 flotation cell with an agitator driven by a 75 kW (100 Hp) electric motor. Rougher copper concentrate in turn flows by gravity to the copper cleaner flotation cells. The copper rougher conditioning and rougher flotation drives are shown in Figure 17-23.

Figure 17-23 Copper rougher conditioning and rougher flotation

Copper rougher tailing reports to the nickel rougher conditioning bank, again two 40 cubic meter Dorr Oliver IMCO units with 11 kW (14 ¾ Hp) MIXTEC mixers. PAX, promoter, and frother are added to the first conditioning tank while CMC is added to the second tank. After conditioning, the slurry flows to the nickel rougher circuit consisting of four 40 cubic meter Dorr Oliver Eimco Smart Cell 40 flotation cells with agitators driven by a 75 kW (100 Hp) electric motor. Nickel rougher concentrate is pumped to the first nickel cleaner bank while nickel rougher tailing is sent to a single 40 cubic meter Dorr Oliver IMCO conditioning tank with an 11 kW (14 ¾ Hp) MIXTEC mixer. PAX and CMC are added to this conditioning tank.

The conditioned nickel rougher tailing slurry reports to the first nickel scavenger bank, once again four 40 cubic meter Dorr Oliver Eimco Smart Cell 40 flotation cells with agitators driven by a 75 kW (100 Hp) electric motors. Nickel first scavenger concentrate reports to the first nickel cleaner circuit along with the nickel rougher concentrate. First nickel scavenger tailing flows to the second nickel scavenger cells, another bank of four 40 cubic meter Dorr Oliver Eimco Smart Cell 40 flotation cells with agitators driven by a 75 kW (100 Hp) electric motors. The second nickel scavenger circuit was added to the process in 2010 to increase retention time and nickel recovery. Concentrate from the second nickel scavengers is pumped to the head-end of the nickel flotation circuit while second scavenger tailing slurry reports to the tailing thickener. The nickel rougher and scavenger banks are shown in Figure 17-24.

Figure 17-24 Nickel rougher and scavenger banks

The first nickel cleaner bank is made up of four 40 cubic meter Dorr Oliver Eimco Smart Cell 40 flotation cells with agitators driven by a 75 kW (100 Hp) electric motors. PAX and CMC are added to the first flotation cells. Concentrate from the first nickel cleaners is pumped to the final nickel cleaner circuit while tailing reports to the nickel first cleaner scavenger bank which is made up of three 40 cubic meter Dorr Oliver Eimco Smart Cell 40 flotation cells with agitators driven by a 75 kW (100 Hp) electric motors. PAX is added here as well. Nickel first cleaner scavenger concentrate is pumped to the head-end of the nickel flotation circuit while tailing reports either to the final tailing thickener or is sent back to upstream nickel circuit tanks. The nickel first cleaner scavenger bank is shown in Figure 17-25.

Figure 17-25 Nickel first cleaner scavenger bank

Final nickel cleaner flotation consists of three 16 cubic meter Dorr Oliver Eimco DO 600 UT flotation cells with agitators driven by a 22 kW (30 Hp) electric motors. While the final nickel concentrate product is pumped to the concentrate thickener, final nickel cleaner tailing is sent back to the first cleaner tailing bank.

17.7 AREA 60 – CONCENTRATE THICKENING AND FILTERING

Concentrate slurry density ranges from 27% to 30% solids by weight. A majority of the water must be removed before concentrates can be shipped to the smelter. The first stage of concentrate dewatering is accomplished by a 14 m (46 foot) Outokumpu high-capacity thickener with a 7.5 kW (10 Hp) rake drive. Flocculant (N-71771) is added to thickener feed slurry to increase solids settling velocity. Clear thickener overflow solution is pumped to a reclaim water holding tank from where it is distributed throughout the process plant as makeup water.

Thickener underflow, with a density of between 55% to 60% solids by weight, is pumped to the filter press feed holding tank where coagulant (N-8105) is added to increase filtration time and efficiency. Coagulated slurry is pumped to one of two FSI Filtration Model B-1210/07-CM 1.2x1.2m (47 ¼ x47 ¼-inch) plate and frame 5.5 kW (7.4 Hp) filter presses fitted with 48 filter plates. Feed slurry is alternately fed to one filter press or the other with one filter press always operating. Filter cake concentrate containing less than 10% moisture is transfer to the concentrate storage area from where it is loaded into trucks and hauled to the smelting facility.

Filter press equipment is housed in a building separate from and adjacent to the grinding and flotation process building. Also included in the filtration building is a non-functional FSI Filtration Model B-1210/04-CM 1.2x1.2m (47 ¼ x47 ¼-inch) plate and frame filter press. This filter press was used to dewater copper concentrates when the nickel and copper concentrates were kept separate. A filter press and the concentrate storage area are shown in Figure 17-26 and Figure 17-27. The concentrate dewatering flowsheet is shown in Figure 17-28.

Figure 17-26 Plate and frame filter press

Figure 17-27 Concentrate storage area

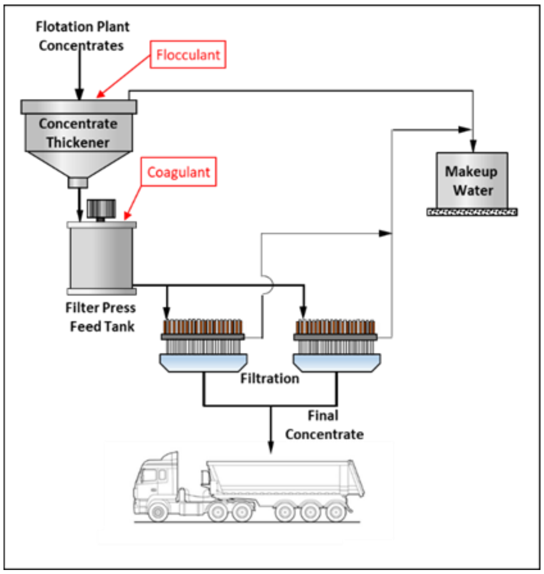


Figure 17-28 Concentrate dewatering

17.8 AREA 70 – TAILING DISPOSAL

Flotation tailing density ranges from 27% to 30% solids by weight. Because hindered settling does not occur in such dilute slurries, solids would settle in tailing delivery pipelines. Also, excess water is needed for process makeup water.

Water is removed from flotation tailing via a 15 m (49 foot) Outokumpu high compression paste thickener. The thickener rake mechanism is driven by a 47 kW (63 Hp) motor. Clear thickener overflow solution is pumped to the reclaim water holding tank from where it is distributed throughout the process plant as makeup water.

Thickener underflow, with a density of between 70% to 72% solids by weight, flows to an Abel HMO-H-160-2000 piston diaphragm positive displacement pump powered by a 160 kW (215 Hp) motor which pumps the thickened slutty to the tailing storage facility. In addition to the Abel pump, there are two Metso HR150 centrifugal pumps driven by 90 kW (120 Hp) motors that are utilized to pump the slurry to the tailing storage facility whenever the piston pump is down for maintenance. The tailing thickener is shown in Figure 17-29. The Abel and Metso pumps are shown in Figure 17-30 and Figure 17-31, respectively.

Figure 17-29 Tailing thickener (Golder)

Figure 17-30 Piston diaphragm pump (Golder)

Figure 17-31 Centrifugal pumps (Golder)

Tailing thickener underflow slurry is pumped through a 200 mm diameter steel pipe with 16 mm wall thickness (8-inch diameter x schedule 20) which transitions to a 200 mm HDPE pipe with 16 mm wall thickness (8-inch x SDR 13.5) for transport to the tailing storage facility. The HDPE line splits and runs along the perimeter of the impoundment. Smaller HDPE spigot lines branch from the 200 mm pipe to deposit tailing slurry along the impoundment beaches. The tailing storage facility elevation is roughly 30 meters (98 feet) above the process plant ground floor elevation and the tailing pipe line total length is approximately 3,000 meters (9,842 feet). Solids settle onto the impoundment beach as the solution flows to a pool from where it is pumped to the process plant makeup water tank for use in the process plant.

The tailing storage facility is lined with HDPE liner over compacted fill. Two Sulzer Model APP 23-40 37 kW (50 Hp) pumps return water from the tailing disposal facility to the process plant. The tailing storage facility is shown in Figure 17-32. A tailing disposal flowsheet is shown in Figure 17-33.

Figure 17-32 Tailing storage facility

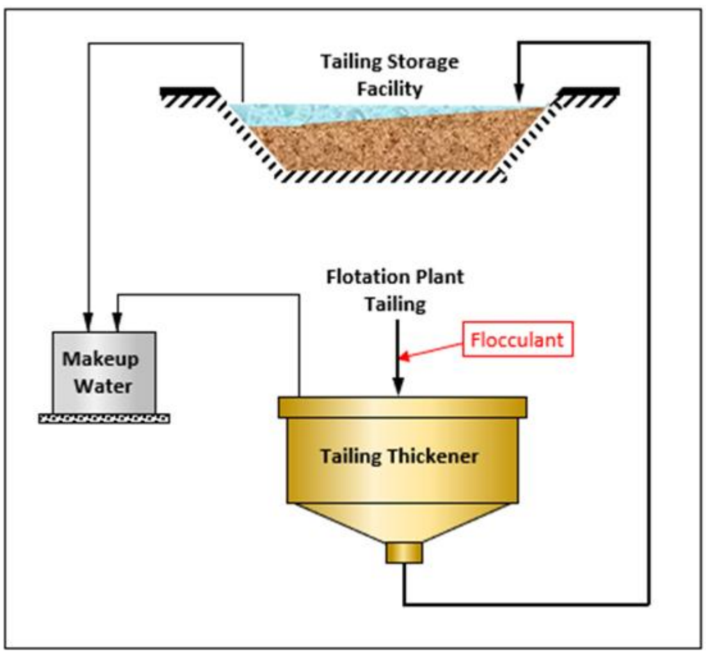


Figure 17-33 Tailing disposal flowsheet

17.9 AREA 82 – SERVICES

This area consists of all the water and air facilities associated with the mineral processing plant including the following systems:

- Process water
- Fresh water
- Firefighting
- Compressed air
- Blowers

17.9.1 PROCESS WATER

The data in Table 17-3, replicated from the Lain Technology report, shows Aguablanca temperature and precipitation data.

Table 17-3 Aguablanca weather data

WEATHER CONDITIONS			
Temperature		Precipitation	
Maximum	45 °C (113°F)	Median	680 mm/year (27 inches)
		Maximum	1100 mm (43 inches)
Minimum	-8 °C (46°F)	Rainy days	70 per year
		Evaporation	1280 mm/year (50 inches)

Thickener overflow solution from the concentrate and tailing thickeners are pumped to the 1,000 cubic meter (264,172 gallon) holding tank shown in Figure 17-34 for distribution throughout the process plant. Water from the tailing impoundment and the pressure filters is also pumped to the process water tank.

Figure 17-34 Process water holding tank

Because of high ambient temperatures and evaporation rates, efficient Aguablanca water management is critical. Process plant water consumption, the difference between plant feed moisture content and the volumes of water retained in the final concentrate and settled tailing material, coupled with the high evaporation rate, necessitate the use of fresh makeup water to achieve neutral water balance.

17.9.2 FRESH WATER

Runoff precipitation and inert water pumped from the mine workings are collected in an earthen pond. Two Axflow NT80-250 55 kW (75 Hp) pumps are in place to pump water from the pond to a 686 cubic meter (181,222 gallon) covered steel storage from where it is distributed to the process plant. When necessary, water from the Caja river is used maintain sufficient fresh water tank volume. Water from the fresh water tank is filtered and pumped for use as pump gland seal water.

17.9.3 FIRE WATER

Site firewater protection systems are charged with fresh water distributed by electric pumps. An auxiliary diesel-powered water pump is also in place for emergency use. The firewater pumping system is shown in Figure 17-35.

Figure 17-35 Firewater pumping system (Lain photo)

17.9.4 POTABLE WATER

Fresh water is filtered on site and chemically treated with hypochlorite for site offices, emergency showers, and change rooms.

17.9.5 COMPRESSED AIR

Process plant compressed air is provided by three Ingersoll Rand Model SRILATA 90 kW (120 Hp) screw compressors. Compressed air is used at the crushing plant, for grind area grease pumps, in the filtration area, and as instrument air as well as for pneumatic maintenance equipment. Compressed air is distributed throughout the plant to receivers with volumes ranging from 0.5 cubic meters (132 gallons) to 5 cubic meters (1,320 gallons). Each receiver is fitted with a drain valve to periodically drain moisture from the tanks. Compressed air is cleaned and dried through several stages of filters and driers. The compressors are shown in Figure 17-36.

17.9.6 EMERGENCY GENERATOR

An Electra Molins 400 kVA stand-by diesel powered generator is in place to provide emergency electrical power in the event of planned or unplanned power outages Figure 17-37.

Figure 17-36 Ingersoll Rand compressors

Figure 17-37 Emergency Generator

Most concentrated reagents are received in dry form and all must be mixed with water to the desired concentration to be distributed throughout the plant. Reagents and their functions are listed in Table 17-4. Note that because Aguablanca slurry pH is typically neutral, hydrated lime addition is rarely necessary.

17.10 REAGENTS

Table 17-4 Aguablanca reagents

Reagent	Function	Area Used
Hydrated Line	pH control	Grind and Flotation
FloMin F 650 Glycol Frother	Frother	Flotation
Depramin 347 CMC Depressant	Depressant	Flotation
FloMin C 3430 Collector	Nickel Sulfide Collector	Nickel Flotation
Aero 3894A Collector	Copper Sulfide Collector	Copper Flotation
Copper Sulfate	Nickel Sulfide Activator	Nickel Flotation
N-8105 Coagulant	Settling Agent	Concentrate Dewatering
N-71771 Flocculant	Settling Agent	Concentrate Dewatering
N-9601 Flocculant	Settling Agent	Tailing Dewatering

Reagent inventory is maintained at a level sufficient for two months of process operation to compensate for ordering and delivery times. Reagents are received properly packaged and labeled with appropriate hazard warnings, product names, and weights. The raw reagent storage area is shown in Figure 17-38.

Figure 17-38 Raw reagent storage area

There is a dedicated system for each reagent which includes a dry reagent holding tank, a mixing tank, screw feeders for transferring dry reagent to the mixing tank, distribution pumps, an emergency shower and eyewash station, and a floor sump equipped with a sump pump with starts and stops controlled by high and low level switches. The systems can be controlled from a local panel or from the central control room.

To eliminate the possibility of reagent solution being contaminated by suspended solids in process solution, only filtered fresh water is used for reagent mixing. Interlocks for each system prevent upstream equipment from starting if downstream equipment is not in operation. The interlocks will also immediately shut down upstream equipment in the event of a downstream equipment stoppage. Any solution spills flow to the sumps and are pumped to the tailing thickener. A photo of a portion of the reagent area is presented in Figure 17-39.

Figure 17-39 Reagent area

18 PROJECT INFRASTRUCTURE

The Aguablanca Mine is a fully self-contained facility comprising all the necessary components to initiate restart without significant capital or schedule constraints. Due to the care and maintenance program established by RNR, the required rebuild repairs, upgrades, modifications, and commissioning identified in this PFS is relatively insignificant.

18.1 SITE BUILDINGS

The general site buildings include a staff office, change room, conference rooms, light vehicle shop, large equipment shop, fuel bays, gate house, and a receiving area with covered/uncovered spare parts storage.

The underground mine facility currently consists of a primary fan, overhead power line, and barricaded portal; viewed in Figure 18-1 and Figure 18-2. The contractor will be required to establish a dry facility and portal offices. In addition, the contractor shall install the underground dewatering system, communication lines, compressed air lines, and powder magazines. The compressor room and compressors are scheduled to be installed by the owners.

Figure 18-1 Primary Fan Installation

Figure 18-2 Barricaded Portal

18.2 PROCESS PLANT ELECTRICAL POWER

Hydroelectrical power is provided by Endesa, one of Spain’s three large electricity providers. Endesa transports electrical power via overhead transmission lines to the Aguablanca sub-station. Power is delivered at 6.6 kV which is reduced by Aguablanca transformers to voltages required throughout the plant and site buildings. The Aguablanca power distribution system and transformers are inspected annually by Elva Electricidad. The MCC rooms are clean and well maintained. A photo of a section of the process plant motor control center (MCC) is presented in Figure 18-3.

Figure 18-3 Process plant ACC cabinets

18.3 TAILINGS STORAGE FACILITY

The Aguablanca tailing storage facility (TSF) covers an area of 4.2 hectares (10.4 acres). When originally constructed, the tailing impoundment capacity was 2,500,000 cubic meters (˜3,270,000 cubic yards), sufficient capacity to contain 4,130,000 dry tailing tonnes, approximately two years' production at original process plant throughput rate. The original impoundment elevation was 496 meters (1,227 feet) above sea level. Lifts added to the embankment during the plant operating years raised the embankment to the current elevation of 520 meters (1,706 feet).

When the process plant was shut down in October, 2015, the impoundment contained 13,422,607 dry tailing tonnes (around 8,137,000 m^3 volume or 10,643,000 yd3). With the embankment elevation at 520 meters, the impoundment can contain an additional 3,800,000 dry tailing tonnes (2,300,000 cubic meters or 3,008,000 cubic yards). Therefore, at least one additional embankment lift will be required to contain the predicted 4,400,823 dry underground ore tailing tonnes.

Third-party TSF inspections are performed twice per year. WSP Golder completed an Aguablanca TSF inspection during September, 2023, and in the inspection report dated January 2024, WSP Golder noted, "*The Aguablanca tailing storage facility was designed by Eptisa Engineering Services S.A. in accordance with the provisions of Supplementary Technical Instruction 08.02.01 of Chapter XII of the General Regulations on Basic Mining Safety Standards "Sludge deposits in treatment processes of extractive industries and national, European and international standards", in such a way as to ensure the best management of the natural resources of the area during the project, exploitation and closure stages*."

The WSP Golder report further states, "*The Tailings Pond is located in an area where alluvial materials, eluvial deposits and soils were identified, as well as outcrops of slate and/or marble with different degrees of alteration and/or competition. The foundation of the dike was made on a firm substrate, for which the edaphic soil, the alluvial deposits and the layers of weathered slate that were in variable thickness in the surface area were excavated, being it supported on healthy rock*."

Regarding construction materials, Scott Wilson Mining stated in their March 2005 "Aguablanca Ni-Cu-PGM Project Tailings Disposal Facility Phase 1 Construction Report":

"*The selection of the materials for construction was based on quality, design requirements and on-site availability*."

"The materials used for the main embankment construction included weathered gabbros and clays, deposited in a 'fines zone', with a minimum horizontal width of 4 m along the lined slope, and weathered gabbros, fresh gabbros, limestone and marble for the rest of the embankment. The presence of the 'fines zone' was justified by the requirement to protect the geomembrane from any risks of puncture. All these materials originate from the mineral extraction operations at the open pit at Aguablanca."

"Materials used for the construction of the saddle embankment were shales originating from the grading of a hill located within the north-east part of the impoundment. No 'fines zone' was allocated for this embankment as shales are a friable material, which after compaction, grading and slope preparation, do not present any risks to geomembrane integrity."

"A campaign of laboratory testing was undertaken in April 2004 for every material identified for the dam construction. This laboratory programme comprised compaction testing in the form of Modified Proctor tests and California Bearing Ratio (CBR) tests. These laboratory tests established a thorough understanding of the geotechnical and mechanical properties of the materials and confirmed their suitability for construction."

The impoundment floor, slopes, and anchor trenches are lined with 1.5 mm High Density Polyethylene sheets (HDPE) placed over a layer of geotextile fabric. Main production seams were double fusion (wedge) welded using a self-propelled wedge welding apparatus. Seams around patches, joints, closing welds, and other minor repairs were made using an extrusion welder. A photo showing liner being placed along the impoundment slope and secured in an anchor trench is presented in Figure 18-4.

Figure 18-4 HDPE liner on slope secured by anchor trenches

Parallel underdrains 16 meters apart placed above the HDPE liner allow for tailing material dewatering. Underdrains were constructed using 200 mm (8-inch) perforated piping covered with gravel and encapsulated in geotechnical filter fabric. The parallel underdrains connect to main drains running perpendicular to underdrains and constructed in the same fashion as the underdrains. The main drains connect with two 250 mm (10-inch) solid HDPE pipes which intersect with the HDPE liner and pass through the embankment. Seepage flows from these lines flows to a lined seepage pond. The underdrain system is illustrated in Figure 18-5 and the two pipes passing through the embankment are shown in Figure 18-6.

Figure 18-5 Underdrain system

Figure 18-6 Impoundment drain pipes

Like the tailing impoundment, the seepage pond was constructed with compacted fill and is lined with 1.5 mm HDPE liner over geotextile fabric. Solution from the seepage pond is pumped back to the tailing impoundment. The seepage pond and seepage pond pump platform are shown in Figure 18-7.

Figure 18-7 Seepage Pond and pump platform

The tailing thickener underflow pumps transport tailing slurry via a 200 mm (8-inch) pipe to the tailing storage facility. The tailing line splits at the embankment and runs in both directions along the perimeter of the embankment. HDPE spigot lines connected to the tailing pipeline run down the impoundment slopes where tailing slurry flows onto the impoundment beaches. As the thick slurry flows across the beaches, solids settle and the slurry water flows to the impoundment pool. From there, the water is pumped back to the process plant to be used as process water. The tailing impoundment pool pump platform is shown in Figure 18-8.

Figure 18-8 Tailing Pond pump platform

19 MARKET STUDIES AND CONTRACTS

No market studies or product valuations were competed as part of the PFS. Market price assumptions were based on a review of public information, industry consensus standard practices and specific information from comparable operations in Spain.

There are markets worldwide for the sale nickel-copper concentrates. Denarius has tendered several international offtake customers for the sale of nickel-copper concentrates. No contracts for the offtake of concentrates have been signed as of the effective date of the Technical Report. The QP has reviewed the terms of a proposed agreement which was the basis of the economic analysis for this report. The proposed terms are within industry accepted norms and the results support the assumptions in the Technical Report.

Similarly, there are no contracts in place for the supply of reagents, equipment, labor, or bulk commodities to operate the Project.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL INFORMATION PUBLISHED IN THE OFFICAL STATE GAZZETTE THROUGH AUGUST 4, 2017

Resolution of June 2, 2003 pf the General Secretariat for the Environment

This resolution formulated an environmental impact statement (EIS) on the mining project of the Aguablanca deposit together with the reclamation plan, within the framework of the research permit covering 95 mining licenses. The EIS was approved for open-pit mining for a period of 11.5 years. The approved project included a beneficiation plant, a tailings storage facility (TSF), and three dumps: number 1 (35 ha and capacity 11.5 Mm3), number 2 (43 ha and capacity of 13.1 Mm3), currently combined into one dump, collectively the West dump, and number 3, now known as the East dump (40 ha and 8.3 Mm3 capacity).

Resolution of July 30, 2004 of the General Secretariat for the Preventions of Pollution and Climate Change

The resolution approved an EIS for the completion of a 66 kV power line, from the Presur Substation in Cala, Huelva, to the Monesterio Substation, passing through the Aguablanca Substation, both in Monesterio, Badajoz. Approved a 18.27 km power line necessary for mining that runs through the municipalities of Cala, Santa Olalla de Cala and Monesterio, including all phases of construction and operation. The EIS did not contemplate its dismantling subsequent to the cessation of mining. Subsequently, by resolution of the Directorate-General for Energy Policy and Mines, Río Narcea Recursos, S.A. was authorized to build a 66 kV overhead power line from the Presur substation, in the municipality of Cala (Huelva) to the Aguablanca substation, in the municipality of Monesterio (Badajoz), in order to supply electricity to the industrial exploitation of the Aguablanca nickel-copper deposit and the installation of the nickel and platinum concentration plant in Monesterio (BOE No. 278 of 18 November 2004).

Resolution of May 21, 2012 of the Secretary of State for the Environment

The East dump in the in the EIS is reduced from 40 to 38.5 ha.

December 26, 2014 Letter from the Ministry of Industry, Energy and Tourism

Acknowledgement that Lundin Mining indicated they are planning to transition from open pit mining to underground mining.

May 7, 2015 letter from the Sub-Directorate General for Environmental Assessment to the Sub-Directorate General for Mines

They indicated that the condition provided for in Article 4.1 of Royal Decree 1032/2003, of July 25, 2003, declaring the Aguablanca area as a definitive reserve in favor of the State, would apply to the change of the method of exploitation from open pit to underground mining, in which it is stipulated that any modification of the approved project will require a new EIS. It was also pointed out that the continuation of the proposed underground operation constitutes in itself a separate operation from open-pit.

July 21, 2015 Ministry of Industry, Energy and Tourism

By order of the Ministry of Industry, Energy and Tourism, underground mining was suspended.

20.2 SUMMARY OF REVISED EIS INITIATED OCTOBER 27, 2015

The following will summarize the results of the current EIS. There are no known environmental issues that could materially affect Denarius' ability to extract the mineral reserves identified in this report.

20.2.1 REQUIREMENTS FOR WASTE AND TAILINGS DISPOSAL

The eastern dump and the tailings pond must accommodate, respectively, the tailings of underground mining and the tailings of the processing of the without increasing their currently authorized dimensions. Authorities may suspend mining activity if higher volumes are generated.

During the mine operations, the tailings ponds will be monitored for quality and control of the size of particles, control of the quality of the supernatant water and the leachate water. Periodic inspections to detect possible movement, displacement, settlement, instabilities and/or leaks, topographic control with periodic surveys, and periodic bathymetric control.

Waste sumps are to be reclaimed to a slope of less than 3.5% and a maximum elevation of 548 m. Waste dumps will be revegetated with stands of local shrubs and trees native to the area. Tailings pods will be reclaimed to an elevation of 219 m. and a slope of no greater than 1.5%. Tailings ponds will be covered with HDPE liner and then covered with topsoil followed by revegetation of native plant species.

20.2.2 SITE MONITORING REQUIREMENTS

Air

Vehicular traffic, mining, and treatment of ore will generate particles that will be dispersed in the atmosphere in the form of dust and polluting gases. To minimize emissions, the mine will employ measures such as watering trucks, placing sprinklers on conveyor belts and crushing the ore by wet means. Traffic speeds will be limited and proper maintenance of vehicles and machinery will be employed.

Soil

For the exploitation phase, the environmental impact study states that the main mechanism of instability will be the sliding of rocky blocks on the lower slopes of the pit, which is already occurring, and could be enhanced with the subvertical walls that derive from the sublevel mining, being able to extend upwards to the exposed pit bottom. The magnitude of the risk was estimated as medium-high and a moderate effect.

There will be no subsidence of waste dumps or tailing ponds due to underground blasting.

In order to minimize the instabilities predicted for the bottom of the pit, the pit will be backfilled, keeping the bottom 181 meters. Material will be sourced from the east dump. This technique can help correct possible local instabilities. In addition, the mine will be continuously drained during the underground mining. Geotechnical monitoring will be carried out on the most unstable slopes of the southeast highwall, thus following the recommendations to mitigate the instabilities identified in the pit from a study carried out by Lundin in 2014.

There has not been any indication that the waste dumps or tailings pond will have any risk of instability.

During mining, the impacts on the soil due to compaction from traffic or machinery and pollution due to accidental spills of hydrocarbons, fuel and hazardous products or waste, are similar to those from open pit mining. Similar preventive and corrective measures contained in the EIS will be applicable.

For the reclamation and closure phase of the mine, the restoration plan establishes that a soil investigation will be carried out in accordance with Decree 49/2015, of March 30, 2015, in the areas where potentially polluting activities have been carried out, such as the area occupied by the beneficiation plant and auxiliary facilities, in which the existence or not of potentially polluting soil is determined as well as the appropriate measures for its management.

Surface and Groundwater

Underground mining will not affect public waterways. As the mine deepens influxes of water will increase from 2 l/s to as much as 10 l/s. This water will be used in the mining and beneficiation plant. There is no indication that the water table will be affected more than 100 m from the current open pit. A detailed hydrogeologic model of the project affirms there will be no effect on the base flow of the nearby Rivera de Cala river.

Regarding water quality, the EIS identifies the risk of water contamination due to underground mining as medium-low, since the current mining operation operates with "zero discharge", reusing the runoff and leachate water generated in its facilities. The system will be maintained throughout reclamation. Results obtained from the monitoring and follow-up do not show remarkable variations, and identifies as moderate the potential impact of groundwater contamination due to contact with the operational areas. The risk of accidental spills or spills during the mining phase that could cause indirect effects on other factors such as fauna or protected areas will be continually addressed.

As measures to minimize or correct impacts on water quality, the mine plans to continue applying the same measures adopted to date: water management focused on zero discharge, separation of flows and differentiated treatment depending on the origin and quality of the water, construction of drainage channels around the perimeter of the auxiliary mine facilities to collect runoff water and reuse it, and waterproofing of any pond or reservoir that accumulates acidic water and heavy metals to prevent leachate.

Biodiversity

For the mining phase, the EIS identifies nuisances for fauna due to an increase in noise levels, although it is estimated that these will not be significant and will hardly differ from those produced during the operation of the open-pit mine. This has not seemed to have prevented the regular presence in the surrounding territory of species with reserved habits such as the black stork and the otter. This is confirmed in the fauna studies carried out in the previous Environmental Monitoring Program.

To mitigate this slight impact on fauna, various measures are proposed: minimization of dust, noise and light emissions, speed limits at 40 km/h at the entrances while maintaining and replacing the appropriate signage, progressive restoration of altered surfaces, and plant restoration aimed at the recovery of the Mediterranean forest.

Likewise, during the mining phase and the post-closure phase, there may be an increase in water pollution that may have an impact on aquatic and riparian fauna and flora, or fauna that drinks or feeds in the surrounding streams and rivers, for which compliance with measures aimed at protecting the waters must be ensured.

Landscape

Restoration and reclamation measures are incorporated in the EIS. This includes the complete dismantling and removal of the process plant, offices, explosives magazines, the entrance to the underground mine and the ventilation shafts. Restoration also includes the underground mine, dumps, tailings pond, water ponds, stockpiles, roads and the fresh water pond. Topography will be remodeled and shaped to as close to original pre-mining topography as can be obtained.

In general, the restoration plan proposes the planting of herbaceous plants for the affected areas where it is planned to generate a pasture (type 1: grasses 60 % and legumes 40 %) while the planting of herbaceous plants (type 2: grasses 55 % and legumes 40 %) and shrubs will be carried out in the areas intended to create a Mediterranean scrubland. On steep slopes a mixture of herbaceous and shrubby crops (60% grasses, 30% legumes and 10% shrubs) will be carried out.

After planting, medium-tall shrubs typical of Mediterranean scrubland (1,100 units/ha) will be planted with rockrose, flax, broom and kermes oak, among others, in groups or groves.

To the southeast of open pit, and in order to reduce its visibility from the access road, it is proposed to create a visual screen of shrubs in a row and staggered (1.2 m apart between feet) with columnar buckthorn. In some water ponds, in order to promote the quality of the water in addition to naturalization, it is proposed to generate a border of hygrophilous vegetation with cattails and reeds (800 units/ha).

Ultimately, the plan is to reestablish a predominantly natural forest, restoring as much as possible the land to its original state.

Community

Anticipated mining production will allow significant employment from the local area. After mining is complete there will be a significant reduction in the workforce. The EIS points to the retraining of the dismissed personnel as a mitigating measure, and also points to a possible subsequent use of the abandoned mining site for social, cultural or educational purposes, but without giving further specificity or developing a specific project.

20.3 RECLAMATION BUDGET

The budget for the execution of this Restoration Plan amounts to €8,289,254, where it is estimated that some €4,237,889 will be allocated to restoration work on mining waste facilities.

Table 20-1 is a summary of the budget for the closure and restoration of the Aguablanca Mine grouped by zones and facilities.

Table 20-1 Mine Site Reclamation Budget

GENERAL SUMMARY OF THE CLOSURE AND RESTORATION BUDGET BY ZONES/FACILITIES		
Code	**Description**	**Budget (€)**
TASK 1	**SURFACE OF MINING AREA**	
TASK 1.1	Trenching and construction of drainage works	384 €
TASK 1.2	Dismantling, demolition and removal of equipment	187,486 €
TASK 1.3	Preparing Soil for Revegetation	8,710 €
TASK 1.4	Revegetation (sewing, planting and other measures)	5,697 €
TASK 1.5	Enclosures and signage	89,357 €
TOTAL TASK 1		**291,633 €**
TASK 2	**UNDERGROUND MINE (Portal and Ventilation Shafts)**	
TASK 2.1	Earthworks	15,380 €

GENERAL SUMMARY OF THE CLOSURE AND RESTORATION BUDGET BY ZONES/FACILITIES		
Code	**Description**	**Budget (€)**
TASK 2.2	Dismantling, demolition and removal of equipment	547,466 €
TASK 2.3	Preparing Soil for Revegetation	16,829 €
TASK 2.4	Revegetation (sewing, planting, and other measures)	10,189 €
TASK 2.5	Enclosures and signage	100,000 €
TOTAL TASK 2		**689,865 €**
TASK 3	**DUMP RECLAMATION**	
TASK 3.1	Earthworks	77,454 €
TASK 3.2	Trenching and construction of drainage works	21,655 €
TASK 3.3	Dismantling, demolition and removal of equipment	2,574 €
TASK 3.4	Preparing Soil for Revegetation	92,858 €
TASK 3.5	Revegetation (sewing, planting, and other measures)	137,562 €
TOTAL TASK 3		**332,102 €**
TASK 4	**TAILINGS STORAGE FACILITY**	
TASK 4.1	Earthworks	1,204,877 €
TASK 4.2	Trenching and construction of drainage works	214,628 €
TASK 4.3	Waterproofing	1,745,839 €
TASK 4.4	Preparing Soil for Revegetation	159,008 €
TASK 4.5	Revegetation (sewing, planting, and other measures)	178,594 €
TOTAL TASK 4		**3,502,946 €**
TASK 5	**BENEFICIATION PLANT AND AUXILIARY FACILITIES**	
TASK 5.1	Earthworks	147,310 €
TASK 5.2	Trenching and construction of drainage works	4,680 €
TASK 5.3	Dismantling, demolition and removal of equipment	411,185 €
TASK 5.4	Preparing Soil for Revegetation	144,685 €
TASK 5.5	Revegetation (sewing, planting, and other measures)	71,711 €
TOTAL TASK 5		**779,571 €**
TASK 6	**WATER PONDS**	
TASK 6.1	Earthworks	728,992 €
TASK 6.2	Dismantling, demolition and removal of equipment	84,421 €
TASK 6.3	Preparing Soil for Revegetation	121,592 €
TASK 6.4	Revegetation (sewing, planting, and other measures)	35,793 €
TOTAL TASK 6		**970,798 €**
TASK 7	**OTHER AREAS**	
TASK 7.1	Dismantling, demolition and removal of equipment	244,930 €
TASK 7.2	Preparing Soil for Revegetation	274,703 €
TASK 7.3	Revegetation (sewing, planting, and other measures)	7,661 €
TASK 7.4	Enclosures and signage	38,600 €
TOTAL TASK 7		**565,894 €**
TASK 8	Maintenance Work	
TASK 8.1	Office Services	120,000 €
TASK 8.2	Facility Maintenace	44,500 €
TOTAL TASK 8		**164,500 €**
TASK 9	**MONITORING AND CONTROL TASKS**	
TASK 9.1	Revegetation work	382,950 €
TASK 9.2	Geotechnical monitoring	201,995 €
TASK 9.3	Water quality monitoring	311,500 €

GENERAL SUMMARY OF THE CLOSURE AND RESTORATION BUDGET BY ZONES/FACILITIES		
Code	**Description**	**Budget (€)**
TASK 9.4	Wildlife monitoring	80,000 €
TOTAL TASK 9		**976,445 €**
TASK 10	**OTHER GENERAL**	
TASK 10.1	Safety and health during closure	15,500 €
TOTAL TASK 10		15,500 €
TOTAL BUDGET FOR RESTORATION AND CLOSURE		**8,289,254 €**

21 CAPITAL AND OPERATING COSTS

Capital and operating costs used for Aguablanca were developed from Zero-Based cost build up from first principles engineering along with vendor and contractor estimates. In addition, all available project technical data, historical operating data, and metallurgical test work were considered to provide an accurate estimate for the restart. The estimated capital costs are listed in Table 21-1. Contingency was applied at a 10% rate to all initial and sustaining capital cost items to account for current market supply fluctuations.

Table 21-1 Aguablanca Estimate capital Cost (US dollars)

Capital Category	2024	2025	2026	2027	2028	2029	2030
Mine Development	438,856	5,584,911	4,044,446	3,852,279	3,903,673	2,394,044	701,633
Exploration/Delineation		968,000	1,232,000	484,000	594,000	352,000	220,000
Mobile Equipment	715,000	150,000	100,000				
Underground Infrastructure	490,000	1,580,000	2,500	2,500	2,500	22,500	
Surface Infrastructure	3,928,819		550,000	550,000			
Subtotal	**5,572,675**	8,282,911	5,928,946	4,888,779	4,500,173	2,768,544	921,633
Contingency	557,267	**828,291**	**592,895**	**488,878**	**450,017**	**276,654**	**92,163**
Total CAPEX	**6,129,942**	9,111,202	6,521,841	5,377,657	4,950,190	3,045,399	1,013,797

21.1 CAPITAL COSTS

21.1.1 MINE

Mine capital costs were developed based on mine design, current mine status, and required operational needs. The capital costs are estimated to bring the existing underground mine into production and provide access development to maintain production levels. Prior to commencing underground activities, the current pit lake must be pumped to the tailing storage facility. An estimate of USD$1.45 M is included in the estimate in Table 21-1 to complete this task.

21.1.2 PROCESS

Lain Technologies completed a study of the idle Aguablanca process plant during 2021 to determine capital costs necessary to return the plant to operating condition and reported labor and material costs totaling US$3,096,803. The Lain Technologies report provided the basis for current capital cost determination.

It was reported that the plant refurbishment project will require 21,020 contractor manhours spanning a period of 19 ½ weeks. A replica of the Lain Aguablanca plant project Gantt chart is presented in Figure 21-1.

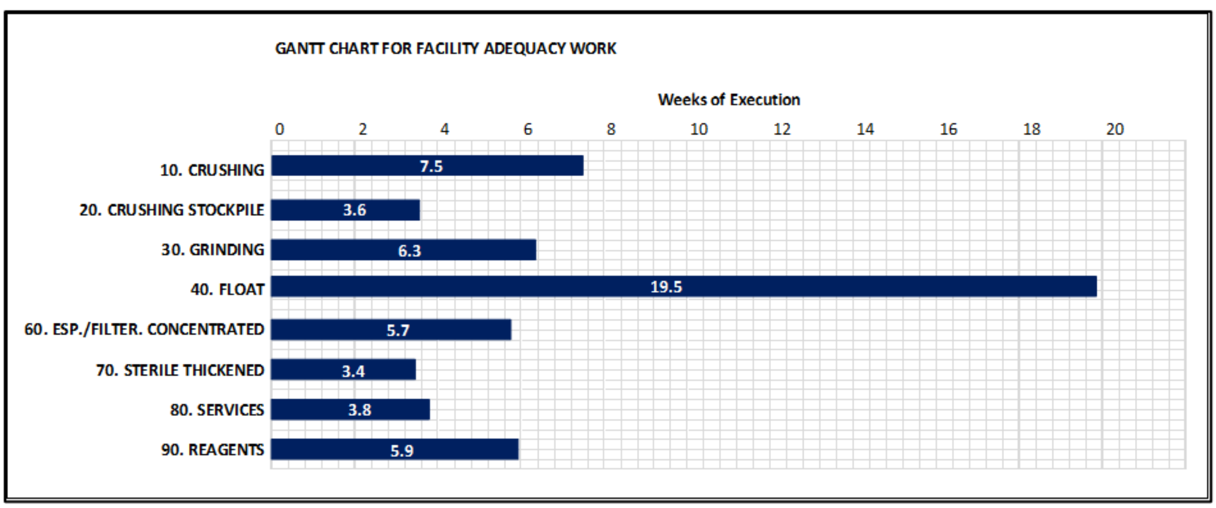


Figure 21-1 Aguablanca plant refurbishment project Gantt chart replica

According to the National Institute of Statistics, the average annual worker earning in Spain during 2021 was US$28,435 compared to 2024 average annual earnings of US$29,700, an increase of 4.5%. Lain report labor costs are compared to the same labor costs inflated by 4.5% in Table 21-2 with inflated contractor labor costs totaling US$831,840.

The Lain report did not include Aguablanca personnel labor costs. However, it is assumed that Aguablanca personnel will oversee all plant refurbishment work. The lower section of Table 21-2 includes one Aguablanca Project Manager and one Aguablanca Project Manager Assistant. Aguablanca personnel hourly costs are based on estimated Mill Manager and Mill Supervisor salaries shown in the Processing Plant Operating Costs section. Based on the inflated Lain contractor costs and Aguablanca personnel costs, current labor costs for Aguablanca process plant refurbishment total $931,662. Table 21-3 shows a breakdown of contractor manhours by process plant area.

Table 21-2 Lain Technology contractor labor costs compared to inflated contractor labor costs

Category	US$ per Manhour	Average Total Men	Distribution of Hours	Total Man Hours	2015 Total Labor Cost, US$	Inflated Labor Cost, US$
Team Leader	31.35	6.74	25%	5,255	149,768	156,485
Official	28.05	9.43	35%	7,357	187,604	196,018
Specialist/Assistant	25.30	10.78	40%	8,408	193,384	202,058
Overheads	20.00%				106,151	110,912
Contingency	25.00%				159,227	166,368
			Total Contractor	**21,020**	**796,133**	**831,840**
Aguablanca Project Manager	60.50	1		780	0	47,188
Aguablanca Project Manager Assistant	41.88	1		780	0	32,669
Contingency	25.00%				0	19,964
			Total Company	**1,560**	0	**99,822**
					796,133	**931,662**

Table 21-3 Contractor manhours breakdown by process plant area

Area	Manhours
10. CRUSHING	3,100
20. CRUSHING STOCKPILE	1,456
30. GRINDING	3,272
40. FLOAT	5,988
60. ESP./FILTER. CONCENTRATED	2,672
70. STERILE THICKENED	1,196
80. SERVICES	600
90. REAGENTS	2,736
TOTAL MANHOURS	21,020

The Lain Technologies report predicted a total of $2,019,143 for mill refurbishment Parts and Material costs. Revised Parts and Materials costs are forecast to total $1,445,157. Table 21-4 shows 2021 Lain Technologies Parts and Materials capital costs compared to updated 2024 Parts and Material costs.

Table 21-4 Lain Technologies 2021 Part and Materials capital costs compared to updated 2024 Parts and Material capital costs

MAIN SPARE PARTS	Lain Costs, US$	Updated Costs, US$
SAG Mill Liners Set	274,310	0
Ball Mill Liner Set	140,000	0
Crusher Jaw Wear Part	14,198	19,740
Secondary Crusher Wear Part	0	6,600
Pebble Crusher Wear Part	0	3,570
Subtotal Main Spare Parts	428,508	29,910
OTHER SPARE PARTS AND MATERIALS		
Other Spare Parts Subtotal	350,000	567,333
GREASE/OIL		
Grease Consumption	18,000	21,964
Oil Consumption	4,500	5,491
Subtotal Grease & Oil	22,500	27,455
FIRST REFILLS		

MAIN SPARE PARTS	Lain Costs, US$	Updated Costs, US$
Balls	334,490	578,557
Reagents	316,250	136,607
Greases & Oils	4,340	5,295
Subtotal First Refills	655,080	720,459
GENERAL CONTINGENCY 25%		
Contingency Subtotal	563,055	0
LABORATORY EQUIPMENT		
Estimated laboratory equipment	0	100,000
Total Capital Material Costs	**2,019,143**	**1,445,157**

Both the SAG and ball mills will require relining prior to plant startup. Thus, Lain Technologies included SAG and ball mill liner costs in their capital cost assessment. However, full sets of liners for both mills ordered prior to the 2015 process plant shutdown are on site so mill liners will not have to be purchased to reline the grinding mills prior to restarting the plant. Liners for the jaw, secondary crusher cone, and pebble crushers are also already on site but new liners should be purchased before the plant goes into production so replacement liners are available when needed after startup. Therefore, the first replacement crusher liner costs are included as a capital cost.

Lain estimated a total of $350,000 for Other Spare Parts. Although the Consumer Price Index inflation rate from 2021 to 2024 is 15.8%, a more conservative inflation factor of 17.9% was used to adjust the Lain Technologies Other Spare Parts costs. In addition, a 25% contingency was applied to update the Lain Other Spare Part capital cost estimate.

Lain applied a 25% contingency to the total capital costs estimate. The contingency was applied here only to the Other Spare Parts category since recent supplier cost proposals were used for all other Parts and Materials costs. Thus, while the Lain contingency totaled US$563,055, the updated parts and materials contingency totals only $154,727 and is included with the revised Other Spare Part cost.

Lain grease and oil costs were inflated by 10.9% based on Macrotrends petroleum inflation data for 2021 to 2024. The Lain First Refill Greases and Oils costs were also inflated by 10.9%.

Grinding media charges were left in the SAG and ball mills from December 2015 plant shutdown until 2018. During that time, based on manufacturers' recommendations, the mills were rolled periodically to prevent trunnion bearing damage. The charges were removed from both mills in 2018 and sold. Therefore, new charges must be purchased prior to startup. Lain forecast a capital cost totaling US$334,490 to replace the grinding media charges. Magottauex recently submitted a grinding media cost of $1,381 per metric tonne for both SAG and ball mill grinding media. The updated grinding media cost includes not only the costs to recharge the SAG and ball mills but also the costs for sufficient grinding media inventory to last three months of operation to compensate for grinding media delivery time.

Lain Technologies First Refill Reagent costs were reduced from the US$316,250 2021 cost to US$136,607. The revised Reagents cost will give Aguablanca sufficient reagent inventories to compensate for delivery times after the plant is restarted.

As mentioned above, Lain applied a 25% contingency to all Parts and Materials costs with a total contingency of US$563,055 while the only contingency applied to the revised Parts and Materials costs were a 25% addition to the Other Spare Parts cost.

Finally, Lain did not specifically list costs necessary to replace equipment removed from the Metallurgical Laboratory since the plant was shut down. An estimated US$100,000 was included for the purchase of metallurgical lab equipment.

21.2 OPERATING COSTS

A summary of the key operating costs, cash costs and AISC are presented below.

Table 21-5 Operating, Cash and AISC Costs

Operating Costs	LOM (US$M)	Per Lb Nickel (US$)
Mining	167.9	3.89
Processing	104.2	2.41
Site administration and social programs	30.8	0.71
Transport	0.3	0.01
Total operating costs	**303.2**	**7.02**
Less: by-product credits for copper, gold, platinum, palladium	(164.9)	**(3.82)**

Total cash costs (**)	**138.2**	**3.20**
Capital and exploration	**36.2**	**0.84**
All-in sustaining costs (**)	**174.3**	**4.04**

Note: Cash costs and all-in sustaining costs ("AISC") per lb of nickel are non-IFRS measures and are computed on a by-product credit basis whereby the net revenue from the sale of copper, gold, platinum and palladium are deducted from operating costs to derive the cash costs. AISC represents the sum of cash costs and capital and exploration costs. Cash costs and AISC are divided by the payable nickel produced to derive the per unit measures

21.2.1 MINING

Operating unit costs are based on first principal engineering buildups and bench marked against similar mining operations. These costs were derived for materials, power, fuel consumption, and labor for each operational unit. Labor costs for the mine were estimated based on historical actual labor costs plus adjustments for inflation. Consumables and fuels are based on vendor quotes; equipment rental fees were primarily based on replacement estimates. Operating costs are listed in Table 21-6. Figure 21-2 projects total site mine costs as currently scheduled.

Table 21-6 Aguablanca Underground Mining Unit Operating Costs

Operating Costs	Unit	Unit Cost USD$
Waste Development	US$/meter	2,234.50
Sublevel Development	US$/tonne ore	43.79
Sublevel/LHOS Extraction	US$/tonne ore	26.39
CRF Backfill (LHOS Only)	US$/tonne ore	14.00
General and Administration	US$/tonne ore	69.5
Concentrate Handling	US$/tonne ore	0.70
UG Mining Unit Cost (w/ Contingency)	**UIS$/tonne ore**	**39.27**

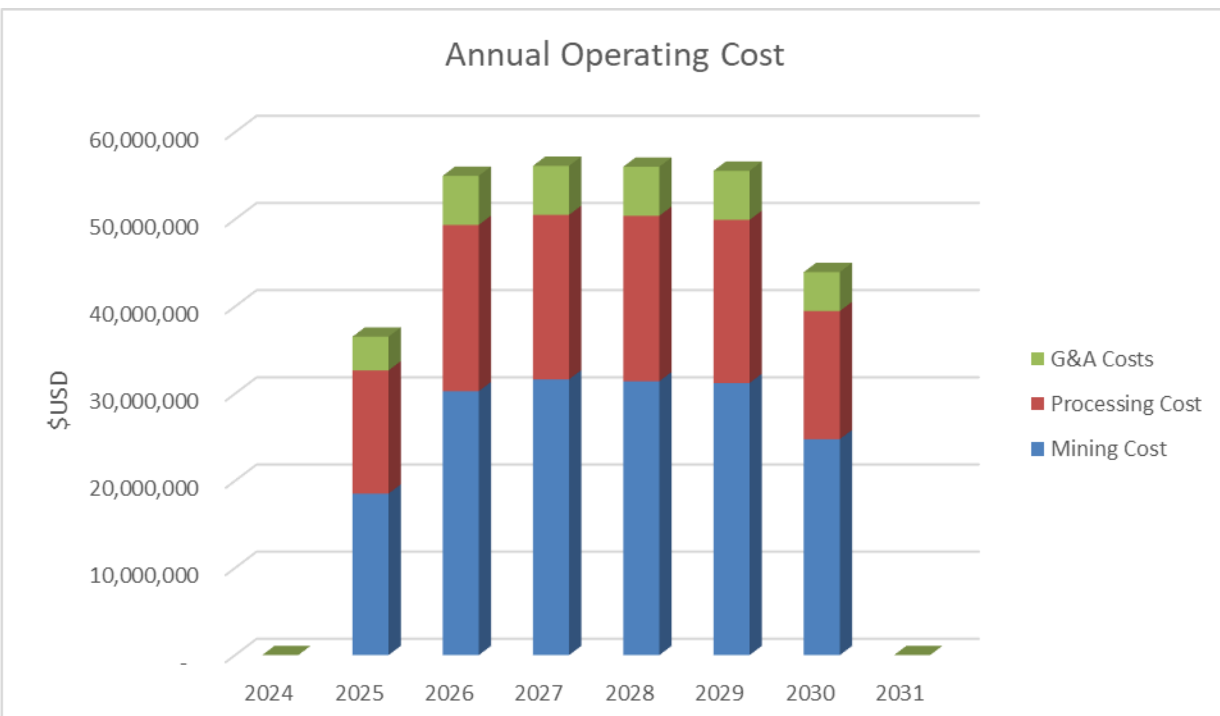


Figure 21-2 Site Operating Costs

21.2.2 PROCESS PLANT

An Aguablanca processing plant budget was prepared using the zero-base method for major operating costs and inflated average 2013, 2014, and 2015 operating costs per tonne milled for lower cost items. A forecast operating costs summary is presented in Table 21-7.

Table 21-7 Aguablanca processing plant operating budget (US dollars)

Element	2025-2030 Total, $	% of Total	2024 $/t	2014 $/t	Variance, $	Variance, %
Ore tonnes Processed	4,807,181					
W&S parts Milling area	5,764,646	5.81	1.20	1.09	0.11	109.8
W&S parts Crushing area	1,748,672	1.76	0.36	0.36	0.00	100.1
W&S parts Flotation area	1,991,421	2.01	0.41	0.40	0.02	103.8
W&S parts Other areas	2,444,619	2.46	0.51	0.36	0.14	139.7
Grinding balls	7,808,228	7.87	1.62	1.12	0.51	145.4
Other supplies & consumables	1,129,861	1.14	0.24	0.03	0.21	825.6
Fuel & lubes	1,035,200	1.04	0.22	0.10	0.12	223.0
Reagents	15,180,493	15.30	3.16	2.64	0.51	119.4
Lab assays	310,944	0.31	0.06	0.04	0.03	178.2
Sample preparation	213,807	0.22	0.04	0.04	0.01	126.0
Rentals	781,098	0.79	0.16	0.12	0.04	131.8
Maintenance	7,654,737	7.71	1.59	1.41	0.18	112.9
Consultants	589,176	0.59	0.12	0.12	0.00	104.2
Transport	118,905	0.12	0.02	0.01	0.01	186.6
Insurances	1,194,089	1.20	0.25	0.19	0.06	134.0
Electricity	20,247,725	20.41	4.21	3.49	0.72	120.6
Supplies	997,441	1.01	0.21	0.10	0.11	203.4
Courier	48,904	0.05	0.01	0.01	0.00	116.6
Travel	129,457	0.13	0.03	0.02	0.01	175.7
Security	425,205	0.43	0.09	0.06	0.03	141.4
Customs	153,552	0.15	0.03	0.02	0.01	146.6
Ore rehandling	3,593,037	3.62	0.75	0.61	0.14	122.7
Other services Milling area	266,955	0.27	0.06	0.03	0.03	207.1
Other services Crushing area	483,413	0.49	0.10	0.04	0.06	275.8
Other services Flotation area	9,290	0.01	0.00	0.00	0.00	221.0
Taxes	253,078	0.26	0.05	0.04	0.02	142.9
Central Office Charge	241,793	0.24	0.05	0.03	0.02	177.4
Safety	133,240	0.13	0.03	0.03	0.00	107.3
Training	220,702	0.22	0.05	0.02	0.02	190.3
Medical Services	65,492	0.07	0.01	0.01	0.00	116.5
Others	797,194	0.80	0.17	0.12	0.05	141.6
AB PITE	472,244	0.48	0.10	0.06	0.04	160.2
Personnel costs	22,716,226	22.89	4.73	3.76	0.97	125.8
TOTAL AB plant costs	**99,220,843**	**100.00**	**20.64**	**16.46**	**4.18**	**125.4**

Zero-base plant operating cost components include grinding balls, reagents, electrical power, and personnel costs. Mill and crusher area wear parts costs are a combination of zero-base costs for liners and inflated historic costs for other wear parts. Zero-base operating costs make up 70% of the 2025 through 2030 process plant operating coats.

Grinding ball and reagent costs are based on 2014 consumption rates and supplier price proposals. Table 21-8 presents a summary of these grinding ball and reagent consumption rates, price sources, and proposed unit costs. Costs shown in Table 21-8 include estimated shipping costs.

Crusher and grinding mill liner costs are also based on recent supplier cost proposals. Presented in Table 21-9 is a summary of liner components, replacement intervals in tonnes milled, price source, liner costs, and liner costs per tonne milled. All liner costs include costs for hardware and accessories. Note that the jaw crusher liner costs do not include shipping costs.

Table 21-8 Grinding ball and reagent consumption rates, price sources, and unit costs

Consumable	Consumption Rate, kg/t processed	Price Source	Price, US$/kg	Price, US$/tonne processed	Total US$/tonne processed
Balls 125 mm	0.604	Magotteaux	1.38	0.83	1.62
Balls 60 mm – 63 mm	0.572	Magotteaux	1.38	0.79	
Dowfroth 250 Frother	0.066	Quadra Chemicals (SNF)	3.47	0.23	3.16
CMC MF 4000 Depressant	0.756	Hebei Double Bulls Cellulose Co. Ltd.	2.86	2.16	
N-8105 Coagulant	0.012	Quadra Chemicals (SNF)	2.51	0.03	
N-71771 Floc (SNF 920 SH)	0.003	Quadra Chemicals (SNF)	4.29	0.01	
N-9601 Floc (SNF 912)	0.027	Quadra Chemicals (SNF)	4.29	0.12	
Promoter Flomin C4132	0.004	Quadra Chemicals (SNF)	3.93	0.01	
Collector Flomin C3430	0.125	Quadra Chemicals (SNF)	3.70	0.46	
Copper sulfate	0.030	Chemical Montana	4.50	0.14	

Table 21-9 Crusher and mill liner consumption rates, price sources, costs, and unit costs

Unit	Component	Change Interval, tonnes processed	Price Source	Liner Cost, US$	Cost, US$ per tonne processed
Jaw Crusher	Cheek plt LOW C160B 471543@ / Lower Side Armor	402,119	Metso	1,826	0.049
	Cheek plt C160@ / Upper Side Armor			2,310	
	Jaw, MVBL fixed quarry 2P@ / Movable Jaw			6,980	
	Jaw, fixed quarry 2P@			8,624	
5 1/2-Foot Symons Cone	Bowl and Mantle Liners	737,218	Columbia Steel Castings	6,600	0.009
HP200 Metso Cone	Bowl and Mantle Liners	76,908	Columbia Steel Castings	1,870	0.024
CH 430 Sandvik Cone	Bowl and Mantle Liners	76,908	Columbia Steel Castings	1,870	0.024
SAG Mill	Bottom Plate X - feed end	1,340,397	Magotteaux	54,552	0.083
	Bottom Plate Y- - feed end			38,592	
	Bottom Plate Z - feed end			18,239	
	Ferrule Plate A - last shell rows	670,199		54,485	0.249
	Ferrule Plate B - last shell rows			56,003	
	B1 Splint Plate - last shell rows			56,340	
	Splint C-Plate - last shell rows	1,742,516		135,877	0.211
	Ferrule Plate D - last shell rows			55,496	
	D1 Ferrule Plate - last shell rows			114,586	
	Splint Plate E - last shell rows			61,899	
	85 x 120 S/Flat YA/EA/OA-2131/1 Y Y External Outlet Grid Plate - frame	1,072,318		23,611	0.130
	External Outlet Grille Plate Y1 Hole, 80 x 120 s/flat AA/EA/OA-2205/1 rev. b - frame			24,594	
	External Outlet Grid Plate Y2 85 x 120 S/Flat AA/EA/OA-2224/1 - frame			23,999	
	Grid plate for internal pos outlet 6+4+2 x 50 sec/flat hole, AA/EA/OA-2131/1 X rev. c - frame			13,917	
	Inner Outlet Grille Plate X2 s/flat AA/EA/OA-2225/11			13,263	

Unit	Component	Change Interval, tonnes processed	Price Source	Liner Cost, US$	Cost, US$ per tonne processed
	Lifter bars pos. K as flat BA/EA/EB-21 (1) rev.A Lifter bar pos. K mod. as flat BA/EA/EB-21 mod.			19,717 20,668	
Ball Mill	Plates Lifters	1,608,477 804,238	Polycorp	17,030 185,541	0.011 0.231

Electrical power cost was calculated based on the Spain electrical power index. Historic Spanish price indices are shown in Figure 21-3. Monthly power indices for 2015 shown in Table 21-10 average US$56.452 per kWh while the March 20, 2024 power index was US$62.634, a 10.95% increase over the 2015 average.

Actual 2015 Aguablanca process plant electrical power cost averaged US$0.084 per kWh. To calculate current process plant power price, US$0.084 was increased by 10.95% to US$0.093 per kWh. Note that a contract is yet to be negotiated with the electrical power provider so the power price shown here may vary from the negotiated power price.

Figure 21-3 Historic Spain electrical price index chart

Table 21-10 Spain Power Indices

Month	Euro/kWh	US$/kWh
Jan-15	33.56	36.92
Feb-15	39.64	43.60
Mar-15	43.49	47.84
Apr-15	53.77	59.15
May-15	54.62	60.08
Jun-15	60.94	67.03
Jul-15	55.64	61.20
Aug-15	63.52	69.87
Sep-15	56.01	61.61
Oct-15	46.77	51.45
Nov-15	60.32	66.35
Dec-15	47.56	52.32
Average	51.32	56.45
Chart 3/20/24	56.94	62.63
2015 to 2024 Increase		10.95%
2015 Price, US$/kWh		$0.084

2015 Price, US$/kWh	$0.093

Historic data shows that the Aguablanca process plant consumes on average 43.39 kWh electrical power per tonne milled while in operation. Therefore, for each month, that month's tonnage was multiplied by 43.39 to determine the month's operating power consumption. Power will also be consumed during startups and shutdowns even with no feed reporting to the grinding mills. Based on equipment motor sizes, each startup and shutdown is estimated to consume an average 2,653 and 4,950 kWh's of electrical power, respectively.

Some equipment, for example some agitators, will remain running while the mill is down. Again, based on equipment motor sizes, it is estimated that 184 kWh's of electrical power will be consumed each hour by running equipment while the process plant is shut down. Additionally, based on site power consumption while the plant is currently idle, an additional 87 kWh's per hour will be consumed during downtimes by additional electrically dependent sources.

Power consumptions were calculated for each month from January 2025 until the plant is shutdown in October, 2030. Table 21-11 shows power consumptions and costs for 2026.

Table 21-11 2026 Electrical power consumption and cost by month

Month	Tonnes Processed	Run Hours	Down Hours	Start Ups	Shut Downs	Run kWh	Down kWh	Start Up kWh	Shut Down kWh	Total kWh	Cost, US$
Jan-26	72,099	332	412	4	5	3,128,375	111,704	10,611	24,750	3,275,441	276,194
Feb-26	72,099	360	312	4	4	3,128,375	84,592	10,611	19,800	3,243,378	273,490
Mar-26	72,099	358	386	5	4	3,128,375	104,655	13,264	19,800	3,266,095	275,406
Apr-26	72,900	404	316	4	5	3,163,133	85,676	10,611	24,750	3,284,170	276,930
May-26	72,900	312	432	4	4	3,163,133	117,127	10,611	19,800	3,310,671	279,165
Jun-26	72,900	406	314	5	4	3,163,133	85,134	13,264	19,800	3,281,331	276,691
Jul-26	73,701	356	388	4	5	3,197,895	105,197	10,611	24,750	3,338,454	281,507
Aug-26	73,701	382	362	5	4	3,197,895	98,148	13,264	19,800	3,329,107	280,719
Sep-26	73,701	360	360	4	4	3,197,895	97,606	10,611	19,800	3,325,912	280,450
Oct-26	73,701	332	412	4	5	3,197,890	111,704	10,611	24,750	3,344,956	282,056
Nov-26	73,701	382	338	5	4	3,197,890	91,641	13,264	19,800	3,322,595	280,170
Dec-26	73,701	380	364	4	5	3,197,890	98,690	10,611	24,750	3,331,942	280,958

Annual labor costs were extrapolated using 2015 labor cost compared to labor costs for each Aguablanca employee position. Labor costs are shown in Table 21-12.

Table 21-12 Annual labor cost summary

Position	Number	Cost per Person, US$	Total Cost, US$
Mill Manager	1	132,128	132,128
Mill Foreman	1	91,473	91,473
Mill Crew Foreman	2	81,309	162,619
Mill Operators	16	51,258	820,130
Reserve Operators	5	50,818	254,092
Reagents & Plant Work	2	50,818	101,637
Weekend Supervisors	2	73,178	146,357
Weekend Operators	2	46,132	92,265
Commercial Supervisor	1	86,391	86,391
Commercial Operator	1	50,818	50,818
Mechanical General Supervisor	1	91,473	91,473
General Supervisor Assistant	1	81,309	81,309
Mechanical Supervisor	2	81,309	162,619

Position	Number	Cost per Person, US$	Total Cost, US$
Mill Mechanics	12	55,900	670,802
Electrical Supervisor	1	86,391	86,391
Electricians	4	55,900	223,601
Lab Foreman	1	76,228	76,228
Sample Prep Supervisor	1	60,982	60,982
Technicians	3	45,737	137,210
Metallurgical Technicians	1	45,737	45,737
Assayer Supervisor	3	60,982	182,946
Technicians	3	45,737	137,210
Totals	**66**		**3,894,415**

22 ECONOMIC ANALYSIS

The economic performance of the Aguablanca Mine was evaluated utilizing a cash flow based economic model using project costs and revenues as the financial basis. The revenue factors for the project are dependent on the metal prices, operating costs, mill recoveries, and the commercial smelter terms. Key parameters and assumptions are shown in Table 22-1.

Table 22-1 Key Economic Parameters and Assumptions

Key Parameters	PFS
Total LOM Production, t	4,807,182
Process Rate, tonnes/month	71,667
Total LOM Waste Production, t	651,575
Diluted LOM Grades	
Nickel Grade, %	0.656
Copper Grade, %	0.592
Palladium Grade, g/t	0.283
Platinum Grade, g/t	0.327
Gold Grade, g/t	0.163
LOM Process Recoveries	
Nickel, %	82.8%
Copper, %	93.6%
Palladium, %	75.0%
Platinum, %	75.0%
Gold, %	75.0%
PAYABLE Metal	
Nickel, t	19,602
Copper, t	15,704
Palladium, oz	13,144
Platinum, oz	15,092
Gold, oz	7,205
Projected Metal Prices	
Nickel, USD$/lb	7.30
Copper, USD$/lb	3.50
Palladium, USD$/oz	1,200.00
Platinum, USD$/oz	900.00
Gold, USD$/oz	2,000.00
Financials	
LOM Gross Payable Revenue, USD$000	480,295
LOM Cash Flow Post Tax (Free), USD$000	105,720
LOM Cash Flow Post Tax (5), USD$000	83,098
LOM Cash Flow Post Tax (8), USD$000	72,442
LOM Cash Flow Post Tax (10), USD$000	66,292
IRR Post Tax	213%
Payback Period, years	1.2

22.1 LOM OPERATING AND FINANCIAL DATA

A summary of the key operating and financial metrics over the approximately 6-year mine life of the Aguablanca Project according to the PFS is set out in Table 22-2 below.

Table 22-2 LOM Operating and financial data

Year	Production [3]		Net Revenue[4]	Operating Costs[5]	EBITDA [6]	Capex & Exploration	Income Taxes	Project Cash Flow	AISC per lb Nickel [7]
	Nickel	Copper							
	Klbs		US$ Millions						US$
2024 [2]	-	-	-	-	-	6.1	-	(6.1)	-
2025	5,857	4,256	61.8	36.5	25.3	9.1	5.8	10.4	4.54
2026	7,932	6,035	86.6	55.0	31.6	6.5	6.9	18.2	4.14
2027	7,931	5,924	86.5	56.2	30.3	5.4	6.1	18.8	4.15
2028	7,842	6,724	89.7	56.1	33.6	5.0	6.6	22.0	3.64
2029	7,858	6,913	90.5	55.5	35.0	3.1	6.4	25.5	3.24
2030	5,784	4,760	65.2	43.9	21.3	1.0	3.4	16.9	3.79
Total	**43,204**	**34,612**	**480.3**	**303.2**	**177.1**	**36.2**	**35.2**	**105.7**	**4.04**

Notes:

1. All figures are rounded to reflect the relative accuracy of the estimate.
2. Activities and spending in 2024 focus on the restart of the processing plant and de-watering of the open pit to gain access to the underground mine workings. Development commences in the underground mine in late 2024.
3. Production represents payable quantities of nickel and copper from the sale of concentrates. Production (not shown in Table 3) will also include payable quantities of gold, platinum and palladium.
4. Net revenue is based on expected long-term prices of US$7.30/lb for Ni, US $3.50/lb for Cu, US$2,000/oz for Au,US $900/oz for platinum and US$1,200/oz for palladium, and is shown net of refining and treatment charges.
5. Refer to Table 21-5.
6. EBITDA is a non-IFRS measure and is calculated as net revenue minus operating costs.
7. Refer to Table 21-5.
8. AISC is a non-IFRS measure and is calculated on a by-product credit basis by deducting revenue from copper, gold, platinum and palladium from the sum of operating costs and capex and exploration, divided by the number of nickel pounds produced. Refer also to Table 21-5.

The Aguablanca Mine Plan annual operational physicals are displayed in Table 22-3.

Table 22-3 Mine Plan Operating Physicals

Denarius Metals Corporation Aguablanca - Mine Plan Physicals		2024	2025	2026	2027	2028	2029	2030
HORIZONTAL DEVELOPMENT	w x h							
Primary Drift Rehabilitation,m		2,000	1,400	-	-	-	-	-
Advance Rate/Date, m/day		13	12	-	-	-	-	-
Primary Declines, m	4.5 x 4.5	80.0	480.0	400.0	400.0	400.0	380.0	-
Advance Rate/Date, m/day		2.6	1.3	1.1	1.1	1.1	1.0	-
Primary Cross Cuts, m	4.5 x 4.5	50.0	1,800.0	1,200.0	1,200.0	1,200.0	600.0	314.0
Advance Rate/Date, m/day		1.6	4.9	3.3	3.3	3.3	1.6	0.9
Primary Ventilation Cross Cuts, m	4.5 x 4.5	60.0	181.0	178.0	92.0	115.0	61.0	-
Advance Rate/Date, m/day		1.9	2.0	1.0	1.0	1.3	0.2	-
Miscellaneous	4.5 x 4.5	6.4	38.4	32.0	32.0	32.0	30.4	-
Advance Rate/Date, m/day		0.2	0.1	0.1	0.1	0.1	0.1	-
Sublevel Development	5.0 x 5.0	-	548.9	630.0	557.0	619.2	586.7	365.6
Advance Rate/Date, m/day		-	2.6	1.7	1.5	1.7	1.6	1.0
Total Horizontal Development, m		**196.4**	**3,048.3**	**2,440.0**	**2,281.0**	**2,366.2**	**1,658.1**	**679.6**
Advance Rate/Date, m/day		**6.3**	**8.3**	**6.7**	**6.2**	**6.5**	**4.5**	**1.9**
Total Waste Produced, t		13,589	172,938	125,238	119,287	120,878	74,132	21,726
Waste Rate, t/day		438	474	343	327	331	203	60
UNDILUTED STOPE PRODUCTION								
Sublevel Drilling			57,607	56,750	46,759	48,404	49,106	38,136
Drilling Rate, m/day			157	156	128	133	135	104
Sublevel Stope, tonnes			499,824	492,389	405,704	419,972	426,069	330,882
Production Rate, t/day			1,364	1,352	1,112	1,150	1,167	907
LHOS Drilling			5,385	36,560	47,224	45,005	44,603	37,151
Drilling Rate, m/day			35.1	99.9	129.4	123.3	122.2	102
LHOS Production, tonnes			46,720	317,211	409,737	390,488	386,997	322,341
Production Rate, t/day			305	867	1,123	1,070	1,060	883
Total Drilling			**62,992**	**93,310**	**93,983**	**93,409**	**93,709**	**75,287**
Drilling Rate, m/day			**172**	**256**	**258**	**256**	**257**	**206**
Total Stope Production, t			**546,544**	**809,601**	**815,441**	**810,460**	**813,066**	**653,223**
Stope Production Rate, t/day			**1,491**	**2,218**	**2,234**	**2,220**	**2,228**	**1,790**
Production from Sublevel Development			**43,911**	**50,400**	**44,559**	**49,539**	**46,934**	**29,245**
Total Production			**590,455**	**860,000**	**860,000**	**859,999**	**860,001**	**682,468**
Total Production Rate, t/day			**1,610**	**2,356**	**2,356**	**2,356**	**2,356**	**1,870**
TOTAL MATERIAL MOVED								
Waste		13,589	172,938	125,238	119,287	120,878	74,132	21,726
Average Loaded Haul,m (One Way)		2,400	2,700	3,170	3,570	3,820	3,300	3,300
Ore		-	590,455	860,000	860,000	859,999	860,001	682,468
Average Loaded Haul,m (One Way)		-	2,000	2,300	2,600	3,050	3,300	3,300
Cemented Rock Fill		-	51,661	330,373	436,628	414,192	410,461	336,103
Average Loaded Haul,m (One Way)		-	2,000	2,300	8,450	3,050	3,300	3,300
Total Tonnes Moved		15,989	871,454	1,337,491	1,440,595	1,422,549	1,351,194	1,046,897
Tonnes Moved Rate, t/day		516	2,379	3,662	3,947	3,898	3,702	2,868

22.2 CASH FLOW

The production schedules presented in Sections 16 and 17 have been used in conjunction with the cost data discussed in Section 21 to create a model for the Aguablanca Mine economic performance. Costs are in constant 2024 US dollars, no escalation of cost has been assumed. Operating costs are generated based on production physicals (tonnes) and unit rates. The detailed cash flow model for the Aguablanca Mine is presented in Table 22-4.

Table 22-4 Cash Flow Summary

Cash Flow, USD$	LOM (USD$000)	2024	2025	2026	2027	2028	2029	2030	2031
Revenue	480,295	0	61,829	86,564	86,463	89,675	90,524	65,240	0
OPEX	303,185	0	36,531	54,989	56,134	56,027	55,560	43,943	0
Royalties	0	0	0	0	0	0	0	0	0
Working Capital Adjustments	36,150	0	2,064	4,183	5,631	7,377	9,145	7,750	0
CAPEX	(36,150)	(6,130)	0	0	0	0	0	0	0
Sustaining CAPEX	0	0	(9,111)	(6,522)	(5,378)	(4,950)	(3,045)	(1,014)	0
Tax Paid	(35,240)	0	(5,809)	(6,848)	(6,174)	(6,568)	(6,455)	(3,387)	0
NET Cashflow	**105,720**	**(6,130)**	**10,378**	**18,205**	**18,777**	**22,130**	**25,463**	**16,896**	**0**

22.3 TAXES, ROYALTIES AND OTHER INTERESTS

Tax calculations in the financial model are based on current Mine Tax laws in Spain which are 25%. Payable royalties and Mine Tax details for the project are outlined in Table 22-4.

22.4 SENSITIVITY ANALYSIS

Metal Price and Operating/Capital Cost sensitivities have been calculated with respect to the after tax NPV and IRR. The results demonstrate little effect in regards to Capital Costs variances; however, Operating Costs have a significant effect on the NPV. As expected, movement in Metal Prices has a noticeable effect on the NPV. Table 22-5 summarize the after-tax sensitivities of NPV and IRR, with respect to metal prices and costs.

Table 22-5 Sensitivity Analysis

NPV (5%) ($M) — Metal Prices

Nickel Price ($/lb)	Copper Prices ($/lb) 2.80	3.15	3.50	3.85	4.20
6.70	53	60	67	75	82
7.00	61	68	75	83	90
7.30	69	76	83	90	98
7.60	76	84	91	98	106
7.90	84	91	99	106	113

NPV (5%) ($M) — Operating & Capital Costs

Total Capital Cost (+/-)	Operating Costs (+/-) -20%	-10%	0%	10%	20%
-20%	124	106	88	70	51
-10%	122	104	85	67	49
0%	120	101	83	65	47
10%	117	99	81	62	44
20%	115	97	78	60	42

IRR (%) — Metal Prices

Nickel Price ($/lb)	Copper Prices ($/lb) 2.80	3.15	3.50	3.85	4.20
6.70	136%	154%	171%	189%	207%
7.00	156%	174%	192%	210%	228%
7.30	177%	195%	213%	231%	249%
7.60	197%	215%	234%	252%	270%
7.90	218%	236%	254%	273%	291%

IRR (%) — Operating & Capital Costs

Total Capital Cost (+/-)	Operating Costs (+/-) -20%	-10%	0%	10%	20%
-20%	400%	343%	286%	229%	171%
-10%	346%	296%	245%	195%	144%
0%	303%	258%	213%	167%	122%
10%	268%	227%	186%	146%	58%
20%	239%	202%	165%	128%	91%

23 ADJACENT PROPERTIES

There are no adjacent properties, as defined by NI 43-101.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 SITE LOCATION AND CONDITIONS

Table 24-1 Site location and conditions

Localization		Near Real de la Jara, Province of Badajoz, Spain (SW)	
Elevation		495	Above sea level (m)
Temperature:	Maximum	45	ºC
	Maximum in January	11,5	ºC
	Maximum in July	34	ºC
	Minimum	-8	ºC
Rains:	Stocking	680	mm/a
	Maximum	1100	mm/a
	Rainy season	October - May	
	Average number of rainy days	70	D/A
Evaporation rate		1,280	mm/a
Wind:	Address	No predominant direction	
	Design Zone		W
	Exhibition area		Normal
Seismic conditions:	Acceleration Factor	<0.04	g
	Contribution coefficient (NCSE_02)	1.3	
	Earthquake Baseline operation (OBE)	0.01	g
	Maximum credible earthquake (MCE)	0.09	g
	Ground Bearing Pressure	250	kPa

24.2 MATERIAL CHARACTERISTICS

Aguablanca material characteristics are listed in Table 24-2

Table 24-2 Material Characteristics

Source/method of extraction of the ore			Cut in the open	
Mineral Blend		Breccia	20	%
		Disseminated	80	%
Nickel Head Grade		Average	0.57	%
		Range	0.2 – 1.5	%
Copper Head Grade		Average	0,44	%
		Range	0.2 – 0.8	%
Moisture Content		Design	4	% w/w
		Maxim	8	% w/w
Bulk density of the mineral		Specific Gravity of the Ore	3.1	t/m3
		Calculated Volume (Dry)	1.75	t/m3
		Calculated Weight (Dry)	1.86	t/m3
Compressive Strength		Breccia	143	Mpa
		Disseminated	149	Mpa
Crushing Work Index		Breccia	14	kWh/t
		Disseminated	22	kWh/t
Ball Mill Work Index		Breccia	13.45	kWh/t
		Disseminated	16.3	kWh/t
Abrasion Index		Breccia	0.24	
		Disseminated	0.53	
JKTech Break Parameters	Breccia	To	66.6	
		b	1.42	
		Axb	94.5	
		Ta	1.31	
	Disseminated	To	49.6 – 70.9	
		b	0.48 – 0.75	
		Axb	34.0 – 39.9	
		Ta	0.29 – 0.32	
Material Flow Properties		Critical 'Rat Hole' Diameter	5600	mm
		Angle of repose	35	Deg
Withdrawal angle		Above Critical Diameter	55	Deg
		Above Critical Diameter	80	Deg

24.3 GENERAL OPERATING PARAMETERS

General operating parameters are listed in Table 24-3.

Table 24-3 General Operating Parameters

	Maximum annual plant yield	1,905,000	T/A
	Monthly Plant Yield (Average)	158,745	T/M
	Public holidays	0	D/A
	Annual Operating Days	365	D/A
	Scheduled Maintenance	576	h/a
	Plant Utilization After Maintenance	96	%
	Annual Plant Operating Hours	7.844	h/a
	Plant Feeding Rate (Average)	235	t/h
	Plant Feed Rate (Design)	195	t/h
	Particle size in crushing (80% step)	150	mm
	Particle size in float (80% step)	80 - 100	Microns
	Particle size in whirlpool (80% step)	Do not whirl	Microns
	Nickel	77 - 83	%
	Copper	91 - 94	%
Average recoveries:	Cobalt	75	%
	Platinum	75	%
	Palladium	75	%

24.4 CRUSHING OPERATING PARAMETERS

Crushing operating parameters are shown in Table 24-4

Table 24-4 Crushing Operating Parameters

Annual Operating Days	273	D/A
Scheduled Schedule Per Day	24	H/D
Start-up and Shutdown	0.5	H/D
Hours of Operation per day	24	H/D
Utilization	75	%
Annual Hours of Operation	6570	h/a
Average Crushing Rate	300	t/h
Design Crushing rate	450	t/h

24.5 CRUSHED ORE STOCKPILE PARAMETERS

Crushed stockpile data is presented in Table 24-5.

Table 24-5 Coarse Ore Stockpile

COARSE ORE STOCKPILE		
Hours of Operation	7	days/week
Crusher downtime	Scheduled maintenance work only	
Crusher Non-Operational Hours	42	hours/week
Required Storage Capacity	206	h
Milling rate	236	t/h
Total Stockpile Capacity	50	t
Live Stockpile Capacity	12	t

24.6 PEBBLE CRUSHING OPERATING PARAMETERS

Pebble Crusher		Cone	
Number of pebble crushers installed		2	
CCS Crusher:	Minimal	8	mm
	Maximum	15	mm
Pebble Production Rate:	Average	70	t/h
	Maximum	110	t/h
Feed Size		150	mm
Moisture content		<1	%
Bulk density of pebbles:	No shredding	1.45	t/m³
	Crushed	1.86	t/m³
Crusher Work Rate		22	kWh/t

24.7 GRIND AREA OPERATING PARAMETERS

Table 24-6 SAG Mill Operating Parameters

Number of SAG Mills		1	
Mill Length		6	m
Mill Diameter		5.5	m
Engine Size		3,400	Kw
Sprocket power		3,000	Kw
Operating Speed:	Nominal	76.5	Critic %
	Rank	60 - 76.5	Critic %
Mill liners		Metal	
Shell drilling		To fit metal and polymers (Standard Liner & Bolts)	
Ball Loading:	Design	8	%
	Maximum	20	%
Total Ball Consumption		0.7	kg/t
Ball Diameter		125	mm
Total Load:	Design	28	%
	Maximum	30	%
Font Size:	D80	50 - 60	mm
	Maximum, D100	150	mm
Mill Discharge Solids		55	% w/w
Hours of Operation		7,844	h/a
Power Design, New Power		195	Dry T/H
Circulating load of pebbles		35	%
Opening the Grille		80x125 / 50x50	mm
Open Grid Area		10	%
Opening of the pebbles reception area		80x125 / 50x50	mm
Pebbles Reception Area Open Area		10	% Grid Opening

Table 24-7 SAG Mill Discharge screening Operating Parameters

Display Type		Horizontal Single Arm	
Screen Operation Type		Moist	Wet/Dry
Number of screens		1	
Screen Width		1870	mm
Screen Length		4800	mm
Rubber Panels Screen Cover Material		Gum Rubber	
Screen Cover Panel Size		300 x 570	mm
Opening Slot Shape		Slot	
Opening Slot Size		12x24 / 10x10	mm
Screen Feed Speed:	Average	320	t/h (dry)
	Maximum	364	t/h (dry)
Specific Gravity of Feed Solids		3.1	t/m³
Feeding Method		Slurry	
Solids content		55	%w/w
Dispersed water		50	m³/h
Feed Size Distribution:	12mm	75	%Intern
	6mm	65	%Intern

Table 24-8 Ball Mill Operating Parameters

Number of Ball Mills			1
Mill Length		7.75	m
Mill Diameter		5	m
Engine Size		3,400	Kw
Sprocket power		3,003	Kw
Operating speed (critical)		78	% Critical
Ball Loading:	Design	34	%
	Maximum	27	%
Consumption of balls		0.5	kg/t
Ball Diameter		60/63	mm
Rubber coating		Gum Rubber	
Feed Size:	D100	<25	mm
	D80	0.35-0.6	mm
Discharge Solids		70	%w/w
Hours of Operation		78,444	h/a
Feed rate, wet new feed		236	Dry T/H
Circulating Load:	Average	290-330	%
	Design	350	%
Output Size		80-100	Microns

Table 24-9 Cyclone Operating Parameters

Number of Hydrocyclones	1	
Number of Cyclones per Hydrocyclone Bank	5	
Number of Cyclones Operating	3	
Number of Replacement Cyclones per Hydrocyclone (installed)	1	
Number of Cyclone Connections in Bank	1	
Cyclone Diameter	500	mm
Vortex Diameter	170	mm
Spigot Diameter	100-110	mm
Cyclone Cone Ratings	20	Degrees
Cyclonic Coating	Replaceable rubber	
Operating Pressure	80_120	kPa
Feed Rate	917	Dry T/H
Feed Rate	1,112	m^3/h
Feed Density	53	%w/w
Feed Size	1,000	Microns
Output Density	30	%w/w
Output Size	80-120	Microns

24.8 FLOTATION OPERATING PARAMETERS

Flotation operating parameters are listed in Table 24-10. Note that talc pre-flotation is now the first stage of copper rougher flotation so pre-flotation data may no longer be reliable.

Table 24-10 Flotation Operating Parameters

TALC PRE-FLOTATION			
	Solid Feed Rate	195	t/h
	Slurry Feed Rate	518	m³/h
	Feed solids content	30	% w/w
	Residence Time	6	min
	Mass pull	4	% w/w
COPPER ROUGHER			
	Solid Feed Rate	187	t/h
	Slurry Feed Rate	504	m³/h
	Feed solids content	30	% w/w
	Residence Time	2	min
	Mass pull	6	% w/w
NICKEL ROUGHER			
	Solid Feed Rate	255	t/h
	Slurry Feed Rate	712	m³/h
	Feed solids content	29	% w/w
	Residence Time	12	min
	Mass pull	27	% w/w
NICKEL SCAVENGER			
	Solid Feed Rate	201	t/h
	Slurry Feed Rate	562	m³/h
	Feed solids content	29	% w/w
	Residence Time	11	min
	Mass pull	11	% w/w
PRIMARY NICKEL CLEANER			
	Solid Feed Rate	90	t/h
	Slurry Feed Rate	346	m³/h
	Feed solids content	22	% w/w
	Residence Time	17	min
	Mass pull	10	% w/w
NICKEL CLEANER SCAVENGER			
	Solid Feed Rate	79	t/h
	Slurry Feed Rate	261	m³/h
	Feed solids content	25	% w/w
	Residence Time	17	min
	Mass pull	13	% w/w
FINAL NICKEL CLEANER			
	Solid Feed Rate	37	t/h
	Slurry Feed Rate	163	m³/h
	Feed solids content	19.5	% w/w
	Residence Time	16.5	min
	Mass pull	7.7	% w/w
COPPER CLEANER			
	Solid Feed Rate	12	t/h
	Slurry Feed Rate	15	m³/h
	Feed solids content	22	% w/w
	Residence Time	20	min
	Mass pull	1.5	% w/w

24.9 CONCENTRATE DEWATERING

Concentrate thickening and filtering parameters are given in Table 24-11. As of 2015 when the process plant was shut down, copper concentrate was combined with nickel concentrate for thickening and filtering.

Table 24-11 Concentrate Dewatering Operating Parameters

NICKEL THICKENING			
	Thickener Type	High Capacity	
	Number of thickeners	1	
	Solid Feed Rate	16	t/h
	Feed solids content	26-38	% w/w
	Solids content of the sunk	65	% w/w
	Settlement	3,2	t/d/m²
	Thickener Diameter (Design)	14	M
NICKEL CONCENTRATE FILTRATION			
	Filter Type	Horizontal Pressure	
	Number of filters	2	
	Solid feed rate (each filter)	180	T/D
	Feed solids content	60-65	% w/w
	Moisture content of the filtered product	11	% w/w
	Filtering Rate	365	kg/h/m²
	Filter Area (Layout)	94	m²
COPPER THICKENING			
	Thickener Type	High Capacity	
	Number of thickeners	1	
	Solid Feed Rate	3	t/h
	Feed solids content	27	% w/w
	Solids content of the sunk	65	% w/w
	Settlement	3.2	t/d/m²
	Thickener Diameter (Design)	5	m
COPPER CONCENTRATE FILTRATION			
	Filter Type	Horizontal Pressure	
	Solid Feed Rate	72	T/D
	Feed solids content	65	% w/w
	Moisture content of the filtered product	8.5	% w/w
	Filtering Rate	365	kg/h/m²
	Filter Area (Layout)	33	m²
CONCENTRATE STORAGE			
	Concentrate Transport Schedule	5	Days/week
	Concentrate storage required	4	days
Concentrate storage required:	Nickel	1,800	t
	Copper	360	t

24.10 TAILING DISPOSAL OPERATING PARAMETERS

Table 24-12 Tailing Thickener Operating Parameters

Thickener Type	Deep Cone High Compression	
Solid Feed Rate	222	t/h
Feed solids content	30	% w/w
Solids content of the sunk	66-70	% w/w
Settlement	26	t/d/m²
Thickener Diameter (Design)	15	m

Table 24-13 Tailing Disposal Operating Parameters

Solids content 70%_75% w/w	Yield Stress	Tba	Pa
	Apparent viscosity @Pseudo shear rate of 100/s	150 - 450	mPa.s
Solids content 60%_70% w/w	Yield Stress	n/a	Pa
	Apparent viscosity @Pseudo shear rate of 100/s	40 - 150	mPa.s
Deposition Method	Sub-aqueous deposit (to be confirmed)		

24.11 SERVICES OPERATING PARAMETERS

Process plant services data are listed in Table 24-14. Note that when the Aguablanca processing plant was shut down in 2015, the copper thickener was no longer in use. Copper concentrate was combined with nickel concentrate in the nickel thickener.

Table 24-14 Services Operating Parameters

AIR			
	Plant Air Volume	3000	m^3/h FAD
	Plant Air Pressure	1000	kPa
	Instrument Air Volume	90	m^3/h @ 700kPa
	Instrument Air Pressure	700	kPa
RAW WATER			
	Source	Caja River	
	Volume	161	m^3/h
	Pressure	350	kPa
GLAND WATER			
	Source	Raw water	
	Volume	40	m^3/h
	Pressure	800	kPa
FIRE WATER			
	Source	Raw water	
	Volume	120	m^3/h
	Pressure	900	kPa
PROCESS WATER			
	Source	Waste thickener overflow	
Low Pressure	Volume	800	m^3/h
	Pressure	350	kPa
	Source	Waste thickener overflow	
High Pressure	Volume	100	m^3/h
	Pressure	600	kPa
NICKEL FLOTATION			
	Source	Nickel Concentrate Thickener Overflow	
	Volume	186	m^3/h
	Pressure	320	kPa
COPPER FLOTATION	Source	Copper Concentrate Thickener Overflow	
	Volume	19	m^3/h
	Pressure	200	kPa

24.12 REAGENT OPERATING PARAMETERS

Table 24-15 Reagents Operating Parameters

COLLECTORS		
Copper collector, S700 or equivalent	15	g/t
Nickel Collector, SIBX	120	g/t
Nickel Manifold, Aero 3477	4	g/t
DEPRESSANTS		
Pyrrhotite Depressant	n/a	g/t
Talc Depressant	1100	g/t
MODIFIERS		
pH Modifiers	n/a	g/t
FROTHERS		
Dowfroth	90	g/t

25 INTERPRETATIONS AND CONCLUSIONS

This report was prepared by a group of independent consultants, all Qualified Persons as defined by NI 41-101, to demonstrate the economic viability of restarting mining and processing operations, based on the estimated Mineral Reserves at Aguablanca. This report provides a summary of the results and findings to the level that should be expected for a pre-feasibility study. Standard industry practices and assumptions have been applied in this study.

Mineral Resources meet the reasonable prospects of eventual economic extraction due two main factors: 1) cut-off grades are based on scientific data and assumptions related to the project, and 2) Mineral Resources are estimated only within geology shapes derived by the scientific data as well as by using generally accepted practices. Confidence in the Mineral Estimate was used to classify Mineral Resources based upon drill hole spacing and geological knowledge of the deposit.

Mineral Resources were converted to Mineral Reserves by developing a mining plan for the deposit using only Measured and Indicated Mineral Resources. Modifying factors in the determination of Mineral Reserves included detailed mine planning, dewatering and mining and processing costs for the production of nickel and copper concentrates.

Mineralization consists of magmatic sulphides within the gabbro and gabbro-norite units. Sulphide mineralization is predominantly represented by pyrrhotite, pentlandite, chalcopyrite and pyrite with minor marcasite and covellite. Magnetite is often present. Lesser amounts of platinum group minerals (PGM) and gold are also associated with the sulphide minerals.

Mineralization is generally hosted within a chaotic magmatic breccia, rich in fragments, in the matrix of which can be found semi-massive sulphides. Mineralization often grades laterally into more continuous disseminated sulphides and irregular patchy mineralization with variable textures. The mineralized body consists of two sub-parallel zones of mineralization, elongated in an east west direction and dipping sub-vertically. The larger South or Main Zone can be traced for 400 metres along an N115º orientation dipping between 75º and 80º to the north. This zone is over 100 metres thick and extends to a depth of more than 600 metres. The mineralization has a plunge ofN65º to N80ºE. The North Zone contacts with the surrounding metamorphic country rocks and strikes N115º and is approximately 125metres long in the strike direction, 50 metres thick and extends to a depth of more than 300 metres.

The MRE is based on 496 diamond drillholes containing 25,025 assay intervals. Drilling includes two exploration holes which were recently drilled in 2022. Outlier grades were capped prior to compositing to 24,250 two-meter intervals. Nickel, copper and cobalt mineralization was estimated using ordinary kriging techniques based on detailed variography analysis of the mineral deposit. Gold, platinum and palladium mineralization was interpolate using inverse distance estimation techniques. Three-dimensional geology models were constructed to identify the mineralized domains of the mineral deposit. Mineralization is constrained geologically to the mineralized domains to accurately reflect in situ mineralization. The MRE was completed using Vulcan scientific software in a 3D block model, with blocks ranging from 4x4x4 meters down to 2x2x2 meters which is a size reflective of the selective mining unit envisioned for underground mining of the deposit. These blocks demonstrate the reasonable prospects for eventual economic extraction. All other material, smaller than the SMU, was reported as non-mineralized material.

Aguablanca Mineral Resources have been re-estimated independently and confirm the past Mineral Resource estimates for the Project. Nearly 89% of the Measured and Indicted Mineral Resources have been converted to Proven and Probable Reserves. The stated Mineral Reserves provide a suitable basis for the existing processing plant to recommence the production of nickel and copper concentrates. The underground mine will be contractor operated. Two mining methods will be implemented. Up-hole Sublevel Extraction (“USE”) will be employed for the upper portion of the deposit. This will essentially extract mineralization from the 75 level up to the bottom of the pit, caving the pit bottom and exposing a void. The void will be back-filled to the current pit bottom of 181 msl as required by the current EIS. Below the 75 level, Long Hole Open Stoping (“LHOS”) will be the primary mining method. Stopes will be supported by cement backfill, with the cement being produced onsite.

Aguablanca process plant design calls for a dry feed throughput rate of 195 tph or approximately 5,000 dry tpd. However, the underground mine will produce on average only approximately 2,400 tpd. The milling schedule will be modified accordingly. The process plant operations cycle will include seven weeks with the plant operating Monday through Thursday. During the eighth week, the plant will operate only on Monday and Tuesday, thus allowing for two extra days during the eighth week to complete large maintenance projects. It is estimated that plant startups will take one hour while shutdowns will require two hours. Thus, the plant will operate 93 hours for seven weeks and 45 hours during the eighth week.

The crushing plant design throughput is 450 tph, the crusher will not operate for as many hours each week as the rest of the plant. Because of this, crusher maintenance projects can be completed while the rest of the plant is in operation.

While the plant is operating, staffing will be similar to 2015 staffing including two reagent and plant employees as well as a commercial supervisor and commercial operator to be responsible for concentrate storage and loading. During periods when the plant is scheduled to be down, the plant will be staffed by one supervisor and one operator on day shifts and one supervisor and one operator on night shifts. These individuals will handle fire watch, monitor any equipment still operating, and complete cleanup projects. The sample preparation, assay, and metallurgical laboratories will be staffed with sufficient manpower to complete mine and plant sample analyses and testing.

The QP is not aware of any additional environmental, permitting, legal, title, taxation marketing or other factors that could affect resources. Mineral Reserve estimates could be materially affected by highwall instability issues related to the USE open mining stope. When complete, the USE stoping will expose an unsupported 100-meter vertical wall below the existing highwall. Inherent geotechnical stresses will be imposed which may cause an unstable operating environment due to spalling of material into the void. Efforts to mitigate the risk may include changing from the USE mining method to 100% LHOS mining of the Project.

A risk to the economic performance of the Project would be higher mining costs for the portion of the deposit that would be converted from USE to LHOS. This would include cemented backfill at a cost us US$14/t.

26 RECOMMENDATIONS

Recommendations for recommissioning the processing plant are as follows:

1. Non-destructive thickness testing should be completed on all tanks, especially where sulfide material has been in contact with the bases of the tanks;
2. A structural steel survey should be completed, especially where sulfide material has been in contact with lower portions of vertical structural members;
3. Non-destructive testing should be completed on SAG and ball mill trunnions;
4. SAG and ball mill gear alignment needs to be measured; and
5. A group including Denarius management personnel and those familiar with the Aguablanca operation should meet and prioritize a list of projects to be completed and categorized into four groups as follows:
    a. **Absolutely** must be completed and **cannot** be completed once the plant is operational;
    b. **Absolutely** must be completed but **can be** completed after the plant is operational;
    c. Not absolutely necessary but can only be completed prior to plant startup; and
    d. Not absolutely necessary and can be completed after the plant is operational.

Implementation of 5a and 5c above may significantly lower the risk of cost overruns during the recommissioning of the processing plant.

An exploration drilling program is recommended. This program would be focused on expansion to depth of the mineral resources as well as expanding the resource footprint of the satellite mineralization of the deposit. A budget is presented below for a 20,000 m drilling program. Results from the drilling would be used to update the MRE for Aguablanca.

Geotechnical and rock mechanics evaluations should be conducted to conclusively support the USE mining which will breach the bottom of the existing open pit.

Budget Item	Description	Cost (US$ 1,000's)
Exploration Drilling	Drilling program to expand the mineral resources beneath the north and main mineralization as well as expansion of the satellite mineralization footprint	US$3,100
	Assaying	US$150
	Technical Services	US$210
Geology	Resource Model Updates	US$125
Geotechnical and rock mechanics assessments	Field and engineering to support the USE mining method	US$300
Subtotal		**US$3,885**
Contingency	Monte Carlo Simulation suggests there is a 20% probability the program will exceed US$3,885,000. A contingency of 8% has been added to the work program budget	US$311
Total		**US$4,196**

The QP is not recommending successive phases.

27 REFERENCES

The following six documents comprise the approved current Environmental Impact Study:

CRS, January 2023, Document 0, Executive Summary of the Reclamation Plan for Aguablanca, 18 pages.

CRS, January 2023, Document I, Aguablanca Mine Underground Project Update Restoration Plan, 327 pages.

CRS, January 2023, Document II, Aguablanca Mine Waste Characterization Study, 113 pages.

CRS, January 2023, Document III, Aguablanca Mine Closure Plan, 980 pages.

CRS, January 2023, Document IV, Water Management Plan, 301 pages.

CRS, January 2023, Document V, Health and Safety Plan, 54 pages.

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Sources for labor cost increases:
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Source for electrical power price increase:
Spain Electricity Price - Quote - Chart - Historical Data - News (tradingeconomics.com)

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.
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.

Term	Definition
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).

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
°	degree
°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
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

Abbreviation	Unit or Term
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
Mt/y	million tonnes per year
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

Abbreviation	Unit or Term
PMF	probable maximum flood
ppb	parts per billion
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
t/d	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