NI 43-101 Technical Report & Feasibility Study on The Las Chispas Project

Location: Sonora, Mexico

Effective Date: January 4, 2021

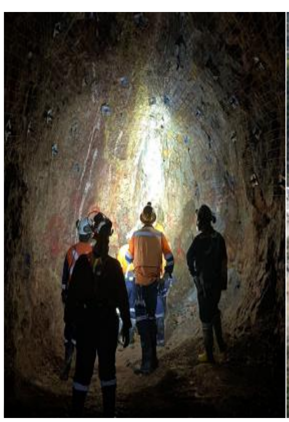
Prepared for: SilverCrest Metals Inc.

570 Granville Street, Suite 501, Vancouver, BC, V6C 3P1

Prepared by: Ausenco Engineering Canada Inc.

List of Qualified Persons:

Robin Kalanchey, P.Eng. – Ausenco Engineering Canada Inc. • Scott Weston, P.Geo. – Hemmera • William Stone, P.Geo. – P&E Mining Consultants Inc.• Eugene Puritch, P.Eng., – P&E Mining Consultants Inc. • David Burga P.Geo.– P&E Mining Consultants Inc. • Jarita Barry, P.Geo. – P&E Mining Consultants Inc. • Yungang Wu, P.Geo. – P&E Mining Consultants Inc. • Andrew J. Turner P.Geol. – P&E Mining Consultants Inc. • Carl Michaud, P.Eng., – G Mining Services Inc. • Michael Verreault, P.Geo. – Hydro-Ressources Inc • Khosrow Aref, P. Eng., – Rockland Ltd. • Humberto Preciado, P.E. – Wood Group PLC •



CERTIFICATE OF QUALIFIED PERSON

Andrew J. Turner, P. Geol.

I, Andrew J. Turner, P. Geol., certify that:

1. I am employed as a Principal of, and senior Geological Consultant with APEX Geoscience Ltd, with an office address of #100, 11450 – 160 Street NW, Edmonton, Alberta, T5M 3Y7.
2. This certificate applies to the technical report titled, *"NI 43-101 Technical Report & Feasibility Study on the Las Chispas Project,"* (the "Technical Report"), that has an effective date of January 4, 2021 (the "Effective Date").
3. I graduated from the University of Alberta in 1989 with a Bachelor of Science, (Honors) in Geology.
4. I am a geological consultant and a registered practising member of the Association of Professional Engineers and Geoscientists of Alberta (License No. 49919) since 1994 as well as the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (License No. L2456).
5. I have practiced my profession for 30 years since graduating. I have extensive experience with exploration for, and the evaluation of intrusive-related gold deposits including low sulphidation epithermal gold-silver mineralization at various projects in Western Canada, the United States and Mexico.
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") for those sections of the technical report that I am responsible for preparing.
7. I visited the Las Chispas Project between November 3, 2020 and November 5, 2020 for a site visit duration of 2 days.
8. I am responsible for co-authoring sub-section 12.2 of the technical report.
9. I am independent of SilverCrest Metals Inc. ("SilverCrest") as independence is described by Section 1.5 of NI 43–101.
10. I have had no previous involvement with the Las Chispas Project.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: February 2, 2021

"Signed and Sealed"

Andrew J. Turner, P. Geol.

CERTIFICATE OF QUALIFIED PERSON
Carl Michaud, P. Eng.

I, Carl Michaud, P. Eng, certify that:

1. I am employed as an Underground Engineering Manager for G Mining Services Inc., with an office address of 7900 Taschereau Blvd., Building D, Suite 200, Brossard, Quebec, Canada, J4X 1C2.
2. This certificate applies to the technical report titled, *"NI 43-101 Technical Report & Feasibility Study on the Las Chispas Project,"* (the "Technical Report"), that has an effective date of January 4, 2021 (the "Effective Date").
3. I graduated from from l'Université du Laval with a B.Sc. (Mine Engineering) in 1996. In addition, I obtained an M.B.A. from the Université du Québec à Chicoutimi, in 2012.
4. I am a Professional Engineer registered with the "Ordre des Ingénieurs du Québec" (OIQ-Licence: 117090).
5. I have worked as a Junior Mine Engineer from May 1996 to 1998. Following this experience, I obtained my professional registration and have worked as a Mine Engineer for a total of 20 years with Placer Dome Canada, Mines McWatters, Resources Campbell Inc., Orica Canada Inc. Goldcorp Inc., Nyrstar and Arcelor Mittal.
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") for those sections of the technical report that I am responsible for preparing.
7. I visited the Las Chispas Project between May 21 and May 24, 2019 and between November 4 and November 11, 2020. The purpose was to visit the active underground at Babicanora Main, Babicanora Central and Las Chispas. To examine the core at the core shack with SilverCrest personnel, have discussions concerning the rock units found in the mining area and rock mechanics, and the geotechnical investigation program.
8. I am responsible for sections 1.12, 1.13, 1.14.3, 15, 16.1, 16.4, 16.5, 16.6, 16.7, 16.8, 16.9, 16.10, 18.12, 21.2.2, 21.4.2, 24.2.4, 24.2.5, 24.3.4, 24.3.5, 25.7, 25.8.3 and 27 of the technical report.
9. I am independent of SilverCrest Metals Inc. ("SilverCrest") as independence is described by Section 1.5 of NI 43–101.
10. I have had no previous involvement with the Las Chispas Project.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: February 2, 2021

"Signed and Sealed"

Carl Michaud, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
David Burga, P. Geo.

I, David Burga, P. Geo., certify that:

1. I am employed as a geological consultant contracted by P&E Mining Consultants Inc., with an office address of 3884 Freeman Terrace, Mississauga, Ontario.
2. This certificate applies to the technical report titled, *"NI 43-101 Technical Report & Feasibility Study on the Las Chispas Project,"* (the "**Technical Report**"), that has an effective date of January 4, 2021 (the "**Effective Date**").
3. I graduated from the University of Toronto with a Bachelor of Science degree in Geological Science in 1997.
4. I am a geological consultant currently licensed by the Association of Professional Geoscientists of Ontario (License No 1836).
5. I have practiced my profession for over 20 years since obtaining my S. Sc. Degree. I have been directly involved as:

• Exploration Geologist, Cameco Gold	1997-1998
• Field Geophysicist, Quantec Geoscience	1998-1999
• Geological Consultant, Andeburg Consulting Ltd.	1999-2003
• Geologist, Aeon Egmond Ltd.	2003-2005
• Project Manager, Jacques Whitford	2005-2008
• Exploration Manager – Chile, Red Metal Resources	2008-2009
• Consulting Geologist	2009-Present

6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") for those sections of the technical report that I am responsible for preparing.
7. I have not visited the Las Chispas Project.
8. I am responsible for section 10 of the technical report.
9. I am independent of SilverCrest Metals Inc. ("SilverCrest") as independence is described by Section 1.5 of NI 43–101.
10. I have had no previous involvement with Las Chispas Project.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: February 2, 2021

"Signed and Sealed"

David Burga, P. Geo.

CERTIFICATE OF QUALIFIED PERSON
Eugene J. Puritch, P. Eng., FEC, CET

I, Eugene J. Puritch, P.Eng., FEC, CET, certify that:

1. I am employed as an independent mining consultant and President of P&E Mining Consultants Inc., with an office address at 44 Turtlecreek Blvd., Brampton, Ontario, L6W 3X7.
2. This certificate applies to the technical report titled, *"NI 43-101 Technical Report & Feasibility Study on the Las Chispas Project,"* (the "**Technical Report**"), that has an effective date of January 4, 2021 (the "**Effective Date**").
3. I graduated from The Haileybury School of Mines, with a Technologist Diploma in Mining in 1977, as well as obtaining an additional year of undergraduate education in Mine Engineering at Queen's University 1978. In addition, I have also met the Professional Engineers of Ontario Academic Requirement Committee's Examination requirement for a bachelor's degree in Engineering Equivalency 2003.
4. I am a mining consultant currently licensed by the: Professional Engineers and Geoscientists New Brunswick (License No. 4778); Professional Engineers, Geoscientists Newfoundland and Labrador (License No. 5998); Association of Professional Engineers and Geoscientists Saskatchewan (License No. 16216); Ontario Association of Certified Engineering Technicians and Technologists (License No. 45252); Professional Engineers of Ontario (License No. 100014010); Association of Professional Engineers and Geoscientists of British Columbia (License No. 42912); and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (No. L3877). I am also a member of the National Canadian Institute of Mining and Metallurgy.
5. I have practiced my profession continuously since 1978. I have been directly involved in:

• Mining Technologist – H.B.M.& S. and Inco Ltd.,	1978-1980
• Open Pit Mine Engineer – Cassiar Asbestos/Brinco Ltd.,	1981-1983
• Pit Engineer/Drill & Blast Supervisor – Detour Lake Mine,	1984-1986
• Self-Employed Mining Consultant – Timmins Area,	1987-1988
• Mine Designer/Resource Estimator – Dynatec/CMD/Bharti,	1989-1995
• Self-Employed Mining Consultant/Resource-Reserve Estimator,	1995-2004
• President – P&E Mining Consultants Inc,	2004-Present

6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") for those sections of the technical report that I am responsible for preparing.
7. I have not visited the Las Chispas Project.
8. I am responsible for sections 1.7, 1.8, 1.10, 1.11, 24.2.3**,** 24.3.3**,** 25.6, 26.3.1, 26.3.2, 26.3.4, 27 and co-authoring section 14 of the technical report.
9. I am independent of SilverCrest Metals Inc. ("SilverCrest") as independence is described by Section 1.5 of NI 43–101.
10. I have had no previous involvement with Las Chispas Project.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: February 2, 2021

"Signed and Sealed"

Eugene J. Puritch, P. Eng., FEC, CET

CERTIFICATE OF QUALIFIED PERSON
Humberto Preciado, PhD, PE

I, Humberto Preciado, PhD, PE, certify that:

1. I am employed as a Principal Geotechnical Engineer by Wood Environment & Infrastructure Solutions, Inc., with an office address at 2000 South Colorado Blvd., Suite 2-1000; Denver, CO 80222-7931, USA.
2. This certificate applies to the technical report titled, *"NI 43-101 Technical Report & Feasibility Study on the Las Chispas Project,"* (the "Technical Report"), that has an effective date of January 4, 2021 (the "Effective Date").
3. I graduated from the Universidad Autónoma de Guadalajara in 1992 with a Bachelor of Science in Civil Engineering and earned a PhD in Civil Engineering from the University of British Columbia in 2005. I have been practicing my profession since 1992 and have been involved in previous NI 43-101 studies such as the AZOD Zinc Oxide Pre-feasibility and Ollachea Gold Feasibility Projects in Peru, the Terronera Pre-feasibility Project and the Cozamin Mine in Mexico.
4. I am a member (#04219767) of the Society for Mining, Metallurgy & Exploration (SME).
5. I have practiced my profession continuously since 1992 and have been directly involved in: Tailings Storage Facilities, Heap Leach Pads, Mine Waste Dumps, Mining Closure, Geo-environmental Site Investigations, Highways, Earthworks, Foundation Design, Deep Foundations, and Soil-Contaminant interactions.
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") for those sections of the technical report that I am responsible for preparing.
7. I visited the Las Chispas Project between March 2, 2019 and March 3, 2019.
8. I am responsible for sections 18.14, 18.15, 20.1.2, 20.4, 21.2.5, 21.2.8, 21.4.4, 24.2.7, 24.2.8 and 27 of the technical report.
9. I am independent of SilverCrest Metals Inc. ("SilverCrest") as independence is described by Section 1.5 of NI 43–101.
10. I have had no previous involvement with Las Chispas Project.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: February 2, 2021

"Signed and Sealed"

Humberto Preciado, PhD, PE

CERTIFICATE OF QUALIFIED PERSON
Jarita Barry, P. Geo.

I, Jarita Barry, P. Geo., certify that:

1. I am employed as an independent geological consultant contracted by P&E Mining Consultants Inc., with an office address of 4 Creek View Close, Mount Clear, Victoria, Australia, 3350.
2. This certificate applies to the technical report titled, *"NI 43-101 Technical Report & Feasibility Study on the Las Chispas Project,"* (the "Technical Report"), that has an effective date of January 4, 2021 (the "Effective Date").
3. I graduated from RMIT University of Melbourne, Victoria, Australia, with a B.Sc. in Applied Geology in 2000.
4. I am a geological consultant currently licensed by Engineers and Geoscientists British Columbia (License No. 40875), Professional Engineers and Geoscientists Newfoundland & Labrador (License No. 08399) and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (License No. L3874). I am also a member of the Australasian Institute of Mining and Metallurgy of Australia (Member No. 305397).
5. I have practiced as a geologist for over 15 years since obtaining my B. Sc. degree. A list of my relevant experience for the purpose of the Technical Report is provided below:
    - Geologist, Foran Mining Corp. — 2004
    - Geologist, Aurelian Resources Inc. — 2004
    - Geologist, Linear Gold Corp. — 2005-2006
    - Geologist, Búscore Consulting — 2006-2007
    - Consulting Geologist (AusIMM) — 2008-2014
    - Consulting Geologist, P.Geo. (APEGBC/AusIMM) — 2014-Present
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") for those sections of the technical report that I am responsible for preparing.
7. I have not visited the Las Chispas Project.
8. I am responsible for sections 11, 12.1 and 12.3 and co-authoring 12.2 of the technical report.
9. I am independent of SilverCrest Metals Inc. ("SilverCrest") as independence is described by Section 1.5 of NI 43–101.
10. I have had no previous involvement with Las Chispas Project.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: February 2, 2021

"Signed and Sealed"

Jarita Barry, P. Geo.

CERTIFICATE OF QUALIFIED PERSON
Khosrow Aref, P. Eng.

I, Khosrow Aref, P.Eng., certify that:

1. I am employed as the Principal Rock Mechanics Engineer and President at Rockland Ltd. with an office address of 1011 West Keith Road, North Vancouver, British Columbia, Canada.
2. This certificate applies to the technical report titled, *"NI 43-101 Technical Report & Feasibility Study on the Las Chispas Project,"* (the "Technical Report"), that has an effective date of January 4, 2021 (the "Effective Date").
3. I graduated from Tehran Polytechnic University in 1980 with a B.Sc. in Mining Engineering. Also, I have an M.Eng. in Geotechnical Engineering from the University of Toronto in 1983, and a Ph.D. in Rock Mechanics from McGill University in 1990.
4. I am a Professional Engineer registered in British Columbia, member number (#16847).
5. I have practiced my profession continuously since 1988 and have been directly involved in geomechanics and mine backfill design.
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") for those sections of the technical report that I am responsible for preparing.
7. I visited the Las Chispas property from March 12-14, 2019, June 25-27, 2019, July 24-26, 2019, and November 5-6, 2019.
8. I am responsible for section 1.14.1, 16.2, 25.8.1 and 27 of the technical report.
9. I am independent of SilverCrest Metals Inc. ("SilverCrest") as independence is described by Section 1.5 of NI 43–101.
10. I have had no previous involvement with Las Chispas Project.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: February 2, 2021

"Signed and Sealed"

Khosrow Aref, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
Michael Verreault, Hydrogeologist, P. Eng.

I, Michael Verreault, Hydrogeologist and P. Eng, certify that:

1. I am employed as a Hydrogeologist and President with Hydro-Resources Inc., with an office address of 1855 des Campanules, Jonquiere, QC Canada.
2. This certificate applies to the technical report titled, *"NI 43-101 Technical Report & Feasibility Study on the Las Chispas Project,"* (the "Technical Report"), that has an effective date of January 4, 2021 (the "Effective Date").
3. I graduated from the University of Quebec at Chicoutimi with a Bachelor of Geological Engineering in 2000 and a master's degree in Hydrogeology in 2003.
4. I am a Professional Engineer registered with the Ordre des Ingénieurs du Québec, member number 125243.
5. I have practiced my profession continuously since 2001 and have been directly involved in Hydrogeology and mine dewatering since then.
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") for those sections of the technical report that I am responsible for preparing.
7. I visited the Las Chispas Project between November 10, 2020 to November 12, 2020.
8. I am responsible for section 1.14.2, 16.3, 25.8.2 and 27 of the technical report.
9. I am independent of SilverCrest Metals Inc. ("SilverCrest") as independence is described by Section 1.5 of NI 43–101.
10. I have had no previous involvement with Las Chispas Project.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: February 2, 2021

"Signed and Sealed"

Michael Verreault, Hydrogeologist and P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Robin Kalanchey, P. Eng.

I, Robin Kalanchey, P. Eng., certify that:

1. I am a Professional Engineer, currently employed as Vice President, Transportation & Logistics with Ausenco Engineering Canada Inc. (Ausenco), with an office at 855 Homer Street, Vancouver, BC V6B 2W2.
2. This certificate applies to the technical report titled, *"NI 43-101 Technical Report & Feasibility Study on the Las Chispas Project,"* (the "Technical Report"), that has an effective date of January 4, 2021 (the "Effective Date").
3. I graduated from the University of British Columbia with a Bachelor of Applied Science degree in Metals and Materials Engineering, with specialization in extractive metallurgy in 1996.
4. I am a Professional Engineer, registered with the Association of Professional Engineers and Geosciences of Alberta, member number 61986.
5. I have practiced my profession continuously since 1996 and have been involved in: mineral processing and metallurgical testing, metallurgical process plant design and engineering, and metallurgical and economic project evaluations for precious metals, including gold and silver, and base metal projects in numerous countries, including Mexico.
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") for those sections of the technical report that I am responsible for preparing.
7. I visited the Las Chispas Project on June 6, 2019 for a site visit duration of one day.
8. I am responsible for sections 1.1, 1.2, 1.9, 1.15, 1.16, 1.17, 1.19 to 1.24, 2, 3.1, 3.4, 3.5, 13, 17, 18.1 to 18.11, 18.13, 18.16, 19, 21.1, 21.2, 21.2.3, 21.2.4, 21.2.6, 21.2.7, 21.3, 21.4.1, 21.4.3, 21.4.5, 22, 24.1, 24.2.1, 24.2.2, 24.2.6, 24.3.1, 24.3.6, 24.3.7, 24.3.8, 25.1, 25.5, 25.9, 25.10, 25.11, 25.13 to 25.17, 26.1, 26.3.3 and 27 of the technical report.
9. I am independent of SilverCrest Metals Inc. ("SilverCrest") as independence is described by Section 1.5 of NI 43–101.
10. I have had no previous involvement with the Las Chispas Project.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: February 2, 2021

"Signed and Sealed"

Robin Kalanchey, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
Scott Weston, P. Geo.

I, Scott Weston, P. Geo., certify that:

1. I am employed as Vice President, Business Development with Hemmera Envirochem Inc, a wholy owned subsidiary of Ausenco Engineering Canada ("Ausenco"), with an office address of 4730 Kingsway, Burnaby, BC, Canad*a*.
2. This certificate applies to the technical report titled, *"NI 43-101 Technical Report & Feasibility Study on the Las Chispas Project,"* (the "Technical Report"), that has an effective date of January 4, 2021 (the "Effective Date").
3. I graduated from University of British Columbia, Vancouver, BC, Canada, 1995 with a Bachelors of Science, Physical Geography, and Royal Roads University, Victoria, BC, Canada, 2003 with a Masters of Science, Environment and Management
4. I am a Professional Geoscientist of Engineers and Geoscientists British Columbia; 124888.
5. I have practiced my profession for 25 years.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") for those sections of the technical report that I am responsible for preparing.
7. I have not visited the Las Chispas Project.
8. I am responsible for sections 1.18, 3.3, 20.1.1, 20.1.3, 20.2, 20.3, 25.12 and 27 of the technical report.
9. I am independent of SilverCrest Metals Inc. ("SilverCrest") as independence is described by Section 1.5 of NI 43–101.
10. I have been involved with the Las Chispas Project.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: February 2, 2021

"Signed and Sealed"

Scott Weston, P. Geo.

CERTIFICATE OF QUALIFIED PERSON

William Stone, PhD, P. Geo.

I, William Stone, PhD., P. Geo., certify that:

1. I am employed as an independent geological consultant with P&E Mining Consultants Inc., with an office address of 4361 Latimer Crescent, Burlington, Ontario, L7M 4R2.
2. This certificate applies to the technical report titled, *"NI 43-101 Technical Report & Feasibility Study on the Las Chispas Project,"* (the "Technical Report"), that has an effective date of January 4, 2021 (the "Effective Date").
3. I graduated from Dalhousie University with a Bachelor of Science (Honours) degree in Geology in 1983. In addition, I earned a Master of Science in Geology in 1985, and a PhD in Geology in 1988 from the University of Western Ontario.
4. I am a geological consultant currently licensed by the Professional Geoscientists of Ontario (license No. 1569).
5. I have practiced my profession for 35 years since I obtained my M.Sc. degree in 1985. I have been directly involved in:
   - Contract Senior Geologist, LAC Minerals Exploration Ltd. 1985-1988
   - Post-Doctoral Fellow, McMaster University 1988-1992
   - Contract Senior Geologist, Outokumpu Mines and Metals Ltd. 1993-1996
   - Senior Research Geologist, WMC Resources Ltd. 1996-2001
   - Senior Lecturer, University of Western Australia 2001-2003
   - Principal Geologist, Geoinformatics Exploration Ltd. 2003-2004
   - Vice President Exploration, Nevada Star Resources Inc. 2005-2006
   - Vice President Exploration, Goldbrook Ventures Inc. 2006-2008
   - Vice President Exploration, North American Palladium Ltd. 2008-2009
   - Vice President Exploration, Magma Metals Ltd. 2010-2011
   - President & COO, Pacific North West Capital Corp. 2011-2014
   - Consulting Geologist 2013-2017
   - Senior Project Geologist, Anglo American 2017-2019
   - Consulting Geoscientist 2020-Present
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") for those sections of the technical report that I am responsible for preparing.
7. I have not visited the Las Chispas Project.
8. I am responsible for sections 1.3, 1.4, 1.5, 1.6, 3.2, 4, 5, 6, 7, 8, 9, 23, 24.3.2, 25.2, 25.3, 25.4, 26.2, and 27 of this technical report.
9. I am independent of SilverCrest Metals Inc. ("SilverCrest") as independence is described by Section 1.5 of NI 43–101.
10. I have had no previous involvement with Las Chispas Project.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: February 2, 2021

"Signed and Sealed"

William Stone, PhD., P. Geo.

CERTIFICATE OF QUALIFIED PERSON
Yungang Wu, P. Geo.

I, Yungang Wu, P. Geo., certify that:

1. I am employed as an independent consulting geologist contracted by P&E Mining Consultants Inc., with an office address of 3246 Preserve Drive, Oakville, Ontario, L6M 0X3.
2. This certificate applies to the technical report titled, *"NI 43-101 Technical Report & Feasibility Study on the Las Chispas Project,"* (the "Technical Report"), that has an effective date of January 4, 2021 (the "Effective Date").
3. I graduated from Jilin University, China in 1992 with a master's degree in Mineral Deposits.
4. I am I am a geological consultant and a registered practising member of the Association of Professional Geoscientists of Ontario (Registration No. 1681).
5. I have practiced my profession for more than 25 years since graduating in 1992. I have been directly involved as a:
   - Geologist –Geology and Mineral Bureau, Liaoning Province, China 1992-1993
   - Senior Geologist – Committee of Mineral Resources and Reserves of Liaoning, China 1993-1998
   - VP – Institute of Mineral Resources and Land Planning, Liaoning, China 1998-2001
   - Project Geologist–Exploration Division, De Beers Canada 2003-2009
   - Mine Geologist – Victor Diamond Mine, De Beers Canada 2009-2011
   - Resource Geologist– Coffey Mining Canada 2011-2012
   - Consulting Geologist 2012-Present
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") for those sections of the technical report that I am responsible for preparing.
7. I have not visited the Las Chispas Project.
8. I am responsible for co-authoring section 14 of the technical report.
9. I am independent of SilverCrest Metals Inc. ("SilverCrest") as independence is described by Section 1.5 of NI 43–101.
10. I have had no previous involvement with Las Chispas Project.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: February 2, 2021

"Signed and Sealed"

Yungang Wu, P. Geo.

Important Notice

This report was prepared as a National Instrument 43-101 Technical Report for SilverCrest Metals Inc (SilverCrest) by Ausenco Engineering Canada Inc., G Mining Services Inc., Hydro-Resources Inc., P&E Mining Consultants Inc., Rockland Ltd., and Wood Environment & Infrastructure Solutions, Inc., collectively the "Report Authors". The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in the report. The report is intended for use by SilverCrest subject to terms and conditions of its contracts with each of the Report Authors. Except for the purposed legislated under Canadian provincial and territorial securities law, any other use of this report by any third party is at that party's sole risk.

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1 Summary

1.1 Introduction

SilverCrest Metals Inc. (SilverCrest) commissioned Ausenco Engineering Canada Inc. (Ausenco) to compile a Technical Report (the Report) and feasibility study (the Feasibility Study) on the Las Chispas Project (the Project), located in Sonora, Mexico. The effective date (the Effective Date) of the Report is January 4, 2021.

1.2 Terms of Reference

The Report supports disclosures by SilverCrest in the news release dated February 2, 2021, entitled "SilverCrest Announces Positive Feasibility Study Results and Technical Report Filing for the Las Chispas Project".

The firms and consultants who are providing Qualified Persons (QPs) responsible for the content of the Report are, in alphabetical order, Ausenco, G Mining Services Inc. (GMS), Hydro-Ressources Inc. (HRI), P&E Mining Consultants Inc. (P&E), Rockland Ltd. (Rockland), and Wood Environment & Infrastructure Solutions, Inc. (Wood).

All units of measurement in the Report are metric, unless otherwise stated. The monetary units are in US dollars, unless otherwise stated.

Mineral Resources and Mineral Reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (the 2014 CIM Definition Standards).

Although the Report assumes a targeted timeline for initial operation and ramp-up of production from the Project, calendar years used in the economic analysis are provided for conceptual purposes only.

1.3 Project Setting

The city of Hermosillo is approximately 220 km southwest of the Project, or a three-hour drive; Tucson, Arizona is approximately 350 km northwest of the Project, or a five-hour drive; and the community and large copper mine in Cananea is located approximately 150 km to the north along Highway 89, or a two-and-a-half-hour drive. The closest villages are Banamichi, 25 km to the southwest, and Arizpe, located approximately 12 km to the northeast. The closest resident to the Project, a single ranch house, is 10 km to the west.

Mining supplies and services are readily available from the towns of Cananea, Hermosillo, and Tucson. Labour and skilled workforces exist in the nearby communities, including Banamichi and Arizpe, for which housing and transportation routes could be established to support a mining operation. Provision of grid power for the planned mining operation is in the permitting process, with construction anticipated to begin in 2021 and be completed before production start-up.

The Project is accessed from the community of Arizpe via secondary gravel roads, approximately 10 km off the paved highway. Currently, crossing the Rio Sonora is required. The water levels in the river are typically low and easily passed but can raise to temporary unpassable levels following major rain events. A road bridge is planned to be constructed in 2021. The remainder of the road has been upgraded by dozer/grader.

The climate is typical for the Sonoran Desert, with a dry season from October to May. Seasonal temperatures vary from approximately 0°C to 40°C. Average rainfall is estimated at 300 mm/year. Operations are planned to be conducted year-round.

The Project is located on the western edge of the north-trending Sierra Madre Occidental mountain range geographically adjacent to the Sonora Valley. Surface elevations range from 950 metres above sea level (masl) to approximately 1,375 masl.

Drainage valleys generally flow north to south, and east to west towards the Rio Sonora. During the rainy season, flash flooding can occur in the area.

Vegetation is scarce during the dry season and limited primarily to juvenile and mature mesquite trees and cactus plants. During the wet season, various blooming cactus, trees, and grasses are abundant in drainage areas and on hillsides.

1.4 Property Description and Location

The Las Chispas Property consists of 28 mineral concessions, totalling 1,400.96 ha, which are held by SilverCrest's Mexico subsidiary Compañía Minera La Llamarada S.A. de C.V. (LLA). Concessions have expiry dates that run from 2022–2067. One concession is in the grant process, and one concession is the subject of legal proceedings following cancellation. The mineral concessions that host the Mineral Resources and Mineral Reserves are in good standing. At the Report Effective Date, all required mining duties were paid.

The surface rights overlying the Las Chispas Property mineral concessions and road access from local highway are either owned by LLA or held by LLA under a negotiated 20-year lease agreement with the Ejido Bamori. LLA has purchased the Cuesta Blanca and Babicanora ranches and signed a 20-year lease agreement for a portion of the Tetuachi Ranch. Surface rights are sufficient for the proposed life of mine (LOM) plan and include the locations of necessary infrastructure as presented in the Report.

A 2% royalty is payable on the Nuevo Lupena and Panuco II concessions for material that has processed grades of ≥0.5 oz/tonnes gold and ≥ 40 oz/tonnes silver, combined. These concessions do not include Mineral Reserves.

The Feasibility Study assumes that production water will be from the 900 level (900 m from surface or 850 masl) of the historical Las Chispas Mine and from the Sonora Valley. This combined source of water is considered to be reflective of the regional water table, has been tested, and is adequate in quantity and quality for production purposes. LLA has sufficient water rights for operations.

1.5 History

Historical records indicated mining around the Project started as early as the 1640s. There are incomplete historical records available on mining activities in the 1800s and 1900s. A number of small mines were operated during the period 1900–1930. There is a gap in mining activity records for Las Chispas between the mid-1930s through to 1974. A small mill operated offsite from 1974 to 1984, treating material from historical mine dumps.

Minefinders Corporation Ltd. (Minefinders) conducted geological mapping and a geochemical sampling program comprising stream sediment and bulk-leach extractable gold (BLEG) samples, underground and surface rock chip sampling, and drilling of seven (7) reverse circulation (RC) drill holes (1,842.5 m) to test potential mineralization adjacent to the Las Chispas mineralized northwest-southeast trend. Drill results were not encouraging.

SilverCrest's subsidiary obtained the rights to the Project in 2015. Exploration work completed to the Effective Date included 1,626 (426,441.5 m) core drill holes, surface and underground mapping and sampling, rehabilitation of underground workings, auger and trench sampling of historical mine dumps, Mineral Resource estimations, environmental baseline and supporting studies, initiation of permitting activities, metallurgical testwork approximately 9 km of underground development and completion of a Preliminary Economic Assessment (the PEA) (Tetra Tech, 2019). The Feasibility Study was commissioned in late 2019, and the Report discusses the results of that study.

1.6 Geological Setting and Mineralization

Mineral deposits in the Las Chispas district are classified as gold and silver, low to intermediate sulphidation epithermal systems, typical of many deposits in Sonora, Mexico.

In northwestern Mexico, much of the exposed geology can be attributed to the subduction of the Farallon Plate beneath the North American Plate and related magmatic arc volcanism. The host rocks to mineralization in the Las Chispas district are generally pyroclastic, tuffs, and rhyolitic flows interpreted to be members of the Lower Volcanic Complex. Locally, volcanic pyroclastic units mapped within the underground workings include rhyolite, welded rhyodacite tuff, lapilli (lithic) tuff, and volcanic agglomerate.

All rock types in the Project area show signs of extensive hydrothermal alteration. Thin section and TerraSpec™ hyperspectral studies identified alteration consistent with argillic and advanced argillic alteration. Alteration minerals identified include smectite, illite, kaolinite, chlorite, carbonate, iron oxy/hydroxides, probable ammonium, gypsum/anhydrite, silica, and patch trace alunite.

Generally, the host rocks are above the existing water table. Oxidation of sulphides is observed from near-surface to depths greater than 300 m and the presence of secondary minerals is recorded from the Las Chispas historical underground workings approximately 60 m to 275 m in depth from the surface. Strong and pervasive near-surface oxidation is noted to occur in the Babicanora Area, where host rocks experienced faulting and advanced weathering to limonite, hematite, and clays.

Regionally, the Project is situated in an extension basin related to a Late Oligocene half-graben of the Rio Sonora basin. Multiple stages of normal faulting affect the basin. The main structures are steep, west-dipping (80°) and sub-parallel to the Granaditas normal fault, which is located along the western margin of the Project, striking approximately 30°. The basin is further cross-cut by younger northwest–southeast trending normal faults that dip to the southwest, creating both regional and local graben structures. Locally, the graben structures are complicated by probable caldera collapse. Three structural controls, excluding bedding contacts, are considered to influence alteration and mineralization:

- 150–170° striking and are inclined at approximately 65–75° to the southwest;
- 340–360° striking and are inclined 75° west to 75° east; and,
- 210–230° striking and are inclined 70–85° to the northwest.

Mineralization is hosted in hydrothermal veins, stockwork, and breccia. Emplacement of the mineralization is influenced by fractures and low-pressure conduits formed within the rocks during tectonic movements. Mineralization can be controlled lithologically along regional structures, local tension cracks, and faulted bedding planes. Brecciated mineralization formed in two ways: 1) in zones of low pressure as hydrothermal breccia; and 2) as mechanical breccias. These breccia types are interpreted to occur at the intersection of two or more regional structural trends. The mineralization is 0.10–10 m in true width, and typically encompasses a central quartz ± calcite mineralization corridor with narrow veinlets within the adjacent fault damage zone. Stockwork and breccia zones are centred on structurally controlled hydrothermal conduits.

Generally, it appears that epithermal mineralization is higher in the system (closer to the paleo-surface) on the west side (e.g., La Victoria Vein and historical mine) of the Las Chispas district compared to the east side (e.g., Granaditas Vein and historical mine), where there is an observed increase in base metal content.

Argentite is the principal silver mineral. Electrum and native silver can be present. Silver is associated with galena, pyrite ± marcasite and chalcopyrite. Gold occurs as native flakes and in association with pyrite and chalcopyrite. Locally, gold and silver values have a strong correlation with each other. Base metal contents are low in veins.

The Las Chispas district is divided into the Las Chispas Area and the Babicanora Area, and currently has 45 separate epithermal veins identified. Mineral Resources were estimated for 21 veins, and Mineral Reserves for 15 veins of which six veins (Babicanora Main, Babicanora FW, Babicanora Norte, Babicanora Sur, Babi Vista and Las Chispas) contain the majority of the Mineral Reserves.

1.7 Drilling and Sampling

SilverCrest completed a number of drilling program phases in the period 2016–2020.

The Phase I (March 2016 to October 2016) drilling program targeted near-surface mineralization, lateral extensions of previously mined areas, and potential deep extensional mineralization proximal to the historical workings.

The Phase II (October 2016 to February 2018) drilling program focused on surface drilling at the Las Chispas, Babicanora Main, William Tell, and Giovanni veins and on underground drilling at the Las Chispas and Babicanora area veins. New targets, such as the La Varela, La Blanquita, Granaditas, and Amethyst veins were drill-tested.

The Phase III (February 2018 to February 2019) drilling program focused on surface drilling at the Babicanora Main, Babicanora FW, Babicanora HW, Babicanora Norte, Babicanora Sur, Granaditas, Luigi, and Giovanni veins and underground drilling at the Las Chispas Vein. Newly tested targets for the Phase II drilling program included the Babicanora Norte, Babicanora Sur, Granaditas, Luigi, Amethyst and Ranch veins.

The Phase III Extended (February 2019 to October 2020) drilling program was an infill program to support increased confidence for Mineral Resource classification upgrades, and test for expansion of multiple veins. A systematic drill hole vein piercing pattern of approximately 35 m by 35 m was used to support conversion of Inferred Mineral Resources to the Indicated category. Newly tested targets for Phase III Extended drilling program included the Babi Vista Vein and Babi Vista Vein Splay.

Surface collar locations were initially surveyed using a handheld global positioning system (GPS) unit and then professionally surveyed by a local contractor. A survey was completed by external consultant David Chavez Valenzuela in October 2018. The most recent surveys were completed by Precision GPS S.A. de C.V. (Precision GPS) from Hermosillo, Sonora, Mexico. The survey provided drill collar locations, information on roads, and additional detail on property boundaries.

Underground drill hole collars were surveyed by Precision GPS using the underground control points established for each of the workings. All holes were downhole surveyed as single-shot measurements with a Flex-it tool starting at 15 m with measurements at every 50 m to determine deviation. The survey measurements were monitoring downhole deviations and significant magnetic interference from the drill rods that would prevent accurate readings.

For a newly discovered vein, the first 10 drill holes were completely sampled. Additional drill holes could be entirely sampled, if such sampling were needed to establish a better understanding of geology and mineralization. Sample intervals were laid out for mineralization, veining, and structure. Approximately 10 m before and after each mineralization zone was included in the sampling intervals. A minimum of 0.5 m sample lengths of mineralization material was taken up to a maximum of 3 m in non-mineralization rock. Each sample interval was either split using a hand splitter or cut using a wet core saw, perpendicular to veining, where possible, to leave representative core in the box and to reduce any potential bias in the sampled mineralization submitted with the sample.

Chip samples and/or channel samples were collected from historical underground workings and newly developed in-vein drifting. Samples were collected using a small sledgehammer, a hand maul/chisel, and a small tarp on the floor to collect the chips, or a power saw for channeling. Sampling was conducted at random within the existing historical muck and material stockpiles in the Las Chispas, William Tell, and Babicanora historical workings. Sample collection was completed by hand or shovel, from near surface material, as non-selective collection to represent both the fine and coarse fragment portions of the muck piles. Auger sampling, completed as a test program, was conducted on nominal 1 m depth intervals in selected surface mineralized rock dumps. Due to a combination of large rocks and low recovery, the auger program was discontinued. Trenches were initially hand excavated to approximately 0.5 m in the face of the mineralization rock dumps with collection of samples every 1 m. Subsequently, mechanical trenching was completed on all accessible historical dumps. A backhoe was used to dig trenches approximately 1.5 m deep and pile the excavated material next to the trench for sampling and description. Sample weights were 3–5 kg.

A total of 641 bulk density measurements were collected on site by SilverCrest using the water immersion method. Seventy-two (72) samples were tested by ALS Chemex (ALS) based in Hermosillo, Mexico for wax-coated bulk density to validate the on-site measurements. In November 2018, two samples were collected and sent by SilverCrest to Geotecnia del Noroeste S.A. de C.V. based in Hermosillo, for wax coated dry bulk density testing. The bulk density ranged from 4.02 t/m^3 with a mean value of 2.52 t/m^3. A uniform mean bulk density of 2.55 t/m^3 was applied to all rock types in the Mineral Resource estimate based on the results of the bulk density test work completed by SilverCrest and the two laboratories.

All primary assays were completed by ALS in Hermosillo, ALS in Vancouver, BC, Canada, and Bureau Veritas Minerals Laboratories (Bureau Veritas, formally Inspectorate Labs) in Hermosillo. Check assays were performed by SGS de Mexico S.A. de C.V in Durango, Mexico (SGS Durango). These laboratories are independent of SilverCrest and hold accreditations for selected analytical techniques.

All samples were crushed to 75% (ALS) or 70% (Bureau Veritas) minus 2 mm, then mixed and split with a riffle splitter. A split from all samples was then pulverized to 80% (ALS) or 85% (Bureau Veritas) minus 75 µm. All pulverized splits were submitted for multi-element aqua regia digestion with inductively coupled plasma (ICP)-mass spectrometry (MS) detection, and for gold fire assay (FA) fusion with atomic absorption spectroscopy (AAS) detection.

Samples returning grades above the upper detection limit of >100 gpt Ag from ICP analysis were re-run using aqua regia digestion and ICP-atomic emission spectroscopy (AES) detection and diluted to account for grade detection limits (<1,500 gpt). Where silver grades were ≥1,500 gpt, the sample was re-run using FA with gravimetric detection. During the Phase II drilling program, where gold values >1 gpt, the samples were re-run using FA with gravimetric detection, and where gold values were >10 gpt, the samples were re-run using 30 g FA with AAS detection. Samples returning grades >10,000 ppm Zn, Pb or Cu from ICP-MS analysis were re-run using aqua regia digestion with ICP-AES finish.

The quality assurance/quality control (QA/QC) program consisted of certified reference material (CRM), and blank sample insertions at a rate of 1:50 for all sample types being collected, and insertion of duplicate samples for some underground chip samples, core pulps and coarse rejects. CDN Resource Laboratories Ltd. Was the source of the CRMs. The blank samples were collected from a local silica cap.

The sample preparation, analysis, and security program implemented by SilverCrest was designed with the intent to support collection of a large volume of data. Sample collection and handling routines were well documented. The laboratory analytical methods, detection limits, and grade assay limits are suited to the style and grade of mineralization. The QA/QC methods implemented by SilverCrest enabled assessment of sample security, assay accuracy, and potential for contamination. The QP reviewed sample collection and handling procedures, laboratory analytical methods, QA/QC methods, and QA/QC program results and believes these methods are adequate to support the current Mineral Resource estimate.

1.8 Data Verification

SilverCrest developed an extensive dataset that is saved and managed using Geospark™ management software. P&E reviewed the data compilation and audited the Geospark™ database. P&E conducted verification of the Las Chispas databases for gold and silver by comparison of the database entries with assay certificates in comma-separated values (csv) file format, obtained directly from ALS Webtrieve. Assay data were verified for five separate datasets: Las Chispas, Las Chispas Underground, Babicanora Underground, William Tell Underground and Babi Vista.

P&E also validated the drill hole database by checking for inconsistencies in analytical units, duplicate entries, interval, length or distance values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals or distances greater than the reported drill hole length, inappropriate collar locations, survey and missing interval and coordinate fields. A few errors were identified and corrected in the database.

The QP believes the database provided by SilverCrest is reliable and the QP does not consider the few minor discrepancies encountered during the verification process to be of material impact to the data supporting the Mineral Resource estimate.

1.9 Mineral Processing and Metallurgical Testwork

Two metallurgical testwork programs were undertaken in August 2017 and November 2018 in support of the previous evaluations of the Project prior to start of the Feasibility Study. Both programs were completed at SGS Durango. These earlier test programs highlighted the preferred process options to be evaluated and provided context for selection of drill cores and preparation of composite samples for further, more detailed testing.

In 2019, selected samples of mineralization from the Las Chispas deposit, which had either been used in the previous two series of tests at the SGS Durango facility or were new samples from various drill programs at site, were shipped to SGS Lakefield Research in Ontario, Canada (SGS Lakefield), for further metallurgical testing to support the Feasibility Study. The SGS laboratory facilities in Mexico and in Canada are well respected for their metallurgical testwork, and are independent of SilverCrest. There is currently no facility that accredits metallurgical testwork methods.

The materials tested in support of the Feasibility Study were considered to be representative of the Las Chispas deposit, both with respect to the global average materials characteristics, and also with respect to high-grade, low-grade (and waste) and known high-clay containing zones within the deposit.

In summary, work completed from 2019 to the Effective Date of the Report, in support of the Feasibility Study included: chemical and mineralogical analysis of the feed samples; comminution testwork; investigation of pre-concentration options, including gravity separation and flotation; cyanide leaching of concentrate and tailings fractions from both gravity and flotation pre-concentration options; solid-liquid separation testing; precious metals recovery testing (Merrill Crowe process); cyanide destruction testing; and variability testing across key unit operations for selected lithologies and veins.

In 2019 and early 2020, over 200 variability and composite leach tests were complete with a gravity pre-concentration step. The testwork followed a proposed flowsheet incorporating: comminution; gravity pre-concentration; cyanide leaching of the gravity concentrate and tailings fractions; counter-current decant (CCD) washing of the leach residue, and ultimately cyanide detoxification of the tailings slurry prior to filtration and long term storage; and Merrill Crowe recovery of precious metals and conventional smelting to generate a doré.

Overall, high recoveries for this work were achieved at laboratory scale by application of conventional, commercially proven processes. However, due to the brittle nature of the silver sulphide mineral, argentite, there was a concern that the mineral could overgrind in practice and result in reduced gravity recovery. Therefore, pre-concentration using flotation was pursued, and favourable results were gained from 40 additional flotation tests:

- Gravity concentration: 40–50% gold and silver recovery and 4% mass pull; and
- Flotation concentration: 60–80% gold and silver recovery and 2% mass pull.

An extensive campaign was commissioned in mid-2020 to provide confidence in gold and silver extractions upon cyanidation. This included evaluation of the response of high-grade mineralization and samples with elevated antimony values. Overall recovery, under the most promising conditions tested, ranged from 98–99% for gold and 91–97% for silver.

For this Feasibility Study, a flowsheet incorporating flotation followed by separate cyanidation of the concentrate and tailings was considered to be the most appropriate design for the mineralization at Las Chispas. Over the range of samples tested, overall gold recovery was determined to be relatively insensitive to grade, such that use of an average recovery value of 97.6% would be appropriate. Silver recovery varied linearly with increasing grade, is relatively predictable, and therefore was estimated at 94.3% on the basis of the grade determined for the LOM.

1.10 Mineral Resource Estimation

Mineral Resource Estimates were prepared by P&E for potential underground mining of in-situ vein deposits at the Las Chispas and Babicanora Areas, and for surface extraction of stockpiles which remain from historical operations.

All drilling, surveying (collar and downhole) and assay data were provided by SilverCrest in the form of Microsoft Excel data files up to and including a data cut-off date of October 16, 2020. The database consisted of surface drill holes, underground drill holes and underground channel and chip samples for the in-situ narrow veins and included surface surveys with assay data for surface channel and RC samples for the historical stockpiles.

Mineralized vein wireframes were interpreted and constructed by SilverCrest and verified by the QPs. Some adjustments to the wireframes were made as a result of the reviews, and the QP considered the wireframes to be reasonable and suitable for Mineral Resource estimation.

Vein models, representing the continuous zone of structurally-hosted gold and silver mineralization and the structural extensions of the veins, were developed using the drill core field logs and assays. All veins were constrained to a minimum true width of 0.5 m. Solids were manually clipped in the initial Mineral Resource modelling stage to include mineralized areas with ≥150 gpt AgEq (where AgEq = Ag gpt + Au gpt * 75.0). In some cases, samples <150 gpt AgEq were included to maintain mineralized continuity and minimum true width. Zones of internal waste were delineated within the mineralization veins where there was a minimum true thickness of 1.5 m of <150 gpt AgEq across two or more adjacent drill holes. A surveyed topographical surface was provided by SilverCrest. All mineralization veins were clipped above the surface. Areas of historical mining and significant internal waste zones were clipped from the related vein wireframes.

A unique rock code was assigned to each rock type in the Mineral Resource model. Assays were constrained by a wireframe and back coded in the assay database with rock codes that were derived from intersections of the clipped mineralized solids and drill holes. A 0.5 m compositing length was selected for the drill hole intervals that fell within the wireframe domains. Grade capping and high-grade transition analyses for gold and silver were undertaken on the composites using log-normal histograms and log-probability plots. The high-grade transition consisted of a restrictive search ellipse and a maximum limiting composite value. Variography analyses were performed using the gold and silver composites within each individual vein wireframe as a guide to determining a grade interpolation search distance and ellipse orientation strategy.

The block model consisted of separate model attributes for estimated gold and silver grades, rock type (mineralization domains), volume percent, bulk density, AgEq value, and classification. All blocks in the rock type block model were initially assigned a waste rock code. The mineralization domain was used to code all blocks within the rock type block model that contained ≥0.01% volume within the domain. These blocks were assigned individual rock type codes.

Gold and silver grade blocks were interpolated using inverse distance weighting to the third power (ID^3). Multiple passes were executed for the grade interpolation to progressively capture the sample points to avoid over-smoothing and preserve local grade variability. Pass 0 was interpolated with underground samples, when available; Pass 1 and 2 were interpolated with capped composites derived from clipped wireframes for blocks coded with clipped solids. Pass 3 was interpolated with composites derived from unclipped solids for blocks coded with unclipped solids. At the intermediate stage of grade block estimation (May 2020), the AgEq block values were calculated with the following formula which were based on prevailing metal price averages and process recoveries at that time:

- AgEq gpt = Ag gpt + (Au gpt x 83.7).

Models were validated using on-screen visual examination of composites and block grades on successive plans and sections, the ID^3 estimate was compared to a nearest-neighbour (NN) estimate, and the ID^3 and NN estimates were compared against the composites using swath plots. ID^3 was chosen as the preferred interpolation method based on maintaining some high-grade variability within a hard boundary wireframe to develop a better mine plan for blending material for processing.

For the Babicanora Main Vein, estimates were generated and compared between ID^3 and ordinary kriging (OK) grade interpolation methods. The results from the comparison showed OK interpolation with more tonnes, less grade, and approximately 4% more AgEq ounces.

An interpolation comparison between estimates that included or excluded the influence of recent in-vein development underground channel samples was completed for the Babicanora Main Vein. Comparatively, higher gold grades are reported from underground channel samples due to better sample recovery of the fine (<75 µm) fraction in relation to the drill hole core. No major biases were noted from this validation procedure and underground channel samples were used for the Babicanora Main Vein Mineral Resource modelling.

In August 2020, the following parameters were used to calculate the AgEq cut-off grades (COG) to support the deposit as a reasonable prospect for eventual economic extraction: selective underground mining methods; silver price of $18.50/oz (approximate two-year trailing average at October 31, 2020); silver process recovery of 95%; marginal mining cost of $40/t; processing cost of $30/t, and general and administrative (G&A) cost of G&A: $15/t. AgEq cut-off for reporting the Mineral Resource estimate amenable to underground mining was calculated as follows:

- ($40+ $30+$15) / ($18.50/31.1035 x 95%) = 150 gpt AgEq.

A Measured Mineral Resource was only classified for the recent underground sampled workings in the Babicanora Main Vein with a 10 m range extended up and down from the samples interpolated with underground channel and chip samples and drill holes within this area. Indicated Mineral Resources were classified for the blocks interpolated with the Pass 1 and 2, which used at least two drill holes within 50 m. Inferred Mineral Resources were classified for all remaining grade blocks within the mineralization vein wireframes >150 gpt AgEq with some lower grade exception based on continuity. The classifications were adjusted on a longitudinal projection to reasonably reflect the distribution of each classification.

In addition to the estimate of material potentially amenable to underground mining, Mineral Resources were estimated for selected historical dumps and stockpiles. The estimated tonnage of each stockpile was calculated using the average thickness, based on trench profiles and auger drill holes, an estimated bulk density of 1.7 t/m^3, and a measured surface area, calculated in GEMS using the dump perimeter. Grade capping was applied to gold and silver assay data for each stockpile area, then average grades were estimated for each stockpile area based on the samples collected. The Mineral Resource Estimate for the historical stockpiles were calculated using a COG of 110 gpt AgEq.

1.11 Mineral Resource Statement

The Mineral Resource estimates are reported with an effective date of October 16, 2020. The QPs for the estimates are Mr. Yungang Wu, P. Geo. and Mr. Eugene Puritch, P.Eng., of P&E.

Mineral Resources considered potentially amenable to underground mining methods are provided in Table 1-1 (Summary) and Table . (Details). Mineral Resources are reported inclusive of those Mineral Resources converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-1: Las Chispas Mineral Resource Estimate Summary at 150 gpt AgEq Cut-off(1-8)

Vein	Classification	Tonnes (k)	Au (gpt)	Ag (gpt)	AgEq (gpt)	Contained Au (koz)	Contained Ag (koz)	Contained AgEq (koz)
Babicanora Area Total	Measured + Indicated	2,214.5	7.35	681	1,319	523.2	48,471	93,939
Las Chispas Area Total	Indicated	445.1	4.20	548	913	60.1	7,844	13,065
Total Undiluted Veins	**Measured + Indicated**	**2,659.6**	**6.82**	**659**	**1,251**	**583.3**	**56,316**	**107,004**
Stockpiles	Indicated	164.2	1.23	108	215	6.5	572	1,135
Total (Veins + Stockpiles)	**Measured + Indicated**	**2,823.8**	**6.50**	**627**	**1,191**	**589.8**	**56,888**	**108,139**
Babicanora Area Total	Inferred	861.6	5.47	409	884	151.6	11,325	24,496
Las Chispas Area Total	Inferred	378.4	1.80	272	428	21.9	3,308	5,209
Total (Undiluted Veins)	**Inferred**	**1,240.0**	**4.35**	**367**	**745**	**173.4**	**14,634**	**29,705**

Notes:
1. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
3. The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It can be reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.
4. The Mineral Resources in the Report were estimated using the 2019 CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines and 2014 CIM Definition Standards for Mineral Resources & Mineral Reserves.
5. Historical mined areas were removed from the wireframes and block model.
6. AgEq is based on gold to silver ratio of 86.9:1 calculated using $1,410/oz Au and $16.60/oz Ag, with average metallurgical recoveries of 96% Au and 94% Ag.
7. Mineral Resources are inclusive of the Mineral Reserves stated in Section 15.
8. Totals may not add due to rounding.

Table 1-2: Las Chispas Mineral Resource Estimate Details (1-8) at 150 gpt AgEq Cut-off

Vein	Classification	Tonnes (k)	Au (gpt)	Ag (gpt)	AgEq (gpt)	Contained Au (koz)	Contained Ag (koz)	Contained AgEq (koz)
Babicanora Main	Measured	143.3	13.52	1,192	2,366	62.3	5,490	10,901
Babicanora Main	Indicated	919.0	5.29	532	992	156.3	15,720	29,302
Babicanora Main	**Measured + Indicated**	**1,062.3**	**6.40**	621	**1,177**	**218.6**	**21,210**	**40,204**
Babicanora FW	Indicated	162.7	6.60	610	1,184	34.5	3,190	6,191
Babicanora HW	Indicated	119.3	2.48	151	366	9.5	579	1,406
Babicanora Norte	Indicated	351.5	9.03	1,067	1,851	102.0	12,051	20,919
Babicanora Norte HW	Indicated	66.9	2.87	236	486	6.2	507	1,045
Babicanora Sur	Indicated	233.4	7.09	372	988	53.2	2,791	7,412
Babicanora Sur HW	Indicated	18.4	2.62	97.5	325	1.5	57	191
Babi Vista	Indicated	179.9	15.81	1,293	2,668	91.5	7,480	15,482

Vein	Classification	Tonnes (k)	Au (gpt)	Ag (gpt)	AgEq (gpt)	Contained Au (koz)	Contained Ag (koz)	Contained AgEq (koz)
Babi Vista FW	Indicated	20.2	9.53	928	1,756	6.2	603	1,141
Babicanora Area Total	**Measured + Indicated**	**2,214.5**	**7.35**	**681**	**1,319**	**523.2**	**48,471**	**93,939**
Las Chispas	Indicated	208.2	5.74	748	1,246	38.4	5,007	8,344
Giovanni	Indicated	70.8	2.76	394	634	6.3	896	1,443
Gio Mini	Indicated	54.9	3.70	466	787	6.5	821	1,388
William Tell Main	Indicated	17.3	1.99	283	456	1.1	157	253
Luigi	Indicated	61.9	2.48	338	553	4.9	672	1,101
Luigi_FW	Indicated	31.9	2.74	281	520	2.8	288	533
Las Chispas Area Total	**Indicated**	**445.1**	**4.20**	**548**	**913**	**60.1**	**7,844**	**13,064**
Total Undiluted Veins	**Measured + Indicated**	**2,659.6**	**6.82**	**659**	**1,251**	**583.3**	**56,315**	**107,004**
Stockpiles	Indicated	164.2	1.23	108	215	6.5	572	1,134
Total (Veins + Stockpiles)	**Measured + Indicated**	**2,823.8**	**6.50**	**627**	**1,191**	**589.8**	**56,888**	**108,139**
Babicanora Main (Inc. El Muerto Zone)	Inferred	342.0	3.02	256	519	33.2	2,819	5,706
Babicanora FW	Inferred	5.4	1.39	154	275	0.2	27	48
Babicanora HW	Inferred	6.0	1.97	79	250	0.4	15	48
Babicanora Norte	Inferred	53.1	2.09	317	499	3.6	541	851
Babicanora Norte HW	Inferred	27.2	1.77	172	326	1.6	151	286
Babicanora Sur	Inferred	79.4	4.94	251	681	12.6	641	1,737
Babicanora Sur HW	Inferred	2.8	2.53	6	226	0.2	1	21
Babicanora Sur FW	Inferred	42.0	1.77	162	316	2.4	219	426
Babi Vista	Inferred	14.1	3.05	222	488	1.4	101	221
Babi Vista Splay	Inferred	211.4	13.00	909	2,039	88.3	6,180	13,857
Babi Vista FW	Inferred	15.1	2.36	214	419	1.1	104	204
Granaditas 1	Inferred	43.5	4.11	295	653	5.8	413	913
Granaditas 2	Inferred	19.7	1.19	182	285	0.8	115	180
Babicanora Area Total	**Inferred**	**861.6**	**5.47**	**409**	**884**	**151.6**	**11,325**	**24,496**
Las Chispas	Inferred	71.7	3.27	469	753	7.5	1,082	1,736
Gio Mini	Inferred	6.8	2.20	535	726	0.5	118	160
William Tell Main	Inferred	155.5	1.49	233	363	7.4	1,166	1,813
William Tell HW	Inferred	55.9	2.00	237	412	3.6	427	740

Vein	Classification	Tonnes (k)	Au (gpt)	Ag (gpt)	AgEq (gpt)	Contained Au (koz)	Contained Ag (koz)	Contained AgEq (koz)
William Tell Mini	Inferred	33.5	1.60	172	311	1.7	185	334
Luigi	Inferred	19.7	1.14	161	260	0.7	102	165
Luigi_FW	Inferred	35.2	0.33	202	230	0.4	229	261
Las Chispas Area Total	**Inferred**	**378.4**	**1.80**	**272**	**428**	**21.9**	**3,308**	**5,209**
Total Undiluted Veins	**Inferred**	**1,240.0**	**4.35**	**367**	**745**	**173.4**	**14,633**	**29,705**

Notes:
1. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
3. The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It can be reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.
4. The Mineral Resources in the Report were estimated using the 2014 CIM Definition Standards on Mineral Resources and Mineral Reserves.
5. Historical mined areas were removed from the wireframes and block model.
6. AgEq is based on Au: Ag ratio of 86.9:1 calculated using $1,410/oz Au and $16.60/oz Ag, with average metallurgical recoveries of 96% Au and 94% Ag.
7. Mineral Resources are inclusive of the Mineral Reserves stated in Section 15.
8. Totals may not add due to rounding.

1.12 Mineral Reserve Estimation

Mineral Reserves were converted from Measured and Indicated Mineral Resources using applicable modifying factors. The Inferred Mineral Resources contained within the Mineral Resource block models were treated as waste at zero grade.

COGs were calculated using input parameters, such as process recovery, processing costs, G&A costs, commodity prices, exchange rate, and marketing costs (Table 1-3).

Table 1-3: COG Input Parameters

Input Parameters		
Processing Costs	$/t milled	35.00
General & Administrative costs	$/t milled	15.00
Gold Price	$/oz	1,410
Silver Price	$/oz	16.60
Conversion	g/oz	31.10
Government Gold Royalty	%	0.5
Gold Recovery	%	96.0
Silver Recovery	%	94.0
Recovered Gold/Silver Ratio	Au/Ag	86.9
Gold Payable	%	99.85
Silver Payable	%	99.85
Transport	$/oz	0.01
Treatment and Refining	$/oz	0.22
Net Value (Ag)	$/oz	15.25

Input Parameters		
Net Value (Ag)	$/g	0.49

Operating costs were based on an extraction and processing rate of 1,250 t/d. Marginal COGs were also calculated without development costs.

COGs were defined by mining method:

- Long hole and Avoca mining methods: COG with dilution = 190 gpt AgEq; COG marginal with dilution = 170 gpt AgEq;
- Cut-and-fill mining methods: COG with dilution = 250 gpt AgEq where the stope mining width is 1.5 m, 220 gpt AgEq where the stope mining width is 2.5 m, 200 gpt AgEq where the stope mining width is 4.5 m; marginal COG with dilution = 210 gpt AgEq where the stope mining width is 1.5 m, 200 gpt AgEq where the stope mining width is 2.5 m, and 190 gpt AgEq where the stope mining width is 4.5 m;
- Cut-and-fill mining methods using resue: COG with dilution = 430 gpt AgEq where the stope mining width is 0.5 m, 330 gpt AgEq where the stope mining width is 1.0 m, 290 gpt AgEq where the stope mining width is 1.5 m; marginal COG with dilution = 300 gpt AgEq where the stope mining width is 0.5 m, 240 gpt AgEq where the stope mining width is 1.0 m, and 220 gpt AgEq where the stope mining width is 1.5 m; and
- Stockpiles: 110 gpt AgEq.

Stopes were designed using Deswik Stope Optimizer (DSO) software. Multiple DSO scenarios were run to obtain the best results in terms of tonnage and grade. The DSO software was first programmed to estimate the economic stopes with the applicable COG. A second set of economic calculations were then completed with the use of the applicable marginal COG. The stopes calculated from the marginal COG that were continuous with the first stopes were considered for reserve inclusion.

Unplanned mining dilution was applied using DSO by allowing for dilution on the hanging wall and footwall. Different thicknesses of dilution were selected depending on the mining method. A 0.2 m shape surrounding the mineralization vein was created in the Mineral Resource block model using drill holes assays to evaluate the dilution grade outside the mineralization zone. Apart from these shapes, the diluting material was assumed to have zero grade. Some of the cemented rock fill (CRF) is expected to fall into the stope and be removed from an adjacent stope and/or be scraped off the stope floors during the mineralized material loading. This material was also assumed to have zero grade.

A 95% mining recovery was used for the various mining methods to account for the under-break and for the mineralized material left in place. Since the mining methods used are largely very selective, this percentage also included some of the rock left in place in the sill pillars.

1.13 Mineral Reserve Statement

The Mineral Reserve estimates have an effective date of January 4, 2021. The QP for the estimate is Mr. Carl Michaud, P.Eng., a GMS employee. COGs were calculated using input parameters, such as process recovery, processing costs, G&A costs, commodity prices, exchange rate, and marketing costs (Table 1-4 and Table 1-7).

Table 1-4: Mineral Reserve Estimate (effective date: January 4, 2021)

Classification	Tonnes (k)	Au (gpt)	Ag (gpt)	AgEq (gpt)	Contained Au (koz)	Contained Ag (koz)	Contained AgEq (koz)
Proven	336.5	6.21	552	1,091	67.1	5,971	11, 806
Probable	3,014.7	4.65	451	855	451.0	43,707	82,898
Proven + Probable	**3,351.2**	**4.81**	**461**	**879**	**518.1**	**49,679**	**94,740**

Notes:

1. The Mineral Reserve is estimated using the 2019 CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines and 2014 CIM Definition Standards for Mineral Resources & Mineral Reserves.
2. The Mineral Reserve is estimated with a variable COG which was calculated by vein width and economic and operating parameters. Refer to Subsection 15.2 for COG estimation details.
3. The Mineral Reserve is estimated using long-term prices of $1,410/oz for gold and $16.60/oz for silver.
4. A government gold royalty of 0.5% is included in the Mineral Reserve estimates.
5. The Mineral Reserve is estimated with a mining recovery of 95%.
6. The Mineral Reserve presented includes both internal and external dilution. The external dilution included a mining dilution of 0.5 m width on the hanging wall and footwall for the long hole mining method and a 0.2 m width on the hanging wall and footwall for the cut-and-fill and resue mining methods. Backfill dilution is also included and represents 7% for the long hole mining method and 10% for cut-and-fill and resue mining methods.
7. A minimum mining width of 1.5 m was used for the long hole and cut-and-fill mining methods. A minimum mining width of 0.5 m was used for the resue mining method.
8. The economic viability of the Mineral Reserve has been demonstrated.
9. AgEq is based on gold to silver ratio of 86.9:1 calculated using US$1,410/oz Au and US$16.60/oz Ag, with average metallurgical recoveries of 96% Au and 94% Ag.
10. The Qualified Person for the estimate is Mr. Carl Michaud, P.Eng., Underground Engineering Manager for GMS. The estimate has an effective date of January 4, 2021.
11. Totals may not add due to rounding.

Table 1-5: Mineral Reserve Estimate by Vein (effective date: January 4, 2021)

Vein	Classification	Tonnes (kt)	Au (gpt)	Ag (gpt)	AgEq (gpt)	Contained Au (koz)	Contained Ag (Moz)	Contained AgEq (Moz)
Babicanora Main	Proven	119.4	13.11	1,168	2,307	50.3	4,486	8,860
Babicanora Main	Probable	1,474.8	3.47	337	638	164.6	15,965	30,273
Babicanora Norte	Probable	514.8	5.87	682	1,192	97.1	11,289	19,732
Babi Vista	Probable	220.7	11.71	955	1,972	83.1	6,774	13,994
Babi Vista FW	Probable	18.8	8.64	867	1,618	5.2	524	978
Babicanora Sur	Probable	305.3	4.98	262	695	48.9	2,569	6,818
Babicanora Sur HW	Probable	6.1	3.12	102	373	0.6	20	73
Total Babicanora Area	**Proven**	**119.4**	**13.11**	**1,168**	**2,307**	**50.3**	**4,486**	**8,860**
Total Babicanora Area	**Probable**	**2,540.6**	**4.89**	**455**	**880**	**399.6**	**37,142**	**71,867**
Las Chispas	Probable	180.6	5.21	661	1,114	30.2	3,841	6,470
Giovanni	Probable	65.8	2.02	301	476	4.3	637	1,007
Gio Mini	Probable	64.6	2.51	318	536	5.2	660	1,112
Luigi	Probable	37.7	2.27	309	506	2.8	374	613

Vein	Classification	Tonnes (kt)	Au (gpt)	Ag (gpt)	AgEq (gpt)	Contained Au (koz)	Contained Ag (Moz)	Contained AgEq (Moz)
Luigi FW	Probable	38.7	1.67	178	323	2.1	222	402
William Tell	Probable	8.9	1.94	273	441	0.6	78	126
Total Las Chispas Area	**Probable**	**396.3**	**3.54**	**456**	**764**	**45.1**	**5,812**	**9,731**
Historical Stockpiles	Proven	162.6	1.23	108	215	6.4	565	1,123
Babicanora Stockpile + Open stope	Proven	54.5	5.93	525	1040	10.4	920	1,823
Babicanora Stockpile + Open stope	Probable	77.8	2.51	301	519	6.3	754	1,300
Total Mineral Reserve Estimate	**Proven**	**336.5**	**6.21**	**552**	**1,091**	**67.1**	**5,971**	**11,806**
	Probable	**3014.7**	**4.65**	**451**	**855**	**451.0**	**43,707**	**82,898**
	Proven+ Probable	**3,351.2**	**4.81**	**461**	**879**	**518.1**	**49,679**	**94,704**

Note:
1. Footnotes to Table 1-4 also apply to this table.
2. Babicanora Main Vein includes Babicanora Central Zone, Babicanora FW Vein and Babicanora HW Vein.
3. Babicanora Norte Vein includes Babicanora Norte HW Vein.

Factors that may affect the Mineral Reserve estimates include: geological complexity, geological interpretation, and Mineral Resource block modelling; COG estimations; commodity prices, market conditions and foreign exchange rate assumptions; operating cost assumptions; sustaining capital costs to develop; rock quality and geotechnical constraints, dilution and mining recovery factors; hydrogeological assumptions; and metallurgical process recoveries.

Dilution grades for dilution were estimated from the Mineral Resource block model for the planned and unplanned dilution. Table 1-6 presents the average expected unplanned dilution by area.

Table 1-6: Average Dilution by Mining Area

Areas	Average Dilution
Babicanora Main	48%
Babi Vista	90%
Babicanora Norte	56%
Babicanora Central	24%
Babicanora Sur	60%
Las Chispas	65%
Las Chispas Average All Areas	52%

There are no other environmental, legal, title, taxation, socioeconomic, marketing, political or other relevant factors known to the QP that would materially affect the estimation of Mineral Reserves that are not discussed in the Report. It is reasonably expected that all necessary government approvals will be issued for the Project to proceed.

1.14 Mining Methods

1.14.1 Geotechnical Considerations

A detailed geotechnical program was carried out in 2019–2020 by Rockland, consisting of site visits, field data collection, laboratory tests, analytical, and numerical modeling investigations.

The majority of resource drill holes were geotechnically logged and point load tested. Two rock mass classification systems were employed for the rock quality data collection program. Geotechnical data were collected from the immediate hanging wall, vein domain, and immediate footwall domains. The results are summarized in Table 1-7.

Table 1-7: Rock Quality Ranges based on the RMR_{76} Rock Mass Classification

Vein Name	Hanging wall Domain Range	Vein Footwall Domain Range	Footwall Domain Range
	25% - 75% Percentile	25%-75% Percentile	25%-75% Percentile
Babicanora Main (upper part) and Babicanora Central Zone	26-41 Poor-Fair	24-45 Poor-Fair	30-62 Poor-Good
Babicanora Main (lower part)	40-51 Poor-Fair	40-47 Poor-Fair	47-60 Fair
Babicanora Norte, Babicanora Norte Northwest, Babi Vista, Babicanora Sur, Babicanora FW, Las Chispas Vein, La Blanquita Zone, and Giovanni Veins	51-84 Fair- V. Good	41-76 Fair-Good	47-84 Fair- V. Good

Therefore, the rock quality of the various Las Chispas veins can be broadly divided into two main domains: "Poor–Fair/Good" and "Fair–Good/V. Good". The lower part of the Babicanora Main vein has a better rock quality than the upper portion. The Babicanora Norte, Babicanora Norte Northwest, Babi Vista, Babicanora Sur, Babicanora FW, Las Chispas, La Blanquita Zone, and Giovanni Veins have a range of "Fair-Good/V. Good" rock quality. Stope dimension analysis was conducted using the collected rock quality data and stability graph method for all veins. The results were used to assist in the selection of mining methods.

Ground support selection considered industry-standard empirical design guidelines and Rockland's experience with variable ground conditions. Ground support was recommended based on rock quality, the period in use (long-term or short-term), and the size of headings (excavation). The ground support consisting of Inflatable bolts (e.g., Swellex) or resin rebars with Split Sets and mesh was specified.

Several of the Babicanora Central stopes will come close to the ground surface; crown pillars are required for these stopes. A stability analysis, consisting of empirical and numerical modeling methods, was carried out to recommend the minimum crown pillar thickness. The crown pillar geometries were based on the three-dimensional solid model, and the rock quality was based on drill hole information from the crown pillar area. Using the scaled crown span empirical method, a minimum factor of safety of 1.5 with a probability of failure of 5–10% was chosen. Subsequently, CPillar limit equilibrium (Rocscience, 2019) and RS2 (Rocscience, 2019) finite element analyses were carried out to verify the empirical assessments. The results show that a crown pillar with strike lengths of 25 m, 50 m, and 100 m should have a minimum thickness of 12 m, 12.5 m, and 13 m, respectively.

1.14.2 Hydrological Considerations

A hydrological and hydrogeological study was completed by HRI. Work completed included: installation of six pressure probes to measure flow elevations; water elevation measurements taken for quality control purposes in three piezometers to verify measurements taken by SilverCrest; slug testing in three piezometers; and pump tests in a stope at the base of the historical workings at Las Chispas that is filled with groundwater, and which is the only known location in the historical operations that has groundwater.

There was insufficient rainfall during the monitoring period to generate any pressure variation between the six pressure probes.

Water elevation measurements indicated the presence of a perched phreatic surface considerably above the natural water table. The water table is at approximately 900 m elevation and the perched phreatic surface is at about 1,032 m elevation. The perched phreatic surface does not impact the historical workings, and for the purposes of the mine plan, will not require dewatering. Pump tests indicated that the host rocks had low permeability. Based on the pump test results, a maximum flow of about 9.4 L/s has been estimated at the end of operation of Las Chispas Area. There is insufficient data to determine if this flow rate will be sustained in the long-term. As a result, the mine plan in this area was designed with a dewatering system in the lower levels with a pumping capacity of 9.4 L/s; however, this pumping system will not be required until late in the mine life.

As the majority of the workings will be above the water table elevation of 900 masl, groundwater inflows are not expected to be a concern to mining operations. No impacts to surrounding perennial streams or valley bottoms are expected from mine dewatering activities, since these are typically dry other than during short-term, low precipitation rainfall events. The Rio Sonora, located 7 km west of the future operation, is considered to be too distant to be affected by any future mine-related pumping.

1.14.3 Mining Methods

The mine design was based on a production rate of 1,250 t/d and will be reached by maintaining a proper balance between productive and selective mining methods. The proposed mining approach will use variations of long hole stoping and cut-and-fill mining methods via several access drifts and ramps. These methods are appropriate to the sub-vertical vein geometry and to veins that have thicknesses ranging from 0.5–10 m.

The long hole stope mining methods will include long hole longitudinal retreat stoping and Avoca. These methods will be used in mining areas where vein thicknesses are >1.5 m and where ground conditions are fair–good. Avoca requires multiple accesses to the veins, whereas long-hole longitudinal retreat typically requires only one access. Variations of cut-and-fill mining methods will include cut-and-fill with uppers, cut-and-fill with breasting and resuing. Cut-and-fill with uppers will be used in mining areas with fair ground conditions and where the vein thickness is >1.5 m. Cut-and-fill with breasting will be used in mining areas with adverse ground conditions, and where the vein thickness is >1.5 m. Resuing will be used in mining areas where the vein thickness is <1.5 m, independent of ground conditions. Mining areas will be accessed via three portals: the Santa Rosa, Babicanora Central, and San Gotardo portals.

The level spacing was selected based on the mining method chosen and the efficiency of long hole drilling. The level distance is generally 18 m for areas where cut-and-fill will be the predominant mining method. A 15 m level spacing was used for the Babicanora Main to reduce the length of the long hole drilling, and thus reduce the deviation of the drilling. This allows for reduced dilution and better vein recovery. The level distance for the cut-and-fill and resuing was set at 18 m to reduce the total development required.

Declines will provide access from the portals to mining levels for all veins and zones. Depending on the mining methods selected, drifts may be required to access the mineralized zones. Excavation dimensions are set at 4.5 m x 4.5 m. Rigid ventilation ducts, freshwater, compressed air, and dewatering pipes and power cable will be installed in declines and main access drifts. The ramp incline is set at -15% grade. Loading and hauling will be carried out to the nearest muck bay that will be located at a maximum distance of 200 m. Dimensions will allow the use of 10 t scoops, 30 t trucks and two-boom jumbos. This equipment will be sufficient to achieve the daily development productivity target.

Pivot drives will be excavated for zones mined using cut-and-fill methods (breasting, uppers and resue). Only Babicanora Main, which will be mined by long hole method, will not have a pivot drive. The pivot drive dimensions are set at 4 m x 4 m. Fan, freshwater and compressed air pipes will be installed. The pivot drive incline will range from -18% to +18% grade. Loading and hauling will be carried out to the nearest muck bay, which will be located at a maximum distance of 60 m. The first pass advance will be conventional development and will be followed by five backslash passes. Dimensions will allow the use of 10 t scoops and two-boom jumbos.

Long hole sills will be excavated in the Babicanora Main Vein. Dimensions are set at 3 m x 3 m. The long hole sill will be developed following geology. Services such as fan pipes, fresh water etc. will be located directly in the long hole sills. Long hole sills will be used to drill and blast stopes and to mine muck. Loading and hauling will be carried out to the nearest muck bay that will be located at a maximum distance of 250 m. Dimensions will allow the use of 3 t scoops and single-boom jumbos.

Development of cut-and-fill sills will include sills developed by breasting, uppers and resuing Dimensions are set at 3 m L x 3 m H for upper and breasting sills and 2.6 m L x 3 m H for resuing sills. Services such as fan pipe, fresh water etc. will be located directly in the cut-and-fill sills. Loading and hauling will be carried out to the use of 1.5 t or 3 t scoops, single-boom or two boom jumbos, or jacklegs, depending on the dimensions.

Several raises will be required for the different mine zones. Raises for all zones will be 3 m in diameter for main raises (drilled by raise boring) and 3 m x 3 m for internal raises. Some ventilation raises will also serve as escape ways. A prefabricated modular manway system will be installed in the emergency exits. This system has been proven to save time and costs to establish the manway. Ventilation raises between the levels will be excavated by the drop raise method.

Mining operations will extract from 16 principal veins: Babicanora Main including Area 51 and Babicanora Central zones, Babicanora FW, Babicanora HW, Babicanora Sur, Babicanora Sur HW, Babicanora Norte including Area 200 Zone, Babicanora Norte HW, Babi Vista, Babi Vista FW, Luigi, Luigi FW, William Tell, Giovanni, Gio Mini and Las Chispas including Area 118 Zone. These veins are grouped into six main mining areas: Babicanora Main, Babicanora Sur, Babi Vista, Babicanora Norte, Babicanora Central Zone and Las Chispas. Each of these mining areas will be serviced by supporting infrastructure including power distribution, compressed air distribution, water supply, ventilation, dewatering and communications.

Three drill types will be used depending on the size of the excavations. Mechanized bolters are planned for the ground support installation. Depending on the excavation size, two types of bolters will be used: standard size bolters for waste development, and narrow vein bolters in other excavations. Jackleg and stoper drills are planned for bolting in raises, drilling safety bays, service drilling, and ground support installation. Electric-hydraulic long hole drills are planned for the production holes.

Based on the mining method and excavation dimensions a fleet of 1.2 t, 3 t, 3.5 t and 10 t load–haul–dump (LHD) vehicles were selected. The 1.2 t, 3 t and 3.5 t LHD units will remove the mineralized material or waste from the stope and transport it to a muck bay, where the 10 t LHD can directly load haul trucks. In long hole stopes, an LHD equipped with remote control will be used to keep personnel away from unsupported ground. LHDs will tram the material to a nearby re-muck bay.

A 30 t diesel truck fleet was selected to bring rock to surface. The trucks will be loaded by the 10 t LHD at the loading point. Each sector of the mine will have loading points on each production level. The trucks will travel to surface where broken (not crushed) mineralized material or waste will be unloaded on a surface transfer pad. From this pad, mineralized material will be transported by surface mining trucks or via loader to the process plant. The waste not used for rockfill will be transported by surface mining trucks to the appropriate waste rock storage facility (WRSF). All mining activities will be completed via a contractor that will supply adequate underground mining equipment for the different mining activities.

SilverCrest will supply fuel, electricity, explosives, explosives accessories, ground support consumables (e.g., rebars, wire mesh), construction consumables (e.g., steel), and services consumables (e.g., piping, rigid ventilation ducts).

Development has already commenced on the Babicanora Main area. Material mined prior 2021 was stockpiled on surface. Including in-vein development and production, a total of 3.35 Mt including historical stockpiles will be mined over a period of approximately 8.5 years. The pre-production period will start in 2021 and will end during the second quarter of 2022. Development during the pre-production period will be carried out in the Babicanora Main, Babicanora Norte and Babi Vista mining areas. The average grade mined is forecast to be 4.81 gpt Au and 461 gpt Ag or 879 gpt AgEq. The production schedule was established for an annual mill feed production rate of 456,200 t/a. The schedule was built to maximize tonnage throughput to the process plant and, where possible, target higher-grade mining blocks in the early stage of the mine life. The single heading advance rate were set a 2.5 m/d with a scheduling maximum constraint of 300 m of lateral advance per month. This performance level was shown to be achievable at the Project in 2019–2020. Total mine underground capital and operating development will be 53,554 m and 32,220 m, respectively. Underground overall development will average 9,500 m/a or 26.0 m/d from Year 1 to Year 8.

The mine will operate seven days per week with two shifts of 12 hours each. Development and production crews will be on a schedule of 14 days working/seven days off, for two 12-hour shifts. The maintenance crew will also be on the same schedule of 14 working days/seven days off, for 12 hr/shift, night and day, or days only. This schedule is equivalent to operating 365 days per year. Staff mine labour, including mine management and the technical department will work on five, eight-hour shifts, per week.

Mine services will include ventilation, water supply, power, provision of cemented rock fill, compressed air, fuel, surface and underground communications networks, explosives storage and handling, and transport for personnel and materials. All major mechanical maintenance will be performed on surface at the existing workshop. Only minor maintenance and emergency work will be performed underground by mobile maintenance crews.

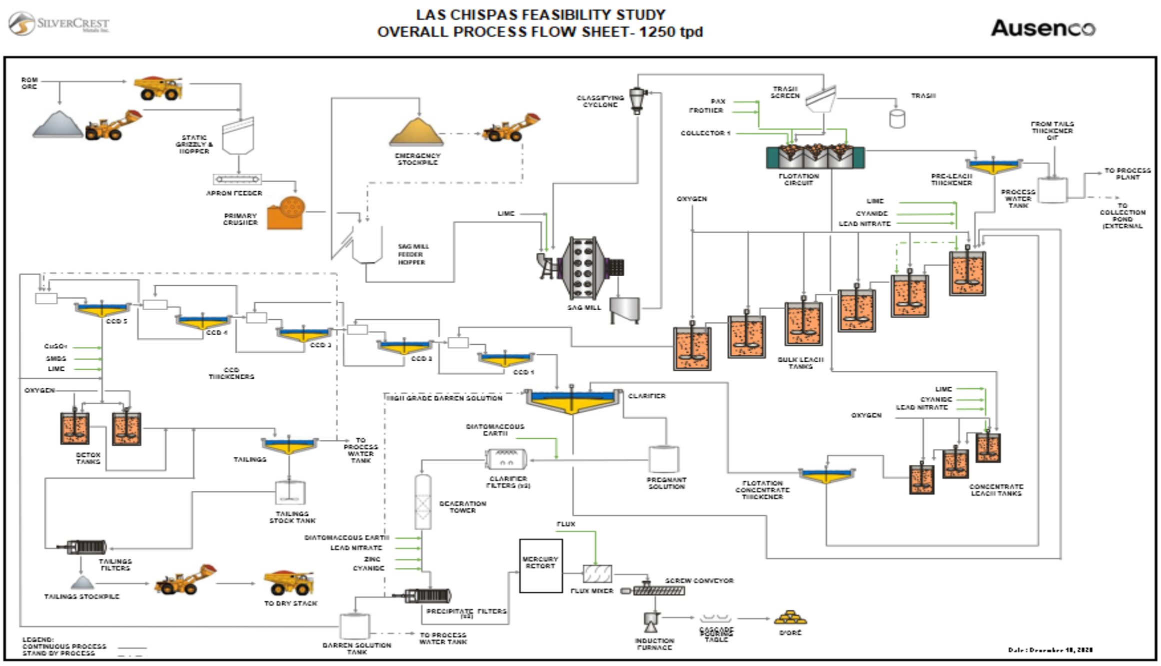
1.15 Recovery Methods

Based on the metallurgical testing results, Ausenco's design expertise and experience from local operations treating similar types of mineralized material, the planned flowsheet, which is designed for treatment of a variety of feed grades, is flexible and robust. The flowsheet is based on well-proven unit operations in the industry and there are no unique or novel processing methods required for gold and silver recovery.

The process plant will be located at the mine site and will receive blended feed material from a number of different mineralized veins. The key project design criteria for the plant are:

- Major equipment designed for nominal throughput of 1,250 t/d with the ability to accommodate increased throughput up to 1,750 t/d via an expansion.
- Crushing circuit availability of 70%, supported by the use of a surge bin, a dedicated feeder and an emergency stockpile to provide continuous feed to the balance of the process plant; and
- Process flowsheet (Figure 1-1) including semi-autogenous grinding (SAG), flotation, independent cyanide leaching circuits for both flotation concentrate and tailings streams, Merrill Crowe circuit, and tailings handling facilities, with an overall availability of 91.3%, given:
  - Axb of 41 and BWI of 19.4 kwh/t;
  - Design head grades of 8 gpt Au and 800 gpt Ag with the ability to handle peak head grades of as much as 13 gpt Au and 1,300 gpt Ag; and,
  - Overall process recovery of 97.6% gold and 94.3% silver for LOM average grades.

The total operating power for the process plant will be 4.6 MW. Provision will be made for raw water to be supplied from the underground mine, the fresh water (storm) pond, the Sonora Valley, or any combination thereof pending availability and requirements. Wherever possible in the plant, process water or barren solution will be used to minimize freshwater consumption. Potable water will be sourced from the sediment-free water in the raw water tanks and treated prior to distribution or shipped to site. Plant consumables will include quick lime, sodium cyanide, lead nitrate, oxygen, flocculants, coagulant, Aerofloat 208, PAX, frother, diatomaceous earth, zinc powder, copper sulphate, sodium metabisulfite, antiscalant, and flux.



Flowsheet designed by Ausenco

Figure 1-1: Overall Process Flow Diagram

1.16 Project Infrastructure

1.16.1 Infrastructure Requirements

Infrastructure that will be required for the mining and processing operations will include:

- Underground mine, including portals, ramps and vents;
- Roads: main access road, site access road, bridge crossing, borrow pit haul road, filtered tailings storage facility (FTSF) haul road, waste rock storage facility (WRSF) haul road, and explosives access road;
- Diversion and collection channels, culverts, and containment structures;
- Site main gate and guard house;
- Construction camp;
- Power and waster distribution;
- Warehouse and truck shop, offices, process plant dry facility, medical clinic, and nursery;
- Explosives magazines;
- Processing plant;
- Control room;
- Doré room;
- Assay laboratory (off-site facility);
- Reagent storage facility;
- Water treatment plant;
- Mineralized stockpiles and WRSFs;
- FTSF;
- Hazardous waste containment facility; and
- Exploration core shack.

Figure 1-2 shows the proposed site layout.

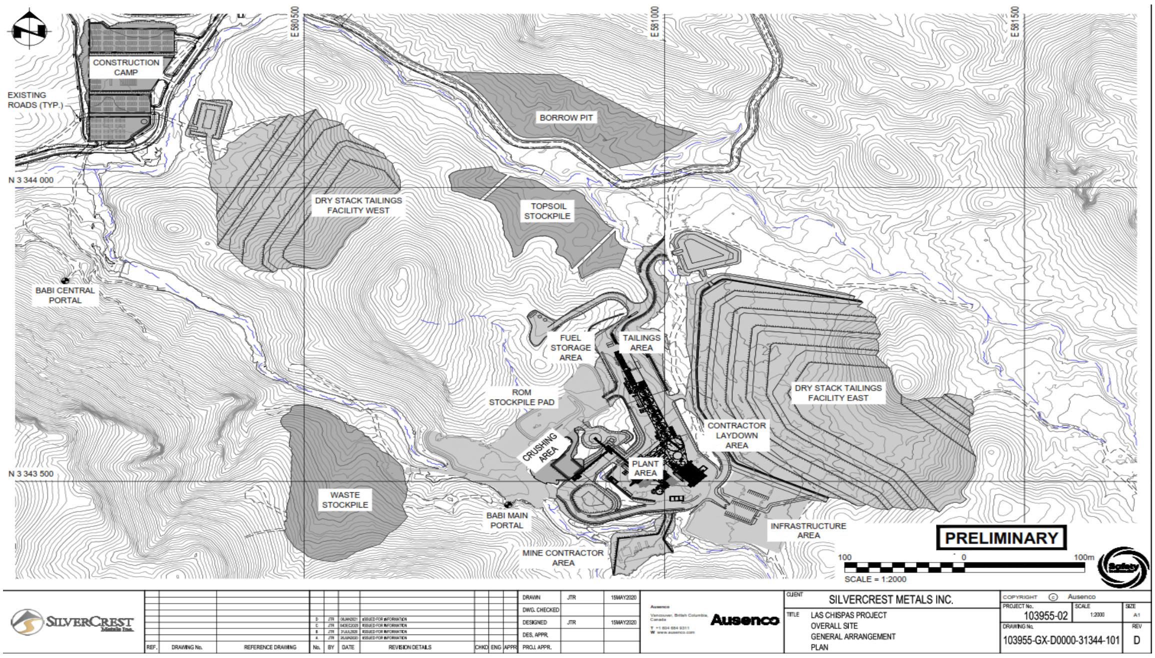


Figure 1-2: Proposed Site Layout

1.16.2 Waste Rock Storage Facility

The WRSF will have a capacity of approximately 0.9 Mt and will be a temporary facility that will store development waste before returning it to be used as rock fill in mined-out stopes. Although current waste rock characterization indicates no potential for acid generation, contact water channels as well as contact water ponds have been considered to capture surface water runoff from this temporary structure and allow for water sedimentation/monitoring before returning it to the process plant as make up.

1.16.3 Stockpile

The mineralized material stockpile will have a capacity of about 0.3 Mt with segregated piles by grade (or clary content). Non-contact water management structures will be used around the stockpile area to minimize contact water generation. Contact water channels and contact water ponds were designed to manage and capture runoff from this temporary structure.

1.16.4 Filtered Tailings Storage Facility

A FTSF concept was adopted based on the mine plan, the limited available construction materials, and to avoid risks associated with storage of conventional slurried tailings behind a dam. Tailings will be thickened, filtered, and delivered by trucks to the FTSF. Two facilities were designed to store approximately 4.5 Mt of tailings. However, based on the current LOM and estimated production only the East FTSF will be constructed, and will have a storage capacity of 3.5 Mt of filtered tailings.

Each facility would have an overall slope of 2.8:1 (H:V), slope between benches of 2.2:1 (H:V), and maximum approximate heights of 50m to 56 m (measured from the lowest portion of the starting buttress to the maximum elevation of the dry stacks). The East FTSF will be located about 530 m northeast of the process plant and will cover an area of 101,932 m^2. The NW FTSF would be about 300 m northwest of the process plant and will cover an area of 47,758 m^2.

The FTSF will include contact water collection channels, contact water collection/storage ponds, sub-drain collection systems, and access roads. Non-contact water diversion channels will be constructed to reduce the amount of surface contact water generated from the FTSF area.

1.16.5 Power and Fuel

Electrical power will be supplied to site from the national grid, by way of an overhead power line, rated to carry 8.5 MVA at 33 kV. Connection to the grid will be at the Nacozari de Garcia substation, which is 74.4 km from the Project. The Comisión Federal de Electricidad (CFE) has approved the transmission line and anticipates completion of the upgrades and construction to be completed with power supply available at Las Chispas by Q1 2022. Approximately 49 km of line construction is required, and these right-of-ways have been acquired.

Diesel fuel requirements for the mining equipment, process and ancillary facilities will be supplied from two modular above-ground diesel fuel storage tanks.

1.16.6 Camps

The Project will require the use of a temporary construction camp. The camp will generally be self-contained and have its own power generation and heating capabilities, potable water treatment plant, and sewage treatment plant. The first phase of the 520 man-camp (320 rooms) is scheduled to be completed in February 2021. The second phase (100 rooms) is scheduled for Q1 2021 and the third phase (100 rooms) is scheduled for Q2 2021. Given the risk associated with the operation of a camp during a pandemic, SilverCrest has implemented a second facility directly in the small town of Arizpe. The facility is already operational and for the most part supports the exploration needs of the company in a confined setting. The facility has a capacity of 150 beds to supplement the construction camp. The camps are intended as temporary facilities only and will be demobilised upon completion of project construction.

1.16.7 Water Management

Water required for the Project will be supplied as groundwater from dewatering of the underground mine and or from the Sonora Valley, as required. The Project design includes water diversion features to divert precipitation and groundwater away from project infrastructure and direct it to natural receiving streams to minimize the generation of contact water. The layout also includes water collection ponds to collect any contact water that is produced, and to store any excess water from the underground workings such that it can be recycled for use in the process plant. There is not expected to be any water discharged from the plant site.

1.17 Market Studies and Contracts

Detailed market studies on the potential sale of gold and silver doré were not completed. The doré bars produced at the Project can be expected to have variable gold and silver contents and a variable gold to silver ratio, depending mainly on the corresponding gold and silver grades of the feed material being processed at any given time. Over the projected LOM, the metal content is expected to be 0.5%-1.5% gold and 90%-95% silver with the balance impurities. Prior to production, SilverCrest will engage with gold and silver buyers and refiners, and make the necessary arrangements to safely transport, refine, and sell the doré.

Gold and silver doré can be readily sold on many markets throughout the world and the market price can be ascertained on demand. Numerous mining operations produce and sell gold and silver doré in Mexico and elsewhere, and there is sufficient information available in the public domain or furnished to SilverCrest directly from third party refiners or comparable doré producers to use as the basis for the economic analysis.

Metal pricing for financial analysis was agreed upon based on consideration of various metal price sources. This included review of consensus price forecasts from banks and financial institutions, three-year trailing average of spot prices, and current spot prices. The metal pricing for the base case economic model was:

- Gold price of $1,500/troy oz payable; and,
- Silver price of $19.00/troy oz payable.

No contracts were entered into at the Report Effective Date for mining, facility operations, refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements. It is envisaged that SilverCrest would sell any future production through contracts with a refiner, or on the spot market, as applicable. It is expected that when any such contracts are negotiated, they would be within industry norms for projects in similar settings in Mexico.

1.18 Environmental Studies, Permitting and Social or Community Impact

1.18.1 Environmental Considerations

Environmental surveys and studies for the Project were completed in support of permit applications. Completed studies include climate, flora, fauna, air quality, noise, surface and groundwater quality. These were compiled by LLA into an environmental baseline report and submitted July 14, 2020 to the Ministry of Environment and Natural Resources of Mexico (SEMARNAT in the Spanish acronym).

Samples of waste rock from exploration drill holes and test pits in the FTSF footprint area were submitted for testing for acid rock drainage (ARD) and metals leaching (ML) potential. Potentially leachable metals included barium and lead, but in concentrations that were well below the maximum allowable limits. The majority of samples showed no ARD potential; those samples that had elevated neutralization potential/acid potential ratios were still below SEMARNAT thresholds. Tailings samples generated from metallurgical testwork at SGS Lakefield were subjected to acid base accounting (ABA) and net acid generation (NAG) testwork. The majority of samples showed non-acid forming (NAF) characteristics in NAG testing.

No known environmental liabilities exist in the Project area from historical mining and processing operations. Soil and tailings testing were conducted as part of the overall sampling that has been ongoing at site. To date, there are no known contaminants in the soils. Water quality testing is currently ongoing through baseline environmental studies.

1.18.2 Permitting Considerations

SEMARNAT requires a number of studies be completed to support award of environmental permits to conduct exploration, or construct and operate a mine. Given the Project setting, these include a Mining Exploration Permit, Environmental Impact Assessment (MIA in the Spanish acronym) and Change in Land Use (CUS in the Spanish acronym) permit. In addition to the SEMARNAT requirements, permits must also be obtained in certain instances from the Comisión Nacional del Agua (CONAGUA), Comisión de Ecología y Desarrollo Sustentable del Estado de Sonora (CEDES), Secretaría de la Defensa Nacional (SEDENA) and local municipal authorities. The final licence requirement is the environmental operating licence (LAU in the Spanish acronym). The LAU sets out operating conditions, including specifications around equipment and processes, production, air emissions, hazardous waste and water impact obligations.

LLA worked with its permitting team in Mexico to identify the key environmental permits and other Mexican regulatory permits required to construct and operate a mine in Sonora state, Mexico, and to identify which regulatory authorities grant such permits. A total of 27 key permits were identified, of which 21 have been granted, four are pending, and the remaining two permits will be applied for in due course. Granted permits have varying terms, ranging from one year to unlimited terms. Permits will be renewed as required. LLA has all permits required to construct the underground infrastructure and process plant.

1.18.3 Closure Considerations

A Conceptual Closure Plan was prepared in general accordance with applicable Mexican standards. Under Mexican law, mining may be initiated under a Conceptual Closure Plan with a Detailed Closure Plan being developed later in the Project life.

Wood prepared a conceptual closure cost estimate for the planned operation, using a combination of information derived from the Feasibility Study, drone imagery of existing facilities and landforms, a database of itemized costs from local contractors working on similar projects in the area, and assumptions derived from Wood's experience in mine closure. The estimated cost is approximately $3.4 M. Closure costs were assumed to be disbursed over a period of approximately three years, following the cessation of production.

1.18.4 Social Considerations

The Sonora Valley is an isolated community set in a region of rugged topography. The areas planned for mining activity are not visible from the local communities or from adjacent roads. As of November 2020, SilverCrest employed 85 people from the Sonora Valley. There are four main ejido groups that SilverCrest have been engaging with, three of which will be impacted by mining operations (Ejido Bamori, Ejido Arizpe, and Ejido Sinoquipe) and the fourth (Ejido Los Hoyos) will be impacted by the powerline. Impacts to Indigenous populations were examined. There are no indigenous populations located within 10 km of the Project.

A social baseline study, completed in 2019–2020, found key areas of community concern were: water usage, and water safety; a lack of information on the Project; concerns around an environmental incident in 2014 that was caused by a different mining company (100 km north of the Project); a wish to see improvements in the local infrastructure; that environmental safety and appropriate mine closure protocols should be in place to protect the region at the end of the LOM; and job creation with a focus on opportunities being made available for women.

In early 2020, SilverCrest engaged two third-party consultants to complete a materiality assessment designed to identify the key risks facing the company including potential risks relating to SilverCrest's relationship with, and impact on, local communities. A detailed stakeholder analysis was completed. Key findings were centered around climate and water risks, community health issues (mining, food, water), environmental safety of the local river and agriculture, employment opportunities, a desire for improved infrastructure (sports, recreation, health) and a concern regarding a potential influx of people from outside the community taxing local infrastructure. The materiality assessment results will be the basis of a company-wide Environmental and Social Management System.

SilverCrest has formalized a communication strategy that employs direct outreach, social media, company-generated videos, flyers, posters and workshops. SilverCrest has set up a whistle blower policy and hotline and, at the Report effective date, was in the process of finalizing a grievance mechanism process.

SilverCrest joined the Sonoran Mining Cluster, an organization consisting of mining companies based in Sonora, that aims to share best practices on social license concerns, innovation, sustainability, community relations and responsible mining.

SilverCrest is one of the major sponsors in a non-profit organization (Impulso Koria A.C.) located in Arizpe. Impulso Koria's objectives include supporting local infrastructure, education and health care needs. SilverCrest communicates with Impulso Koria representatives on a regular basis as part of local CSR efforts.

1.19 Capital and Operating Costs

1.19.1 Capital Cost Estimates

The capital cost estimate was prepared with a base date of Q3-2020, except for Owner's costs that were based on Q4-2020. The accuracy range of the capital cost estimate is ±15%. The estimate assumes US$1:CAD1.325, and US$1:MXN20.00. Pre-production costs (operating costs) were not included in the initial capital cost estimates.

Project LOM capital costs total $265.0 M, consisting of:

- Initial capital costs: include all costs required to construct the surface facilities and underground development required to commence a 1,250 t/d operation. The remaining initial capital cost is estimated to be $137.7 M after the subtraction of $25.8 M of sunk capital expensed in 2020 (from a total of $163.5 M). The sunk capital was expensed for initial earthworks, some surface infrastructure, initial and detailed engineering, procurement and contract management (EPCM) milestone payments to finance long-lead equipment item purchases;
  - On December 31, 2020, SilverCrest signed an EPC lump sum turnkey price contract for $76.5 M to construct the process plant. Construction is expected to start in February 2021 with production start up in Q2-2022.
- Sustaining capital costs: include all the costs required to sustain operations, with the most significant component being underground mine development. Sustaining capital costs total $123.9 M over the LOM.
- Closure costs: include all of the costs required to close, reclaim, and complete ongoing monitoring of the mine once operations conclude. Closure costs total $3.4 M.

Initial capital costs are summarized in Table 1-8.

Table 1-8: Initial Capital Cost Summary

Project Scope	Total Cost ($ M)
Mine	27.7
Process plant	68.0
Tailings management	3.1
Infrastructure	23.3
Owners costs	18.2
Subtotal	**140.3**
Contingency	23.3

Project Scope	Total Cost ($ M)
Total Initial Capital Cost	**163.5**
Sunk Capital	25.8
Total Initial Capital Cost (remaining)	**137.7**

Note: Totals may not add due to rounding.

Sustaining capital costs consisted of the direct costs of mine development, process plant, site infrastructure, FTSF development, and mobile equipment.

The sustaining capital cost estimate from the start of operations to the end of the LOM is provided in Table 1-9.

Table 1-9: Sustaining Capital Cost Summary ($ M)

Calendar Year	2022	2023	2024	2025	2026	2027	2028	2029	2030	LOM
Production Year	1	2	3	4	5	6	7	8	9	
Process plant		1.4								1.4
Mobile equipment			0.3	0.3	0.3	0.3	0.3			1.3
Dry stack tailings						0.2	0.1			0.4
Mine	10.2	17.7	20.0	19.2	17.0	15.0	13.1	8.0	0.6	120.9
Total sustaining capital costs	**10.2**	**19.1**	**20.2**	**19.4**	**17.2**	**15.5**	**13.5**	**8.0**	**0.6**	**123.9**

Note: All numbers are rounded.

1.19.2 Operating Cost Estimates

The operating costs were completed in US dollars, unless specified, and where required were converted to US dollars using the same exchange rates as for the capital cost estimate.

The projected mine operating costs are $71.40 per tonne of material milled. The average LOM operating cost, at a design mill feed rate of 1,250 t/d, was estimated at $118.49/t of material milled. The operating cost is defined as the total direct operating costs including mining, processing, and G&A costs.

It was assumed that once construction is complete, operations personnel will reside in, or be available from nearby towns or villages. There would be no accommodation provided at site; Personnel will be transported to site by SilverCrest. It is assumed that the mining contractor would hire personnel throughout Mexico and be responsible for lodging and catering of those personnel.

The operating costs exclude doré shipping and refining charges. Costs associated with doré transport and refining were included in the financial analysis in the applied payabilities for gold and silver values recovered.

The operating cost estimate is provided in Table 1-10.

Table 1-10: Operating Cost Summary

Area	LOM Average Operating Cost ($/t milled)
Mining	71.40*
Process and tailings management	31.69
G&A	15.40
Total LOM operating cost	**118.49**

Notes: * Includes stope development but excludes capitalized underground development.

1.20 Economic Analysis

The results of the economic analysis represent forward-looking information as defined under Canadian securities law. The economic information that is forward looking includes the following:

- Proven and Probable Mineral Reserves that have been modified from Measured and Indicated Mineral Resource estimates;
- Cash flow forecasts;
- Assumed commodity prices and exchange rates;
- Proposed mine and process production plan;
- Projected mining and process recovery rates;
- Ability to have doré refined on favourable terms;
- Proposed capital and operating costs;
- Assumptions as to closure costs and closure requirements; and,
- Assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what is assumed;
- Unrecognised environmental risks;
- Unanticipated reclamation expenses;
- Unexpected variations in quantity of mineralization, grade or recovery rates;
- Geotechnical or hydrogeological considerations during operations being different from what was assumed;
- Failure of mining methods to operate as anticipated;
- Failure of plant, equipment or processes to operate as anticipated;
- Changes to assumptions as to the availability and or generation of electrical power, and the power rates used in the operating cost estimates and financial analysis;
- Ability to maintain the social licence to operate;
- Accidents, labour disputes and other risks of the mining industry;
- Changes to interest rates, tax rates or applicable laws;
- Receipt of any required permits, beyond those already held by SilverCrest; and,

- Impacts to manpower availability and delays to the construction schedule due to the COIVID-19 global pandemic.

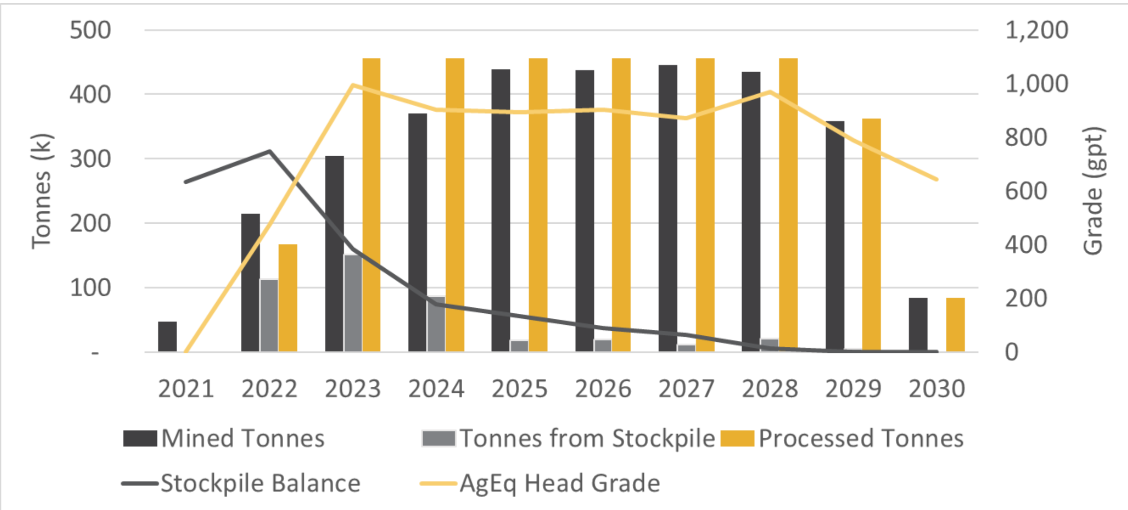
A pre- and post-tax economic analysis was completed on the basis of a discounted cash flow model featuring a 5% discount rate. The analysis used constant (real) 2020 US$ and the project cash flows were modelled in annual periods. The model assumed a 17-month physical construction period, and production period of 8.5 years, including the first year and final year that will see production for only a portion of those two years. Table 1-11 provides the LOM doré production forecast. Figure 1-3 illustrates the annual material movements.

Table 1-11: LOM Doré Production Forecast

	Unit	Total	2022	2023	2024	2025	2026	2027	2028	2029	2030
Mill Feed	kt	3,351	167	456	456	456	456	456	456	363	84
Mill Feed Grade	gpt Au	4.81	2.53	5.23	5.3	5.05	4.9	5.12	4.89	4.16	3.23
	gpt Ag	461	254	541	442	455	478	428	544	427	362
	gpt AgEq	879	474	996	903	893	904	873	968	789	643
Process Recovery	% Au	97.0	90.1	96.2	97.6	97.6	97.6	97.6	97.6	97.6	97.6
	% Ag	93.7	87.0	92.8	94.3	94.3	94.3	94.3	94.3	94.3	94.3
Production in Doré	**koz Au**	**504**	**10**	**74**	**76**	**72**	**70**	**73**	**70**	**47**	**10**
	koz Ag	**46,629**	**1,004**	**7,367**	**6,116**	**6,291**	**6,608**	**5,920**	**7,524**	**4,695**	**1,104**
	koz AgEq	**90,392**	**1,912**	**13,786**	**12,715**	**12,571**	**12,708**	**12,284**	**13,602**	**8,812**	**2,002**

Notes:

- The AgEq is based on Au:Ag ratio of 86.9:1, calculated using metal prices of $1,410/oz Au and $16.60/oz Ag, and metal recovery values of 96% Au and 94% Ag.
- All numbers are rounded.



Note: Figure prepared by SilverCrest, 2021.

Figure 1-3: Material Movement Schedule

The economic model was based on a gold price of $1,500/oz and a silver price of $19.00/oz. The freight, and treatment and refining terms for the doré were based on local rates and industry standard terms, respectively.

Initial capital expenditures were based on the required construction and development beginning in 2021 and continuing until plant start-up is achieved in 2022; these initial expenditures also included underground development expenses. However, underground development and construction expenses incurred prior to January 2021 were not included in the financial model as these were considered sunk costs at the point of a construction decision. Total construction and development expenses considered sunk capital prior to January 2021 were estimated to total $25.8 M.

Sustaining capital costs were incorporated on a year-by-year basis over the LOM, and operating costs were deducted from gross revenue to estimate annual mine operating earnings.

Royalties and fees included the following:

- Government earnings before income, taxes, and depreciation and amortization (EBITDA) royalty of 7.5% of income less authorized deductions, applicable to mining companies;
- Extraordinary government NSR of 0.5%, applicable to gold and silver operations;
- Concession fees (included in G&A operating costs).

Working capital for the project in 2021 and 2022 was estimated at $25.6 M. This estimate was reached based on consideration of required inventory, and taxes and duties.

Allowable deductions were applied to cash flows based on estimated capital costs and expenses that SilverCrest has incurred to date, which include:

- Capital costs depreciated at 12%;
- Non-fixed development capital depreciated at 10%;
- Sustaining capital expenses, depreciated in the year expensed;
- Pre-development exploration costs of $91.5 M depreciated at a rate of 10%; and
- Historical net operating losses (NOLs) applied in 2023 of $55.9 M.

The resulting taxable income was estimated at $715.1 M. SilverCrest applied a tax rate of 30% to this amount over the LOM for an estimated tax amount of $214.5 M over the LOM. A review was completed with the NOLs excluded, and the Project economics remained positive.

No salvage value was assumed for any items. No consideration of financing was made. The model considers the cash flow only at an asset level and assumes 100% equity ownership. The economic analysis demonstrated that the mine plan had positive economics under the assumptions used. The Project post-tax net present value (NPV) at a 5% discount rate was estimated to be $486.3 M with an internal rate of return (IRR) of 52%. The Project would achieve payback in 1.0 years.

The production schedule was incorporated into a pre-tax financial model to develop the annual recovered metal production. The annual at-mine revenue contribution of each metal was determined by deducting the applicable treatment, refining, and transportation charges (from mine site to market) from gross revenue.

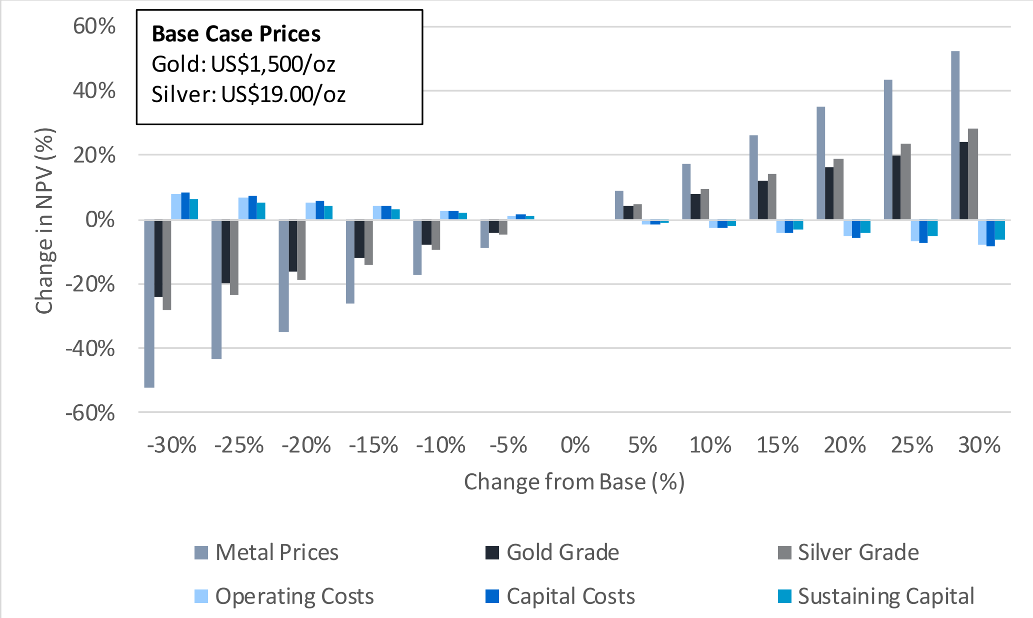
The pre- and post-tax cash flows were based on 90.3 M payable ounces of AgEq (based on gold to silver conversion ratio of 86.9 to 1 gold). A project financial summary is shown in Table 1-12.

Table 1-12: Economic Analysis Summary

Project Metric	Unit	Value
Gold Price	$	1,500
Silver Price	$	19.00
Mine Life	Year	8.5
Nominal Process Capacity	t/d	1,250
Average Annual Gold Production (LOM)	koz Au	55.96
Average Annual Silver Production (LOM)	koz Ag	5,181
Average Annual Silver Equivalent Production (LOM)	koz AgEq	10,044
Average Annual Gold Production (2023-2029)	koz Au	68.97
Average Annual Silver Production (2023-2029)	koz Ag	6,360
Average Annual Silver Equivalent Production (2023-2029)	koz AgEq	12,354
Initial Capital Expenditure	$M	137.7
LOM Sustaining Capital Expenditure	$M	123.9
LOM C1 Cash Costs (LOM)	$/oz AgEq	4.40
LOM C1 Cash Costs (2023-2029)	$/oz AgEq	4.13
Pre-Tax NPV (5%)	$M	655.9
Pre-Tax IRR	%	63
Post-Tax NPV (5%)	$M	486.3
Post-Tax IRR	%	52
Undiscounted Post-Tax Cash Flow (LOM)	$M	656.4
Payback Period (undiscounted, post-tax cash flow)	Year	1.0

Note: C1 cash costs represent costs incurred at each processing stage, from mining through to recoverable metal delivered to market, less net by-product credits.

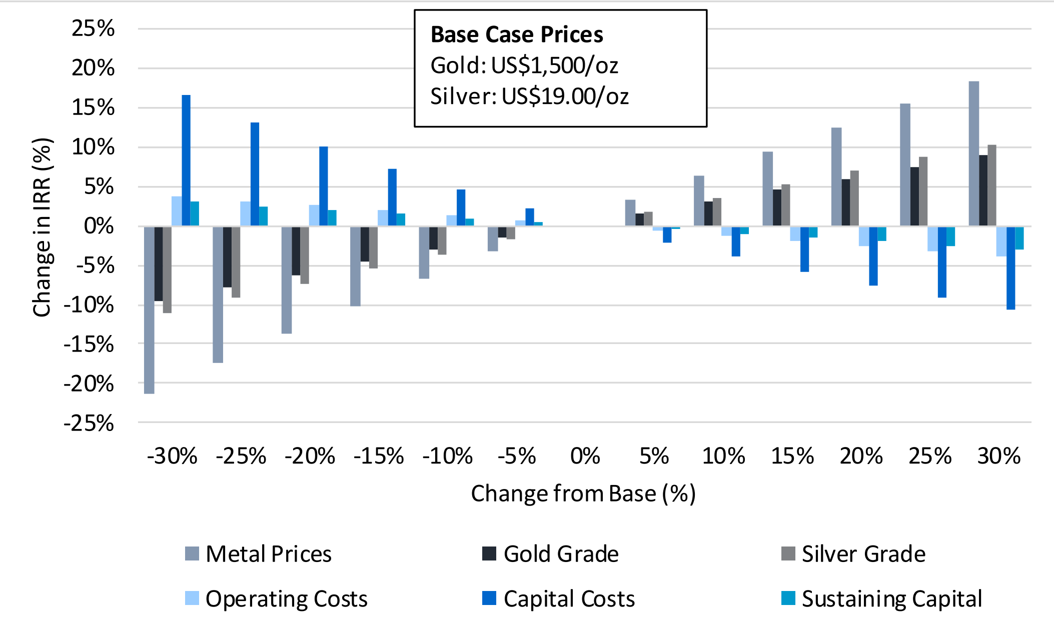
Sensitivity analysis was completed to evaluate the response of the project NPV and IRR to changes in assumptions on key inputs of metals prices, grades, and capital costs and operating costs. The post-tax results across a range of ±30% are shown in Figure 1-4 and Figure 1-5. The Project is most sensitive to changes in metal prices, less sensitive to changes in capital and sustaining costs, and least sensitive to changes in operating capital costs. Grade sensitivity mirrors the sensitivity to metal prices. The base case metal prices of $1,500/oz Au and $19.00/oz Ag were used in this sensitivity analysis.



	NPV ($ M)						
		Metal Prices	Gold Grade	Silver Grade	Operating Costs	Capital Costs	Sustaining Capital Costs
Percent Change from Base (%)	-30%	232.2	369.3	349.3	524.7	527.7	517.1
	-25%	274.5	388.8	372.1	518.4	520.8	512.0
	-20%	316.9	408.3	395.0	512.0	513.9	506.9
	-15%	359.3	427.8	417.8	505.6	507.0	501.7
	-10%	401.6	447.3	440.7	499.2	500.1	496.6
	-5%	444.0	466.8	463.5	492.8	493.2	491.5
	0%	**486.3**	**486.3**	**486.3**	**486.3**	**486.3**	**486.3**
	5%	528.7	505.9	509.2	479.9	479.4	481.2
	10%	571.0	525.4	532.0	473.5	472.6	476.1
	15%	613.3	544.8	554.8	467.1	465.7	471.0
	20%	655.6	564.3	577.6	460.7	458.8	465.8
	25%	697.8	583.8	600.4	454.3	451.9	460.7
	30%	740.1	603.2	623.2	447.8	445.0	455.6

Note: Table prepared by SilverCrest, 2021.

Figure 1-4: Post-tax NPV Sensitivities (base-case is bolded)



		IRR (%)					
		Metal Prices	Gold Grade	Silver Grade	Operating Costs	Capital Costs	Sustaining Capital Costs
Percent Change from Base (%)	-30%	30%	42%	40%	55%	68%	55%
	-25%	34%	44%	42%	55%	65%	54%
	-20%	38%	45%	44%	54%	62%	54%
	-15%	41%	47%	46%	53%	59%	53%
	-10%	45%	48%	48%	53%	56%	53%
	-5%	48%	50%	50%	52%	54%	52%
	0%	**52%**	**52%**	**52%**	**52%**	**52%**	**52%**
	5%	55%	53%	53%	51%	49%	51%
	10%	58%	55%	55%	50%	48%	51%
	15%	61%	56%	57%	50%	46%	50%
	20%	64%	58%	59%	49%	44%	50%
	25%	67%	59%	60%	48%	42%	49%
	30%	70%	60%	62%	48%	41%	49%

Note: Table prepared by SilverCrest, 2021.

Figure 1-5: Post-tax IRR Sensitivities (base-case- is bolded)

A sensitivity analysis was performed to assess the impact of changing gold and silver prices on the Project, as outlined in Figure 1-5. The base-case is bolded in Table 1-13 below.

Table 1-13: Economic Results for Different Metal Price Scenarios

Price Case	Gold Price ($/oz)	Silver Price ($/oz)	Post-Tax NPV 5% ($M)	Post-tax IRR (%)
Base Case	**1,500**	**19.00**	**486.3**	**52**
Three-year Trailing Average	1,788	17.73	530.7	55
Upside (Spot Case)	1,946	27.36	802.5	74
Downside (PEA Base Case Prices)	1,269	16.68	370.4	42

Note: Five-year Trailing prices and Spot Prices are based on data as of January 4, 2021. PEA Case is based on the pricing from the PEA report with effective date of May 15, 2019 and as amended July 8, 2019.

1.21 Risks

1.21.1 COVID-19

The major risk identified to construction of the process plant and infrastructure is disruption due to a COVID-19 outbreak on site or in the local community. To reduce the likelihood of this risk materializing, the construction workforce will be accommodated at the Project site and isolated from the local community. Access to and from site will be strictly controlled, including quarantine, and testing prior to authorising access to site, and ongoing randomized testing will continue to be implemented to site.

1.21.2 Mineral Resource Estimates

The drill sample spacing varies by vein and the classification of Mineral Resource estimates was assigned based on the level of confidence based on drill core sample spacing. Risk is associated with all classifications of Mineral Resource estimates; however, the greatest risk is associated with the Inferred Mineral Resource estimate.

There is a risk that the Mineral Resource estimate wireframes (>150 gpt AgEq) may be moderately biased with respect to the representative volume, and subsequent estimated tonnage and metal content. This potential bias could be where the wireframes extend somewhat too far into lower-grade (<150 gpt AgEq) assay areas of influence. When the first stope mining operation commences in each vein, a follow-up rolling reconciliation is recommended to allow for any mine call factor adjustments to be made.

Localized extremely high-grade samples were encountered in drill core sampling as part of the mineralization system. Locally, this represents a risk in the accuracy of grade estimation for Mineral Resource and subsequent Mineral Reserve estimation, and to operational grade control.

Where only widely spaced sampling is available, the spatial extent of the high-grade mineralization may be uncertain. This risk can be reduced through future close-range sampling to better delineate high-grade shoots within the vein systems, thereby allowing the highest-grade material to be domained to constrain spatial influence of these samples within delineated shoots. Closely spaced pre-production definition drilling in combination with duplicate sampling protocols for high-grade samples should be implemented to mitigate excessive extrapolation of high-grade values and to better inform the local, short-range, grade variability.

1.21.3 Mineral Reserve Estimates and Mine Plan

The main risks that can affect the Mineral Reserves are the decrease in mining recovery and the increase in mining dilution due to the narrow veins that make up the deposit. To mitigate this risk, the mine ramp-up will be gradually increased to design level and stabilized in 2025.

There is a known void area in the Babicanora Central Zone that could cause recovery problems. Although the general area is known, the exact size and geometry of the void area is unknown. To mitigate the possible impact of this risk, mining recovery was tested in Q3 and Q4 2020 with successful results. This area will require additional backfilling and grade monitoring during operations.

Historical excavations may be encountered during mining. To mitigate this risk, a test hole program will be needed during development and stoping.

A portion of the mine will require stringent measures to maintain/control good ground conditions. In addition, the mine plan requires a significant amount of development and a portion of production will rely on lower-productivity mining methods. This combination of factors represents a risk to ensure the plant operates at its designed capacity. Planned mitigation measures include early commencement of development to provide information on ground conditions, productivity, costs, and mining methods; early hire of site management personnel; stockpiling material so that the plant has an early supply of material for treatment; and gradual ramp-up.

1.21.4 Metallurgical Testwork and Recovery Plan

There is a risk that spikes of high clay/mica content material, with poor settling and filtration characteristics, may occur in the process plant and cause reduced capacity through the tailings filters. During detailed design, SilverCrest can further characterise the mineralized material types and identify areas that have high clay/mica contents and thus will be able to better plan for the treatment of these materials in the plant.

When treating very high-grade silver–gold–copper grades that require high cyanide and zinc reagent additions, there is potential for impurities to build up in the recirculating process water. To mitigate this, the cyanide detoxification circuit was designed to treat an additional barren bleed stream to purge impurities from the process water.

There is a risk that a larger portion of the tailings could exhibit higher clay contents than anticipated. This could translate into greater moisture than the target at the filter plant and longer times and greater effort to process and compact the filtered tailings at the FTSF. This risk will be mitigated by providing sufficient area for the FTSF, in the early stages of stacking, where tailings that do not meet the design specifications or higher clay content tailings can be temporarily placed in the interior portion of the FTSF. Filtered tailings could then be extended and compacted when conditions allow without the need to stop tailings disposal. This design assumed that an additional facility, the NW FTSF, could be used as an alternative for temporary, non-specification tailings storage.

1.22 Opportunities

1.22.1 Exploration and Mineral Resource Estimates

The most significant upside is the potential for conversion of Inferred Resources to Indicated Resources and possible Reserves, conversion of excluded Indicated Resources to Reserves, and discovery of additional mineralization that may support Mineral Resource estimation.

Inferred Mineral Resources are estimated at 1.24 Mt grading 4.35 gpt Au, and 367 gpt Ag, or 745 gpt AgEq, for 29.7 Moz AgEq. The majority of these resources is located in the Babi Vista Vein Splay, Granaditas 1 and 2 veins, El Muerto Zone and the Babicanora Norte Vein. The most significant potential for adding resources and reserves is the Babi Vista Vein Splay with an estimated 211.4 kt grading 13.00 gpt Au and 909 gpt Ag, or 2,039 gpt AgEq for 13.85 Moz AgEq in Inferred Resources.

Indicated Mineral Resources that were not used for vein reserve calculation, not converted to Mineral Reserves, nor included in the Feasibility Study mine plan are estimated at 14.8 Moz AgEq (14.8 Moz 1.04 X 86.9 + 102.9 = 193 gpt AgEq) contained in 2.4 Mt at 1.04 gpt Au and 102.9 gpt Ag, and are either adjacent or proximate to the proposed mine plan. The most significant of these is the Babicanora Main Vein which has an estimated excluded Indicated Resource of 7.0 Moz Ag Eq contained in 1.0 Mt at 1.18 gpt Au and 108.6 gpt Ag or 8.41 Moz AgEq.

Through October 16, 2020, 45 veins had been identified, but only 21 of those veins have had sufficient drilling to support at least an Inferred Mineral Resource estimate. Surface exploration and drill-testing has identified over 30 km of potential vein strike length that remains to be tested. Future drilling should focus on step-out drilling within the known mineralization zones and testing deeper host lithologies, parallel veins and newly identified areas that had limited historical workings.

In some areas of the deposit, bulk densities are higher than the 2.55 t/m^3 value used in Mineral Resource and Mineral Reserve estimation. If higher bulk densities are confirmed, there is potential to slightly increase the tonnages in the estimates.

1.22.2 Mine Plan

With the mine expected to showcase good ground conditions and cemented rock fill being employed in mined-out stopes, there is potential that some pillars could be recovered.

Positive exploration drilling results may present an opportunity for further optimization of the mine design and schedule ahead of commercial production. Several of the high priority exploration opportunities are within or close to the proposed footprint of underground development.

1.22.3 Recovery Plan

While additional studies and engineering would be required to execute a future plant expansion; conceptually, such an expansion could be achieved through the addition of a ball mill, pebble crusher and additional flotation capacity.

1.23 Interpretation and Conclusions

Under the assumptions and parameters discussed in the Report, the Project shows positive economics.

On the basis of the Project’s positive economics, SilverCrest has elected to proceed with construction and has entered into an EPC agreement for delivery of the process plant and associated infrastructure.

1.24 Recommendations

A two-phase program is recommended. The work recommended in the first phase relates to additional drilling, comprising infill and step-out drilling in the area where Mineral Resources have been estimated. This work includes exploration expansion and infill drilling based on already defined Inferred Resources for potential re-classification. The second phase focuses on studies including additional metallurgical testwork to identify areas of high mica/clay content in the veins in the mine plan, additional bulk density measurements, and updating the Mineral Resource and Mineral Reserve estimate using results of drilling, bulk density and geometallurgical testwork.

The majority of the second work phase can be completed in conjunction with the first work phase. A portion of the density determination, scanned using a portable shortwave infrared (SWIR), work suggested in Phase 2 require channel samples from the in-vein development.

Resource estimation would be completed once results of the Phase 1 drilling are available and would be updated to incorporate information from the proposed density and SWIR programs as those data became available.

The Phase 1 work program is estimated at $39 M. The Phase 2 program is estimated at $235,000.

2 Introduction

2.1 Introduction

SilverCrest Metals Inc. (SilverCrest) commissioned Ausenco Engineering Canada Inc. (Ausenco) to compile a Technical Report (the Report) and feasibility study (the Feasibility Study) on the Las Chispas Project (the Project), located in Sonora, Mexico. The effective date (the Effective Date) of the Report is January 4, 2021.

2.2 Terms of Reference

The Report supports disclosures by SilverCrest in the news release dated February 2, 2021, titled, "SilverCrest Metals Announces Positive Feasibility Study Results and Technical Report Filing for the Las Chispas Project".

The firms and consultants who are providing Qualified Persons (QPs) responsible for the content of the Report are, in alphabetical order, Ausenco, G Mining Services Inc. (GMS), Hydro-Ressources Inc. (HRI), P&E Mining Consultants Inc. (P&E), Rockland Ltd. (Rockland), and Wood Environment & Infrastructure Solutions, Inc. (Wood).

Although SilverCrest has announced in a previous news release a targeted timeline for initial operation and ramp up of production from the Project, calendar years used in the economic analysis are provided for conceptual purposes only.

The Report presents Mineral Resource and Mineral Reserve estimates for the Project, and an economic assessment based on underground mining operations and a conventional processing circuit that would produce gold-silver doré bars.

All units of measurement in the Report are metric, unless otherwise stated.

The monetary units are in US dollars, unless otherwise stated.

Mineral Resources and Mineral Reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards).

Terms used by the P&E QPs in the Report include the following:

- The Las Chispas Property: this encompasses all mineral occurrences and land underlying the mineral concessions 100% owned or optioned to SilverCrest;
- The Las Chispas District: this is a general term used in historical context for the various mines which operated in the area prior to the 1930s. The District has an approximate footprint of 4 km north to south and 3 km east to west. It consists of the Las Chispas Area and Babicanora Area, which are approximately 1.5 km apart;
- The Las Chispas Area: this consists of the Las Chispas Vein containing Area 118 Zone and Historical Mine, Giovanni Vein, Gio Mini Vein, La Blanquita Vein, William Tell Vein, Luigi Vein, Luigi FW Vein, Varela veins, Chiltepin veins, El Cumaro Vein, and various other lesser or unnamed veins;
- The Babicanora Area: this consists of the Babicanora Main Vein containing Area 51 Zone and Babicanora Central Zone, Babicanora FW Vein, Babicanora HW Vein, Babi Vista Vein, Babi Vista FW Vein, Babi Vista Vein Splay, Babicanora Norte Vein containing

Area 200 Zone, Babicanora Norte HW Vein, Babicanora Sur Vein, Babicanora Sur HW Vein, Amethyst Vein, La Victoria Vein, Granaditas Vein, Granaditas Dos Vein, Ranch Veins and various other lesser or unnamed veins;

- Area 118 Zone (Area 118): the southeast extension of the Las Chispas Vein discovered in 2020 by drill hole LC20-118. The hole intersected 8.6 m (true width) grading 44.30 gpt gold and 4,551.5 gpt silver;
- Area 51 Zone (Area 51): the southeast extension of the Babicanora Main Vein discovered in late 2017 by drill hole BA17-51. The hole intersected 3.1 m (true width) grading 40.45 gpt gold and 5,375.2 gpt silver.
- Babicanora Central Zone (Babicanora Central): the northwest, near surface, extension of the Babicanora Main Vein.
- Area 200 Zone (Area 200): the southeast extension of the Babicanora Norte Vein discovered in 2020 by drill hole BA219-200. The hole intersected 2.0 m (true width) grading 39.77 gpt gold and 3,472.5 gpt silver;
- The Las Chispas (Historical) Mine: this refers to a historical shaft and series of underground developments believed to be sunk under the original discovery outcrop that was located in the 1640s; and
- Vein: this is a current term used by SilverCrest for geological features consisting of semi-continuous structures, quartz veins, quartz stockwork, and breccia.

2.3 Qualified Persons

The following serve as the qualified persons for this Technical Report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in accordance with Form 43-101F1:

- Mr. Robin Kalanchey, P. Eng., Ausenco;
- Mr. Scott Weston, P. Geo., Ausenco;
- Dr. William Stone, P. Geo, P&E;
- Mr. Eugene J. Puritch, P. Eng., P&E;
- Mr. David Burga, P. Geo., P&E;
- Ms. Jarita Barry, P. Geo., P&E;
- Mr. Yungang Wu, P. Geo., P&E;
- Mr. Andrew J. Turner, P. Geol., P&E;
- Mr. Carl Michaud, P. Eng., GMS;
- Mr. Michael Verreault, P. Geo., HRI;
- Dr. Khosrow Aref, P. Eng., Rockland; and,
- Dr. Humberto F. Preciado, P.E., Wood

2.4 Site Visits and Scope of Personal Inspection

Mr. Kalanchey, representing Ausenco, completed a site visit on June 6, 2019. The objectives of the site visit were to develop an understanding of the Project including proposed locations for plant infrastructure and process facilities, to review the availability and proximity of required infrastructure for construction and operation including access to site and water supply, and to highlight any critical issues with respect to logistics that would significantly influence implementation of the project.

Dr. Aref, representing Rockland, conducted four site visits during the Feasibility Study. He visited the Las Chispas Project on March 12-14, 2019, June 25-27, 2019, July 24-26, 2019, and November 5-6, 2019. The main objective of the first visit was to initiate the geomechanical program by reviewing typical Las Chispas drill hole cores and tours of underground developments. The second visit was intended to train Las Chispas geotechnical engineers, geologists, and geotechnical practitioners for the geotechnical data collection program. Follow up visits were carried out to examine new drill holes, review collected data, and inspect underground developments.

Mr. Michaud, representing GMS, conducted two site visits during the Feasibility Study between May 21-24, 2019 and November 4-11, 2020. The purpose of these two visits was to examine the active underground work at Babicanora Main Vein, Babicanora Central Zone and historical underground working at the Las Chispas Vein. To examine the core at the core shack with SilverCrest personnel, have discussions concerning the rock units found in the mining area and rock mechanics, and the geotechnical investigation program.

Mr. Turner, representing P&E, conducted a site visit from November 3-5, 2020. The objectives of the visit were to observe the Project layout and the historical underground workings at at the Las Chispas Vein and active underground working at Babicanora Vein, review the drill core intersections, and meet with SilverCrest technical personnel on site. Thirty-six (36) samples were collected during the site visit for assay data verification purposes.

Mr. Verreault, representing P&E, conducted a site visit from November 10-12, 2020. The objectives of the visit were to observe the following aspects: 1) general layout at site, 2) underground veins and geological structures, 3) surface hydrology, 4) current pump testing situation and 5) general geology. Meetings were held with various SilverCrest technical staff members to discuss basin tectonics, structural geology including veins and other discontinuities, overall geology and historical water records.

Dr. Preciado, representing Wood, conducted a site visit from April 2-3, 2019. The purpose of this visit was to walk the potential filtered tailings storage facility (FTSF) locations, meet with SilverCrest´s environmental and construction team, gather geotechnical information available at the time of the visit, and observe site conditions such as: surface native soils, rock outcrops, ephemeral arroyos and creek beds crisscrossing the Project, the existing waste rock and mineralized pile. As a result of this visit, Wood prepared a conceptual layout of various FTSF options and proposed site investigation program for the selected locations.

2.5 Effective Dates

The Report has a number of effective dates as follows:

- Date of information on mineral tenure, surface rights and agreements: December 7, 2020;
- Mineral Resource estimate: October 16, 2020;
- Mineral Reserve estimate: January 4, 2021; and,
- Financial analysis: January 4, 2021.

The overall Effective Date of the Report is the effective date of the financial analysis which is January 4, 2021.

2.6 Information Sources and References

Report and documents listed in Section 2.7, Section 3, and Section 27 were used to support the preparation of the Report. Additional information was sought from SilverCrest personnel where required.

2.7 Previous Technical Reports

SilverCrest has previously filed the following technical reports on the Project:

- Barr, J., Ghaffari, H., and Horan, M., 2019: Technical Report and Preliminary Economic Assessment for the Las Chispas Property, Sonora, Mexico: report prepared by Tetra Tech Canada Inc. for SilverCrest Metals Inc., effective date May 15, 2019, amended July 19, 2019;
- Barr, J., and Huang, J., 2019: Technical Report and Mineral Resource Estimate for the Las Chispas Property, Sonora, Mexico: report prepared by Tetra Tech Canada Inc. for SilverCrest Metals Inc., effective date February 8, 2019.
- Fier, N.E., 2018: Technical Report and Updated Mineral Resource Estimate for the Las Chispas Property, Sonora, Mexico: report prepared for SilverCrest Metals Inc., effective date September 13, 2018;
- Barr, J., 2018: Technical Report and Mineral Resource Estimate for the Las Chispas Property, Sonora, Mexico: report prepared by Tetra Tech Canada Inc. for SilverCrest Metals Inc., effective date February 12, 2018, amended May 9, 2018; and,
- Barr, J., 2016: Technical Report on the Las Chispas Property, Sonora, Mexico: report prepared by Tetra Tech Canada Inc. for SilverCrest Metals Inc., effective date September 15, 2016.

2.8 Unit Abbreviations

Table 2-1: Unit Abbreviations

Abbreviation	Description
$	United States dollar
MXN	Mexican peso
°C	degree Celsius
°F	degree Fahrenheit
%	percent
µ	micro
µm	micrometre
C$	Canadian dollar
cm	centimetre
ft	feet
ft^2	square feet

Abbreviation	Description
g	gram
gpt	grams per tonne
ha	hectare
hr	hour
HP	horsepower
km	kilometre
koz	thousand ounces
kV	kilovolt
kW	kilowatt
kWh	kilowatt-hour
kWh/t	Kilowatt per tonne
kN/m^3	kilonewton per cubic metre
MW	megawatt
kPa	kilopascal
kcmil	thousand circular mills
kN	kilonewton
masl	metres above sea level
mamsl	metres above mean sea level
L/s	litre per second
M	million
m	metre
m/a	metres per annum
m^2	square metre
m^3	cubic metre
mm	millimetres
t	metric tonne
Mt	million tonnes
oz	ounce
Moz	million ounces
Mt	mega tonne

Abbreviation	Description
ppb	parts per billion
ppm	parts per million
ton	short ton
t/hr	tonnes per hour
t/d	tonnes per day
t/a	tonnes per annum
w/w/ w/s	gravimetric moisture content (weight of water/weight of soil)
wt	weight

2.9 Name Abbreviations

Table 2-2: Name Abbreviations

Abbreviation	Description
3D	three-dimensional
AgEq	silver equivalent
AAS	atomic absorption spectroscopy
AES	atomic emission spectrometry
ALS	ALS Chemex
ARD	acid rock drainage
ABA	acid base accounting
BAN	Babicanora Agrícola Del Noroeste S.A de C.V
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
COG	cut-off grade
CPI	consumer price index
CSR	corporate social responsibility
CNCF	cumulative cash flow
CRM	certified reference materials
CSS	closed side setting
CCTV	closed circuit television
CSV	comma-separated value

Abbreviation	Description
CCD	counter-current decantation
CDN Labs	CDN Resource Laboratories Ltd.
CRF	cemented rock fill
CFE	Comisión Federal de Electricidad (Federal Electricity Commission of Mexico)
CONAGUA	Comisión Nacional del Agua (National Water Comission of Mexico)
CEDES	Comisión de Ecología y Desarrollo Sustentable del Estado de Sonora (Commission of Ecology and Sustainable Development of the State of Sonora, Mexico)
CMT	construction management team
DCS	distributed control system
DSO	Deswik Stope Optimizer software
EBITDA	earnings before interest, taxes, depreciation and amortization
ESMS	environmental and social management system
ELOS	equivalent linear overbreak/slough
EPCM	Engineering, Procurement and Construction Management
ESG	Environmental, Social and Governance
FW	footwall
FA	fire assay
FTSF	filtered tailings storage facility
G&A	General and Administration
GIS	geographic information system
GPS	global positioning system
GMS	G Mining Services
HDPE	high density polyethylene
HRI	Hydro-Ressources Inc.
HW	hanging wall
HR	hydraulic radius
ID	inverse distance

Abbreviation	Description
ID^3	inverse distance weighting to the third power
ICP	inductively coupled plasma
IP	Preventive Report (Informe Preventivo)
IRR	internal rate of return
LLA	Compañía Minera La Llamarada S.A. de C.V (subsidiary of SilverCrest in Mexico)
LOM	life of mine
LAN	local area network
LiDar	light detection and ranging data
LRS	long-hole retreat stoping
MS	mass spectrometry
MC-HS	master composite historical stockpile
MED Comp	medium grade composite
ML	metals leaching
MGM	mine general manager
MS	mass spectrometry
MIA	Spanish acronym for Environmental Impact Assessment
NN	nearest neighbour
N’	stability number
NCF	net cash flow
NAG	net acid generation
NAF	non-acid forming
NPV	net present value
NW	northwest
OEM	original equipment manufacturer
OIS	operator interface station
ORP	operation readiness plan
PCR	polymerase chain reaction

Abbreviation	Description
P&E	P&E Mining Consultants Inc.
PEA	preliminary economic analysis
PEP	project execution plan
PMZ	precious metal zone
PLS	pregnant leach solution
Q’	mass quality
Q (1, 2, 3, 4)	calendar quarter (1, 2, 3, 4)
QA/QC	quality assurance/quality control
QEMSCAN	quantitative evaluation of materials by scanning electron microscopy
QP	Qualified Person
RQD	rock quality designation
RPD	relative percent difference
RDCLF	rhyodacitic crystal tuff
ROM	run of mine
SAG	semi-autonomous grinding
SGS Lakefield	SGS Lakefield Research
SD	standard deviation
SilverCrest	SilverCrest Metals Inc.
SEDENA	Secretaría de la Defensa Nacional (Ministry of Defense)
SG	specific gravity
SLS	solid to liquid system
SEMARNAT	Secretaría del Medio Ambiente y Recursos Naturales (Ministry of Environment and Natural Resources of Mexico)
SUCS	Unified Soil Classification System
SWMS	Safe Work Method Statement
SWIR	shortwave infrared
UTM	Universal Transverse Mercator

Abbreviation	Description
USMCA	United States-Mexico-Canada Agreement
WGS	World Geodetic System
VOIP	voice over internet protocol
VHF	very high frequency
VSA	vacuum swing adsorption

2.10 Reporting of Grades by Silver Equivalent

SilverCrest has reported AgEq calculated on a Au:Ag ratio of 75:1 in several recent news releases, based on contemporaneous understanding of metal recoveries and the prevailing long term metal price trends. Due to recent volatility in metal prices and improved understanding of metal recoveries, the AgEq ratio has varied throughout the Feasibility Study. Unless otherwise stated the AgEq presented in this Report has been determined using an Au:Ag ratio of 86.9:1, as calculated based on long-term gold and silver prices of $1,410/oz Au and $16.60/oz Ag, the approximate metal recovery values discussed in Section 13 (96% Au and 94% Ag), and assuming no smelter charge reductions or metal losses. The Mineral Resource Estimate, Mineral Reserve Estimate and the Economic Analysis have been developed using values from Au and Ag grades; AgEq is used only for reporting purposes.

3 Reliance on Other Experts

3.1 Introduction

The QPs have relied upon the following other expert reports, which provided information regarding mineral rights, surface rights, property agreements, royalties, taxation, and marketing sections of the Report.

3.2 Mineral Tenure and Surface Rights

The QPs have not independently reviewed ownership of the Project area and any underlying mineral tenure, and surface rights. The QPs have fully relied upon, and disclaim responsibility for, information derived from SilverCrest and legal experts retained by SilverCrest for this information through the following document:

Urias Romero y Asociados, S.C., 2020: Legal Opinion: December 7, 2020.

This information is used in Section 4 of the Report. It is also used to support the Mineral Resource estimates in Section 14, the Mineral Reserve estimates in Section 0, and the economic analysis in Section 22.

3.3 Environmental

The QPs have not independently reviewed environmental baseline, permitting, and social information for the Project. The QPs have fully relied upon, and disclaim responsibility for, information derived from SilverCrest and experts retained by SilverCrest for this information through the following documents:

ALS, 2019; Waste Rock Samples Analytical Reports, prepared for Altadore Energía, S.A. de C.V., November 2019, Hermosillo, Sonora, Mexico.

ALS, 2020; Tailings Samples Analytical Reports, prepared for Tinto Roca Exploración S.A. de C.V, March 2020, Hermosillo, Sonora, Mexico.

Ontiveros A, 2019; Various Reports on Surface Water Quality Results from Sampling Points Located Upstream and Downstream from the Mine Project, prepared for Compañía Minera la Llamarada S.A. de C.V and Altadore Energía, S.A. de C.V., Hermosillo, Sonora, Mexico.

SilverCrest, 2019; Water Quality Results in Monitoring Wells Located within Las Chispas Mine Project, prepared by SilverCrest´s Environmental Team, November 2019, Sonora, Mexico.

SGS, 2020a; Modified Acid Base Accounting & Net Acid Generation Testing Results on Various Rock Core Samples from the Babicanora Central, Babicanora Norte, Babicanora Sur, Babi Vista, William Tell, and Las Chispas Mining Areas, Report prepared by SGS Minerals Services for SilverCrest Metals Inc., Project 17337-1, January 2020, Lakefield, Ontario, Canada.

SGS, 2020b; Recovery of Gold and Silver from the Las Chispas Project, prepared by SGS Minerals Services for SilverCrest Metals Inc., Project 17337-1, March 2020, Lakefield, Ontario, Canada.

This information is used in Section 20 of the Report. It is also used to support the Mineral Resource estimates in Section 14, the Mineral Reserve estimates in Section 0, and the economic analysis in Section 22.

3.4 Market Studies and Contracts

The QPs have fully relied upon and disclaim responsibility for information supplied by SilverCrest's staff for information related to market assumptions as applied to the financial model. This information is used in support of the marketing assumptions presented in Section 19, and the financial analysis discussed in Section 22. To some extent this information was also included in the estimation of the Mineral Reserve, detailed in Section 15.

The QPs consider it reasonable to rely upon the information provided by SilverCrest for gold and silver doré marketing and marketing assumptions. Mr. Pierre Beaudoin, COO of SilverCrest, solicited quotations and preliminary offers from doré refiners both in Mexico and around North America for refining of the Las Chispas product and an average of the toll and refining charges was used in the financial model. Mr. Beaudoin has considerable experience in the operation of precious metals mines which produce doré and has had firsthand experience in arranging refining contracts for these operations.

3.5 Taxation

The QPs have fully relied upon and disclaim responsibility for information supplied by ORBE Advisors, S. de R.L. de C.V. (Orbe) who were retained by SilverCrest for information related to taxation assumptions as applied to the financial model through the following document:

Analysis of Mexican Income Tax, Mexican Mining Environmental Erosion Fee and Mexican Special Mining Duty in connection with the Feasibility Study Report on SilverCrest Metals Inc.'s Las Chispas Project.

This information is used in support of the economic analysis discussed in Section 22.

The QPs consider it reasonable to rely upon information provided by Orbe, as they are a well reputable tax advisory service provider, based in Mexico, with considerable experience and expertise in providing tax advice to industrial and corporate clients with mining operations in Mexico.

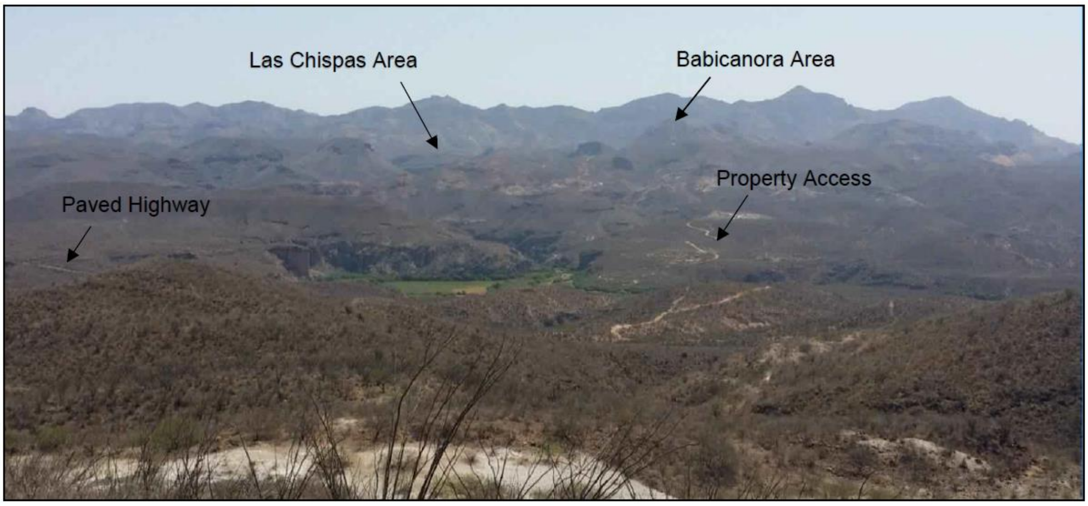
4 Property Description and Location

4.1 Description and Location

The Las Chispas Property (Photo 4-1) is located in the State of Sonora, Mexico, at approximate 30.233902°N latitude and 110.163396°W longitude (Universal Transverse Mercator [UTM] World Geodetic System [WGS]84: 580,500E, 3,344,500N), within the Arizpe Mining District. The city of Hermosillo is approximately 220 km, or a three-hour drive to the southwest; Tucson, Arizona is approximately 350 km via Cananea, or a five-hour drive, to the northwest; and the community and mine of Cananea is located approximately 150 km, or a two-and-a-half-hour drive, to the north along Highway 89. The general topography of the area surrounding Las Chispas is shown in Photo 4-1. A location map for the Property is shown in Figure 4-1. The Property area is covered by the 1:50,000 topographic map sheet "Banamichi" H12-B83.

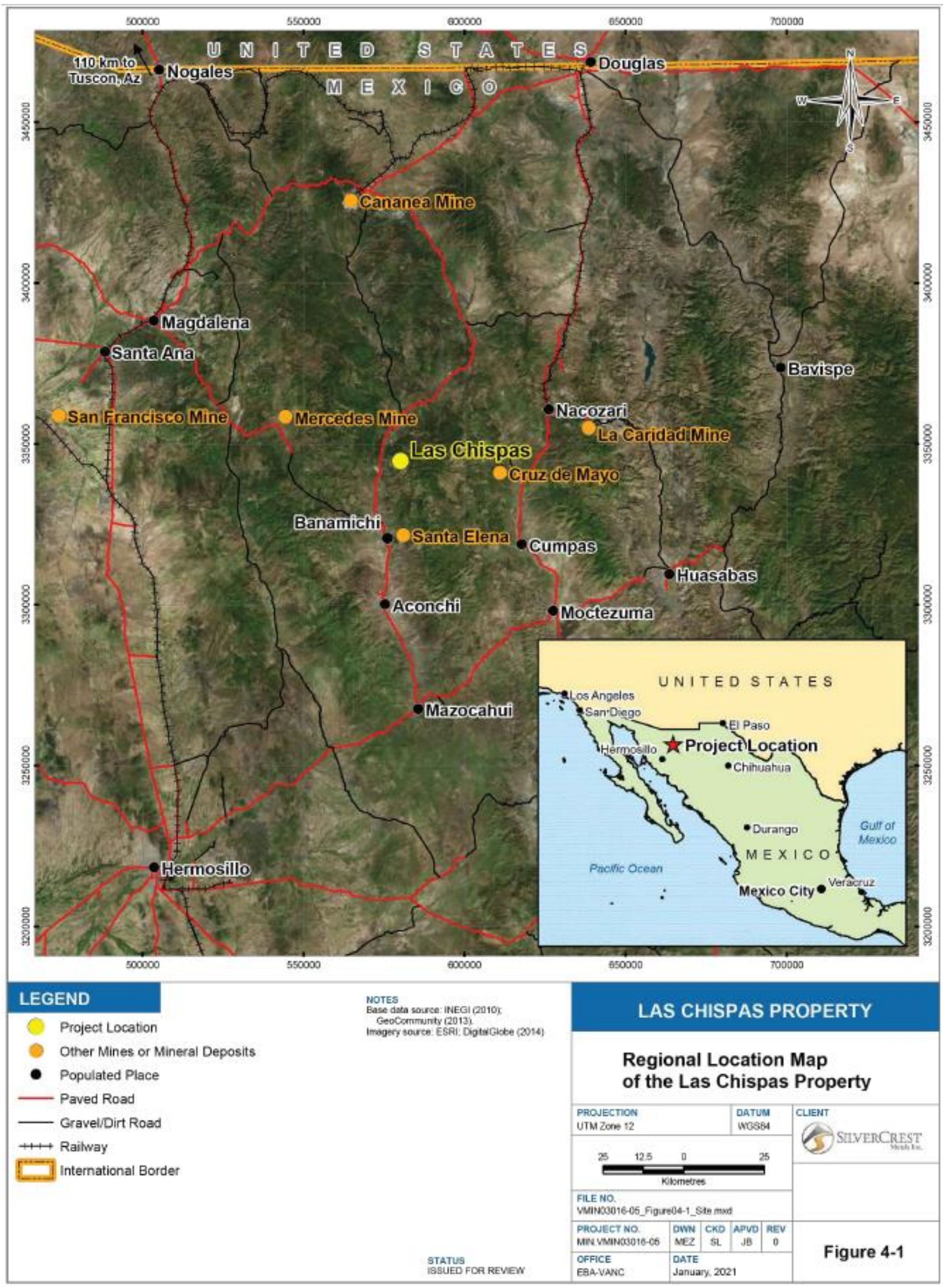
4.2 Project Ownership

Compañía Minera La Llamarada S.A. de C.V. (LLA) is a subsidiary of SilverCrest and is the in-country operating entity.



Note: Length of field of view in photograph is approximately 4 km. Source: PEA, 2019.

Photo 4-1: View Looking Eastwards Across the Las Chispas Property



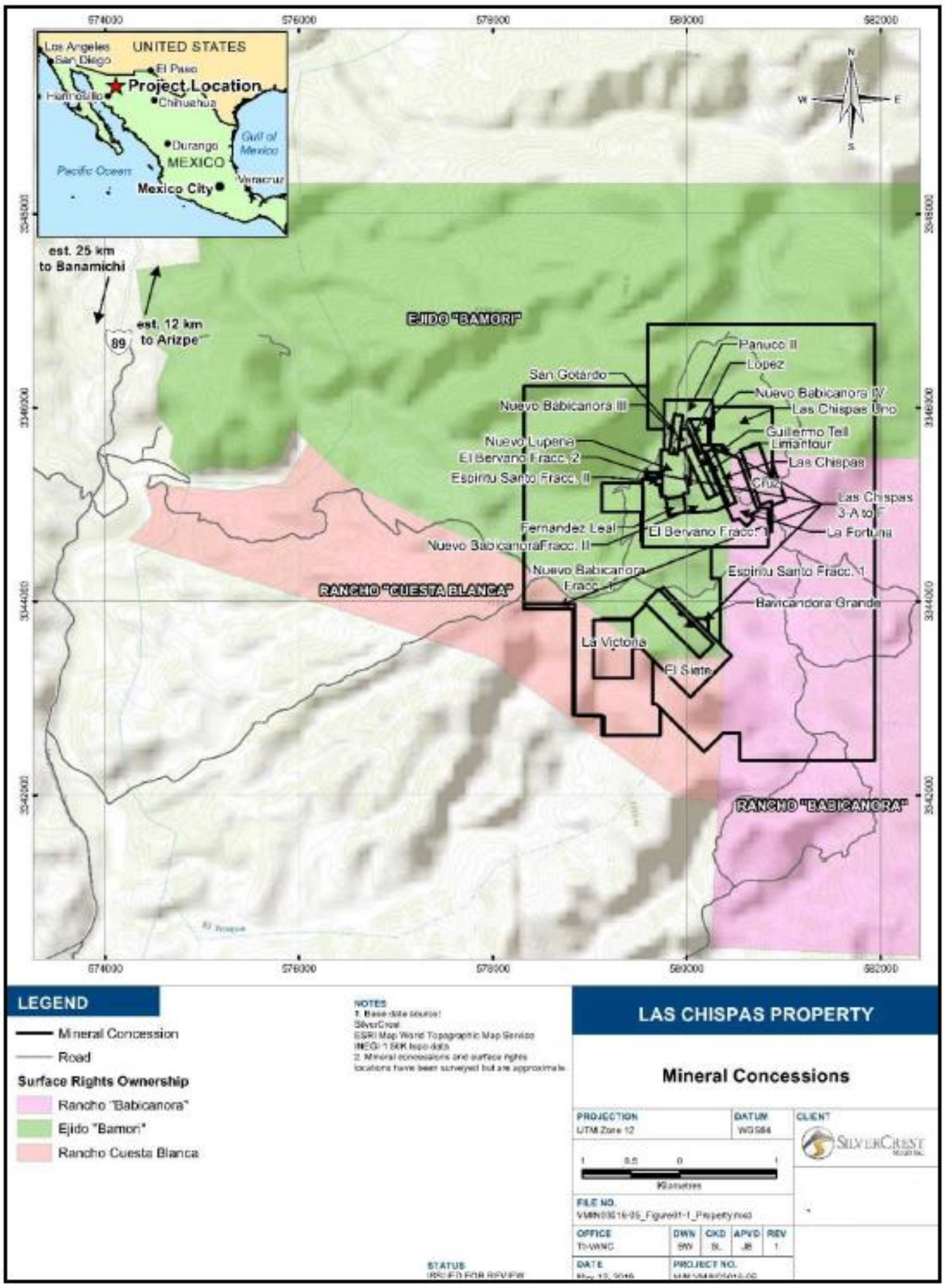
Note: Figure from Barr et al., 2019.

Figure 4-1: Regional Location Map of the Las Chispas Property

4.3 Mineral Tenure

The Property consists of 28 mineral concessions, totalling 1,400.96 ha, as shown in Figure 4-2 and listed in

Table **4-1**. LLA, has acquired the title or has entered into option agreements to purchase the concessions. Three option agreements are described in Section 4.4.



Note: Figure from PEA, 2019.

Figure 4-2: General Map Showing Mineral Concessions and Surface Rights for Las Chispas Property

Table 4-1: Mineral Concessions held by SilverCrest for the Las Chispas Property

Concession Name	Title Number	Registration Date	End Date	Surface Area (ha)	Concession Holder/Option Agreement Number
El Bervano Fraccion 1	212027	8/25/2000	8/24/2050	53.4183	LLA
El Bervano Fraccion 2	212028	8/25/2000	8/24/2050	0.9966	LLA
Las Chispas Uno	188661	11/29/1990	11/28/2040	33.7110	LLA
El Siete	184913	12/6/1989	12/5/2039	43.2390	LLA
Babicanora Grande	159377	10/29/1973	10/28/2023	16.0000	LLA
Fernandez Leal	190472	4/29/1991	4/28/2041	3.1292	LLA
Guillermo Tell	191051	4/29/1991	4/28/2041	5.6521	LLA
Limantour	191060	4/29/1991	4/28/2041	4.5537	LLA
San Gotardo	210776	11/26/1999	11/25/2049	3.6171	LLA
Las Chispas	156924	5/12/1972	5/11/2022	4.4700	LLA
La Fortuna	EXP 082/39410	N/A	N/A	15.28	(1) Tarachi
Espiritu Santo Fracc. I	217589	8/6/2002	8/5/2052	733.3232	LLA
Espiritu Santo Fracc. II	217590	8/6/2002	8/5/2052	0.8770	LLA
La Cruz	223784	2/15/2005	2/14/2055	14.4360	LLA
Lopez	190855	4/29/1991	4/28/2041	1.7173	(2) LLA – Lopez
Nuevo Babicanora Fracc. I	235366	11/18/2009	11/17/2059	392.5760	LLA
Nuevo Babicanora Fracc. II	235367	11/18/2009	11/17/2059	9.8115	LLA
Nuevo Babicanora Fracc. III	235368	11/18/2009	11/17/2059	2.2777	LLA
Nuevo Babicanora Fracc. IV	235369	11/18/2009	11/17/2059	3.6764	LLA
Nuevo Lupena	212971	2/20/2001	2/19/2051	13.0830	LLA
Panuco II	193297	Cancelled	Cancelled	12.9286	(3) LLA
La Victoria	216994	6/5/2002	6/4/2052	24.0000	LLA
Las Chispas 3-A	245423	01/24/2017	01/23/2067	1.0809	LLA
Las Chispas 3-B	245424	01/24/2017	01/23/2067	0.3879	LLA
Las Chispas 3-C	245425	01/24/2017	01/23/2067	0.3413	LLA
Las Chispas 3-D	245426	01/24/2017	01/23/2067	0.3359	LLA
Las Chispas 3-E	245427	01/24/2017	01/23/2067	0.4241	LLA

Concession Name	Title Number	Registration Date	End Date	Surface Area (ha)	Concession Holder/Option Agreement Number
Las Chispas 3-F	245428	01/24/2017	01/23/2067	5.6112	LLA
Total (28)	**-**	**-**	**-**	**1,400.9600**	**-**

Notes: * As of December 7, 2020 Date of the Opinion for Las Chispas by Urias Romero Y Asociados, S.C., for SilverCrest Metals Inc.
** Legal recourse pending (Section 4.4.3)

Mining duties are based on the surface area and date of issue of each concession and are due in January and July of each year at a total annual cost of approximately $10,000 (adjusted scale). At the Report Effective Date, all required mining duties were paid.

4.4 Concession Titles and Option Payments

Payment terms under each option agreement are included in the following subsections. All Mineral Reserves stated in the Report are on mining concessions 100% owned by LLA with no NSR applied.

In

Table 4-1, all mining concessions are 100% owned by LLA except for those noted and described below.

4.4.1 Option 1

On February 21, 2018, LLA acquired from Minerales Tarachi, S. de R.L. de C.V. an option to purchase the rights to the La Fortuna mining concession applications No. 082/39410 and 082/38731, which cover the Panuco II and Carmen Dos Fracción II mineral lots on payment of MXP500,000 (paid) and $150,000 payable on acquisition of title by LLA (Title Opinion dated December 7, 2020). Title transfer of concessions are pending until the applications are issued as mining concessions.

4.4.2 Option 2

On June 29, 2018, LLA acquired 67% ownership from Jose Cruz Lopez Mejia (34%); Eliseo Espina Guillen (33%); and Jesus Cruz Lopez (33%) who still owns 33%.

4.4.3 Option 3

On December 1, 2018, LLA acquired 100% rights to Panuco II, pending title for ownership. This concession was cancelled in 1999 but, by law, a public notice of cancellation was not announced, therefore, a legal recourse for reinstatement was filed by LLA. This process is ongoing as of the Effective Date of the Report (Title Opinion dated December 7, 2020).

All Mineral Reserves stated in the Report are on mining concessions 100% owned by LLA with no NSR applied.

4.5 Surface Rights

The surface rights overlying the Las Chispas mineral concessions and road access from local highway are either owned by LLA or held by LLA under a negotiated 20-year lease agreement. Surface rights are sufficient for the life of mine plan and include the locations of necessary infrastructure as presented in the Report.

4.5.1 Ejido Bamori

On November 18, 2015 (as amended June 3, 2018), LLA signed a 20-year lease agreement with the Ejido Bamori for surface access and use of facilities. Compensation for exploration activities will be paid at a rate of MXN700/ha, up to a total of 360.60 ha. After exploration and announcement of mine construction/production, compensation will be paid on a scaled timeframe at a rate of MXN2,000/ha during construction and Years 1 to 4 of production and MXN4,000/ha from the fifth production year to the end of the mine life. On April 2019, LLA expanded its surface rights from 360.6 ha to 400 ha.

4.5.2 Cuesta Blanca Ranch

In February 2018, LLA purchased the Cuesta Blanca Ranch covering 671.9 ha of land situated in the municipality of Arizpe, Sonora.

4.5.3 Babicanora Ranch

In April 2017, LLA purchased from Maprejex Distributions Mexico, S.A. de C.V. the Babicanora Ranch covering 2,500 ha of land situated in the municipality of Arizpe, Sonora.

4.5.4 Tetuachi Ranch

In November 2017, LLA signed a lease agreement for a term of 20 years with Maria Dolores Pesqueira Serrano for the lease of the Tetuachi Ranch covering 32.3 ha of land situated in Arizpe, Sonora, for payment of an annual rental fee of $2,000 during the exploration phase and $7,000 during the exploitation phase.

4.6 Royalties

A 2% royalty is payable to Gutierrez-Perez-Ramirez, on the Nuevo Lupena and Panuco II concessions for material that has processed grades of≥0.5 oz per tonne gold and ≥40 oz per tonne silver combined (Title Opinion dated December 7, 2020). Currently no reserves are estimated within these concessions.

4.7 Permitting Considerations

Permitting considerations for operations are discussed in Section 20.

4.8 Environmental Considerations

Environmental considerations for operations are discussed in Section 20.

Remnants exist on the Project that show the active mining history and community development that once existed in this district. There are numerous historical mine portals and shafts that are partially overgrown with vegetation, which have been flagged and/or fenced.

4.9 Social License Considerations

Social licence considerations for operations are discussed in Section 20.

4.10 Comment on Property Description and Location

To the extent known to the QP, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the Project that have not been discussed in the Report.

5 Accessibility, climate, local resources, infrastructure, and physiography

5.1 Accessibility

The Project is accessed from the community of Arizpe, a distance of 12 km, and Banamichi, a distance of 25 km, both by paved Highway 89. The Project itself is accessed via secondary gravel roads, approximately 10 km off the paved highway. Currently, crossing the Rio Sonora is required.

The water levels in the Rio Sonora are typically low and easily passed but can raise to temporary unpassable levels following major rain events. A road bridge is planned to be constructed in 2021. The remainder of the road has been upgraded by dozer/grader. Net elevation gain from the highway to the Project area is approximately 400 m.

5.2 Climate

The climate is typical for the Sonoran Desert, with a dry season from October to May. Seasonal temperatures vary from 0°C to 40°C. Average rainfall is estimated at 300 mm/year. There are two wet seasons; one in the summer (July to September) and the second in the winter (December). The summer rains are short with heavy thunderstorms, whereas the winter rains are longer and lighter. Summer afternoon thunderstorms are common and can temporarily impact the local electrical service. Operations are planned to be conducted year-round.

5.3 Local Resources and Infrastructure

5.3.1 Water Supply

Current water requirements during exploration are minimal; core drilling requires the greatest capacity. Some wells have been established to supply local ranches. Hydrogeological testing was conducted to determine the depth to the water table. Twelve pilot water wells were drilled, and initial results show that the most prospective potential area, located on the Project, is in the Sonora Valley. A well in the valley was pump-tested at 12 L/s. A geophysical investigation was completed in 2019 to determine the ultimate location of the operational wells in the valley. During this geophysical investigation, areas near the proposed mine site were also surveyed, but no significant water-bearing targets were identified.

The Report assumes that production water will be pumped from two main sources. First, from the 900 level (feet from surface or 850 metres above sea level (masl)) of the historical Las Chispas mine. This water is considered to be the regional water table and is already equipped with a pumping system connected to four surface water storage tanks adjacent to the proposed mill site (Photo 5-1). The second source of water is located from the valley basin within the property and near the sites main access road. Together the two sources of water are adequate in quantity and quality for production. The second pumping station in the valley basin (10 km from site) will be developed during construction and powered by a local nearby 33 kV power line.

Note: Figure from SilverCrest, 2020.

Photo 5-1: Storage Water Tanks for Operations (Santa Rosa Portal in background)

5.3.2 Community Services

Mining supplies and services are readily available from Cananea and Hermosillo (Mexico), and Tucson (USA).

Labour and skilled workforces exist in the nearby communities, including Banamichi and Arizpe, for which housing and transportation routes could be established to support a mining operation.

5.3.3 Infrastructure

No surface infrastructure from the historical mining industry remains in the Project area, except for roads and a few eroding rock foundations. Several ranch buildings, corrals, and fencing were acquired from the purchase of ranches.

Information on the infrastructure is discussed in Section 0.

5.3.4 Power

Low-voltage power lines and generators supply local ranches. This amount of power is sufficient for exploration requirements.

Provision of grid power for the planned mining operation is in the permitting process, with construction anticipated in 2021 and completed before production start-up.

Information on power requirements is provided in Section 0.

5.4 Physiography

The Project is located on the western edge of the north-trending Sierra Madre Occidental mountain range geographically adjacent to the Sonora Valley.

Surface elevations range from 950 masl to approximately 1,375 masl.

The San Gotardo portal to the Las Chispas and William Tell veins is located at 980 masl, the Santa Rosa decline portal to the Babicanora Vein in Area 51 Zone at 1,180 masl, and the Babicanora Central adit portal to the Babicanora Vein in Babicanora Central Zone at 1,170 masl.

Hillsides commonly have steep colluvium slopes or subvertical scarps resulting from fractures through local volcaniclastic bedrock units.

Drainage valleys generally flow north to south, and east to west towards the Rio Sonora. Flash flooding is common in the area.

Vegetation is scarce during the dry season and limited primarily to juvenile and mature mesquite trees and cactus plants. During the wet season, various blooming cactus, trees, and grasses are abundant in drainage areas and on hillsides.

There are no known indigenous communities or protected areas in the Project vicinity.

5.5 Sufficiency of Surface Rights

There is sufficient surface area for all required facilities as stated in the Report, including top soil stockpiles, underground portals, mineralized stockpiles, and waste rock storage facilities (WRSFs), mill, filtered tailings storage facilities (FTSFs), associated infrastructure, and other operational requirements for the planned life of mine (LOM) and mine plan discussed in the Report.

6 History

6.1 Introduction

Historical records indicated mining around the Project started as early as the 1640s. There are incomplete historical records available on mining activities in the 1800s and 1900s. There is also a gap in mining activity records for Las Chispas between the mid-1930s through to 1974. In 2008, modern exploration activities in the area commenced with Minefinders Corporation Ltd. (Minefinders). SilverCrest acquired the property in 2015 and completed what is believed to be the first core drill hole in early 2016.

The history of Las Chispas as summarized in the following sub-sections was extracted from the limited documentation available to SilverCrest in the public domain and private libraries.

6.2 Exploration History

6.2.1 1800s and Early 1900s

Mining interest in the Project area is believed to have begun in 1640, when outcrop of the Las Chispas Vein was discovered by a Spanish General named Pedro de Perra (Wallace 2008), which led to the development of the Las Chispas Mine. Through to 1880, small-scale mining, including the excavation of the Babicanora (Central) Adit, was intermittently conducted along this trend.

A French company under the name Camou Brothers is reported to have re-developed the Babicanora Mine around 1865 (SilverCrest, 2015). The Babicanora area was mined by Chinese immigrants who originally settled in Baja, relocated to the State of Sinaloa in the late 1800s for agriculture, and were eventually pushed inland by competition to the State of Sonora. Here they found occupation in the mines. The Babicanora Adit portal construction and dimensions of underground development was notably different than that of the Las Chispas and William Tell workings. The adit is a 4 m by 4 m drift and approximately 230 m in length to intersect the Babicanora Vein.

From 1870s to the 1920s, the Las Chispas Mine was initially operated by the Santa Maria Mining Company, which went bankrupt after the crash of silver price in 1893 (Russell, 1908). John (Giovanni) Pedrazzini was given the rights to the mine and founded the Minas Pedrazzini Gold & Silver Mining Company (Photo 6-1, Photo 6-2). From the 1890s to the 1920s, the Pedrazzini family-maintained control of the Las Chispas area together with the Las Chispas William Tell, Luigi, Varela, and Cumaro veins.

Antonio Pedrazzini (Photo 6-3), nephew of Giovanni, maintained an active role in the operation and management of the mine into the 1920s. In 1904, Edward Dufourcq, a well-known mining engineer, was appointed as general manager of the mine. Minas Pedrazzini was the first operator to drive an adit into the Las Chispas Vein known as The San Gotardo Tunnel, or 600 level, an estimated length of 1,250 m, was excavated in the early 1900s. Referenced historical levels (i.e., 600 level) are marked as the depth in feet from the Las Chispas shaft collar (Figure 6-1).

Photo 6-1: Giovanni Pedrazzini and Family at Las Chispas, Circa Early 1880s

Photo 6-2: Minas Pedrazzini Stock Certificate

Photo 6-3: Antonio Pedrazzini and Family at Las Chispas, Circa Early 1900s

At least two other companies focused efforts on the El Carmen Mine, located approximately 5 km southeast of the Las Chispas Mine, and on the Babicanora area, approximately 1.5 km south of the Las Chispas Mine. Little is known about the historical production and operations of those companies; however, it is understood that small mills were installed at Babicanora and El Carmen to process ores of the Babicanora, El Carmen, and Granaditas veins in a similar manner to the San Gotardo (Las Chispas) Mill (Russell, 1908). The district had at least six operating flotation and cyanidation mills from the late 1880s to 1984.

The San Gotardo Mill, operated by Minas Pedrazzini, was located at the northern portal to the 600 level of the Las Chispas and William Tell veins. The mill consisted of rock breakers, five gravity stamps, two Wilfley tables, and three amalgamation pans, with reported recovery of 70% to 75% (Russell, 1908). The mill expanded with up to 20 operating stamps and four pans in 1910, when total recovery was noted then to be between 71% and 84%. An estimate of approximately 26,000 t were treated in the mill, and over 12,000 t of tailings were estimated to have been deposited as tails into ponds below the mill. In 1910, a 24-inch gauge tramway was built from the San Gotardo portal to the new mill, anticipating daily production to increase to 60 t/d. Wallace (2008) reports that in the 1970s, the mill was salvaged and hauled away with old mine buildings and much of the tailings for reprocessing.

In 1910, the decision was made to install a cyanide plant at the Las Chispas Mine in an effort to reduce overall processing costs, enable reprocessing of the earlier deposited tailings, and attempt higher metal recoveries with a throughput of 30 t/d to 40 t/d. Construction of the plant occurred during the Mexican Revolution, with delays in building (Dufourcq 1912). Mulchay (1935) indicates that this plant was used for less than six months, due to interference from sulphides in the mineralized material with cyanidation. A small flotation plant was installed prior to 1926 (Mulchay 1935).

Water for the operations was supplied via a 5 km-long pipeline from the Rio Sonora and power reportedly from a small power line running from a diesel generator at Nacozari. In 1918, the pumping station along the Rio Sonora was destroyed by a flood; the mine resorted to pumping from within the mine to supply the mill with water (Wallace 2008). Dufourcq (1910) indicates that water was originally intersected below the 900 level of the mine.

In 1917, it is reported that the mine was confiscated by the local government who operated and extracted "rich ore" before eventually returning the mine back to Pedrazzini (Montijo, 1920).

From 1900 to 1926, production from the Las Chispas and William Tell veins is reported to have been interrupted several times due to numerous interventions, including theft of high-grade ore, the Mexican revolution from 1910 to 1920, the Mexican National Catholic Church revolution in 1925, mill flooding/fire, and the government take-over of the mine with no economic plan (Montijo, 1920).

Two versions exist regarding how the Las Chispas Mine was taken over and eventually closed. Mulchay (1935) suggests that in 1935, Minas Pedrazzini was taken over under option by Douglas-Williams associated with the Phelps-Dodge Corporation. The mine was managed by Henry Bollweg at this time. However, Wallace (2008) reports the mine was acquired by a French corporate subsidiary Corporación Miñera de Mexico, S.A. in 1921. This company was reported to have remodelled the power plant and continued mining until closure in 1930.

The very limited information available on metal production suggests approximately 200,000 oz of gold and 100 Moz of silver and (Dahlgren 1883) were recovered from mines within the loosely-defined Las Chispas district, including approximately 20–40 Moz of silver estimated to have been recovered from the Las Chispas and William Tell veins. Wallace (2008) estimates that in the period between 1907 and 1911, annual production at the Las Chispas Mine achieved approximately 22,000 t producing 10,000 oz of gold per year and 1.5 Moz of silver and, with an estimated average grade of 1.1 oz per ton gold and 146.8 oz per ton silver (Table 6-1). Reports indicate that gold and silver were produced from both quartz/amethyst veinlets less than 5 cm thick and local high-grade shoots up to 4 m thick.

Table 6-1: Las Chispas Mine Production, 1908 to 1911 (Dufourcq 1910)

	1908	1909	1910	1911(1)	Total
Tonnes	3,286	3,064	3,540	12,000	21,890
Gold ounces per tonne	1.5	1.4	1.0	1.0	1.1
Silver ounces per tonne	199.9	187.2	136.9	125.0	146.8
Gold ounces	4,876	4,189	3,615	12,000	24,680
Silver ounces	656,882	573,448	484,746	1,500,000	3,215,076

Note: Estimated projected budget for 1911.

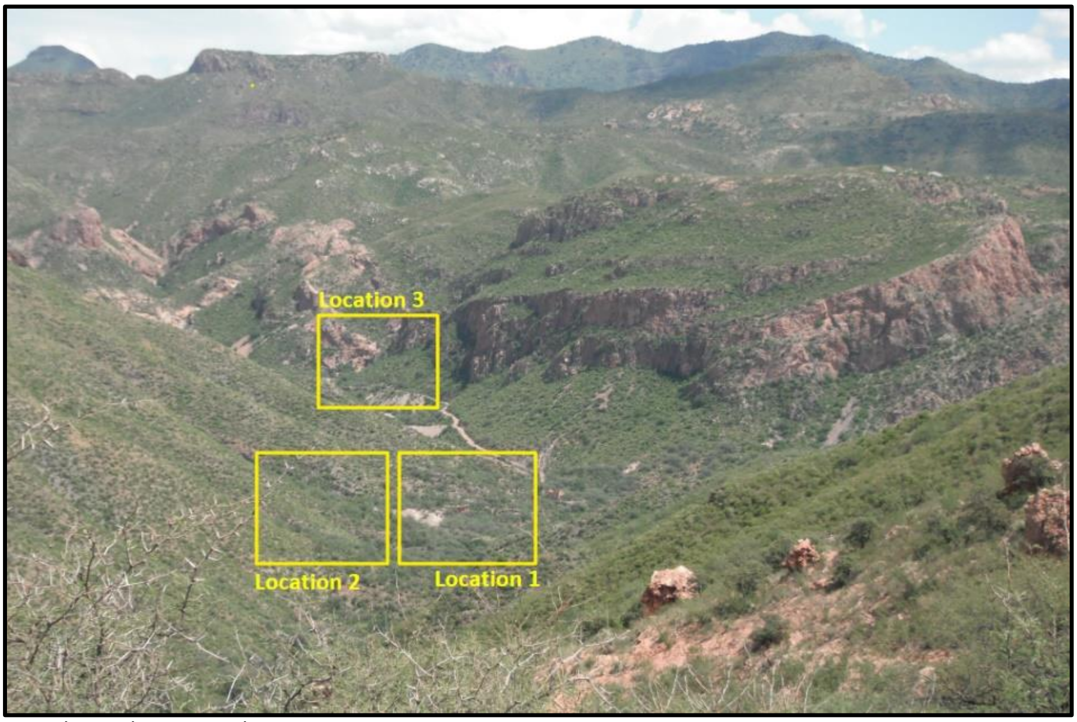
Some records suggest that small-scale mining at Espiritu Santo and operation of a small mill at Babicanora occurred in 1935 (Mulchay 1935). Espiritu Santo workings consisted of a small, inclined shaft approximately 80 cm wide, which declined below a small drainage to two short mineralized drifts. Approximately 13.2 tons of mineralized material was reported to have been shipped from this small mine in 1934 (Table 6-2).

Table 6-2: Espiritu Santo Mine Production, 1934 (Mulchay 1935)

October 1934:	Tons	Ag (oz.)	Au (oz.)
Oct. 9, 1934	3.6	75.2	0.17
Nov. 21, 1934	1.2	149.7	2.63
Jan. 23, 1935	1.8	159.3	0.66
Jan. 25, 1935	2.1	490.0	1.36
Feb. 22, 1935	2.3	160.3	0.56
Apr. 3, 1935	1.2	132.3	0.44
Total	1.0	131.8	0.82

Another small mining operation at La Victoria was in operation around 1940. The workings consisted of three short mineralized drives on separate levels approximately 30 m in length (Mulchay 1941).

Photo 6-4 provides an overview of the Las Chispas Valley and highlights the locations where the community of Las Chispas once stood, in addition to the original San Gotardo mill and the later developed rail-connected mill near the community. Historical Photo 6-4 through Photo 6-8 are from various locations around the historical operation. Photo 6-9 shows the 2017 view of the historical Babicanora (Central) portal and surface workings. Photo 6-10 is a long section of the historical Las Chispas underground development.



Note: Photo taken September 2015. Source: PEA, 2019.

Photo 6-4: View looking North down the Main Valley where Las Chispas Community and Processing Plants were located.

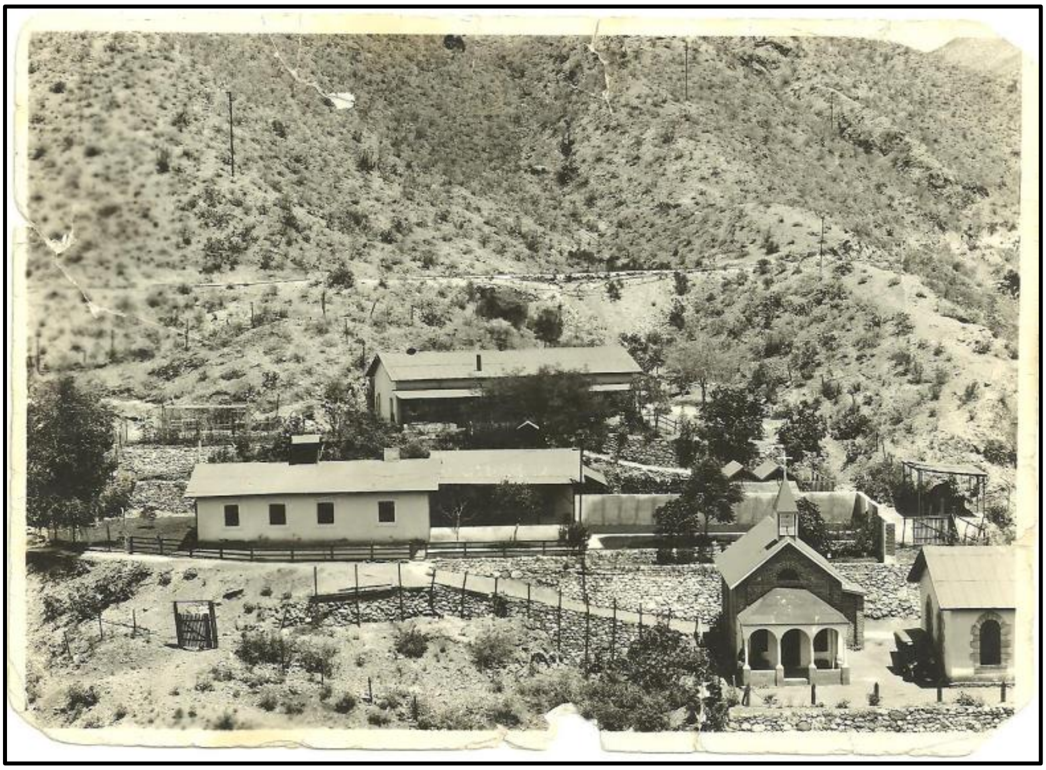


Photo 6-5: Historical Photo of former Las Chispas Community (Location 1, circa 1920s)

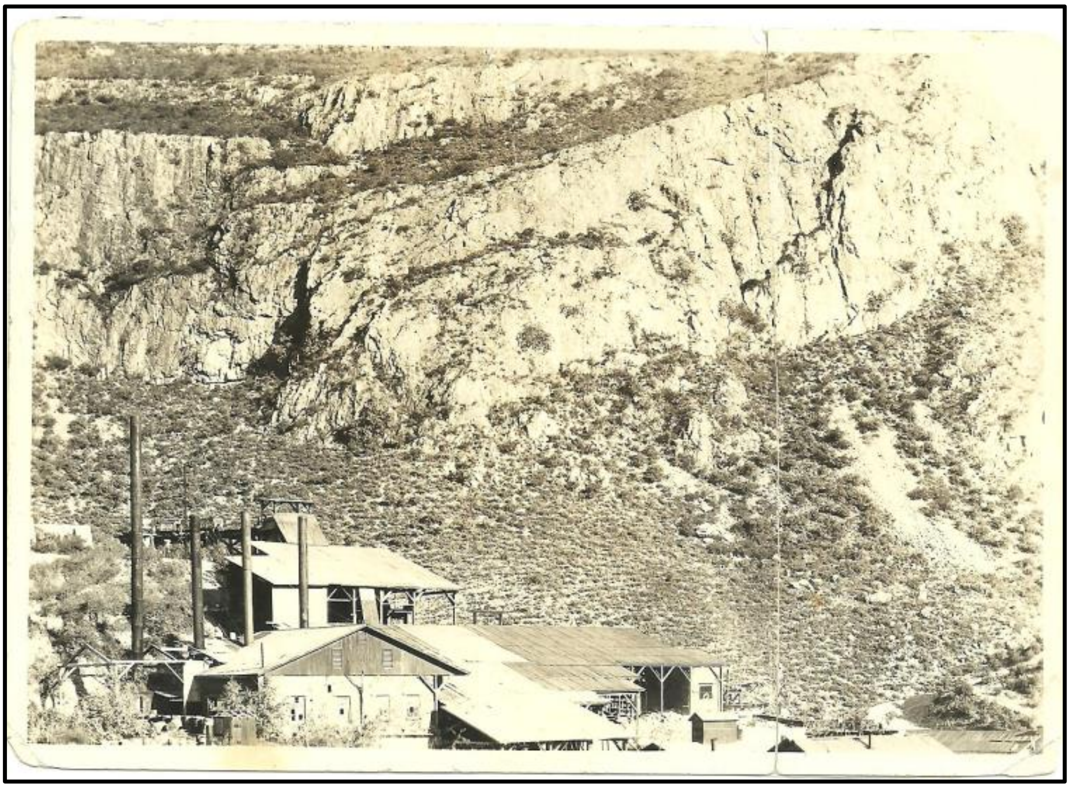


Photo 6-6: Historical Photo of a Processing Facility to Northwest of Community (Location 2, circa 1920s)

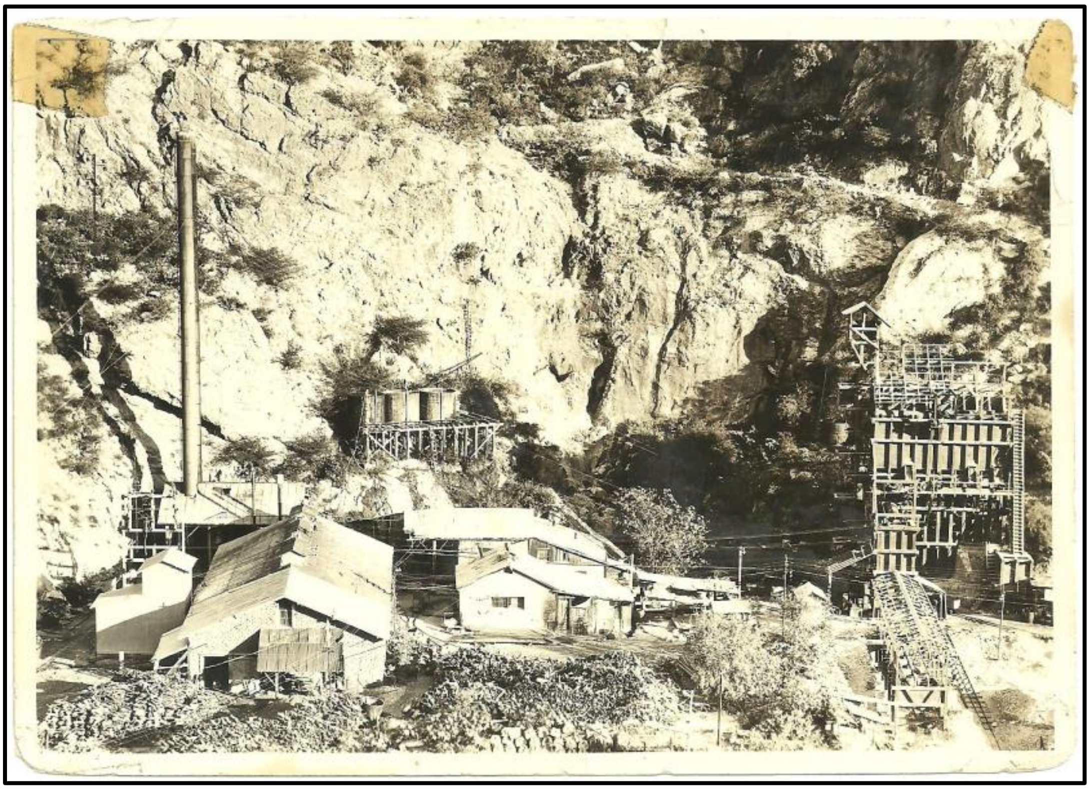


Photo 6-7: Historical Photo of San Gotardo Mill (Location 3, circa 1910s)

Photo 6-8: Photo of Historical Processing Facility at Babicanora, Established in 1921

Photo 6-9: View of Babicanora Central Portal and Site of Historical Processing Facility, November 2017

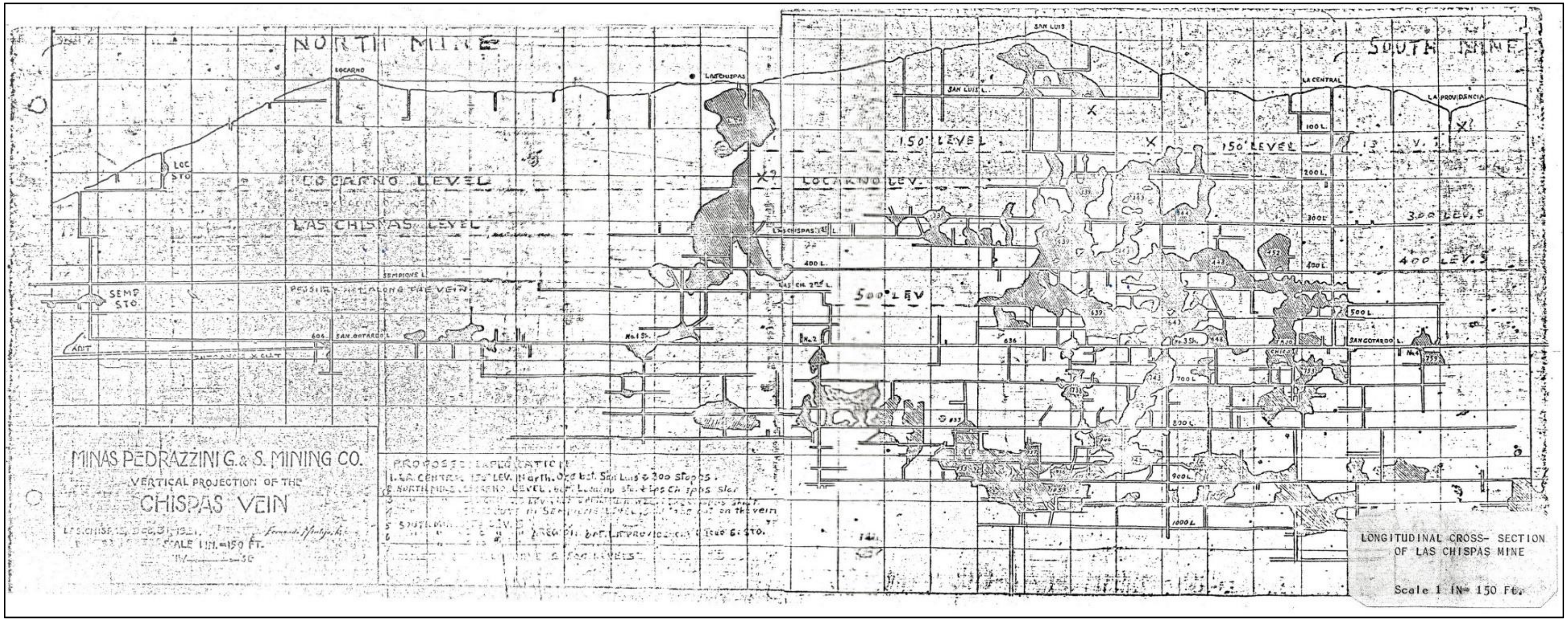


Photo 6-10: Long Section of the Historical Las Chispas Underground Development (circa 1921, looking northeast)

6.2.2 Mid- to Late-1900s to Early 2000s

No written documented information is available for the Project area during this period. Verbal discussions with Luis Perez, a local operator, indicate that from 1974 to 1984 a small cyanide leach mill was constructed near the highway entrance to the Project. During this period, approximately 75,000 t of historical waste was processed with doré poured on-site. No production estimation is available.

It is assumed that sometime between the mid-1930s and 2008, the historical and 1974 processing plants were dismantled and transported from the area and that both concession and surface ownership likely changed hands at least once from the mining companies to their current owners.

6.2.3 Minefinders Corporation Ltd. (2008–2011)

6.2.3.1 Overview

In 2008 Minefinders Corporation Ltd (Minefinders) operating under their Mexican affiliate, Minera Minefinders, acquired concessions within the current SilverCrest ground holdings, but were unable to negotiate with the main district concession owners. Subsequently, Minefinders completed initial exploration work on the district, which they referred to as the Babicanora Project. They drilled seven reverse circulation (RC) holes off the main mineralization trends with negative results, and then dropped the option in 2012.

Minefinders conducted a systematic exploration program across these concessions between 2008 and 2011. Regional activities consisted of geological mapping and a geochemical sampling program totalling 143 stream sediment and bulk-leach extractable gold (BLEG) samples, 213 underground rock chip samples, and 1,352 surface rock chip samples. The work was successful in identifying three gold targets along a 3 km-long structural zone. The most prospective of these targets was interpreted to be an area between the Las Chispas Vein and the Babicanora Vein. Minefinders focused on the western extension of the Babicanora Vein called El Muerto, which is the only part of the trend that was acquired by concession and accessible for exploration work.

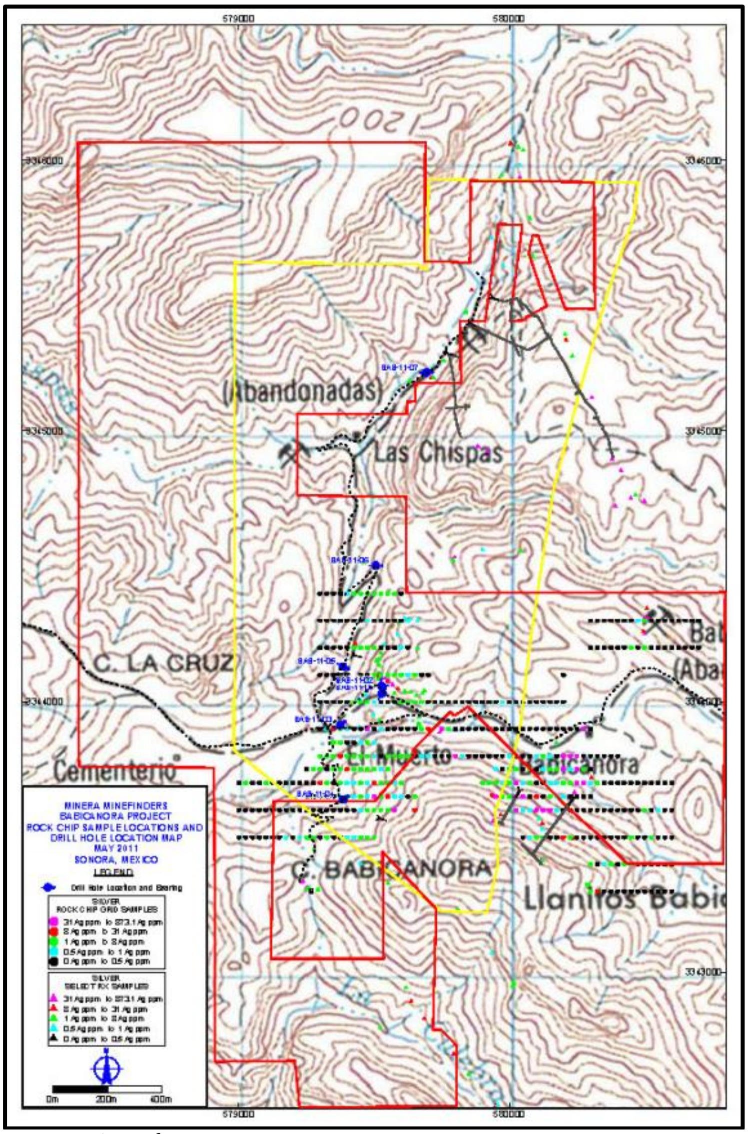
Targeted exploration conducted solely within the Babicanora Project area included the collection of 24 stream sediment and BLEG samples, 184 select surface rock chip samples, 474 grid rock chip samples, and drilling of seven RC drill holes for a total of 1,842.5 m. The drill hole locations are shown in Figure 6-1 and Figure 6-2.

6.2.3.2 Minefinders Surface Sampling

Turner (2011) describes the work by Minefinders on the Babicanora Project in detail. Outcrop in the area is variable and the sampling was adjusted based on terrain limitations. Minefinders determined that high-grade gold and silver occurrences noted in mine workings and outcrops occurred mainly as discontinuous and narrow quartz stockwork zones. Notable exceptions were a 5 m wide zone and narrow veins in the El Muerto area, northwest of the Babicanora Mine workings.

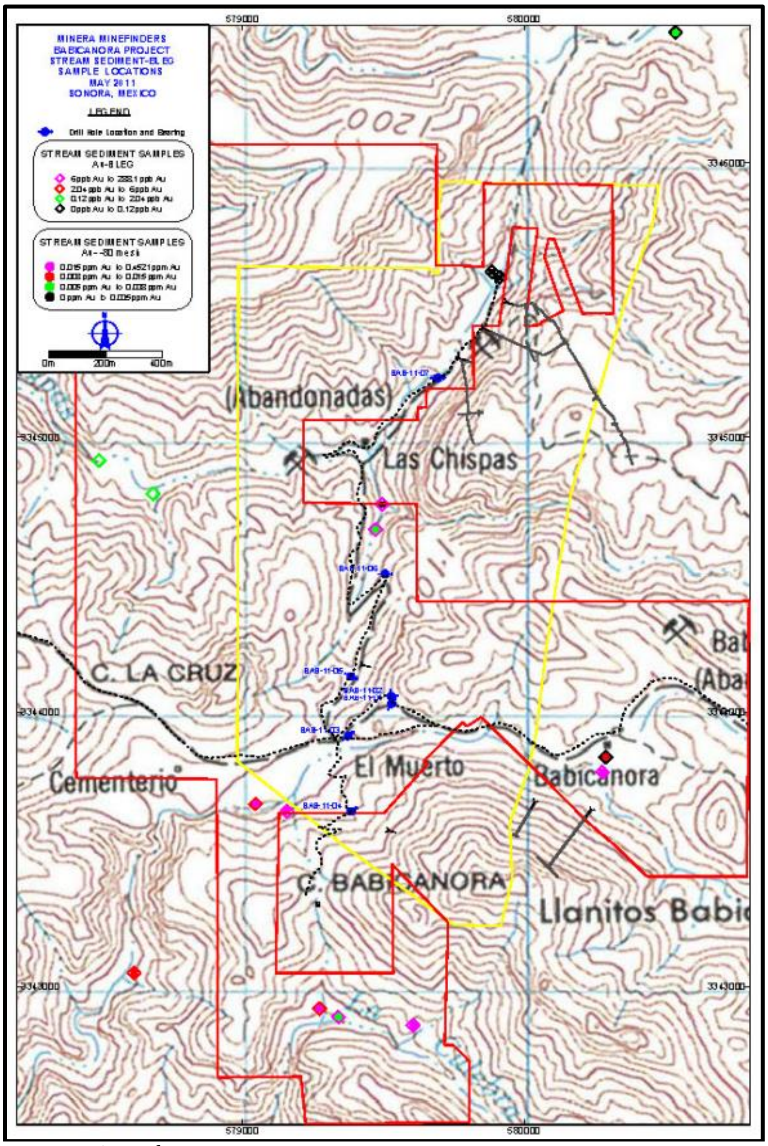
Twenty-four (24) stream sediment samples were collected from drainages in the Las Chispas Area, as part of a regional sampling program. The large samples were analysed as both 2 kg BLEG samples and via a more conventional analysis of a -80 mesh sieved product. The material utilized for the -80 mesh analysis was obtained after splitting the initial 2 kg used for BLEG analysis. Anomalous zones defined by the regional stream sediment program were later confirmed by a follow-up rock-chip grid sampling program.

All surface rock chip and stream sediment samples were collected by the staff of Minefinders and submitted to ALS Chemex in Hermosillo. Sampling coverage and results are also illustrated in Figure 6-1of Minefinders Rock Chip Sample Locations and Gold Results (Turner, 2011) and Figure 6-2.



Note: Figure from Turner, 2011

Figure 6-1: Minefinders Rock Chip Sample Locations and Gold Results



Note: Figure from Turner, 2011.

Figure 6-2: Minefinders Stream Sediment Sample Gold Results – BLEG and -80 Mesh

6.2.3.3 Minefinders Drilling 2011

Minefinders carried out a seven-hole RC drill program in 2011. The purpose of the program was to test a porous volcanic agglomerate (i.e., lithic tuff) unit located west of the main mineralization trend the Babicanora and Las Chispas areas. The area initially drilled by Minefinders is now known as the El Muerto Zone.

Minefinders contracted Drift Drilling to drill the seven (7) holes utilizing an MPD-1000 RC drill rig. The drilling was conducted from existing roads with drill pads enlarged to allow for safe and effective operations. Environmental permitting with the Ministry of Environment and Natural Resources of Mexico (SEMARNAT in the Spanish acronym) was prepared by Bufete Minera y Servicios de Ingeniería S.A. de C.V. and completed on March 23, 2011. All assay work was conducted by Bureau Veritas of Hermosillo, Mexico and Reno, Nevada.

The drill program was conducted between April 7, 2011 through May 3, 2011, with a total of 1,842.5 m drilled. The drill holes were oriented to intercept a range of host rocks in areas of anomalous precious metals or adjacent to mine workings. The plan was that bulk tonnage targets might exist within the more porous or chemically reactive rocks. Table 6-3 shows a summary of the drilling.

Table 6-3: Summary of Minefinders 2011 RC Drill Program

Hole ID	Easting	Northing	Elevation (m)	Dip (°)	Azimuth (°)	Depth (m)	Depth (ft)
BAB11-01	579527	3344033	1,135	-60	30	304.80	1,000
BAB11-02	579526	3344060	1,130	-90	0	324.60	1,065
BAB11-03	579372	3343914	1,091	-60	50	242.30	795
BAB11-04	579382	3343638	1,132	-55	60	350.50	1,150
BAB11-05	579386	3344130	1,053	-45	115	198.12	650
BAB11-06	579507	3344503	1,009	-70	90	182.90	600
BAB11-07	579693	3345216	977	-70	90	239.30	785
Total						**1,842.52**	**6,045**

The drill results were disappointing in that none of the holes were interpreted to have intersected the mineralization structure beneath the historical workings. Only narrow zones of gold mineralization at scattered depths were encountered and only one hole, BAB11-02, intercepted significant mineralization greater than 900 ppb of gold in four narrow intervals. This mineralization interval occurs within basal volcaniclastic sandstones and rhyodacitic tuffs cut by propylitic altered dacite dykes.

Results of the drilling indicate that several phases of quartz veining, accompanied by broad zones of argillic and propylitic alteration, are present in the 1.5 km long target zone. Mineralization was determined to occur as low sulphidation gold-silver epithermal quartz and calcite veins and stockwork within an Oligocene volcanic sequence consisting of volcaniclastic sediments interbedded with rhyolitic tuff and andesitic dykes/flow cut by dacitic dykes.

In 2012, Minefinders dropped their interest in the Babicanora Project.

6.2.4 SilverCrest (2013 to October 2020)

Following Minefinders' retreat, SilverCrest Mines Inc. (now a subsidiary of First Majestic), through its subsidiary Nusantara de Mexico S.A. de C.V. (Nusantara), initiated their interest in Las Chispas in 2013. Legal issues in the main Las Chispas district were settled and SilverCrest Mines Inc. could negotiate option agreements with all the concession holders through their Mexican subsidiary Nusantara. By the end of September 2015, SilverCrest Mines Inc. executed option agreements to acquire rights to 17 concessions.

On October 1, 2015, pursuant to an arrangement agreement, SilverCrest Mines Inc. was acquired by First Majestic and these mineral concessions were transferred to a new spin-out company, the newly registered SilverCrest Metals Inc., and its subsidiary LLA. These companies subsequently obtained rights to 11 additional mineral concessions for a total of 28 concessions.

From March 2016 to November 2016, the Phase I exploration program consisted of initial drilling comprising 22 drill holes, surface and underground mapping and sampling, and rehabilitating an estimated 6 km of underground workings.

From November 2016 to February 2018, the Phase II exploration program consisted of 161 drill holes, additional surface and underground mapping and sampling, further rehabilitation of 4 km of underground workings, plus auger and trenching of approximately 174,500 t in 42 surface historical mine dumps. This work program supported an initial mineral resource estimate (Barr 2018). The February 2018 initial Mineral Resource estimate of Barr (2018) encompassed vein-hosted material at the Babicanora, Las Chispas, William Tell, and Giovanni veins and surface stockpiled material remaining from historical operations such as mine dumps, waste tailings deposits, and recovered underground muck material.

From February 2018 to February 2019, the Phase III exploration program consisted of 256 drill holes, additional surface and underground mapping and sampling, and finalizing approximately11 km of underground rehabilitation, the majority of which is located on the Las Chispas Vein and historical mine. This work program resulted in updated Mineral Resource estimates (Fier 2018; Barr and Huang, 2019 – see Table 14-1 in the Report). The Mineral Resource Estimate of Barr and Huang (2019) encompassed vein material from the Babicanora, Babicanora FW, Babicanora HW, Babicanora Norte, Babicanora Sur, Granaditas, Las Chispas, William Tell, Giovanni, and Luigi veins and previously reported surface stockpiled material. Environmental baseline and supporting studies were undertaken, permitting activities commenced, and metallurgical testwork was initiated in support of a Preliminary Economic Assessment (PEA) (Barr et al., 2019).

The PEA results were announced in a SilverCrest press release dated May 15, 2019 and are summarized in Table 6-4.

Table 6-4: Summary of Las Chispas PEA Summary (Base Case)

Metric	Value
Throughput (t/d)	1,250
Mine Life	8.5 years
Diluted Resource (tonnes)	3,861,000
Average Diluted Gold Grade (gpt)	4.05
Average Diluted Silver Grade (gpt)	411
Average Diluted AgEq[(1)] Grade (gpt)	714
Contained Gold oz[(4)]	502,200
Contained Silver oz [(4)]	51,004,000
Contained AgEq oz[(1)(4)]	88,666,000
Gold Recovery (%)	94.4
Silver Recovery (%)	89.9
Payable Gold oz (LOM)	473,100
Payable Silver oz (LOM)	45,765,000

Metric	Value
Total AgEq(1) oz	81,247,000
Average Annual Production (LOM)	
-Gold oz	55,700
-Silver oz	5,384,000
-AgEq(1) oz	9,559,000
Average Annual Production (Years 1-4)	
-Gold oz	81,600
-Silver oz	7,575,000
-AgEq(1) oz	13,694,000
Mining Cost ($/t) (3)	$50.91
Processing Cost ($/t)	$32.61
G&A Cost ($/t)	$15.14
Total Operating Cost ($/t)	$98.66
Initial Capital Cost ($ M)	$100.5
LOM Sustaining Capital Cost ($ M)	$50.3
LOM AISC ($/oz AgEq(1))	$7.52
Years 1-4 AISC ($/oz AgEq(1))	$4.89
After-Tax IRR	78%
NPV (5%, $ M)	$406.9
Undiscounted LOM net free cash flow ($ M)	$522.5
Payback period	9 months

(1) AgEq based on 75 (Ag):1 (Au), calculated using long-term gold and silver prices of $17 per ounce silver and $1,225 per ounce gold with average metallurgical recoveries of 95% gold and 90% silver.
(2) Note: base case silver price was above the spot price of silver and the base case gold price was below current spot price for gold as of May 15, 2019.
(3) Includes expensed lateral development but excludes capitalized ramp and vertical development.
(4) Contained ounces for gold and silver were estimated to include 35% indicated resources and 65% inferred resources.
(5) Metal prices used for the economic model were $1,269/oz Au and $16.68/oz Ag (~3-year historical average).
(6) Mexico Peso/$ exchange rate of 20:1.

The PEA envisaged an underground mechanized and cut-and-fill with resue mining operation. The 1,250 t/d process plant would use gravity concentration, cyanidation, and Merrill Crow-CCD processes. Under the assumptions in the PEA, the Project showed positive economics over an 8.5-year mine life. The authors of the PEA recommended advancing the Project to feasibility level, along with extending the Phase III exploration program.

The Feasibility Study was commissioned in late 2019. The Phase III Extended exploration program was completed between February 2019 to October 2020 and the results are presented in Section 0 of the Report.

7 Geological Setting and Mineralization

7.1 Regional Geology

The Project is located in north-western Mexico where much of the exposed geology can be attributed to the subduction of the Farallon Plate beneath the North American Plate and related magmatic arc volcanism. The east-directed subduction of the Farallon Plate began in early Jurassic (approximately 200 Ma) with the tectonic rifting of the supercontinent Pangea (Rogers, 2004). The resulting northwest-trending Sierra Madre Occidental extends over 1,200 km from the US-Mexican border to Guadalajara in the southeast.

Delgado-Granados et al. (2000) proposed that subduction of the Farallon Plate occurred at a relatively shallow angle, resulting in continental uplift across northern Mexico and docking of accretionary terranes along the western fringes of the pre-existing Jurassic continental and marine sediments and crystalline Cambrian basement rocks.

Volcanism is related to fractional crystallization of mantle-derived basalts during subduction (Johnson, 1991; Wark et al., 1990). The widespread volcanic deposits and intrusive stock development from emplacement of the regional batholith typify the upper Cretaceous record in the area, which was followed by voluminous accumulation of volcanic flows, pyroclastics, and volcano-sedimentary rocks during the Upper Cretaceous through to the Eocene.

Continental arc volcanism culminated with the Laramide orogeny in the early to late Eocene (Alaniz-Alvarez and Nieto-Samaniego, A.F., 2007). The waning of compression coincided with east–west-directed extension between late Eocene to the early Oligocene (Wark et al., 1990; Aguirre-Diaz and McDowell 1991; 1993), along the eastern Sierra Madre Occidental flank and is considered to be the first formation stage of the Basin and Range province.

By early to mid-Miocene, extension migrated west into Northern Sonora and along the western flank of the Sierra Madre Occidental, resulting in north–northwest to south–southeast trending, west dipping, and normal faults. This extensional regime caused major deformation across the Sierra Madre Occidental, resulting in localized exhumation of pre-Cambrian basement rocks within horst structures, especially in the Northern Sierra Madre Occidental (Ferrari et. Al., 2007). Bimodal volcanic flows capped the volcano–sedimentary deposits of the late Eocene. Migration of later hydrothermal fluids along the pre-existing structures are related to the cooling of the orogenic system.

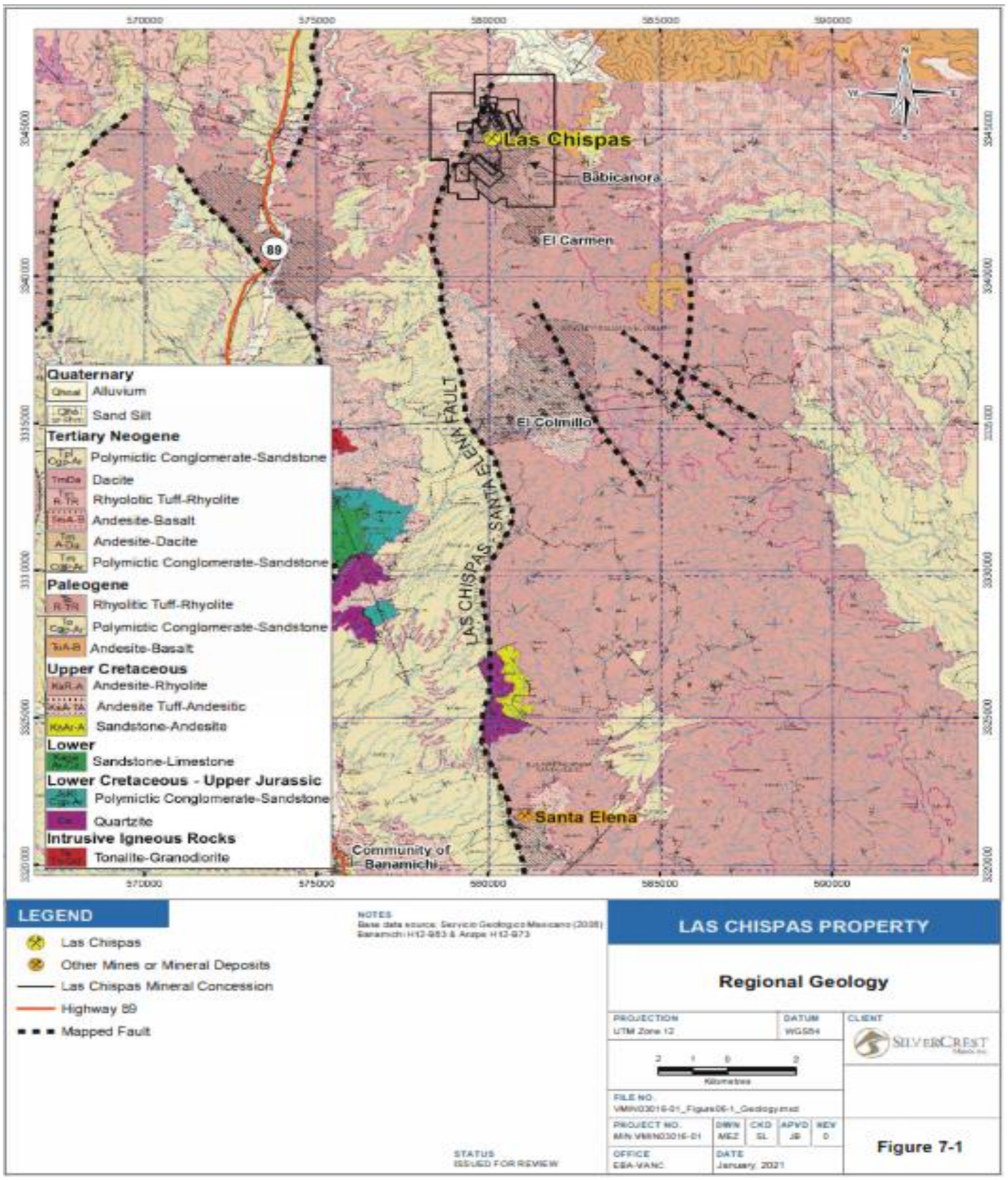
The Pliocene–Pleistocene is characterized by a general decrease in volcanic activity, with deposition of some basalt flows, and accumulation of conglomerate, locally known as the Baucarit Formation.

Ferrari et al. (2007) summarizes five main igneous deposits of the Sierra Madre Occidental:

1. Plutonic/volcanic rocks: Late Cretaceous–Paleocene;
2. Andesite and lesser dacite–rhyolite: Eocene (Lower Volcanic Complex);
3. Felsic dominant and silicic ignimbrites: Early Oligocene and Miocene (Upper Volcanic Complex);
4. Basaltic-andesitic flows: late stage of and after ignimbrite pulses; and,
5. Alkaline basalts and ignimbrites: Late Miocene–Pleistocene (Post-subduction volcanism).

Mineralizing fluids likely originated from mid-Cenozoic intrusions. The structural dilation along the faults formed conduits for mineral-bearing solutions. The heat source for the mineralizing fluids was likely the plutonic rocks that commonly crop out in Sonora.

Porphyry deposits of the Sierra Madre Occidental occur in the Lower Volcanic and are correlated with the various Middle Jurassic through to Paleogene age intrusions. Examples of these deposits are Cananea, Nacozari and La Caridad (Ferrari et. Al., 2007). In Sonora, formation of these deposits is considered to be influenced by east–west and east–northeast-to west–southwest-directed extension. Early Eocene tectonic activity, which resulted in northwest-trending shear and fault zones, appears to be an important control on mineralization in the Sonora region (Figure 7-1).



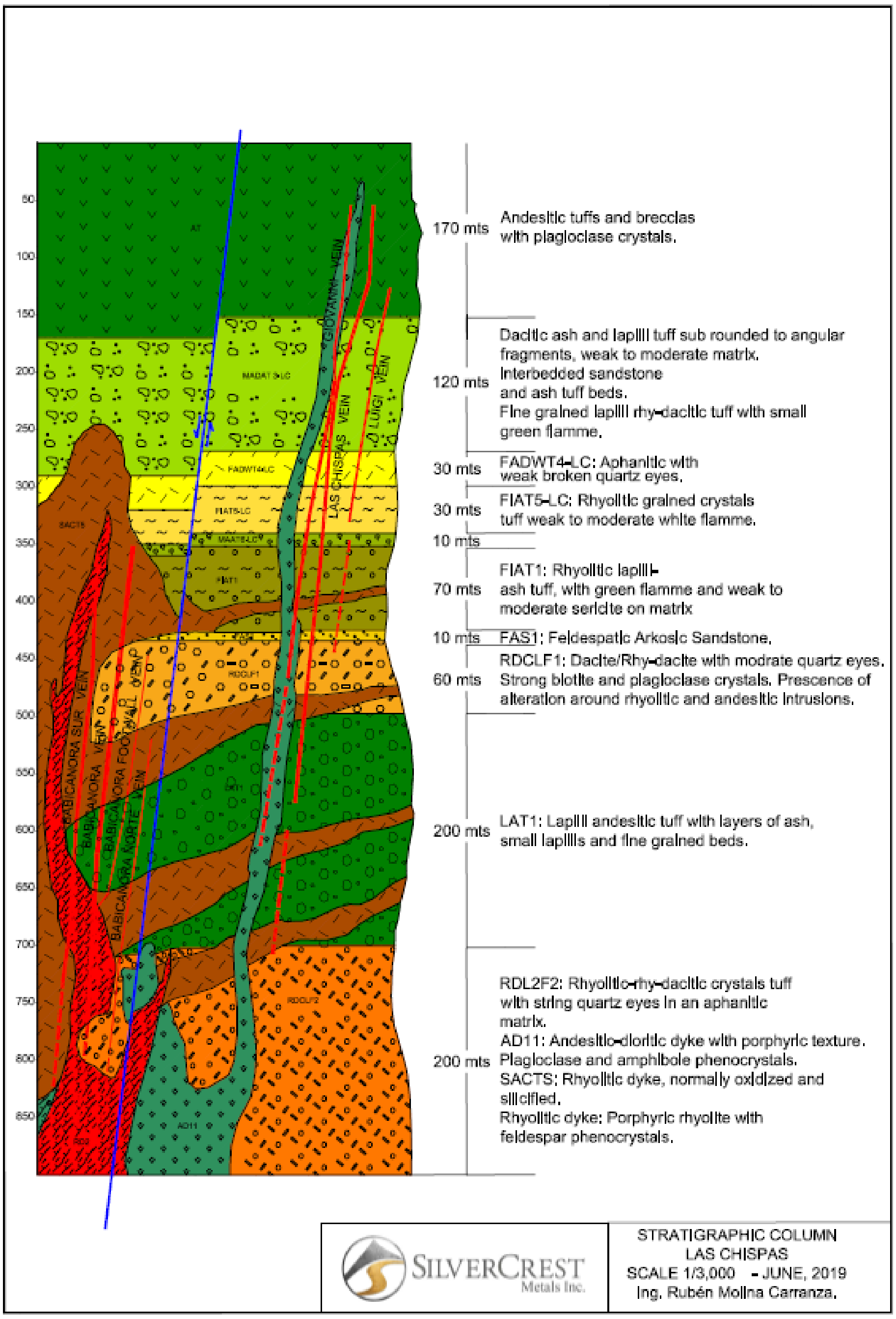
Source: SilverCrest, 2019.

Figure 7-1: Regional Geology Showing Major Graben of the Rio Sonora and Continuous Normal Fault between Santa Elena and Las Chispas

7.2 Local Geology

7.2.1 Lithologies

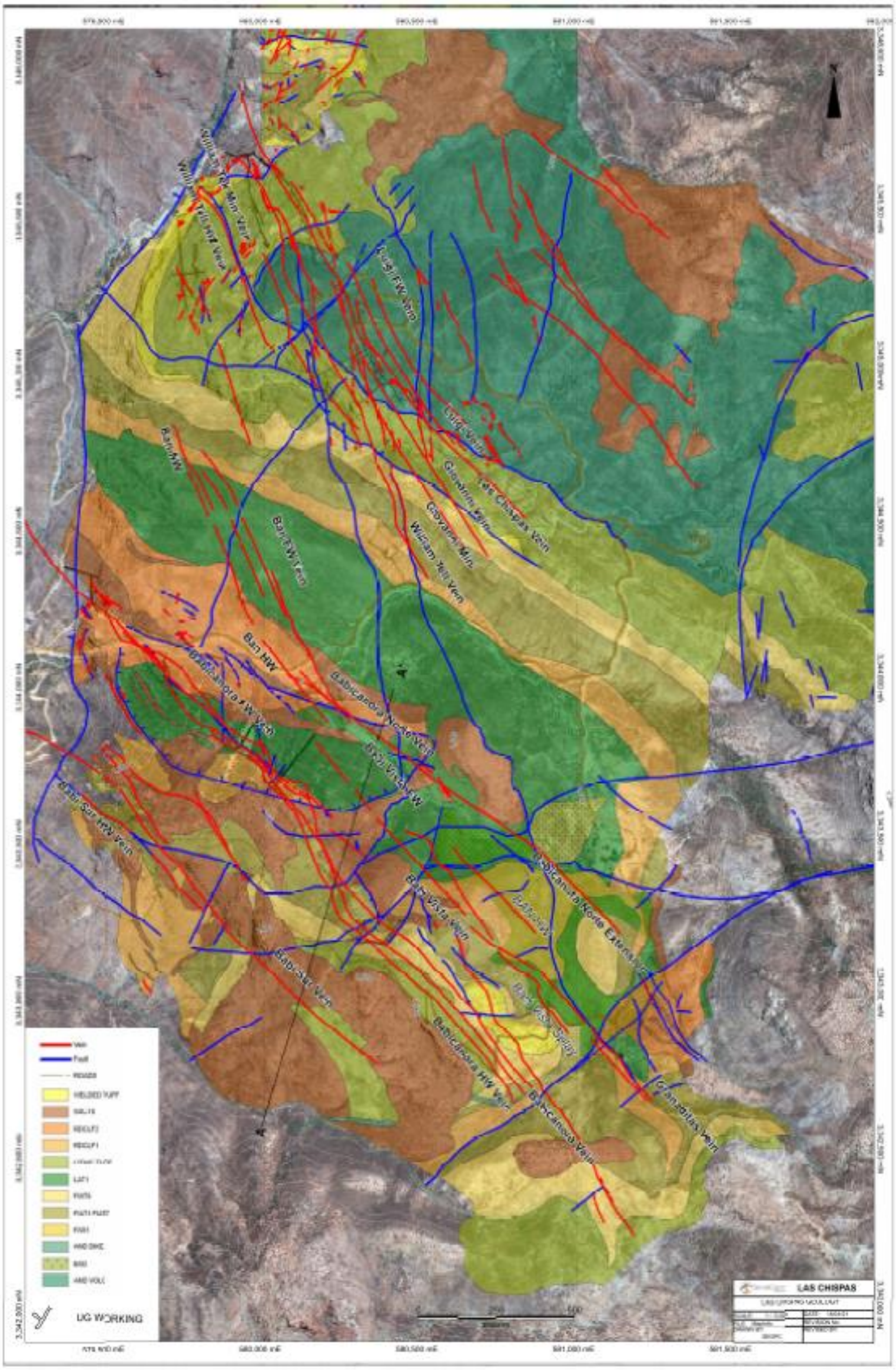
The host rocks in the Las Chispas Area are generally pyroclastic, tuffs, and rhyolitic flows interpreted to be members of the Lower Volcanic Complex. Locally, volcanic pyroclastic units mapped within the underground workings include rhyolite, welded rhyodacite tuff, lapilli (lithic) tuff, and volcanic agglomerate. Figure 7-2 provides a schematic summary of the regional and local stratigraphy.



Note: Figure prepared by SilverCrest, 2021.

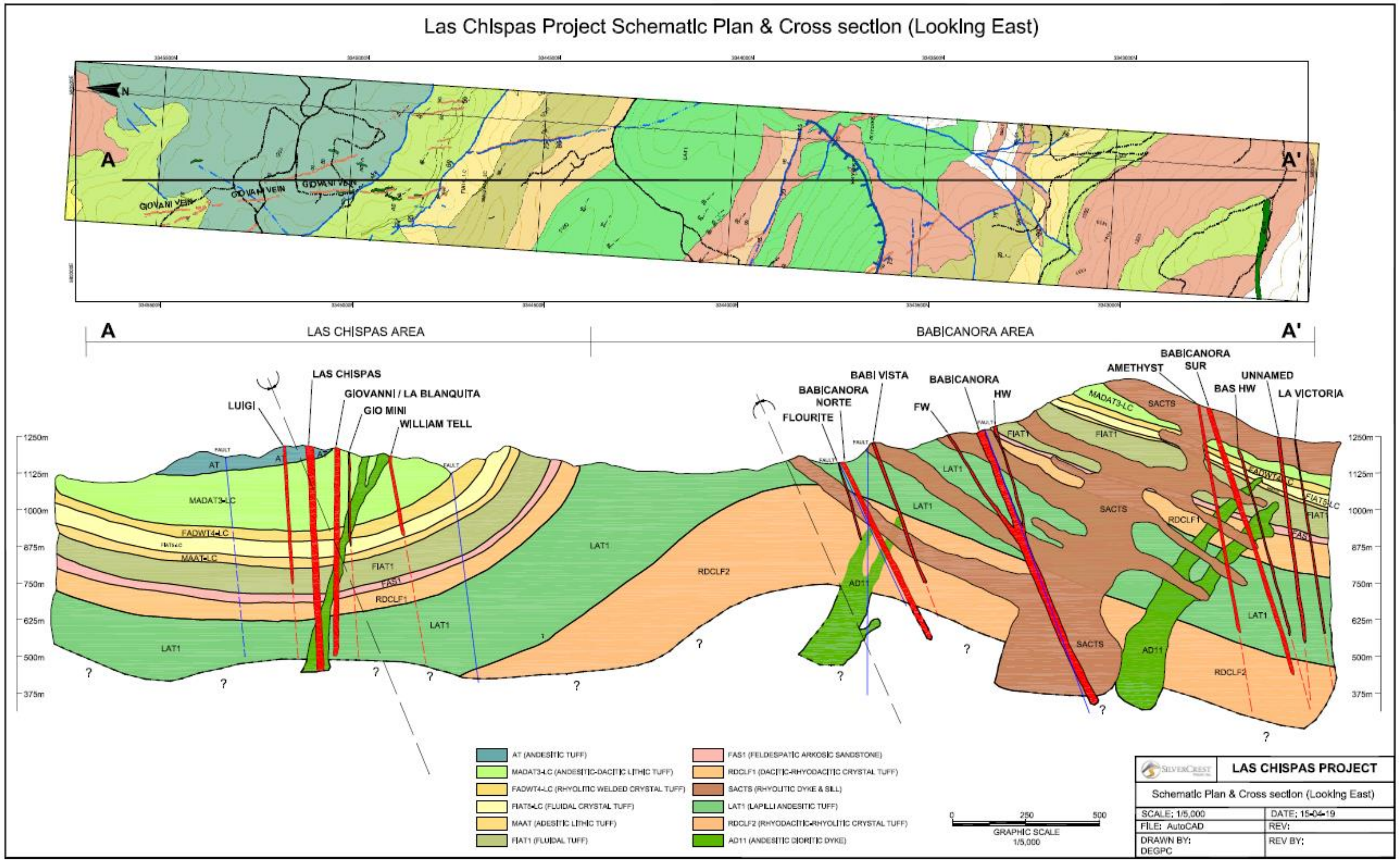
Figure 7-2: Stratigraphic Column for Las Chispas Property

The volcanic units form a gentle syncline and anticline complex across the Project, which is cross-cut nearly perpendicular to the fold axis by the dominant vein trend (Mulchay, 1935). Figure 7-3 and Figure 7-4 show the district geology and a typical section looking eastwards through the Project.



Note: Figure prepared by SilverCrest (2019). Refer to Figure 7-3 for legend explanation.

Figure 7-3: Las Chispas District Geology Map



Note: Figure prepared by SilverCrest, 2021.
Major mineralization lithologic units for this geology plan map are defined as: LAT1, Lithic andesitic tuff and the most significant host for vein-related gold-silver mineralization; RDCLF 1 and 2, rhyodacitic flows which restrict mineralization but can be mineralization; SACTS, silicic andesitic to rhyolitic fragments which occur in sill and dyke form with dykes associated with mineralization.

Figure 7-4: Las Chispas District Cross-Section

7.2.2 Geochemistry

Thin section and TerraSpec™ hyperspectral studies indicate that the alteration generated during the mineralization events are dominantly multi-pulse neutral and consistent with low-sulphidation mineralization. The typical alteration assemblage is montmorillonite-illite ± kaolinite ± MgFe chlorite ± pyrite. However, more acidic species of minerals and clays are also present, such as alunite, dickite and ammonium. In conjunction with the more acidic alteration, magmatically derived orthoclase occurs in thin sections as fine-grained interlobate aggregates that occupy the interstices between the coarse-grained quartz. This relationship indicates that the quartz-rich mineralizing fluids and the orthoclase are syngenetic, and therefore part of the same event (Colombo, 2017a). To produce these near neutral clays and minerals in conjunction with the more highly acidic species, two or more distinct fluid pulses are plausible.

A review of the core database was undertaken in January 2018, which included information available from the 46,925 samples from the various mineralization veins that had been sampled to date by SilverCrest within the Project. The review focused on the correlation coefficient (Table 7-1) and descriptive statistics for modal abundance (Table 7-2) of the anomalous and expected elements typically associated with low to intermediate sulphidation deposits.

Gold and silver have a strong positive correlation coefficient. Emplacement of both gold and silver seems to be strongly related, although there is thin section evidence of a quartz + gold-only event at Babicanora. The principal economic low- to intermediate-sulphidation elements (gold, silver, copper, lead, zinc, and antimony) all have a strong affinity for one another. Mercury does not have a conclusive positive or a negative correlation and has negligible values. Lead and zinc have a very high correlation coefficient 0.870. However, base metals and accessory minerals have low abundance within all the targets.

There is a slight increase in base metal content in the targets located in the eastern portion of the Project, such as Granaditas, which is interpreted to be a deeper section of the epithermal system. This increase may indicate an evolution of the fluids as they ascend or separate base metal-rich pulses, though the mode of emplacement is unclear. Sulphur has a moderate correlation with zinc and lead, likely due to sulphur in their respective sulphide minerals. The gold and silver sulphide hosted mineralization in the uppermost portion of the targets has been oxidized and mobilized as sulphate, resulting in reduced total sulphur contents.

A fluid inclusion study determined that depths of emplacement of mineralization ranged from approximately 100 m to >2 km. The shallow depths of emplacement are outside the current main mineralization zones. Depth of emplacement in the main mineralization zone is well below 1,000 m, with a maximum depth of >2 km (Pérez, 2017). These deeper depths of emplacement are complicated by possible caldera collapse and a change in the paleo-surface.

Overprinting of low- and high-sulphidation mineralization and alteration with conflicting depths of formation are noted in the fluid inclusion, TerraSpec™ hyperspectral and thin section studies, which point towards caldera collapse as a mechanism of emplacement.

Table 7-1: Correlation Coefficient Table, Anomalous Values Highlighted, >0.25 and <-0.25 (January 2018)

	Au	Ag	Cu	Pb	Zn	As	Ba	Cd	Co	Fe	Hg	Mn	Mo	S	Sb
Au	1.00	0.87	0.33	0.20	0.17	0.04	0.00	0.23	-0.01	0.00	0.11	0.00	0.01	0.01	0.52
Ag	0.87	1.00	0.31	0.18	0.16	0.03	0.00	0.20	-0.01	0.00	0.09	0.00	0.02	0.01	0.41
Cu	0.33	0.31	1.00	0.14	0.14	0.06	0.01	0.19	0.09	0.05	0.08	0.01	0.14	0.04	0.33
Pb	0.20	0.18	0.14	1.00	0.39	0.21	0.00	0.43	0.00	-0.03	0.08	0.01	0.09	0.07	0.17
Zn	0.17	0.16	0.14	0.39	1.00	0.20	0.00	0.93	0.10	0.07	0.12	0.06	0.03	0.17	0.16
As	0.04	0.03	0.06	0.21	0.20	1.00	0.00	0.20	0.07	0.07	0.11	0.08	0.06	0.18	0.12
Ba	0.00	0.00	0.01	0.00	0.00	0.00	1.00	0.00	-0.01	-0.01	0.04	0.39	0.02	-0.07	0.21
Cd(1)	0.23	0.20	0.19	0.43	0.93	0.20	0.00	1.00	0.03	-0.04	0.13	0.04	0.05	0.12	0.21
Co	-0.01	-0.01	0.09	0.00	0.10	0.07	-0.01	0.03	1.00	0.74	0.03	0.21	0.02	0.10	0.05
Fe	0.00	0.00	0.05	-0.03	0.07	0.07	-0.01	-0.04	0.74	1.00	-0.03	0.15	-0.02	-0.25	0.04
Hg(1)	0.11	0.09	0.08	0.08	0.12	0.11	0.04	0.13	0.03	-0.03	1.00	0.02	0.03	0.05	0.14
Mn	0.00	0.00	0.01	0.01	0.06	0.08	0.39	0.04	0.21	0.15	0.02	1.00	-0.02	-0.03	0.31
Mo(1)	0.01	0.02	0.14	0.09	0.03	0.06	0.02	0.05	0.02	-0.02	0.03	-0.02	1.00	0.02	0.17
S	0.01	0.01	0.04	0.07	0.17	0.18	-0.07	0.12	0.10	-0.25	0.05	-0.03	0.02	1.00	0.00
Sb(1)	0.52	0.41	0.33	0.17	0.16	0.12	0.21	0.21	0.05	0.04	0.14	0.31	0.17	0.00	1.00

Note: (1) Low statistical population

Table 7-2: Basic Statistics for Trace Elements (January 2018)

Parameter	Count	Minimum	Maximum	Mean	Total	Variance	Standard Deviation	Coefficient of Variation	Skewness	Kurtosis
Weight (kg)	45,944	0.22	12.94	3.899	179,149	3.77	1.942	0.5	0.81	-0.23
Length (m)	46,925	0.1	7.5	1.113	52,249	0.28	0.527	0.47	0.83	0.94
Au (ppm)	45,934	0.001	305	0.122	5,611	5.7	2.387	19.54	77.06	7,654
Ag (ppm)	45,934	0.2	21,858	11.068	508,393	34,356	185.353	16.75	68.64	6,237
Cu (ppm)	29,184	1	10,250	10	290,069	5,810	76	7.67	91.07	11,398
Pb (ppm)	29,184	2	8,150	37	1,089,937	36,473	191	5.11	19.58	526.5
Zn (ppm)	29,060	2	17,700	58	1,699,437	45,639	214	3.65	38.92	2477
Ba (ppm)	29,091	1	10,000	151	4,386,336	78,966	281	1.86	9.57	207.5
Ca (pct)	28,933	0.01	25	1.086	31,420	1.87	1.366	1.26	5.69	64.74
Cd (ppm)	3,740	0.5	130	2.023	7,568	25.96	5.095	2.52	13.74	248
Co (ppm)	24,678	1	176	4	101,027	31.29	6	1.37	3.45	41.09
Hg (ppm)	4,311	0	41	1	4,692	1.03	1	0.93	22.57	705.3
Mn (ppm)	29,064	1	50,000	564	16,399,438	991,598	996	1.76	26.17	1,063
Mo (ppm)	11,304	0	1,670	4	43,432	623.7	25	6.5	44.69	2,531
S (pct)	24,815	0.01	34	0.388	9,636	0.9	0.947	2.44	16.65	381.9
Sb (ppm)	13,910	1	1,045	5	75,476	316.2	18	3.28	36	1,717

7.2.3 Alteration

All rock types in the Project show signs of extensive hydrothermal alteration. The thin section and TerraSpec™ hyperspectral studies identified alteration consistent with argillic and advanced argillic alteration. Alteration minerals identified include smectite, illite, kaolinite, chlorite, carbonate, iron oxy/hydroxides, probable ammonium, gypsum/anhydrite, silica, and patch trace alunite.

Generally, the majority of the mineral deposits drilled to date are above the existing water table; however, it is understood that paleo-water levels have fluctuated and may have previously been higher. Oxidation of sulphides is observed from near surface to depths greater than 300 m and the presence of secondary minerals is recorded from the Las Chispas underground workings approximately 60 m to 275 m depth from surface. Hematite mineralization occurs as halos around small veins, due to percolated meteoric water along small faults and fractures of oxidized iron sulphides. Strong and pervasive near-surface oxidation is noted to occur in the Babicanora Central Area, where host rocks experienced faulting and advanced weathering to limonite, hematite, and clays.

7.2.4 Mineralization

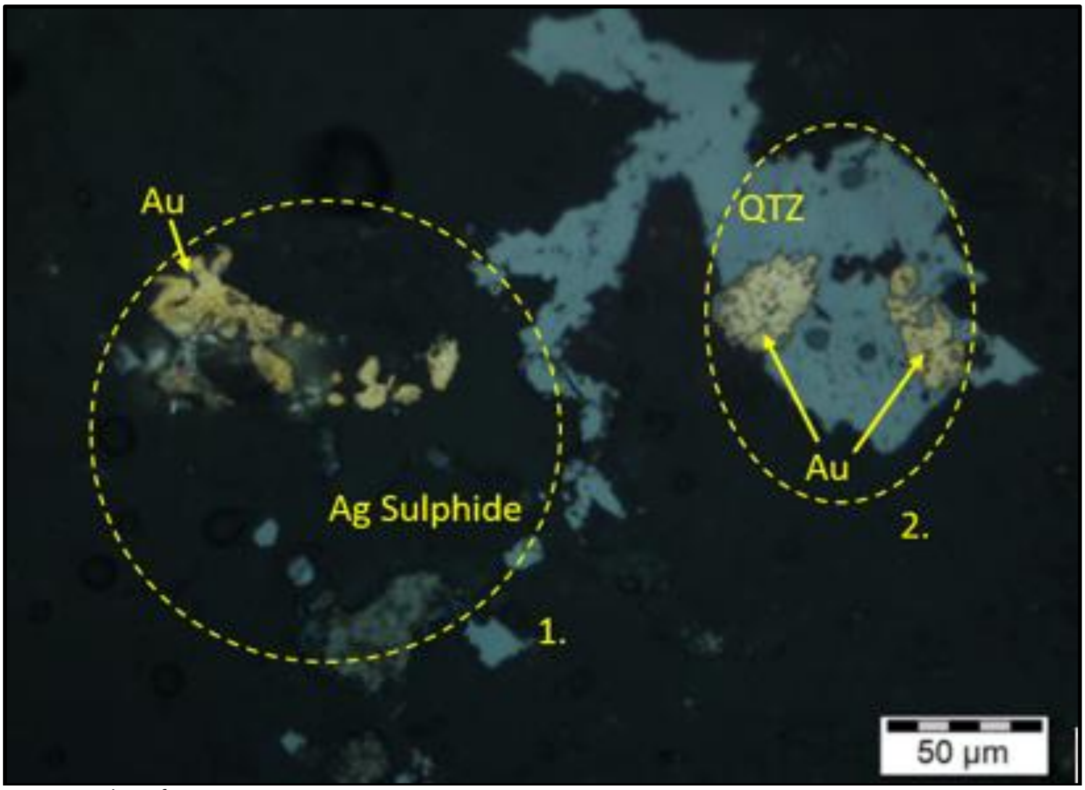
Mineralization is interpreted to be a deeply emplaced, low to intermediate-sulphidation system, with mineralization hosted in hydrothermal veins, stockwork, and breccia. Emplacement of the mineralization is influenced by fractures and low-pressure conduits formed within the rocks during tectonic movements. Mineralization can be controlled lithologically along regional structures, local tension cracks, and faulted bedding planes.

Historical reports and work conducted by SilverCrest have further investigated the gold, silver, base metals, and gangue minerals associated with the mineralization. The mineralization is 0.10 m to 9.30 m in true width and typically encompasses a central quartz ± calcite mineralization corridor with narrow veinlets within the adjacent fault damage zone. Stockwork and breccia zones are centred on structurally controlled hydrothermal conduits.

Historical reporting has identified economic mineralization in the form of silver sulphides and sulfosalts as the primary silver mineral species, and in association with pyrite. Secondary silver enrichment is indicated by the gradation from chlorargyrite near the surface to pyrargyrite at depth.

Silver mineralization is dominant throughout the Project. Typical ratios of silver to gold using a COG of 150 gpt silver equivalent (AgEq) are approximately: Babicanora Main Vein at 90:1, Babi Vista Main Vein at 56:1, Babicanora Norte Main Vein at 117:1, Babicanora Sur Main Vein at 53:1, Granaditas Vein at 102:1, Las Chispas Vein at 142:1, Giovanni Vein at 172:1, and William Tell Vein at 140:1. Overall, a 1:100 gold to silver modal ratio is considered for the Project.

Stronger gold mineralization is noted within the Babicanora Area than within the Las Chispas Area. The modes of gold mineralization currently identified are threefold: 1) gold associated with pyrite and chalcopyrite; 2) gold emplacement with silver sulphides (typically argentite and electrum); and 3) native gold flakes in quartz (Photo 7-1).



Note: Photo from PEA, 2019.

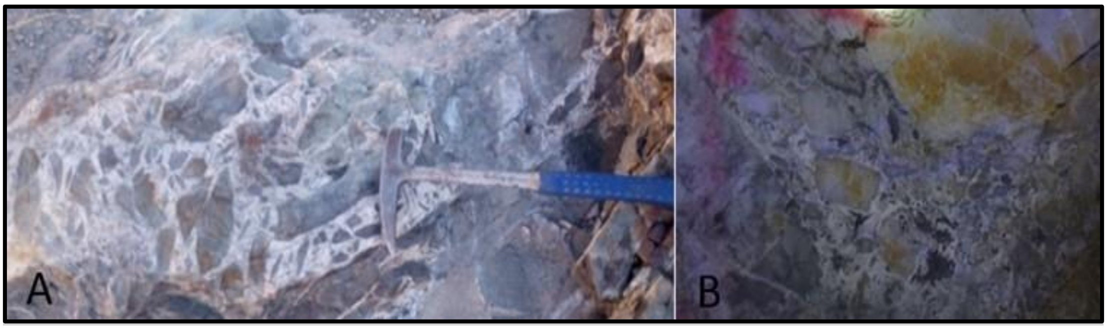
Photo 7-1: Thin Section of Gold and Silver Emplacement at Las Chispas

Additional sulphide species identified are minor chalcopyrite, sphalerite and galena. The veins are low in base metal mineralization, except for the far south-eastern extensions of the Babicanora Norte, Babi Vista and Granaditas veins, in the south-eastern part of the district. In addition to the petrographic findings in Babicanora, samples of an early sphalerite phase were followed by a later galena phase of mineralization and visual inspection of the base metal mineralization showed galena and sphalerite emplaced at the same time within the same discrete vein. Multiple pulses of base metal-rich fluids of variable composition formed the mineralization at the Project. There seems to be an increasing base metal content to the southeast and at depth. Government geophysical maps show a large magnetic anomaly to the east of the Project area, which could be a buried intrusive and potentially the main source of the mineralization in the district.

The veins and stockwork within the Las Chispas Vein consist of fine- to medium-grained, subhedral to euhedral interlocking quartz with minor cavities lined by comb quartz (typically crystals are 5 to 10 mm in length). SilverCrest geologists have not observed any quartz-pseudomorphed blades after platy carbonate or other textures that indicate a shallow environment. Vein emplacement and form are structurally and lithologically controlled. The rheology of the host rock plays an important role in structural preparation and emplacement of the mineralization. Within the fine-grained welded tuff, veining is narrow, typically with sharp narrow contacts, and chaotic. Veins and breccia emplacement in the more competent, medium-grained lapilli tuffs are wider commonly with parallel splays along the main structure, with denser veining in the adjacent fault damage zone.

Brecciated mineralization formed in two ways: 1) in zones of low pressure as hydrothermal breccia: and 2) as mechanical breccias.

In the hydrothermal breccia, mineralization is hosted in a siliceous matrix of hydrothermal quartz ± calcite and previously formed vein clasts that have been brecciated and re-cemented (Photo 7-2 A and B). Clasts are typically homolithic, angular, and show minimal signs of milling and rounding by hydrothermal processes. Although heterolithic breccias are present, they tend to be at the intersection points of the cross-cutting faults (striking 360°) to the main trend and at depth. Where breccia clasts are mineralization, mobilization of the clasts within conduits during multi-episodic pulse events is indicated. Gold values increase with increasing pyrite and chalcopyrite within the quartz matrix.



Notes: (A) Hydrothermal angular homolithic breccia, siliceous matrix with calcite and fine-grained sulphides weathering red.
(B) Heterolithic breccia with minor rounding of clasts and open space filling. Fine grained black sulphides and manganese hosted in the crystalline quartz matrix.

Photo 7-2: Breccias at Las Chispas

Re-cemented mechanical breccia generated by the reactivation of the fault hosting the mineralization is also present. These breccias consist of fault gouge, have a cataclasite texture, and are re-cemented with quartz and calcite. This reactivation mechanism also produces open space filling ores, including narrow stockwork quartz ± calcite ± adularia veins. Additional textures include banding, crustiform, comb, and chalcedonic silica-calcite veins. The matrix commonly has fine disseminated to coarse-grained banded sulphides associated with the cement.

Argentite is the principal silver mineral and has shown association with galena, pyrite ± marcasite and chalcopyrite. Gold and silver values have a strong correlation with each other and likely precipitated together during the crystallization of quartz. Base metals contents are low in veins. Minor zinc and lead are principally found in black sphalerite and galena as blebs and veinlets. Arsenic and mercury are conspicuously absent from the geochemistry. Minor antimony is present. Minor secondary copper minerals as chrysocolla and malachite occur underground in association with oxidized chalcopyrite, however, is rare.

Styles of mineralization present in the Project include laminated veins (Photo 7-3), stockworks, and quartz-calcite filled hydro-brecciated structures (Photo 7-4). The presence of epithermal textures, such as bladed calcite (replaced by quartz), miarolitic cavities, and chalcedony/crustiform banding mapped underground, suggest multiple phases of fluid pulses contributed to the formation of the mineral deposits.

Generally, it appears that epithermal mineralization is higher in the system (closer to the paleo-surface) on the northwestern side of the Las Chispas district compared to the southeastern side, where there is an increase in base metal content.

Note: Photo from PEA, 2019.

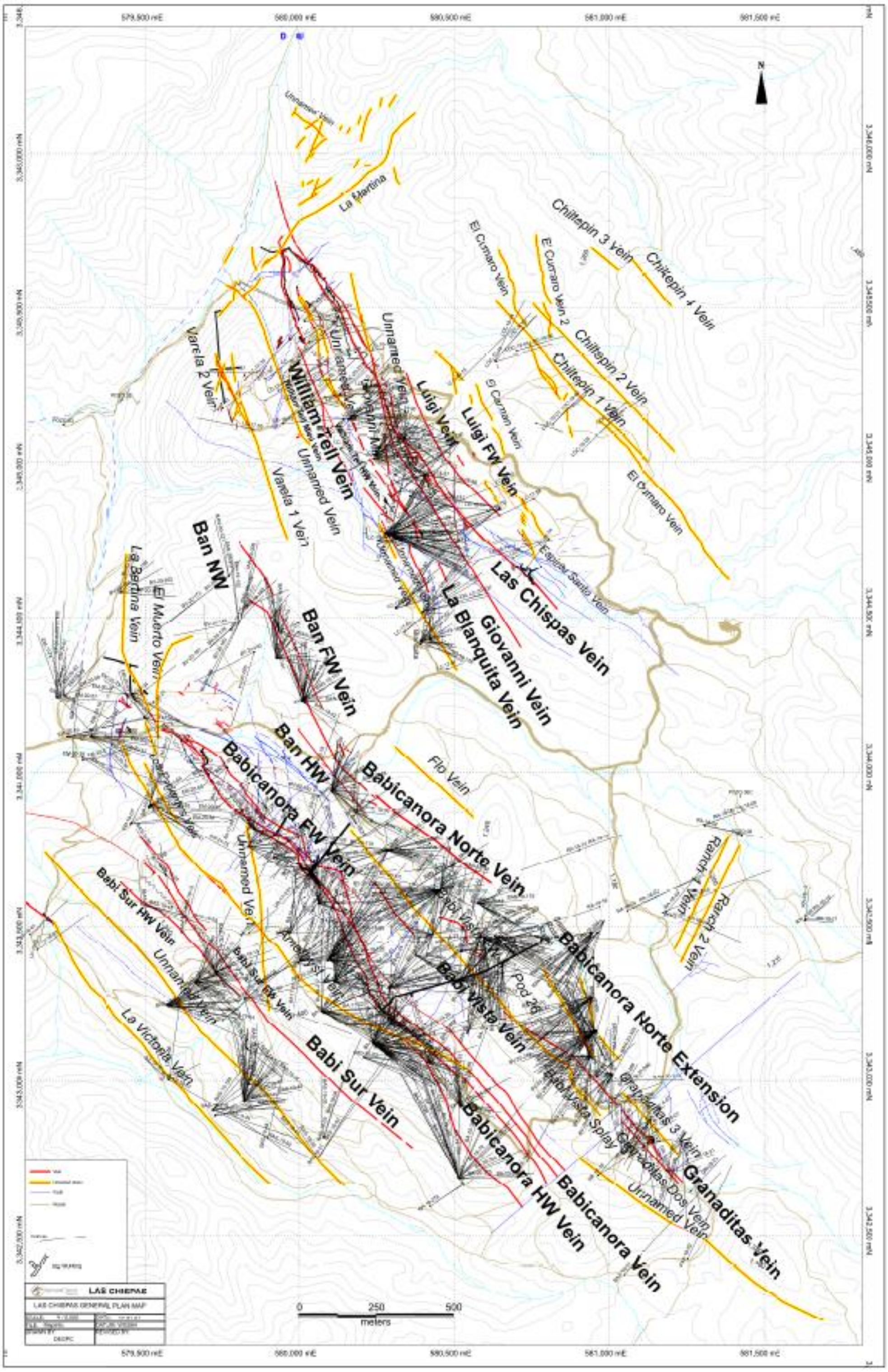
Photo 7-3: Laminated (Banded) Vein Style Mineralization Along Las Chispas Vein, Tip of Rock Hammer Shown on Upper Left (Near SilverCrest Sample 227908, 1.04 gpt Au and 197 gpt Ag over 1.33 m)

Note: Photo from PEA, 2019.

Photo 7-4: Breccia Style Mineralization Along Las Chispas Vein (Base of Las Chispas Gallery Near Silver Crest Sample 617179, 2.34 gpt Au and 344 gpt Ag)

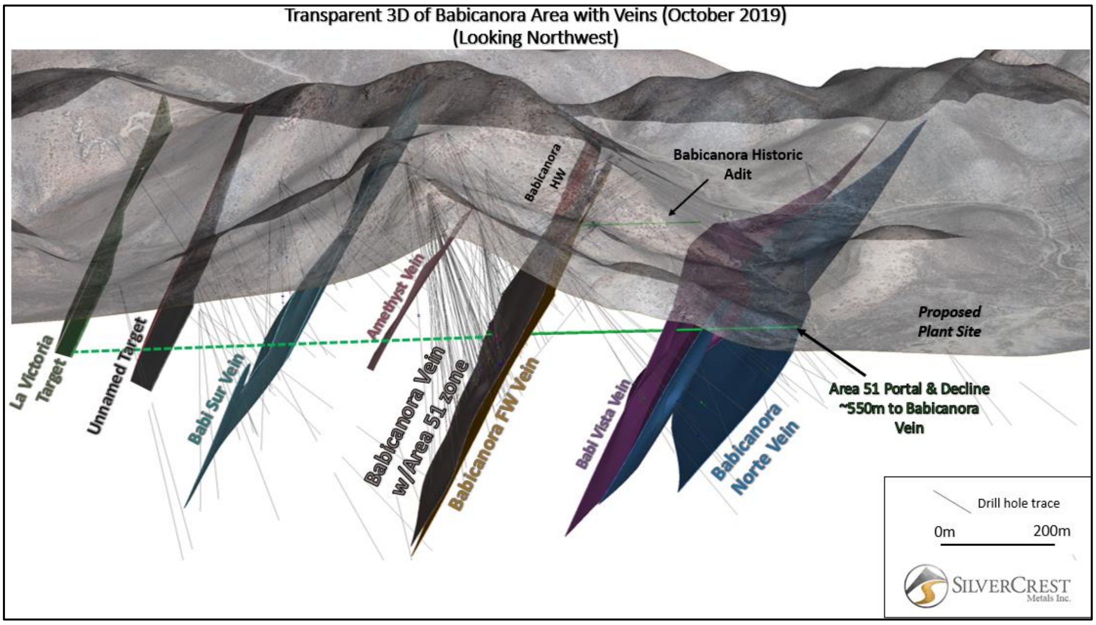
7.2.5 Structural Geology

Mapping and interpretation of the structural controls on mineralization and post-mineral displacement is presented in Figure 7-5, Figure 7-6, and Figure 7-7. Multiple stages of normal faulting has affected the basin. The main structures are steep, west-dipping (80°) and sub-parallel to the Granaditas normal fault, which is located along the western margin of the Project, striking approximately 030°. The area is further cross-cut by younger northwest-trending normal faults that dip to the southwest, creating both regional and local graben structures (Carlos et al., 2010).



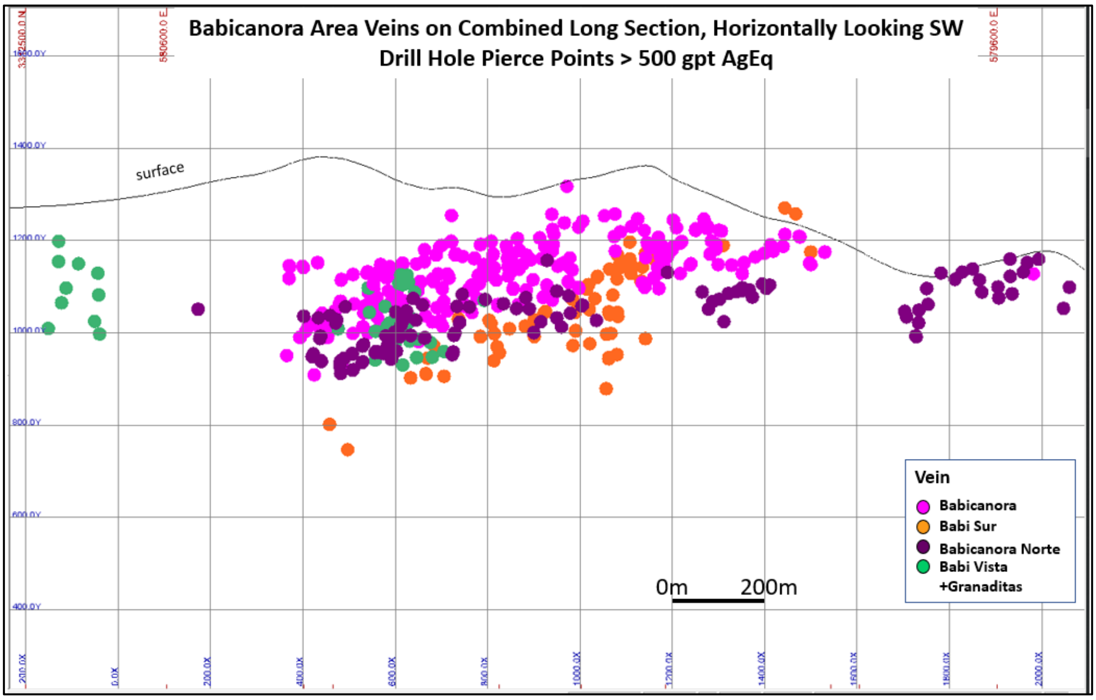
Note: Figure by SilverCrest, 2021.

Figure 7-5: Overview of the Las Chispas and Babicanora Area Veins



Note: Figure by SilverCrest, 2019.

Figure 7-6: Transparent 3D of Babicanora Area with Veins



Note: Figure by SilverCrest, 2019.

Figure 7-7: High-Grade (>500 gpt AgEq) Drill Hole Pierce Points for Babicanora Veins

Three major structural corridors have been identified to date in the Project area and are referred to as the Las Chispas, Babicanora and El Carmen grabens. All three corridors may be related to horst-and-graben style displacement within broad antiformal-synformal folded stratigraphy, such that the graben structures are bounded by:

- Steeply-dipping (80° to 90°) oblique strike-slip sinistral faults trending northeast and south–southwest; and

- Oblique strike-slip dextral faults trending southeast and dipping (60° to 80°) to the northeast.

Locally, the graben structures are complicated by probable caldera collapse. Circular structures identified in the lineament analysis in conjunction with locally derived, immature volcanic fill containing sharp primary quartz clasts, indicate local volcanism (Colombo, 2017b). Within a collapsed caldera, telescoping, juxtaposing or overprinting deep mineralization, is common. Paleo-surfaces may be down-dropped by 1.0 km, leading to vertical compression of contained mineralized deposits (Sillitoe, 1994).

Current understanding suggests that mineralization structures are oriented along a northwest–southeast trend. Three structural controls, excluding bedding contacts, are considered to influence alteration and mineralization:

1. 150° to 170° and are inclined at approximately 65° to 75° to the southwest;
2. 340° to 360° and are inclined 75° west to 75° east; and,
3. 210° to 230° and are inclined 70° to 85° to the northwest.

High-grade vein mineralization in the district has overall plunge along strike at approximately 20° to 30° to the southwest. Figure 7-7 shows this plunge of high-grade (>500 gpt AgEq) for multiple veins and drill pierce points on a combined long section for the Babicanora Area.

Locally, the mineralization structures terminate against the northeast-trending regional Las Chispas–Santa Elena Fault which is a normal fault, on the west side of which rocks have down-dropped. Absolute direction and magnitude of movement along the fault in this area is not known. Recent drilling results have indicated structural continuity and veining that is throughgoing this regional fault. At the nearby Santa Elena mine, drilling indicates that movement along this normal fault is post-mineralization and is west side down by approximately 400 m. This normal fault is also considered a major controlling feature for important regional aquifers.

7.2.6 Deposits and Mineral Occurrences

The Las Chispas district is divided into the Las Chispas Area and the Babicanora Area, and currently of 45 epithermal separate veins have been identified (Figure 7-5). Mineral Resources are estimated for 21 veins, and Mineral Reserves for 15 veins (Table 7-3).

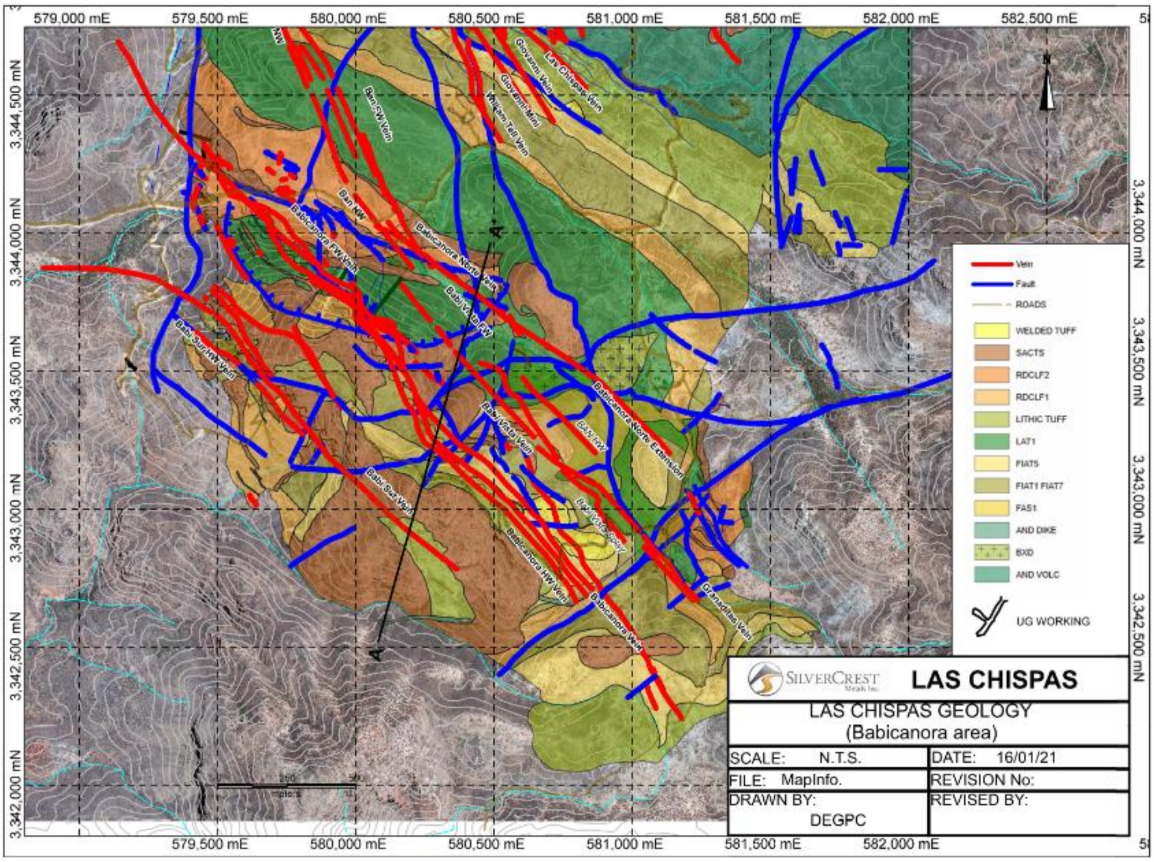
Table 7-3: Las Chispas Epithermal Veins in Mineral Resource and Reserve

Vein Name	Mineral Resource	Mineral Reserve
Babicanora	X	X
Babicanora FW	X	X
Babicanora HW	X	X
Babicanora Norte	X	X
Babicanora Norte HW	X	X
Babicanora Sur	X	X
BAS HW	X	X

Vein Name	Mineral Resource	Mineral Reserve
BAS FW	X	
Babi Vista	X	X
Babi Vista Splay	X	
Babi Vista FW	X	X
Granaditas1	X	
Granaditas2	X	
Las Chispas	X	X
Giovanni	X	X
Gio Mini	X	X
William Tell	X	X
William Tell HW	X	
William Tell Mini	X	
Luigi	X	X
Luigi FW	X	X
Total Veins	**21**	**15**

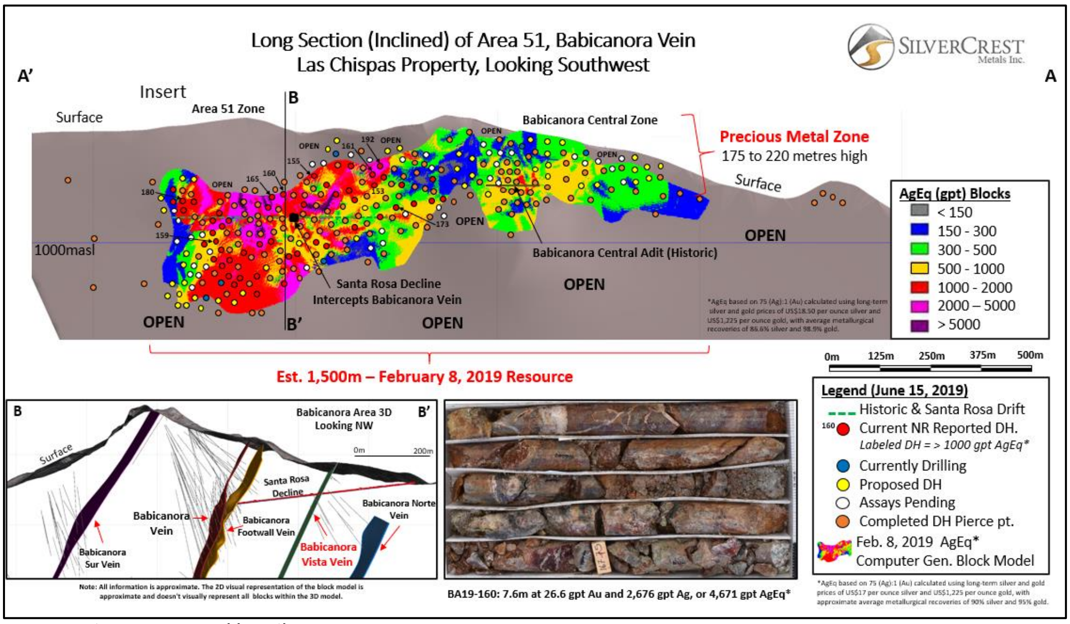
7.2.6.1 Babicanora Main Vein, HW & FW Veins

The Babicanora Main Vein has a continuous mineralization strike length of approximately 3.2 km with an average estimated width of 2 m and includes the Area 51 Zone, Babicanora Central Zone and the El Muerto Zone. The Precious Metal Zone (PMZ) has been drilled continuously to at least 300 m down-dip within the Babicanora Main Vein, and mineralization is exposed on surface to a depth of 500 m. Geological mapping in the Babicanora Area is shown in Figure 7-8. A December 2020 longitudinal section and typical cross-section of the Babicanora Area veins are shown in Figure 7-9 and Figure 7-10, respectively.



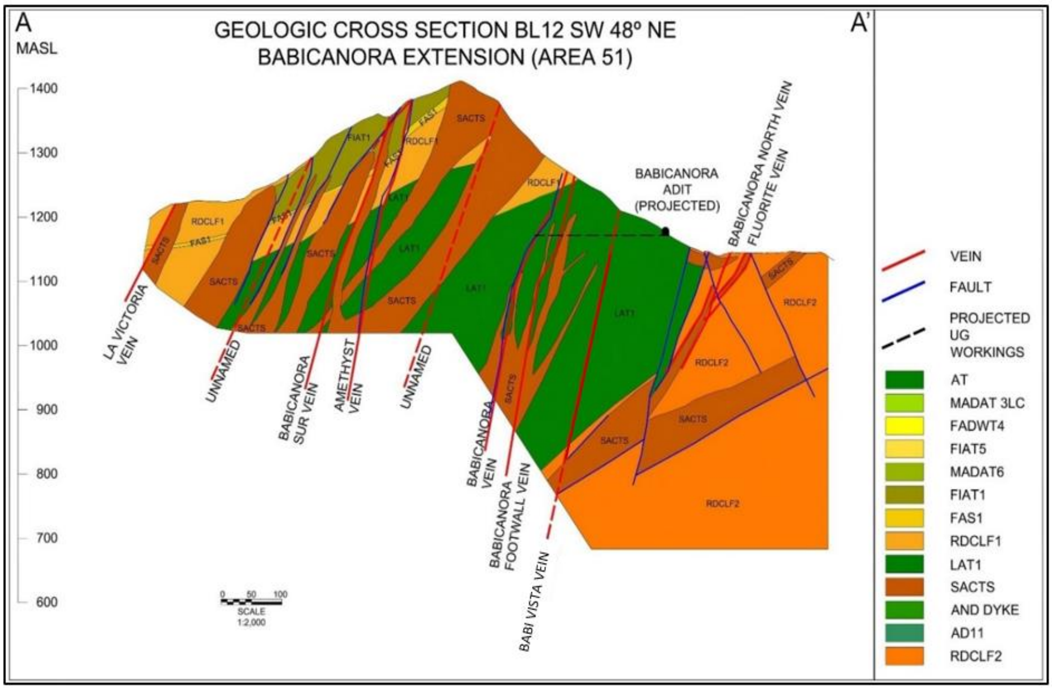
Note: Figure by SilverCrest, 2021. For legend explanation, Refer to Figure 7-4

Figure 7-8: Plan View of Geological Mapping at the Babicanora Area



Note: Figure prepared by SilverCrest, 2019.

Figure 7-9: Babicanora Main Vein Long Section



Note: Figure prepared by SilverCrest, 2019. For legend explanation, refer to Figure 7-4.

Figure 7-10: Vertical Cross Section through Babicanora Area, Line 1 + 300N (looking northwest)

Mineralization is hosted in structurally controlled veins with associated stockwork and breccias. A majority of the high-grade mineralization is located within medium- to coarse-grained lithic tuff (LAT1).

Historical underground workings along the Babicanora Main Vein are located to the northwest portion of the vein in the Babicanora Central Zone. The historical adit is accessible and has been rehabilitated by SilverCrest as a 4 m by 4 m adit that continues as a 230 m horizonal access. Future underground development will be accessed from the historical Babicanora Central adit in addition to a new decline from the newly constructed Santa Rosa portal. Details of the underground development are presented in Section 16.

Mineralization is characterized as quartz veins, stockwork and breccias. The mineralization structural zone strikes between 140° to 150° and dips approximately 60° to 70° to the southwest. Several 200° to 220° cross-striking faults and dense fracture sets intersect the Babicanora Main Vein. These intersections appear to influence mineralization by the development of high-grade, steeply southeast-plunging shoots. From observations underground at the nearby Las Chispas Vein, these cross-cutting faults or dense fracture sets can be mineralization along an approximate 220° strike for up to 20 m.

The historical Babicanora Central Mine had hanging wall stoping from the main adit level (1,152 masl) to the surface, approximately 130 m above (Dahlgren, 1883). The depth of historical underground workings is approximately 25 m below the Babicanora Central adit level. SilverCrest is currently mining and stockpiling historical mineralized material from historical underground mine shoots in preparation for geotechnical controls to mine the main vein. The Babicanora Main Vein is in the footwall of the historical stoping.

Major mineralization lithological units are defined as:

- LAT1: lapilli or lithic andesitic tuff and the most significant host for vein-related gold–silver mineralization;

- RDCLF 1 and 2: rhyodacitic flows, which restrict mineralization with narrow (0.2 m to 1.0 m) high-grade mineralization veining; RDCLF 1 stratigraphically overlies LAT1 and RDCLF 2 stratigraphically underlies LAT1; and,

- SACTS: silicic tuffs and ignimbrite which can be in sill and dyke form cutting both LAT1 and RDCLF1 and RDCLF2. Andesitic and rhyolitic dykes are associated with the mineralization.

General lithologies are andesite to dacite with rhyolitic interbeds. These units are crosscut by andesitic dykes to the southeast and rhyodacitic dykes to the northwest of the Babicanora Main Vein. Strong to intense silicification caps the ridges in the area with a 300 m by 400 m horizontal zone interpreted as possibly sinter (Photo 7-5, A) that covers the slopes in the northwestern portion of the Project area.

Mineralization within the Babicanora veins is characterized as a low (northwest portion) to intermediate (southeast portion) sulphidation system. SilverCrest has identified numerous sulphidation features including sinter capping on the ridges, quartz after calcite bladed textures (Photo 7-5, B), and massive chalcedonic-textured silica (Photo 7-5, C). These high-level features and textures point to the preservation of the mineralization system at depth.

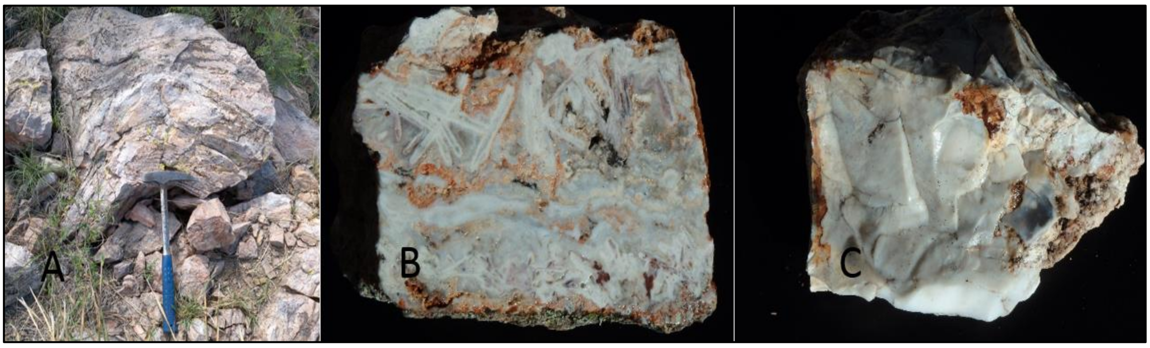
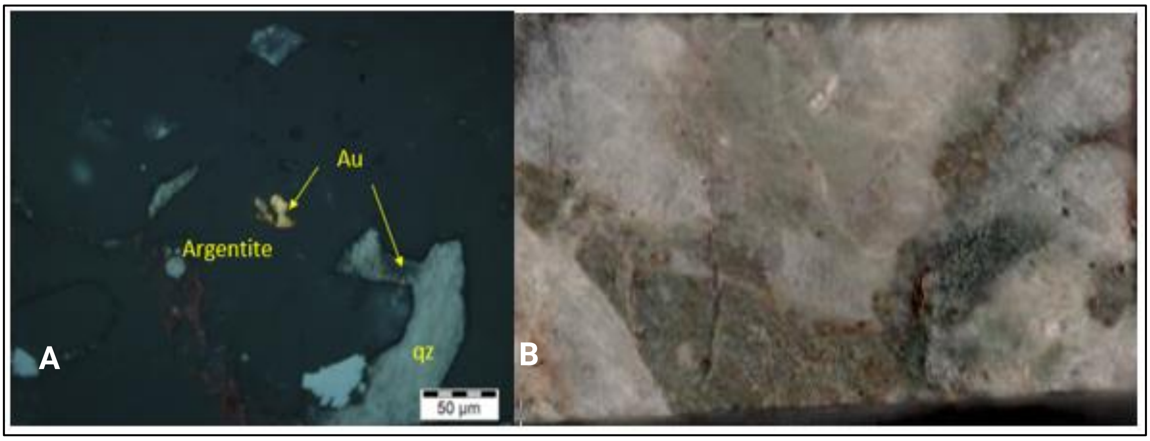


Photo 7-5: A. Sinter Lamina, B. Quartz Replacement of Bladed Calcite with Minor Amethyst, C. Massive Chalcedonic Quartz

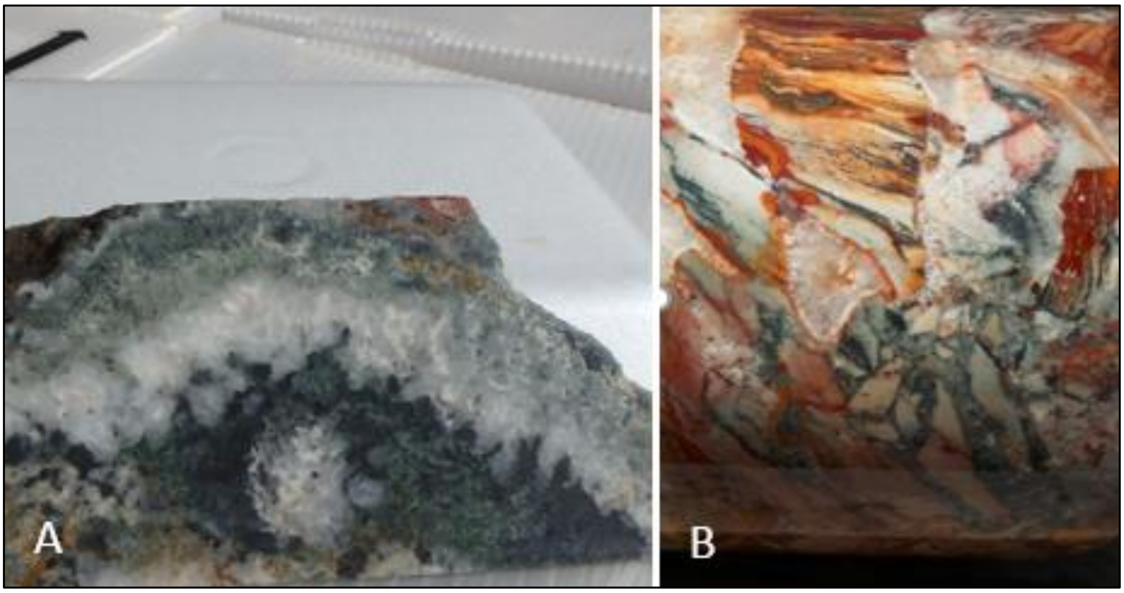
The mineralization of the Babicanora veins has a strong magmatic component. The potassic alteration observed in thin section is crystalline, orthoclase and of magmatic origin. Adularia is also present, but in limited zones. Argentite is the principal silver mineral, electrum and native silver are present, and gold occurs as native flakes and in association with pyrite and chalcopyrite (Photo 7-6). Gold and silver values have a strong correlation to each other and likely precipitated together during the crystallization of quartz.



Notes: Figure by SilverCrest, 2017.
(A) Thin section. A very fine particle of gold is dispersed within the quartz, and it is spatially associated with the argentite. Plane-polarized reflected light.
(B) Core, taupe, brecciated fine grained quartz brecciated and recemented with coarse white quartz, fine grained disseminated pyrite throughout.

Photo 7-6: Babicanora Thin Section with Gold and Argentite

Contents of base metals are low in the Babicanora area. Zinc and lead are found principally in green sphalerite and galena. Gold and silver mineralization can be characterized with three end-member types; 1) breccia hosted; 2) vein hosted; and 3) vuggy quartz hosted (Photo 7-7 and Photo 7-8).



Note: Photo by SilverCrest, 2018.

Photo 7-7: Babicanora Vein Textures

Note: Photo by SilverCrest, 2017.

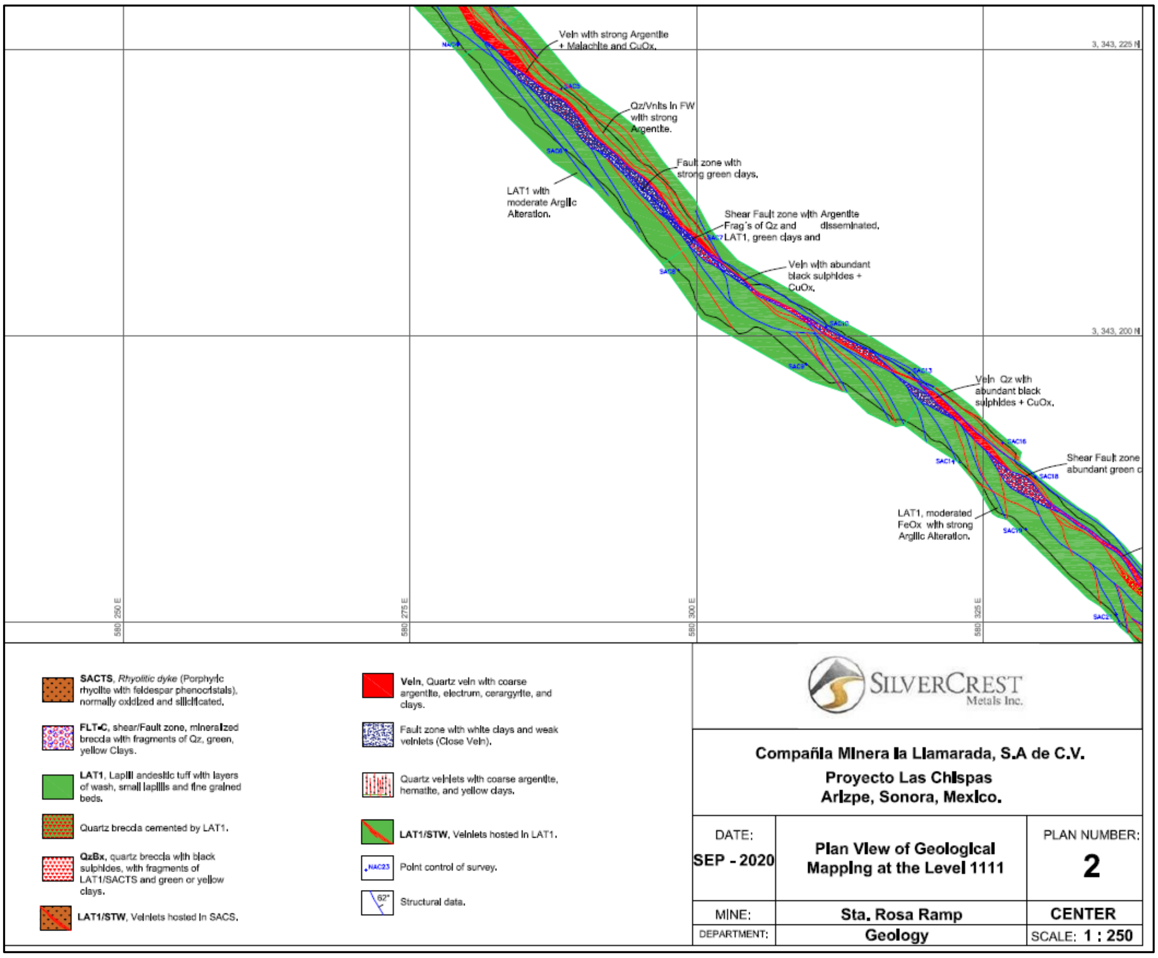
Photo 7-8: Drill Hole BA17-51 (Discovery Hole for Area 51 Zone); from 265.9 to 269.2 m, 3.3 m (3.1 m True Width) Grading 40.45 gpt Au and 5,375 gpt Ag, with Hematite Breccias, Coarse Banded Argentite, Native Silver, Electrum, and Native Gold

In June 2019, the Santa Rosa Decline intercepted the Babicanora Main Vein with banded textures and high-grade gold and silver mineralization (Photo 7-9) within 5 m of the predicted location, based on the computer-generated resource model using surface drilling.

Note: Channel sample grading 336 gpt Au and 26,435 gpt Ag. Photo by SilverCrest, 2019.

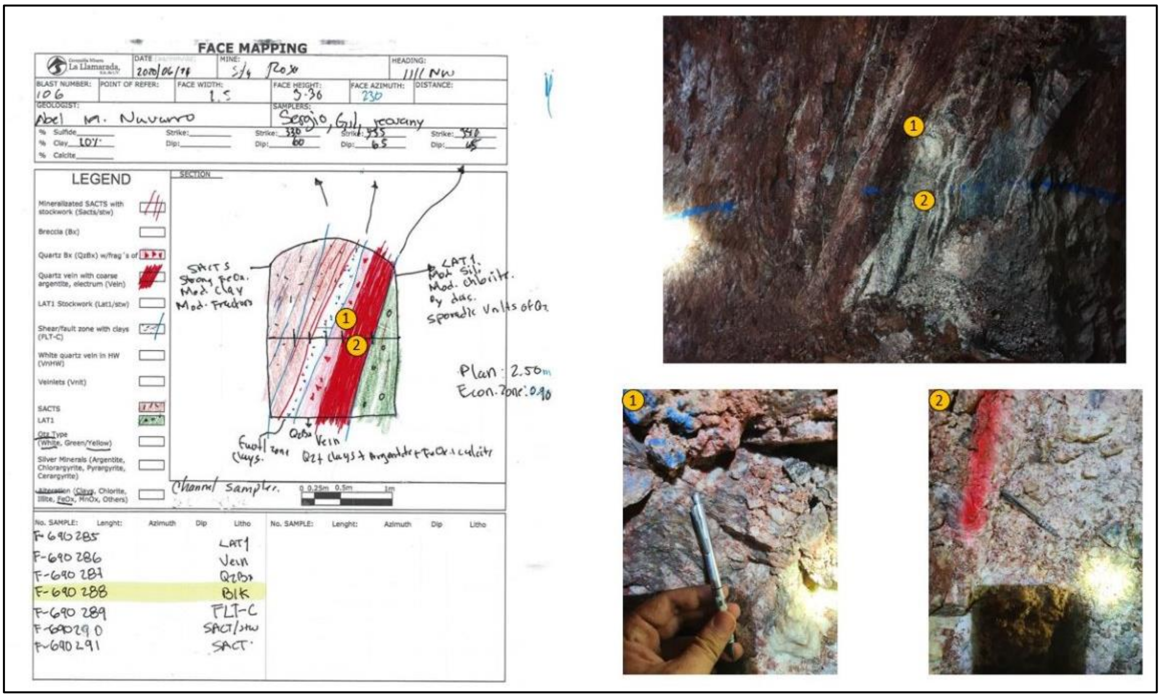
Photo 7-9: Babicanora Vein Intercepted by Santa Rosa Decline in June 2019

Following the Babicanora Vein intersection, underground in-vein development proceeded by drifting to the northwest and southeast along the strike of the vein. While drifting, a significant part of the vein was observed to be fractured and faulted and to contain clay material. This material was not logged in core drilling (low recovery intercepts) and sampled for geochemical analysis or metallurgical testing. This fault zone is a prominent, continuous feature within the vein. It ranges in width from a few centimetres up to 2 m and appears to span the entire length of the vein. The fault zone meanders within the vein from the hanging wall to footwall and back (Figure 7-11 and Figure 7-12). Post-mineralization relative movement is apparent, but the amount of displacement is unknown.



Note: Figure prepared by SilverCrest, 2020. Fault zone in blue.

Figure 7-11: Underground Plan Map of Babicanora Main Vein, Area 51 Zone, Level 1111 (masl)



Note: Figure by SilverCrest, 2020. Fault zone as white speckle texture.

Figure 7-12: Babicanora Main Vein, Area 51 Zone, Face Map of Vein with Fault Zone (looking northwest)

The fault zone is oxidized, and contains kaolin clays, limonite, hematite, minor manganese oxides, fine-grained native gold, and the silver halides chlorargyrite and idiorite.

The Babicanora HW and FW veins are sub-parallel to the Babicanora Main Vein. These veins are 5 m to 50 m from the Babicanora Vein and appear to intersect the Babicanora Main Vein near Area 51, potentially causing near-vertical high-grade shoots.

7.2.6.2 Babicanora Norte (Main), HW & FW veins

Babicanora Norte Vein has a semi-continuous mineralization strike length of approximately 2.2 km with an average estimated width of 1.0 m. Mineralization is exposed on surface to a depth of 300 m.

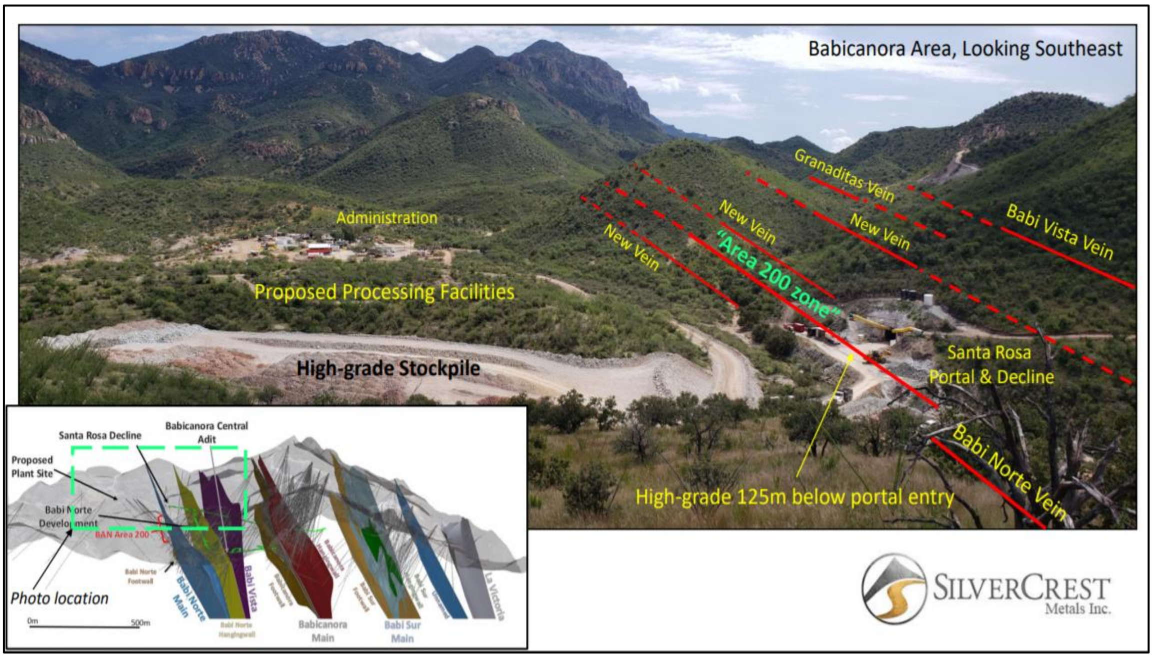
The mineralization of the Babicanora Norte veins resembles that at the adjacent Babicanora Main Vein, but without a significant fault zone. A majority of the high-grade mineralization is located within the RDCLF1 (rhyodacitic flow), near intersections of cross-cutting 220° striking faults and dense fracture sets. The RDCLF shows more brittle fracture compared to LAT1, which has constrained the mineralization vein as a consistently narrower and banded vein compared to Babicanora Main. Argentite is the principal silver mineral, pyrargyrite is present, and gold occurs in electrum and as native gold flakes in association with pyrite and chalcopyrite (Photo 7-10).

Note: Photo by SilverCrest, 2018.

Photo 7-10: Drill Hole BAN18-10, From 93.0 to 95.5 m Grading 61.36 gpt Au and 2,834 gpt Ag with Visible Argentite, Pyrargyrite, Electrum, Native Silver, and Native Gold

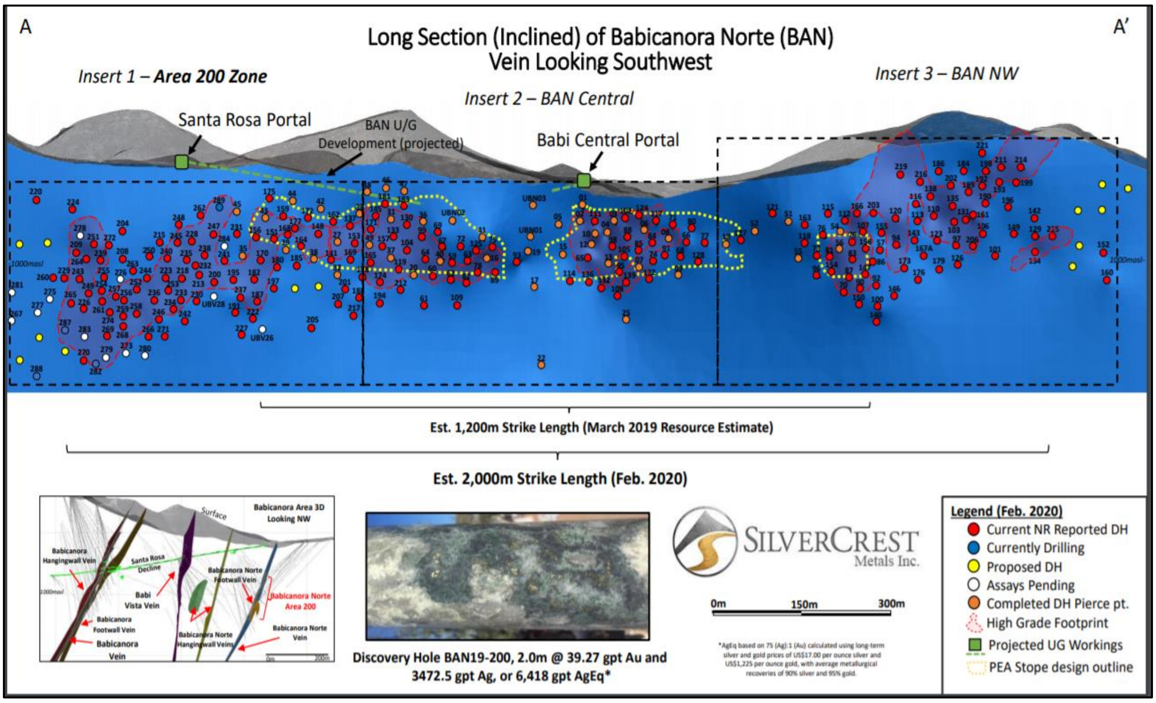
Base metals in Babicanora Norte veins are similar in nature to those described from the Babicanora veins, but the base metal contents are higher (up to 3%). Zinc and lead are principally found in green sphalerite and galena. A chalky white mineral is immediately adjacent to high-grade silver and may be a silver halide similar to that in the Babicanora Main Vein. Geochemical analyses lack detectable arsenic and mercury. Gold and silver mineralization can be characterized as occurring as three end-member types; 1) breccia hosted, 2) vein hosted, and 3) vuggy quartz hosted.

In February 2020, the Babicanora Norte Vein and Area 200 zone were discovered while completing step-out drilling down-plunge of the high-grade mineralization Photo 7-11; Figure 7-13 and Figure 7-14). Hole BAN-19-200 grades 39.27 gpt Au and 3,473 gpt Ag, or 6,418 gpt AgEq (AgEq based on 75:1 Ag/Au) over 2.5 m downhole length, which includes 79.80 gpt Au and 7,380 gpt Ag, or 13,365 gpt Ag Eq over 0.7 m.



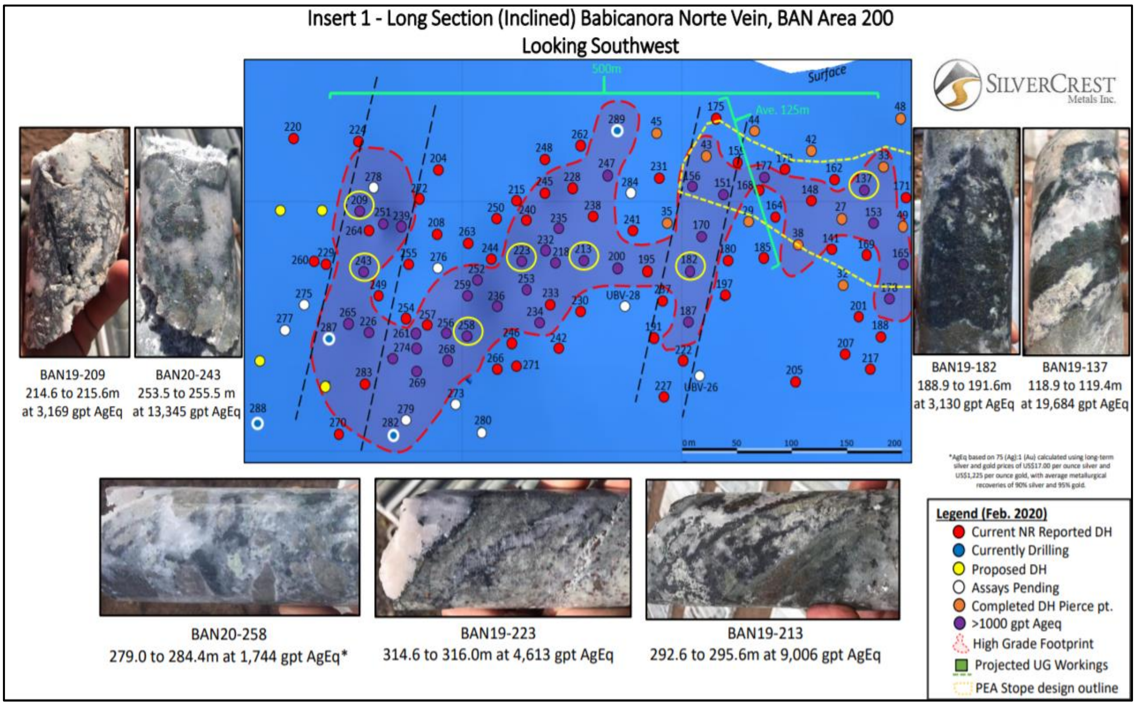
Note: Photo by SilverCrest, 2020.

Photo 7-11: Location of Babicanora Norte Vein, Area 200 Zone



Note: Figure by SilverCrest, 2020.

Figure 7-13: Long Section of the Babicanora Norte Vein



Note: Figure by SilverCrest, 2020.

Figure 7-14: Long Section of the Babicanora Norte Vein Area 200

The Babicanora Norte HW and FW veins are sub-parallel and located 20 m to 30 m from the main vein.

7.2.6.3 Babicanora Sur (Main), HW & FW Veins

The Babicanora Sur Vein has a semi-continuous mineralization strike length of approximately 1.0 km with an average estimated width of 1.0 m. The Precious Metal Zone (PMZ) typical for epithermal veins has an average estimated height of 150 m. Mineralization is exposed on surface to a depth of 300 m.

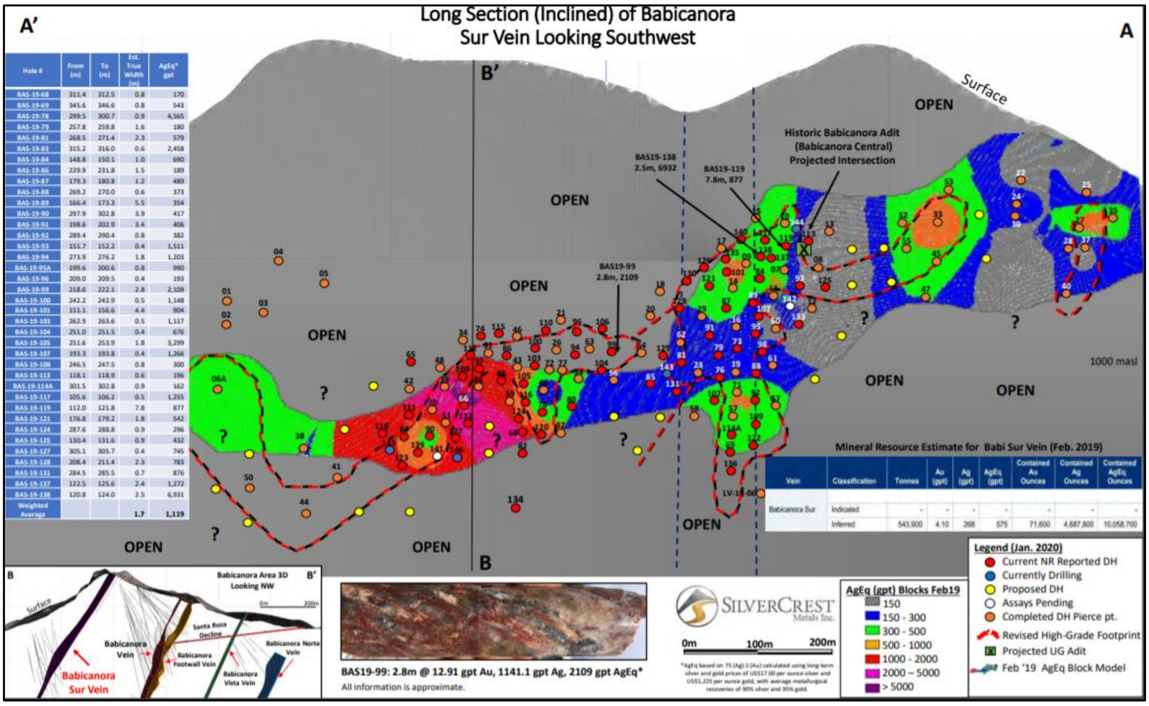
The Babicanora Sur Main Vein is located approximately 300 m southwest of the Babicanora Main Vein and is parallel to that vein. This vein is similar to Babicanora Main with a fault zone containing clays and associated minerals. The fault zone strikes between 140° to 150° and dips approximately 55° to 65° to the southwest. It is cross-cut by several 220° trending faults and dense fracture sets. Mineralization at Babicanora Sur is hosted in lapilli or lithic tuff and breccia with moderate to strong overprinting alteration (Photo 7-12).

As with the Babicanora Main Vein, core loss occurred when drilling the Babicanora Sur Vein and reconciliation while mining may show an impact on grade and width.

Note: Photo by SilverCrest, 2018.

Photo 7-12: Drill Hole BAS18-31, from 230.6 to 232.8 m at 2.2 m (True Width) Grading 18.78 gpt Au and 2,147 gpt Ag with Argentite in black

In January 2020, the Babicanora Sur Vein was expanded while completing step-out drilling down-plunge of the high-grade mineralization (Figure 7-15).



Note: Figure prepared by SilverCrest, 2020

Figure 7-15: Long Section of the Babicanora Sur Vein

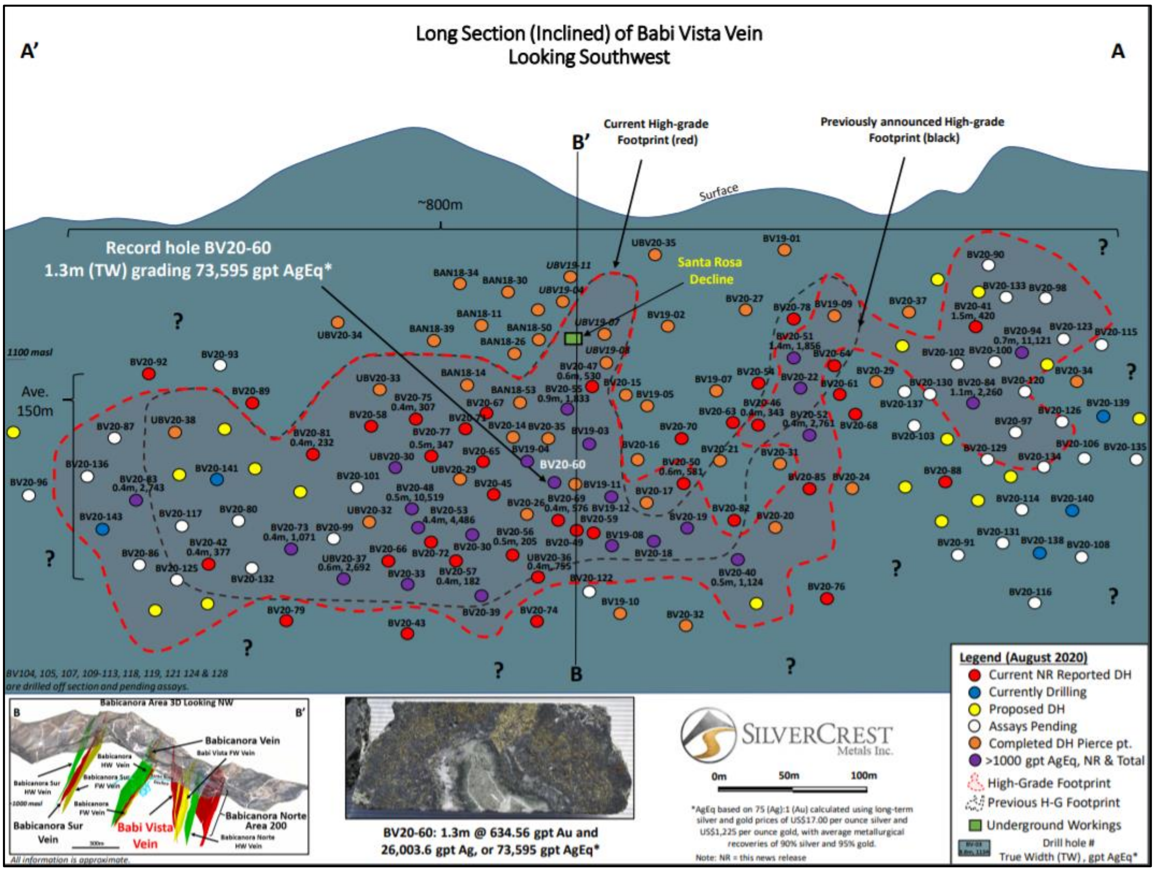
The Babicanora Sur HW and FW veins are sub-parallel and 10 m to 40 m from the vein and are generally narrower than Babicanora Norte.

7.2.6.4 Babi Vista (Main) Vein, HW & FW Veins

The Babi Vista Vein has a semi-continuous mineralization strike length of approximately 1.0 km with an average estimated width of 1.0 m. The structure is exposed on surface and extends to a depth of 300 m.

The Babi Vista Vein is located approximately 100 m north of the Babicanora Main Vein and is parallel to it. The vein strikes from 140° to 150° dips approximately 55° to 80° to the southwest. It is cross-cut by several 220° trending faults and dense fracture sets. Mineralization at Babi Vista is hosted in lapilli or lithic tuff and rhyodacitic weld tuffs with moderate to strong overprinting alteration.

In August 2020, the Babi Vista Vein was expanded while completing step-out drilling down-plunge of the high-grade mineralization (Figure 7-16).



Note: Figure by SilverCrest, 2020.

Figure 7-16: Long Section of the Babi Vista Vein

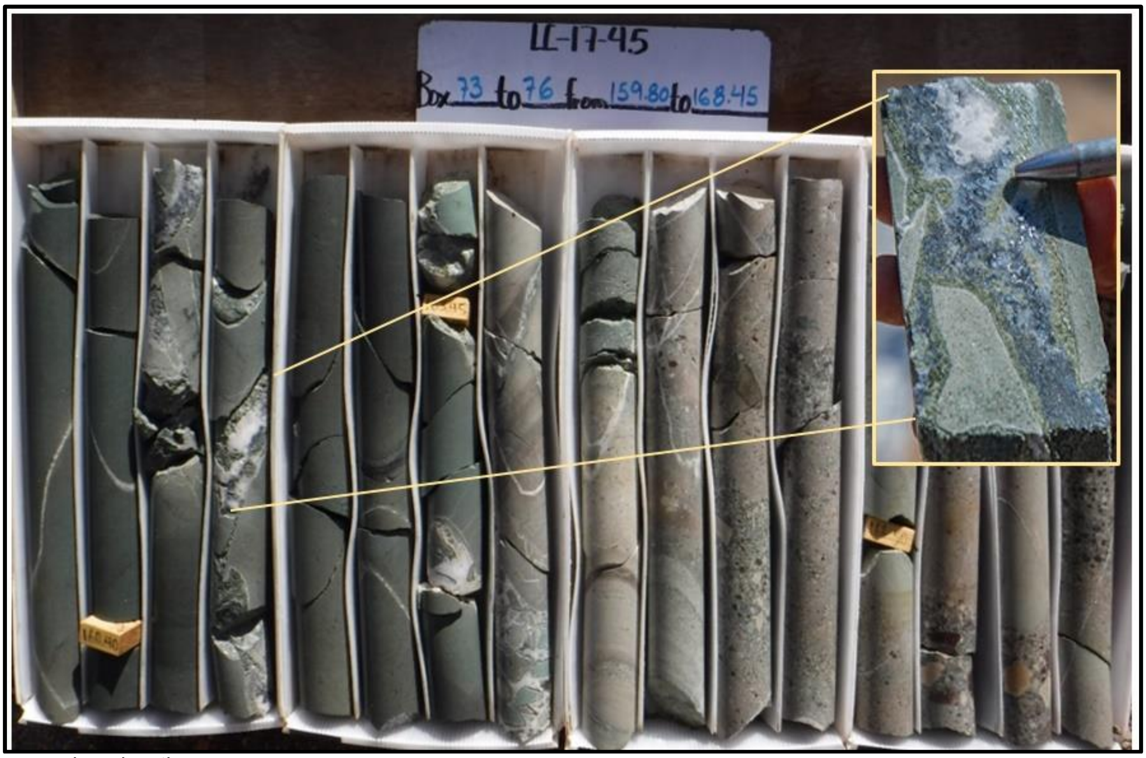
This vein is similar to the Babicanora Norte veins but lacks a significant associated fault zone. The Babi Vista HW and FW veins are sub-parallel and 10 m to 40 m from the Babi Vista Main Vein. An area referred to as Pod 26 is located in the Babi Vista FW vein and is centered around hole BAN-18-26 grading 103.24 gpt Au and 5,583 gpt Ag over 1.60 m.

7.2.6.5 Las Chispas Vein

The Las Chispas Vein has a continuous mineralization strike length of approximately 1.5 km with an average estimated width of 3 m. Mineralization is exposed on surface to a depth of 400 m.

The Las Chispas Vein is located in the Las Chispas Area, in the northern portion of the Project. It is the most extensively historically mined vein in the district.

SilverCrest's exploration work focused on defining the lithology, structure, alteration, mineralization, and channel sampling in unmined pillars and surrounding intact vein. Vein mineralization is described as an undulating and dilating quartz stockwork and breccia zone (as defined in underground mapping and in drill core) of 0.10 m to 7.9 m in true width, which typically encompasses narrow veins of quartz, visible sulphides, and calcite (Photo 7-13).



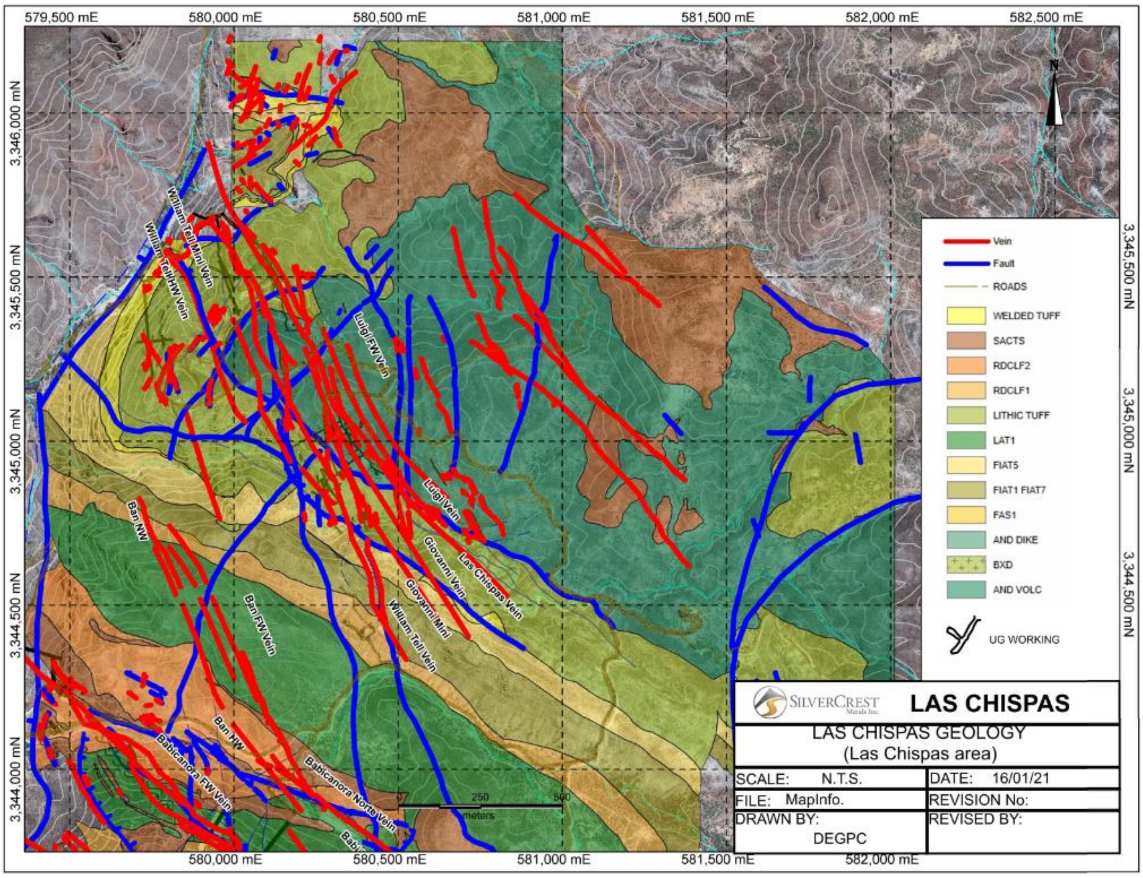
Note: Photo by SilverCrest, 2017.

Photo 7-13: Drill Hole LC17-45; from 159.6 to 161.9 m at 2.3 m (1.9 m True Width) Grading 50.56 gpt Au and 5,019 gpt Ag with Coarse Argentite and Electrum

The Las Chispas Vein strikes 150° and dips at approximately 75° to the southwest. Cross cutting the Las Chispas Vein are normal secondary faults trending 220° and dipping 65°. These secondary faults seem to have had an important role in generating zones of dilatation for emplacement of the high-grade shoots and breccia zones. Flat to steeply inclined, bedding-parallel faults are also present and offset the late stage andesitic dykes by 10 m to 20 m and are associated with drag folds (Schlische 1995). A majority of the high-grade mineralization is within the lithic tuff units. Geological mapping in the Las Chispas Area is shown in Figure 7-17 and a typical cross-section is shown in Figure 7-18.

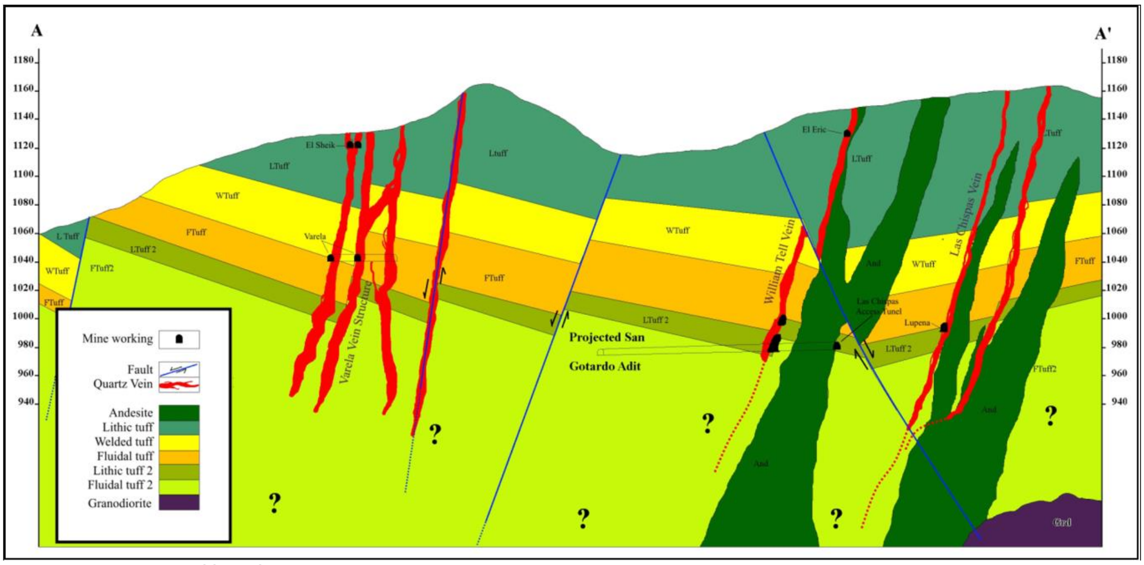
Alteration is similar to the other veins in the Project. Silicification is extensive in mineralization zones. Multiple generations of quartz and chalcedony are commonly accompanied by calcite with minor adularia. Pervasive silicification in vein envelopes is flanked by sericite and clay alteration of the host rock. Intermediate argillic alteration (likely kaolinite–illite–smectite) forms adjacent to some veins. Advanced argillic alteration (kaolinite–alunite) is suspected within the Las Chispas Vein and confirmation studies of the alteration mineralogy have not been completed. Propylitic alteration dominates at depth and peripherally to the mineralization, with abundant fine-grained chlorite and pyrite proximal to the mineralization. Iron-oxyhydroxides, manganese after pyrite, and other fine-grained sulphides are closely associated with the mineralization. Reactivation of the central fault hosting the mineralization provided a conduit from surface for deep weathering of the sulphides and possible supergene enrichment of the silver mineralization. The andesitic dykes are weakly to moderately clay-altered with minor epidote along their narrow chill margins.

Mapping by SilverCrest confirms that the location and extent of mining indicated on historical sections is representative and accurate. From 2017 to 2019, underground rehabilitation, mapping and extensive sampling (over 8,000 samples) were completed from the 50 level to the 900 level (850 masl), covering most of the historical workings. Mapping and sampling on all levels is near completion.



Note: Figure by SilverCrest, 2021. Refer to Figure 7-4 for legend explanation.

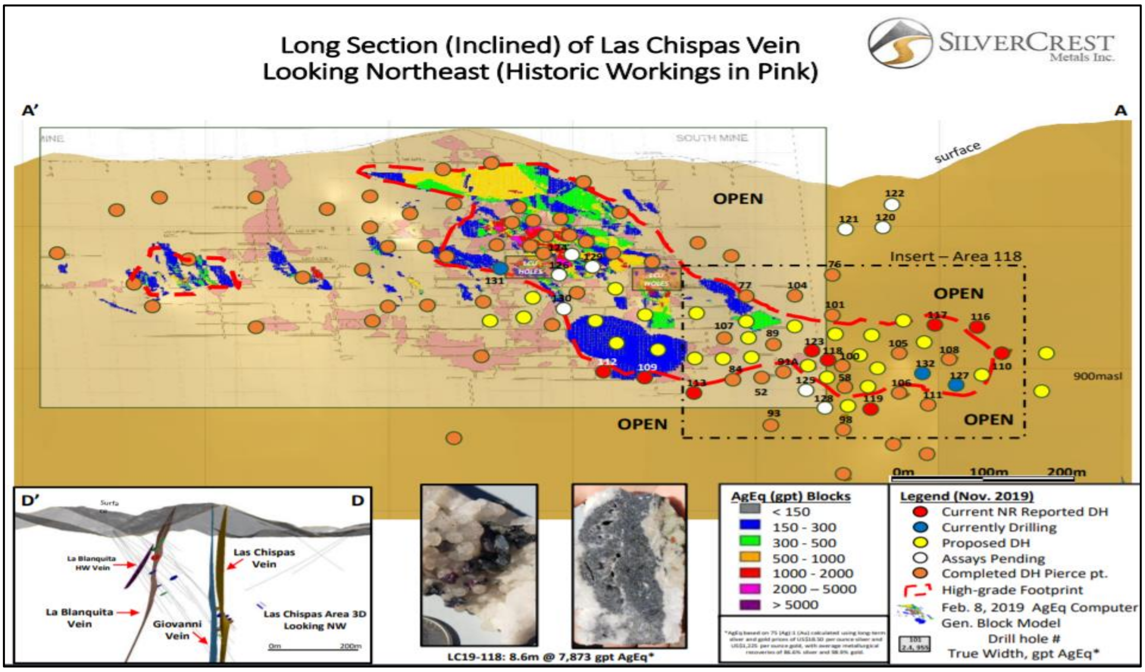
Figure 7-17: Plan View of Geological Mapping at the Las Chispas Area



Note: Figure prepared by SilverCrest, 2020.

Figure 7-18: Typical Geological Cross Section through the Las Chispas Property (looking northwest)

In November 2019, the Las Chispas Vein, Area 118 zone was discovered while completing step-out drilling down-plunge of the high-grade mineralization (Figure 7-19), based on the intersection in drill hole LC-19-118 of 44.30 gpt Au and 4,551 gpt Ag over an estimated 8.6 m true width. This area occurs just below the current groundwater elevation and experienced little to no historical mining.



Note: Figure by SilverCrest, 2019.

Figure 7-19: Long Section of Las Chispas Vein with Area 118 Zone

7.2.6.6 William Tell Vein

The William Tell Vein is located 115 m to the west of, and oriented roughly sub-parallel to the Las Chispas Vein.

The mineralization is characterized as a quartz stockwork zone in the footwall of a continuous northeast–southwest fault striking 140° and dipping 65°. Underground mapping by SilverCrest indicates that mining from the main San Gotardo adit terminated against a cross-cutting fault (220°/70°) that SilverCrest interprets, based on drilling results, to have approximately 10 m of left lateral displacement.

The William Tell Vein is hosted in the same sequence of course- to fine-grained volcaniclastic, flows, and pyroclastic units that are described for the Las Chispas Vein. Alteration consists of white clays, sericite, and fine-grained chlorite with strong silicification. Within the mineralization structure and central vein, fine-grained pyrite, limonite, and iron oxides are present.

Historical mining of the vein is contemporaneous with mining within the Las Chispas Vein, although there is limited historical documentation available. The northern portion of the historical workings can be accessed from the same adit that connects with the San Gotardo level of the Las Chispas Vein. The extents of mapped workings total approximately 3 km horizontally over three levels and approximately 60 m vertically (450 level to 650 level). A shaft or a small stope exists from the lower working level. The vertical extent of this shaft/stope cannot be confirmed but based on historical documentation and drilling in the area it is not considered to be significant.

Mining activity along this structure south of the projected fault cannot be confirmed; however, no open stopes were intersected by SilverCrest drilling where the structure was interpreted to be, and no surface workings are apparent.

In 2016, underground channel sampling by SilverCrest was completed with high-grade mineralization defined in pillars and intact exposures (Photo 7-14 and Photo 7-15).

Note: Photo by SilverCrest, 2016.

Photo 7-14: William Tell Underground Channel Sample No. 144840 Grading 13.4 gpt Au and 1,560 gpt Ag

Note: Photo by SilverCrest, 2016.

Photo 7-15: William Tell Vein, Drill Hole LC16-03, from 172 to 176 m, 4 m (1.5 m True Width) Grading 2.03 gpt Au and 683 gpt Ag

7.2.6.7 Granaditas Vein

The Granaditas Vein is located to the southeast of Babicanora Main Vein in the eastern portion of the Project area. The Spaniards discovered the Granaditas Mine in 1845 (Dahlgren 1883) and subsequently mined it. Little information is available on this historical mine. After a local rancher provided an 1882 district map, SilverCrest was able to locate several adits, shafts, and dumps in the area. However, the shafts have not been entered due to unstable conditions.

The showing is located within 75 m of the confluence of two major lineaments interpreted as faults. The first trends 220° and has a strike length of 3.5 km and is interpreted to be the eastern bounding structure to the Las Chispas graben. The second fault is mineralization, strikes 145°, and parallels the Babicanora trend, approximately on strike with Babicanora Norte Vein. The interpreted mineralization strike length is over 500 m. Several drill holes have intersected fractured zones and encountered mafic andesitic dykes at depth.

Alteration at the Granaditas vein is consistent with the intermediate sulphidation model, with strong silicification in patches and strong clay alteration with zones of pervasive sericite and chlorite.

During the Phase II exploration program, two diamond drill holes were completed on the Granaditas vein. The highest assay was from GR17-02, which returned values of 8.15 gpt Au and 387 gpt Ag, with highly anomalous Pb (600 ppm), Cu (10,250 ppm), and Zn (595 ppm) over 0.7 m (Photo 7-16). Furthermore, copper and base metals are elevated over 20 m to 40 m with grades of 0.5% Pb and 0.3% Zn.

During the Phase III exploration program, 19 diamond drill holes were completed on the Granaditas vein. The highest assay was from GR17-04, which returned values of 47.5 gpt Au and 5,620 gpt Ag, with highly anomalous Pb (2,610 ppm), Cu (1,010 ppm), and Zn (3,130 ppm) over 0.5 m (Photo 7-17).

These elevated base metals values suggest that base metals increase to the southeast and may indicate deeper depths of emplacement of the mineralization.

Photo 7-16: Drill Hole GR17-02; from 139.85 to 140.55 m, 0.7 m Grading 8.15 gpt Au and 387 gpt Ag, and 1.02% Cu with Coarse Argentite, Pyrite and Chalcopyrite

Photo 7-17: Drill Hole GR17-04; from 133.8 to 134.3 m, 0.5 m Grading 47.5 gpt Au and 5,620 gpt Ag, with Coarse Argentite, Sphalerite and Galena

7.2.6.8 Other Structures or Mineral Occurrences of Significance

Amethyst Vein

The Amethyst (Amatista) Vein is located 200 m southeast of, and parallel to, the Babicanora Main Vein. Historical information is limited, but there are many historical workings pits and trenches along the 1 km strike length of the surface lineament.

The Amethyst Vein is steeply dipping and strikes 140°. It is cross-cut by several 200° to 220° trending faults and dense fracture sets that intersect the vein and hosts high-grade mineralization near the intersections. The mineralization is hosted in a sequence of 10° to 15° striking, northeast dipping lithic tuffs (LAT1). Drill hole BA17-20 intercepted high-grade mineralization from 75.7 m to 78.2 m grading 3.05 gpt Au and 78 gpt Ag, or 306 gpt AgEq (Photo 7-18).

Note: Photo by SilverCrest, 2017.

Photo 7-18: Drill Hole BA17-20, from 75.7 to 78.2 m Grading 3.05 gpt Au and, 77.8 gpt Ag

La Victoria Vein

This area is defined by small workings near surface in the southwest portion of the Project. The workings consist of three short and vertically off-set tunnels, each approximately 30 m in length. The vein trends 140° and dips approximately 70° to the northeast. In 2016, SilverCrest rehabilitated the access underground, due to the highly oxidized and soft nature of the host rock, which is strongly clay altered breccia. SilverCrest's sampling suggests that this mineralization structure is gold-dominated.

Historical sampling from three levels of the La Victoria Mine by Ronald Mulchay in 1941 assayed as high as 6.5 opt Au (approximately 220 gpt Au), with minor silver and a gold to silver ratio of 1:1 for high-grade mineralization.

In June 2016, SilverCrest drilled three drill holes down-dip of the workings. Significant mineralization was not intersected by the drill holes, suggesting a possible offset in the mineral continuity at depth or epithermal zonation. Significant alteration was encountered in the drill holes, along with evidence of multiple stages of intrusive activity. The nature of the mineralization and alteration at La Victoria is currently not well understood. SilverCrest proposes additional work in the future.

Espiritu Santo Vein

The Espiritu Santo workings are developed to the southeast of the Las Chispas and William Tell Veins. Two historical adits and a shaft are accessible and have been mapped and sampled.

Two structural trends appear to have been mined in the workings. The first, on an upper level, strikes 150° and dips 60°. The second structural trend, on the lower level, strikes 290° and dips 48°. The latter mineralization is a stockwork within the footwall that parallels the volcanic bedding contact. At surface, the exposed andesitic volcanics are strongly silicified with moderate to strong clay alteration focused along the structural trends. Historical selective underground sampling shows grades at Espiritu Santo to be as high as 500 opt silver (Mulchay 1941). Historical dump samples returned seven samples greater than 111 gpt Au and 100 gpt to 892 gpt Ag (Mulchay 1941). Three drill holes were completed at the target with negligible results.

La Varela Veins

The La Varela workings are located approximately 300 m to the west of the William Tell Vein. Two veins strike 170° and are near vertical with an average width of 1 m. Higher grade precious metal mineralization is dominant in the southern part of the two veins. SilverCrest has rehabilitated the existing underground workings (an estimated 400 m) with mapping and sampling. Three drill holes have been completed in this area. Drill hole LC17-55 intersected 0.8 m grading 2.67 gpt Au and 272 gpt Ag.

La Bertina

The La Bertina Vein is located to the northwest of the Babicanora Area and is juxtaposed along the regional normal fault lineament. It is a northeast-directed quartz breccia structure with strike between 15° to 20°azimuth and dip of 80° to 85°. The vein has been mapped on surface for approximately 100 m with width ranging from 0.10 m – 0.15 m and has been sampled from surface chips with grades up to 0.42 gpt Au and 10 gpt Ag.

El Cumaro

The El Cumaro Vein is located to the northeast of the Las Chispas Area and is a northwest-directed quartz vein structure with strike between 330° to 340° azimuth and with dip of 80°. The vein has been mapped on surface for approximately 600 m with width ranging between 0.30m to 0.50 m and has been sampled with grades up to 3.43 gpt Au and 329 gpt Ag from surface.

Los Parientes

The Los Parientes Vein is located near the northwestern extent of the Babicanora Vein and is approximately subparallel to the strike extension of the Amethyst Vein. It is a southeast-directed quartz breccia structure striking approximately 150° azimuth and dipping between 65° to 70° to the southwest. The vein has been mapped on surface for approximately 250 m with width of approximately 0.30 m and has been sampled from surface chips with grades up to 27.7 gpt Au and 42 gpt Ag.

Los Chiltepines

The Los Chiltepines Veins are located to the northeast of the Las Chispas Area and are a southeast directed set of sub-parallel quartz vein structures with strike of approximately 155° azimuth and with dip of approximately 85°. The vein has been mapped on surface for approximately 500 m with width of approximately 0.30 m to 0.50 m and has been sampled from surface chips with grades up to 0.91 gpt Au and 270 gpt Ag.

El Rancho and El Rancho Dos

The El Rancho Veins are located to east of the Las Chispas area. The El Rancho Vein is a quartz vein and quartz breccia structure with estimate southwest directed strike and with width of approximately 0.40 m to 0.50 m. El Rancho Dos Vein is located immediately east of, and is parallel to, El Rancho and is a quartz breccia structure with width of up to 3.0 m. The veins have not been mapped on surface. The veins were discovered in 2018 as part of the infrastructure condemnation program, which led to completion of 16 holes totalling 4,660.8 m by early 2019. Assay results were low grade with several wide intersections of calcite veins and veinlets. The best assay was sampled from hole RA-18-05, which graded 1.5 gpt Au and 252 gpt Ag over 0.51 m.

Flo Vein

The Flo Vein is located east of Babicanora Norte and is a southeast-directed quartz vein structure with strike of approximately 220° to 230° azimuth and dip of approximately 65°. The vein has been mapped on surface for approximately 320 m with width of approximately 1.5 m and has been sampled from surface. However, no assays returned values above detection limit.

La Martina

The La Martina Vein is a southwest-directed quartz breccia structure containing massive white quartz located northwest of the Las Chispas Area and is juxtaposed approximately along the regional normal fault. It strikes between 220° to 230°azimuth and dips 65° toward the east. The vein has been mapped on surface for approximately 320 m with width up to 1.0 m. The vein has not yet been sampled.

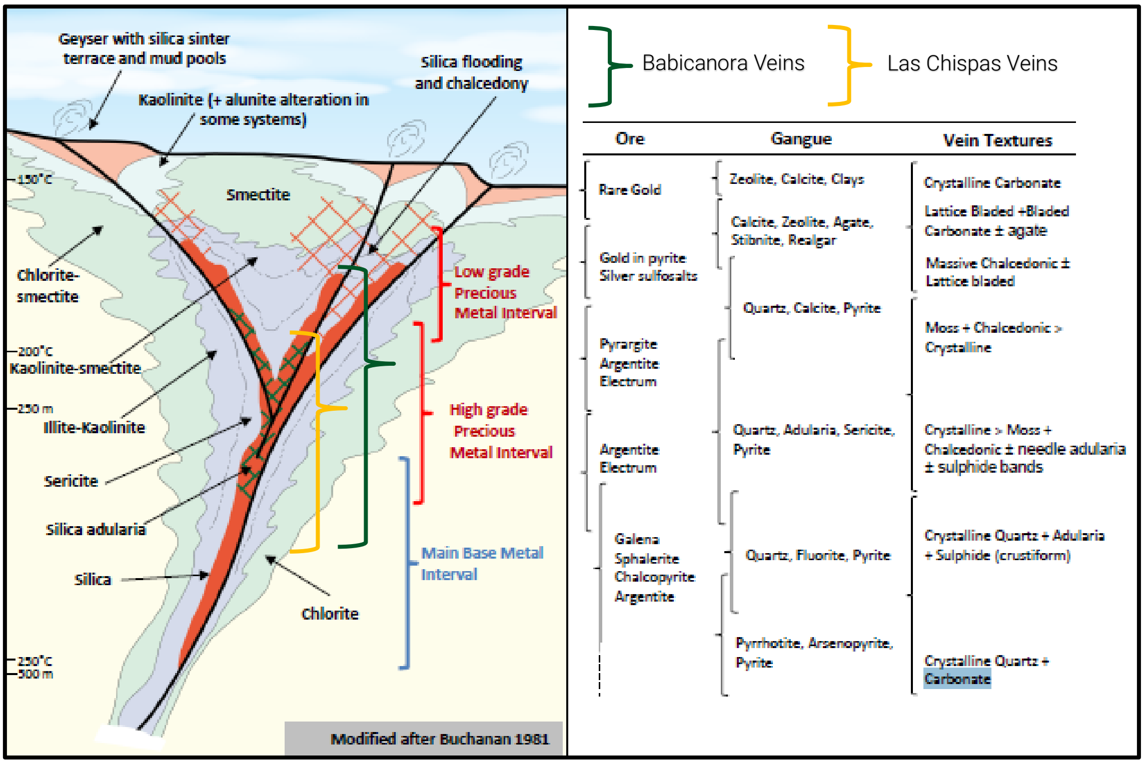
8 Deposit Types

Mineral deposits in the Las Chispas district are classified as gold and silver, low to intermediate sulphidation epithermal systems, typical of many deposits in northeastern Sonora, including the nearby Santa Elena Mine (operated by First Majestic Silver Corp.) and the Mercedes Mine (Premier Gold Mines Ltd.). Elsewhere in the Sierra Madre, additional examples include the Dolores Mine (Pan American Silver Corp.) and Piños Altos Mine (Agnico Eagle Mines Ltd.) in the State of Chihuahua.

8.1 Low Sulphidation

The terms low and intermediate sulphidation are based on the sulphidation state of the sulphide assemblages. Low sulphidation epithermal deposits are formed at shallow depths from hydrothermal systems related to volcanic activity (Figure 8-1). Low sulphidation deposits typically display all or most of the following characteristics (e.g., Sillitoe 1991; White and Hedenquist 1990):

- Hosted in volcanic rocks ranging from andesite to rhyolite in composition;
- Hydrothermal fluids are characterized to be lower temperatures, have circum-neutral pH and are reduced;
- Alteration consists of quartz, sericite, illite, kaolin, adularia and silica. Barite and fluorite may also be present;
- Mineralization hosted in quartz and quartz-carbonate veins, veinlets and silicified zones;
- Silica types range from opal through chalcedony to massive quartz. Textures include crustiform and colloform banding, drusy, massive and saccharoidal varieties. Calcite may form coarse blades and is frequently replaced by quartz;
- Deposits of this type may be overlain by barren zones of opaline silica;
- Overall, sulphides typically comprise less than 5% by volume;
- Sulphides may selectively average up to several per cent and comprise very fine-grained pyrite, with lesser sphalerite, galena, tetrahedrite and chalcopyrite sometimes present;
- Gold may be present as discreet, very fine grains or may be silica or sulphide refractory;
- Gold and silver grades are typically low, but may form extremely high-grade mineralized shoots; and
- Common associated elements include mercury, arsenic, antimony, tellurium, selenium, and molybdenum.



Note: Figure from Buchanan, 1981.

Figure 8-1: Detailed Low-sulphidation Deposit with Ore, Gangue and Vein Textures with Estimated Location of Las Chispas Epithermal Mineralization

Low sulphidation gold-silver epithermal systems commonly precipitate gold from hydrothermal fluids in near-surface hot spring environments. The mechanism most commonly evoked for gold precipitation is boiling. Boiling occurs as pressure decreases on ascending fluids. The physical and chemical changes that accompany boiling cause breakdown of the gold-bearing chemical complexes and precipitation of the gold. Because pressure from the overlying fluid column or rock column controls the level at which boiling occurs, the location of the boiling zone commonly occurs within a particular vertical depth range. However, this depth can change significantly with changes in the water table, sealing of the system, burial of the system through deposition of volcanic rocks, and emergence due to tectonic uplift. The boiling zone is typically within 500 m and rarely more than 1 km below surface at the time of mineralization.

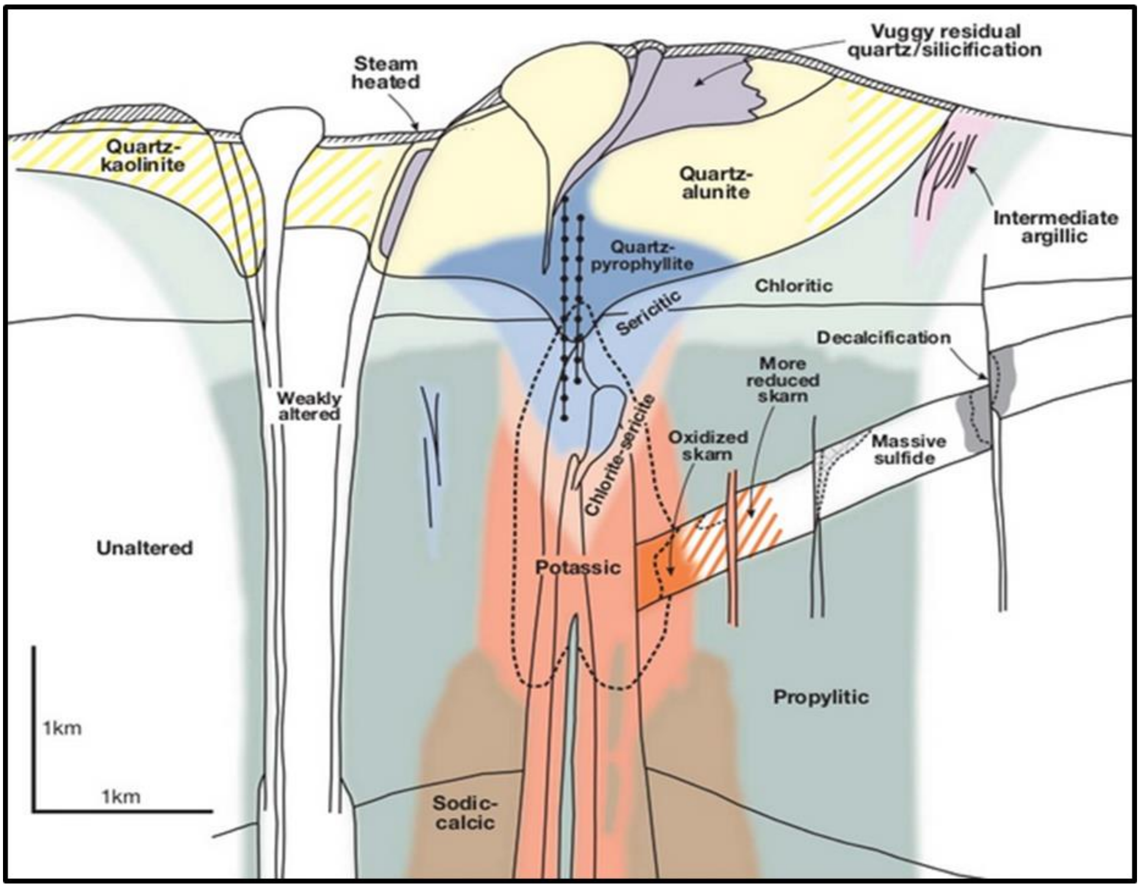
8.2 Intermediate Sulphidation

Intermediate sulphidation epithermal systems are less common but share some characteristics of the low and high sulphidation types. Like the high sulphidation types, intermediate types also occur in mainly in volcanic sequences of andesite to dacite composition within volcanic arcs.

Like the low sulphidation type, intermediate type mineralization normally occurs in veins, stockworks and breccias. The veins can be rich in quartz, with manganiferous carbonates like manganese-rich calcite or rhodochrosite plus adularia, which typically hosts the gold mineralization. Gold occurs as native metal, tellurides and in a variety of gold-rich base metal sulphides and sulfosalts. Low-iron sphalerite, tetrahedrite-tennantite and galena often are the dominant sulphide minerals. The overall sulphide content of the deposits is in the range of 5% to 20% by volume.

Alteration associated with intermediate type deposits consists of a mixture of high- and low-sulphidation assemblages that may overprint each other, depending on the evolution of the fluids. Silica (vuggy), advance argillic (alunite, pyrophyllite, diaspore, dickite, and sericite), argillic (kaolinite), anhydrite, barite, sericite, illite, and adularia may be present or absent within the system (Figure 8-2).

Permeable rocks that host intermediate type deposits may allow the mineral fluids to form a large tonnage of low-grade, bulk-minable stockwork mineralization (Ralf 2017).



Note: Figure from Sillitoe, 2010.

Figure 8-2: **Illustration of Intermediate Sulphidation Hydrothermal Systems**

9 Exploration

Prior to SilverCrest acquiring the Project in 2015, no drilling had been completed on the northwest to southeast-oriented Las Chispas and Babicanora mineralization corridors. This trend is approximately 3.5 km long and 3.5 km wide.

SilverCrest exploration began work on the Project in March 2016, with a primary focus on the Las Chispas, William Tell, and Babicanora Veins. From March to October 2016, the Phase I exploration program consisted of initial drilling, surface and underground mapping and sampling, and rehabilitating an estimated 6 km of underground workings. Core drilling of 22 holes during Phase I is described in Section 0.

From October 2016 to February 2018, the Phase II exploration program consisted of drilling, additional surface and underground mapping and sampling, further rehabilitation of 4 km of underground workings, plus auger and trenching of 42 surface historical mineralization dumps. Drilling of 161 additional holes during Phase II is described in Section 0.

From February 2018 to February 2019, the Phase III exploration program consisted of drilling, additional surface and underground mapping and sampling, and completing approximately 11 km of underground rehabilitation, a majority of which is located on the Las Chispas Vein and historical mine. Core drilling of 256 additional holes during Phase III is described in Section 0.

The continuation of Phase III exploration program (Phase III Extended) from February 2019 to present, consisted of infill drilling to potentially support upgrade of Inferred to Indicated Resources. This phase also includes in-vein development mine face-mapping and chip-channel sampling on the Babicanora Vein in the Area 51 Zone. The following exploration information was collected during this period:

- Additional core drilling of 1,137 holes; approximately 70% infill and 30% expansion. The cut-off date for drill hole assays was October 16, 2020. Infill drilling provides support for upgrade of Inferred to Indicated Resources;
- Babicanora Vein underground in-vein development for approximately 800 m and 2,671 m of continuous sampling, and 646 chip-channel samples collected for geochemical analysis. Results of this work were used in resource modelling, reconciliation for model verses mining, and tracking tonnes and grade for stockpiling.
- Las Chispas Vein underground historical in-vein pillars and historical development for approximately 10 km, with 6,739 chip-channel samples collected for geochemical analysis. Results of this work were used in mineral resource modelling; and,
- Survey and mapping of historical workings, Santa Rosa underground decline, Babicanora (Area 51 Zone) in-vein and waste development.

From the start of drilling in March of 2016 at Las Chispas to October 16, 2020, 1,626 core holes for 426,441.5 m were completed with 145,657 drill samples collected and analyzed (Section 0). A total of 7,385 underground samples were collected and analyzed.

9.1 Underground Exploration at Las Chispas Historical Mine

Initial access to the underground historical workings, the majority located in the Las Chispas (Historical Vein) Mine, commenced with an underground rehabilitation program in February

2016. Rehabilitation included removal of backfill, construction of a network of bridges and ladders across open stopes, installation of safety cables, removal of obstructions and unsafe overhead supports, construction of new overhead supports, rough rock scaling, and development of a control survey (Photo 9-1).

Note: Figure from Barr et al., 2019.

Photo 9-1: Photos of Las Chispas Underground Rehabilitation Activities

As of the Report effective date, SilverCrest estimates that approximately 11.0 km of underground workings was rehabilitated.

As part of the rehabilitation program, an underground mapping and sampling program began in February 2016. Collection of a series of select chip samples was followed by a systematic and continuous saw cut channel sampling program along the rehabilitated underground workings (refer to lower left image in Photo 9-1). Samples were collected perpendicular to mineralization, as transverse samples and as longitudinal samples taken along footwall or hanging wall contacts through stopes. A total of 6,739 chip and channel samples have been collected and used for resource modeling as of the Effective Date of the Report. Of these, 1,094 sample results graded above a cut-off of 150 gpt AgEq, with averages of 4.05 gpt Au and 504 gpt Ag, or 807 gpt AgEq.

A total of 94 samples have been collected from historical underground backfill muck at Las Chispas, grading an average 2.1 gpt Au and 256 gpt Ag, or 414 gpt AgEq. These samples and volumes are excluded from the Mineral Resource Estimate.

Table 9-1 shows summary statistics of underground chip and channel sampling for the Las Chispas workings, Table 9-2 and Figure 9-1 shows other historical underground workings in the Las Chispas Area, and Table 9-3 shows sample results from the Babicanora Area workings.

Table 9-1: Las Chispas Vein – Significant Channel Sampling Results Before February 2019

Las Chispas	Mean Au	Mean Ag	Mean AgEq(1)
200L	0.050	7	11.1
300L	1.008	141	216.6
350L	2.329	333	507.9
400L	1.688	266	392.8
450L	3.237	440	682.6
500L	2.549	337	527.8
550L	1.784	256	389.9
600L	0.410	57.6	88.3
700L	0.121	15.5	24.5
743L	0.615	118	164.3
Average	0.903	131	199.17
Number of Samples	3,923	3,923	3,923
Maximum Value	136	10,000	20,200
Minimum Value	0.002	0.2	0.575
Standard Deviation	3.713	444	704.0
Number of Samples >150 AgEq	-	-	805.0

Note: (1) AgEq is based on a silver to gold ratio of 75:1, calculated using long-term gold and silver prices and of $1,225/oz gold and of $17.0/oz silver, with average metallurgical recoveries of 95% gold and 90% silver.

Table 9-2: Las Chispas Area, Other Vein Targets – Significant Channel Sampling Results Before February 2019

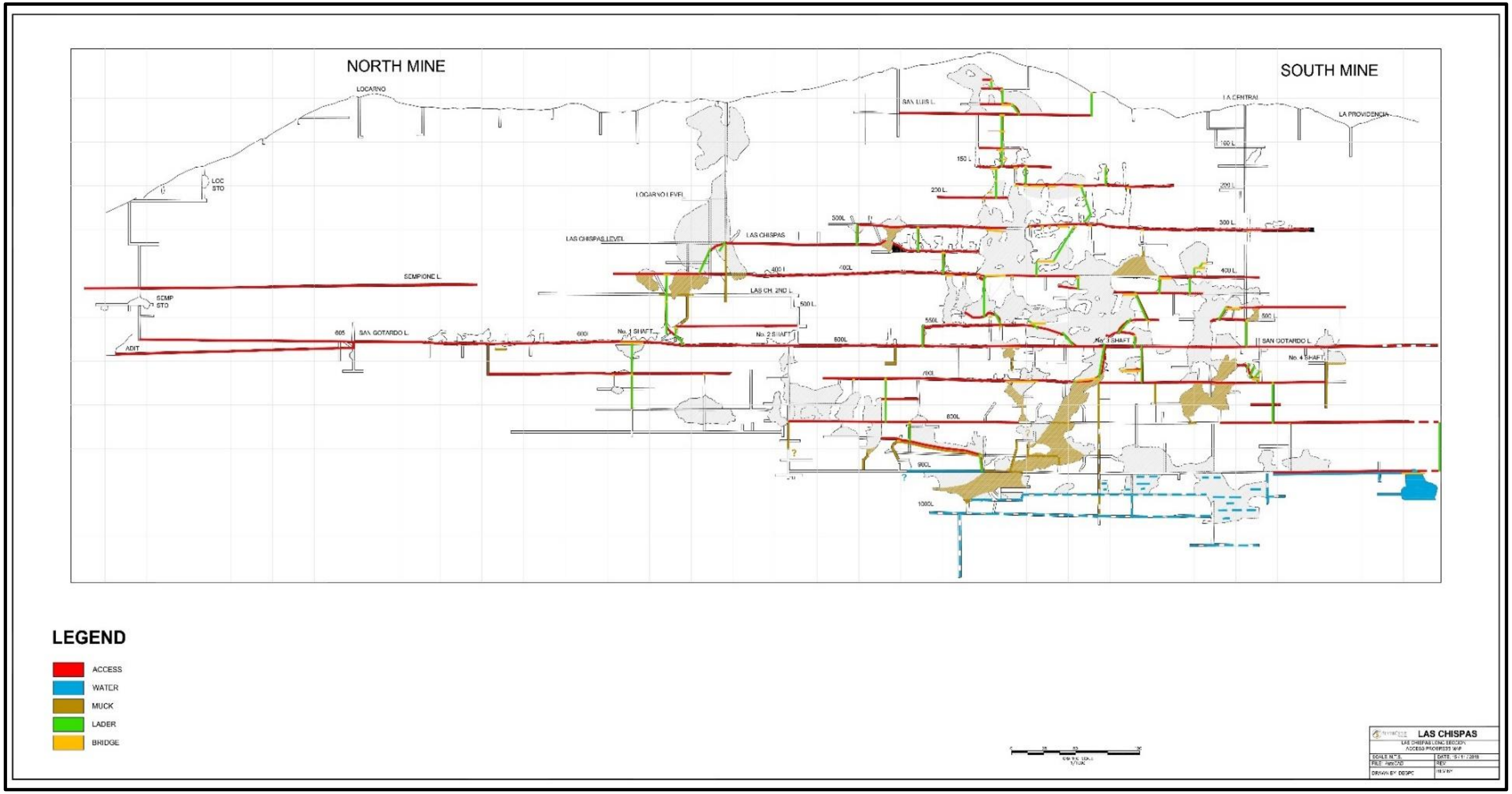
Las Chispas	Mean Au	Mean Ag	Mean AgEq*
El Erick	1.85	118	256.4
El Sheik	1.16	75.8	162.8
Espiritu Santo	0.02	11.2	12.4
Lupena	0.45	39.4	73.0
Varela	0.22	26.5	43.1
WT500L	1.05	62.8	141.4
WT600L	1.29	146	242.4
Average	0.91	73.9	142.0
Number of Samples	1,292	1,292	1,292
Maximum Value	52.2	3,220	5,455
Minimum Value	0.01	0.2	0.0
Standard Deviation	3.44	221	431.1
Number of Samples >150 AgEq	-	-	237

Note: (1) AgEq is based on a silver to gold ratio of 75:1, calculated using long-term gold and silver prices of $1,225/oz gold and $17.0/oz silver, with average metallurgical recoveries of 95% gold and 90% silver.

Table 9-3: Babicanora Main Vein, Other Vein Targets – Significant Channel Sampling Results Before February 2019

Las Chispas	Mean Au	Mean Ag	Mean AgEq(1)
Babicanora	0.41	26.1	56.6
Babicanora de abajo	0.07	7.7	12.6
Bertina	0.08	4.6	10.9
Buena Vista	0.03	7.1	9.1
El Muerto	0.62	33.4	80.1
Jabali	0.15	10.3	21.9
Sementales	0.49	18.7	55.0
Average	0.31	16	39
Number of Samples	756	756	756
Maximum Value	20.80	821	2,381
Minimum Value	0.01	0.2	1.0
Standard Deviation	1.22	51.9	135.8
Number of Samples >150 AgEq	-	-	52

Note: (1) AgEq is based on a silver to gold ratio of 75:1, calculated using long-term gold and silver prices of $1,225/oz gold and $17.0/oz silver, with average metallurgical recoveries of 95% gold and 90% silver.



Note: Figure prepared by SilverCrest, 2020. Based on schematic from Pedrazzini circa 1921 (Photo 6-9).

Figure 9-1: Las Chispas Vein Long Section with 2018 Underground Infrastructure (looking northeast)

9.1.1 Underground Surveying for Las Chispas Historical Mine

A network of control points was first established by a SilverCrest surveying crew when access to workings were rehabilitated and secured. Control points were established at approximately 15 m intervals using portable drills, survey chains, distance lasers, and a handheld Brunton compass. The control network was then re-surveyed by third-party contractor Precision GPS, with professional surveying crew using a Trimble VX Total Station on level 600 to level 150. The centre line of each drift was collected, this included a data set of 178 points. The survey purpose was to adjust the tape and Brunton survey completed by the SilverCrest staff. This underground control network is the base reference for all underground sampling and drilling activities.

9.2 Surface Exploration

Surface exploration focused on geological mapping and delineation of the numerous historical shafts and portals present in the Project area. As of the Report Effective Date, a total of 8.0 km^2 was mapped by SilverCrest geologists.

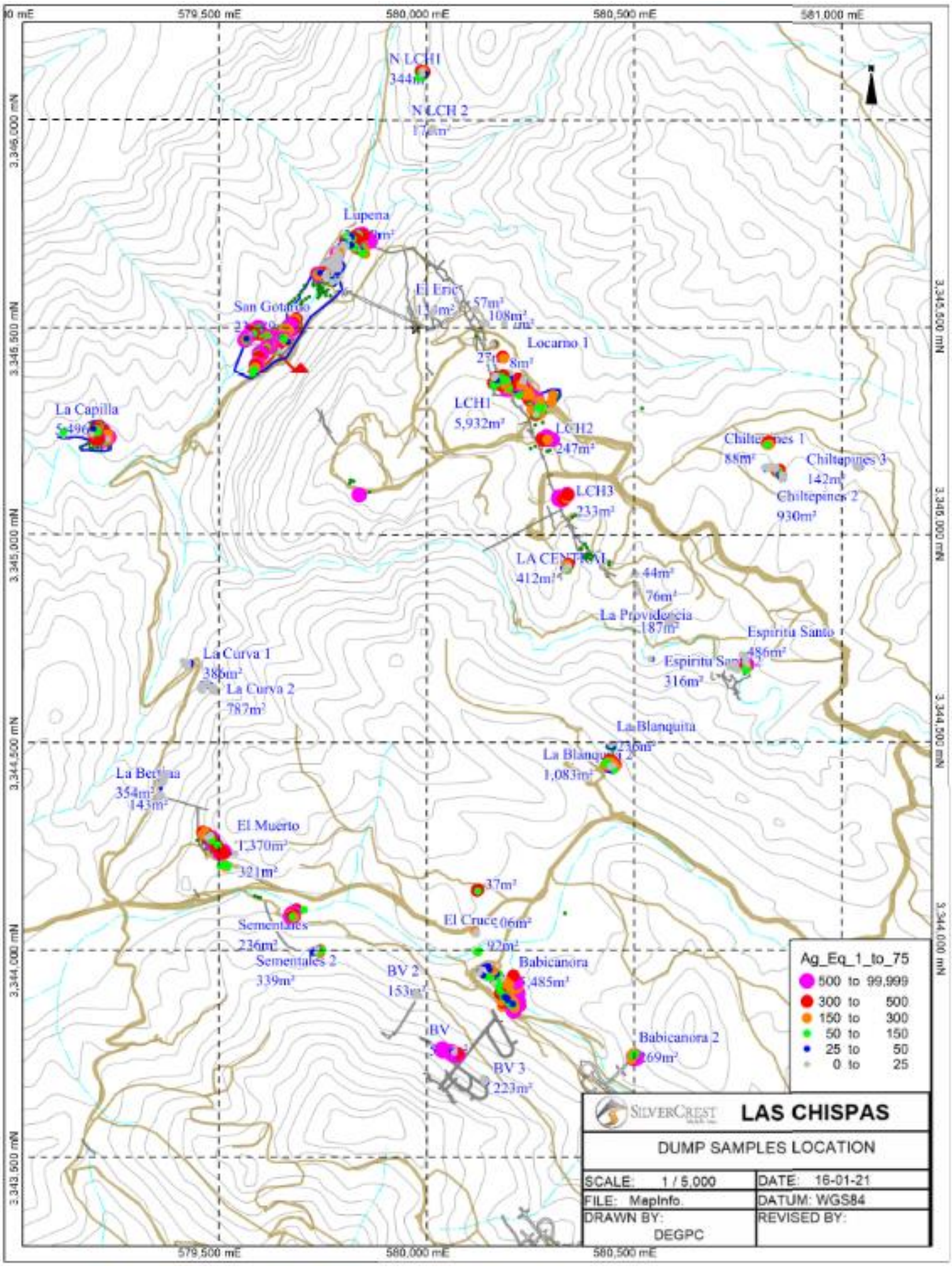
Surface dump augering, trenching, and sampling were completed. Analytical results have been received as of the Report Effective Date for 1,340 surface dump samples, averaging 1.12 gpt Au and 107 gpt Ag, or 185 AgEq. Select grades from the dump sampling range up to 4,548 gpt AgEq. The mapping data are georeferenced and being used to develop a geographic information system (GIS) database for Las Chispas.

In 2017, historical mineralization rock dumps were sampled by a trenching and auger program to collect data, identify dump volumes, and calculate precious metal grades. Data were collected from field measurements using a global positioning system (GPS) instrument and trenching rock and sediment material in the dumps. The dumps were later surveyed between December 14, 2017 and January 26, 2018 using a Trimble Spectra Total Station Model TS-415. Samples were sent to ALS Chemex (ALS) based in Hermosillo for preparation, and then sent to the ALS based in Northern Vancouver laboratory for gold and silver analysis.

In total, 41 dumps at 20 locations within the Project area were sampled by an auger or by trenching between July 2017 and January 2018. Table 9-4 summarizes the dump names and Figure 9-2 shows the locations.

Table 9-4: List of Surface Stockpiles (Dumps, Muck and Tailing) Mapped in the Las Chispas Project Area

Dump Name	Sample Style
North Chispas 1, 2	Trench
La Capilla (LCA), tailings	Auger
San Gotardo (LCD)	Trench
Lupena (LUP)	Trench
El Eric	Trench
Locarno 1, 2, 3, 4	Trench
Las Chispas 1, 2, 3 (LCH)	Trench
La Central	Trench
Maria	Trench
Chiltepines 1, 2, 3	Trench
La Providenca 1, 2, 3	Trench
Espiritu Santo 1, 2	Trench
La Blanquita 1, 2	Trench
La Curva 1, 2	Trench
La Bertina 1, 2	Trench
El Muerto 1, 2	Trench
Sementales 1, 2	Trench
Buena Vista 1. 2, 3	Trench
Babicanora 1, 2	Trench
El Cruce 1, 2, 3	Trench
Total	**41**



Note: Figure by SilverCrest, 2021.

Figure 9-2: Location of Surface Stockpiles and Historical Mine Dumps Mapped and Sampled by SilverCrest Phase III Surface Geological Mapping and Lithological Model

To initially determine the feasibility of evaluating historical dumps, an auger program was tested in July 2017. A standard mechanical gas-powered auger was used to complete the test program. Auger drilling was only found to be useful for one dump (La Capilla tailings), due to issues with large rocks and low sample recovery.

The auger program began by setting up north-south oriented base lines near the centre of a dump. First, a compass, a GPS, and tape were used to mark a hole, then flag and tag it with 10 m between each flag. Depending on the size of the site, a specific number of parallel gridlines were emplaced running east-west and 10 m apart. Second, a tripod was situated over the surface of a flagged hole and a pulley attached at the top. Next, the standard penetration test equipment was aligned at the tripod's centre and the initial hole within 1 m proximity to the flagging. Two personnel manned the sampler with one on the capstan, to drive the sampler into the soil surface and down until either the sampler hits a fixed depth of 1 m or until it could not go any deeper. If a rock prevents downward movement of the auger, it must either drill down by uplifting it or pushing it into the wall, or the piercer can be used to pulverize the rock. When a fixed depth or bedrock reached, the sampler is pulled up to the surface placing the contents on a tarp to spread and homogenize the mixture. Each interval was bagged with the hole ID and interval. The process of three personnel manning the sampler and capstan was repeated at 1 m interval depths.

In 2016 and early 2017, initial testing of waste dump material was completed by hand-cut trenches for sample collection. Trenches were hand excavated to approximately 0.5 m in the face of dumps with collection of samples every 1 m. This program identified that most dump had significant precious metals that warranted further evaluation.

From mid-2017 to January 2018, mechanical trenching was completed on all accessible historical dumps. A backhoe was used to dig trenches approximately 1.5 m deep and pile the excavated material next to the trench for sampling and description. Sample weights were 3 kg to 5 kg. Samples were labelled with an interval ID, GPS coordinate and depth sampled. The backhoe continued to work on an interval until either the soil was reached, or the walls collapsed into the trench. The excavation process continued until the backhoe reached the marked end of the trench. Additionally, a supervisor analyzed the piles for quartz percentage and historical trash, while describing the grain size and rock type.

SilverCrest initiated a comprehensive surface mapping and drill core relogging program in November 2018 to support development of a detailed stratigraphic section and three-dimensional (3D) lithological model across the Babicanora and Las Chispas Areas. The work resulted in an improved understanding of the regional and local structures, location of various intrusive phases, and an understanding of the relationship between host rock lithology and mineralization styles observed in drill core. The resulting 3D model was used to guide exploration targeting in areas not previously considered and additional discoveries were made during the Phase III program, as follows:

- Mineralization footprint expansion of the Babicanora, Babicanora Sur, Babicanora Norte, Babi Vista, Las Chispas and various other lesser veins;
- Deep targets under Las Chispas and Babicanora areas related to specific lithology host rocks, crosscutting structures, and zonation in geochemistry;
- Chiltepin Area, northeast of the Las Chispas Area;
- La Victoria Vein mineralization within respect to host lithologies; and,
- Mineralization along the Babicanora caldera ring structure and associated rhyolite/andesite dikes.

9.3 Exploration Decline in the Babicanora Vein

SilverCrest is in the process of underground exploration and development in the Babicanora Vein Area 51 Zone. With the first blast on February 27, 2019, SilverCrest commenced development of the exploration decline, named the Santa Rosa Decline. On June 20, 2019, the Babicanora Vein was intersected. In the following days, in-vein development began with surveying, geological mapping, and establishment of sampling protocols. High grades were mapped and sampled on the first day of intersection (Photo 9-2). From June 20, 2019 to June 15, 2020, approximately 800 m of in-vein underground development was completed.

Source: SilverCrest, 2020

Photo 9-2: Underground Intersection of Babicanora Main Vein, Area 51 Zone, Santa Rosa Decline, 1096 masl

On October 16, 2019, SilverCrest announced underground Babicanora Vein, Area 51 Zone, sampling results showing positive reconciliation compared to the February 2019 Mineral Resource estimate (Barr et al., 2019). Reconciliation results between the actual mined vein and the resource model from the underground mining of the Babicanora Vein along the first 180 m of vein strike length, or approximately 20% of the mineralization length in Area 51, are shown in Table 9-5, Table 9-6, Figure 9-3, Figure 9-4, Figure 9-5, Figure 9-6, and Figure 9-7.

Table 9-5: Babicanora Main Vein Resource Model (TetraTech, 2019) to Actual Mined Reconciliation Results October 2019

Item	Comment
Weighted average grade of U/G sampling results in vein:	14.89 gpt Au and 1168 gpt Ag, or 2,284 gpt AgEq*
Estimated grade (cut) of actual mined Babicanora Vein:	15.18 gpt Au and 1071 gpt Ag, or 2,209 gpt AgEq
Feb. 2019 mineral resource model grade (cut) for mined area:	4.48 gpt Au and 556 gpt Ag, or 892 gpt AgEq
AgEq* grade percent difference between actual versus model:	147.7%
Estimated tonnage of the actual mined Babicanora Vein:	4,170
Feb. 2019 mineral resource model tonnage for mined area:	6,226
Tonnage percent difference between actual versus model:	-33.00%
Estimated actual AgEq contained ounces within the mined area:	299,000
Feb. 2019 resource modelled AgEq contained ounces in mined area:	179,000
AgEq ounce difference between actual versus model:	65.7%
Estimated cut & diluted actual mined & stockpiled:	7,431 t grading 8.62 gpt Au and 613.4 gpt Ag, or 1,259 gpt AgEq for 301,000 AgEq oz
Cumulative surface stockpiles (new Babicanora Vein and Historical):	182,000 t grading 1.68 gpt Au and 139 gpt Ag or 264 gpt AgEq for 1,546,000 AgEq oz

Note: All numbers are rounded. Composites based on a COG of 150 gpt AgEq. Composites are approximate true widths. Sample results stated in table are only from 1.5 m above the drift floor. Sample results in the block model for reconciliation also include samples from 0.5 metres above floor and on the back (roof) of the drift.
*AgEq based on gold to silver ratio 75:1, calculated using long-term gold and silver prices of $1,225/oz and $17.0/oz, with average metallurgical recoveries of 95% gold and 90% silver.

The positive difference in the actual mined vein and the February 2019 Mineral Resource estimate is based on:

- The presence of a high-grade clay shear zone within the vein that was not recovered in core drilling during exploration,
- 25 m long high-grade portion (3.5 m wide grading 5,680 gpt AgEq) of the mined vein that was not intersected in previous exploration drilling, and
- Overall, there is more consistency in the quartz-argentite portion of the vein than previously modelled.

A relative 33% decrease in Mineral Resource modelled tonnes (undiluted) was determined. In a few areas during development, the vein was only partially mined and future additional excavation may be warranted. Several high-grade splays trending into the hanging wall and footwall of the Babicanora Vein were also observed and may warrant future excavation.

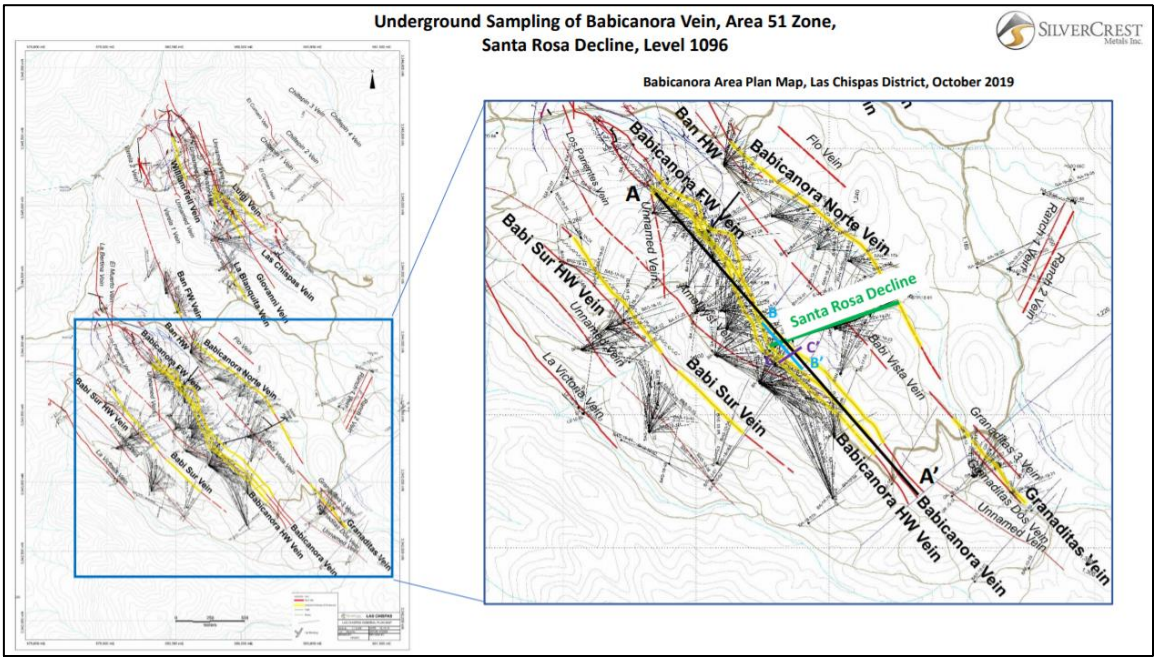
A total of 725 underground channel sample results were used for reconciliation (Table 9-6). In this case, the Mineral Resource estimate in the 2019 PEA (Barr et al., 2019) was used for comparative purposes.

Table 9-6: Babicanora Main Vein, Level 1096, Underground Channel Sample Composite Results

U/G Sample Composite No.	From (m)	To (m)	Length (m)	Au gpt	Ag gpt	AgEq gpt*
B207-SE	0.0	3.1	3.1	8.94	983	1,653
includes			0.6	28.10	3,280	5,388
B207-NW	0.0	2.1	2.1	37.33	4,599	7,399
includes			0.6	125.50	13,961	23,373
B208-SE	0.0	3.9	3.9	8.85	975	1,638
includes			0.6	16.60	2,170	3,415
B208-NW	0.0	3.3	3.3	50.07	1,772	5,527
includes			0.6	166.00	6,140	18,590
B210-SE	0.0	4.3	4.3	51.50	3,161	7,023
includes			0.8	125.00	2,690	12,065
B211-SE	0.0	4.8	4.8	17.46	1,458	2,768
includes			0.3	137.50	5,740	16,053
B212-SE	0.0	3.5	3.5	20.09	1,030	2,537
includes			0.7	60.10	1,440	5,948
B212-NW	0.0	1.3	1.3	9.24	731	1,424
includes			0.5	11.25	864	1,708
B213-SE	0.0	4.0	4.0	18.55	1,399	2,790
includes			0.7	26.30	2,850	4,823
B215-SE	0.0	3.2	3.2	61.89	2,503	7,145
includes			0.8	123.50	3,770	13,033
B215-NW	0.0	1.0	1.0	15.92	2,205	3,399
includes			0.7	22.10	3,060	4,718
B216-SE	0.0	3.5	3.5	25.27	4,737	6,632
includes			0.7	44.80	17,092	20,452
B216-NW	0.0	1.3	1.3	10.71	872	1,675

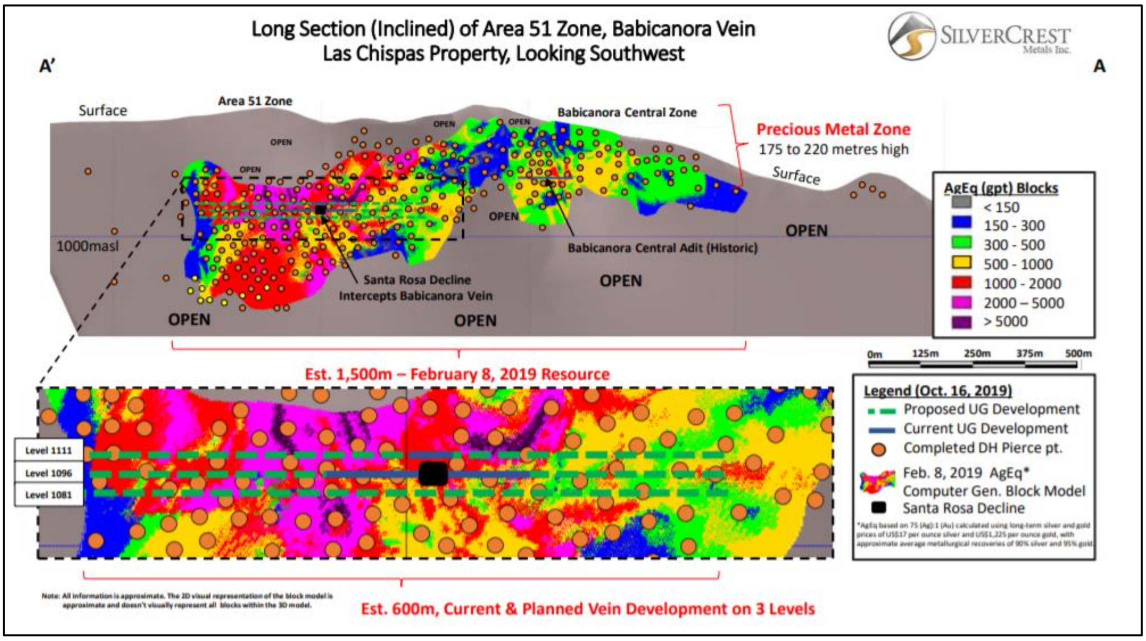
U/G Sample Composite No.	From (m)	To (m)	Length (m)	Au gpt	Ag gpt	AgEq gpt*
includes			0.7	13.10	1,460	2,443
B217-SE	0.0	2.8	2.8	94.52	8,320	15,409
includes			1.0	229.00	21,128	38,303
B218-SE	0.0	3.1	3.1	17.00	1,549	2,823
includes			0.6	63.20	4,570	9,310
B218-NW	0.0	2.4	2.4	11.93	440	1,335
includes			0.5	40.80	1,185	4,245
B220-NW	0.0	2.6	2.6	6.46	812	1,296
includes			0.3	24.90	3,010	4,878
B224-NW	0.0	1.4	1.4	9.38	436	1,139
includes			0.5	15.05	1,030	2,159
B227-SE	0.0	1.7	1.7	6.41	592	1,073
includes			0.6	12.95	695	1,666
B230-SE	0.0	1.8	1.8	9.52	916	1,630
includes			0.5	16.90	1,215	2,483
B232-SE	0.0	1.2	1.2	10.76	1,691	2,498
includes			0.4	25.70	3,320	5,248
B233-SE	0.0	0.8	0.8	5.73	1,267	1,695
includes			0.5	8.16	1,250	1,862
B233-NW	0.0	2.8	2.8	20.51	491	2,029
includes			0.6	60.00	655	5,155
B234-SE	0.0	2.4	2.3	8.94	1,605	2,275
Includes			0.5	14.75	3,570	4,676
B235-SE	0.0	3.3	3.3	7.96	905	1,502
Includes			0.4	50.00	4,710	8,460

Note: All numbers are rounded. Composites based on a COG of 150 gpt AgEq. Composites are approximate true widths. Sample results stated in table are only from 1.5 m above the drift floor. Sample results in the block model for reconciliation also include samples from 0.5 m above floor and on the back (roof) of the drift.
* AgEq based on 75 (Ag):1 (Au) calculated using long-term gold and silver prices of $1,225 /oz Au and $17.0 /oz Ag, with average metallurgical recoveries of 95% Au and 90% Ag.



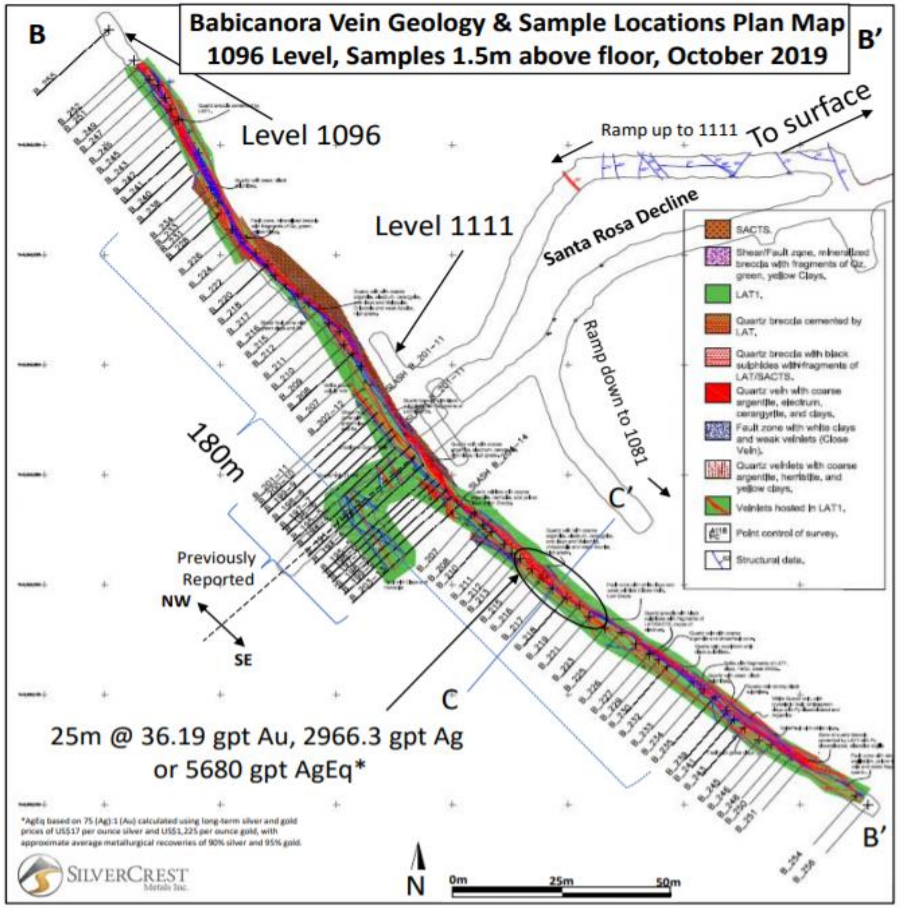
Note: Figure prepared by SilverCrest, 2019.

Figure 9-3: Location of Las Chispas District Veins, Santa Rosa Decline and Intersection with Babicanora Main Vein



Note: Figure prepared by SilverCrest, 2019.

Figure 9-4: Long Section of Babicanora Main Vein with Underground in Vein Development Location for Reconciliation

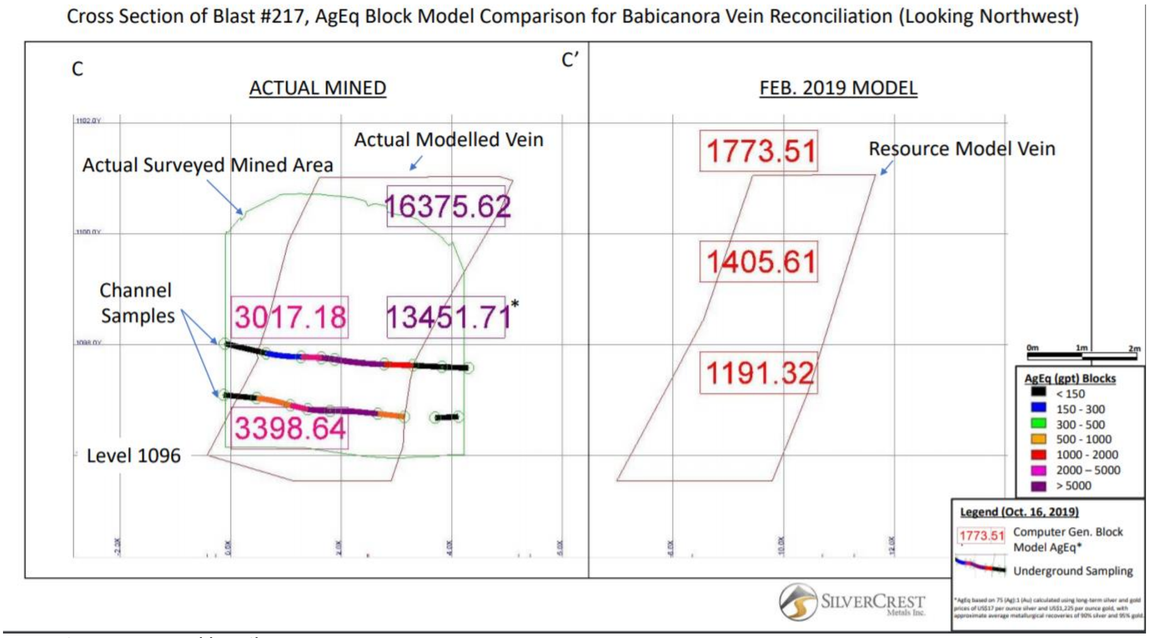


U/G Sample Composite No.	From (m)	To (m)	Length (m)	Au gpt	Ag gpt	AgEq gpt*
B207-SE	0.0	3.1	3.1	8.94	983.1	1,653
B207-NW	0.0	2.1	2.1	37.33	4,599.0	7,399
B208-SE	0.0	3.9	3.9	8.85	974.6	1,638
B208-NW	0.0	3.3	3.3	50.07	1,772.4	5,527
B209-NW	0.0	4.2	4.2	1.47	554.7	665
B210-SE	0.0	4.3	4.3	51.50	3,160.6	7,023
B210-NW	0.0	3.6	3.6	1.45	289.5	399
B211-SE	0.0	4.8	4.8	17.46	1,458.1	2,768
B211-NW	0.0	2.5	2.5	1.20	286.1	376
B212-SE	0.0	3.5	3.5	20.09	1,030.0	2,537
B212-NW	0.0	1.3	1.3	9.24	730.8	1,424
B213-SE	0.0	4.0	4.0	18.55	1,398.8	2,790
B215-SE	0.0	3.2	3.2	61.89	2,503.7	7,145
B215-NW	0.0	1.0	1.0	15.92	2,205.0	3,399
B216-SE	0.0	3.5	3.5	25.27	4,737.4	6,632
B216-NW	0.0	1.3	1.3	10.71	871.5	1,675
B217-SE	0.0	2.8	2.8	94.52	8,319.8	15,409
B217-NW	0.0	2.2	2.2	3.09	192.5	425
B218-SE	0.0	3.1	3.1	17.00	1,548.7	2,823
B218-NW	0.0	2.4	2.4	11.93	439.7	1,335
B219-SE	0.0	1.3	1.3	3.17	412.9	651
B220-NW	0.0	2.6	2.6	6.46	812.0	1,296
B221-SE	0.0	0.3	0.3	0.80	95.0	155
B222-NW	0.0	3.3	3.3	4.69	376.0	727
B223-SE	0.0	1.0	1.0	1.32	145.0	244
B224-NW	0.0	1.4	1.4	9.38	435.7	1,139
B225-SE	0.0	1.3	1.3	0.48	123.0	159
B226-SE	0.0	1.7	1.7	0.50	160.5	198
B226-NW	0.0	2.2	2.2	2.97	151.2	374
B227-SE	0.0	1.7	1.7	6.41	592.0	1,073
B228-NW	0.0	2.7	2.7	1.19	238.5	328
B229-SE	0.0	1.9	1.9	1.55	597.6	714
B230-SE	0.0	1.8	1.8	9.52	915.7	1,630
B231-NW	0.0	1.5	1.5	1.73	286.7	417
B232-SE	0.0	1.2	1.2	10.76	1,691.2	2,498
B233-SE	0.0	0.8	0.8	5.73	1,265.6	1,695
B233-NW	0.0	2.8	2.8	20.51	490.7	2,029
B234-SE	0.0	2.4	2.3	8.94	1,604.5	2,275
B234-NW	0.0	3.6	3.6	5.05	158.7	537
B235-SE	0.0	3.3	3.3	7.96	905.2	1,502
B238-NW	0.0	2.2	2.2	2.55	317.8	509

Note: This table is not representative of the weighted average grade of actual mined tonnage. Excludes samples for 0.5m above floor and back (roof).

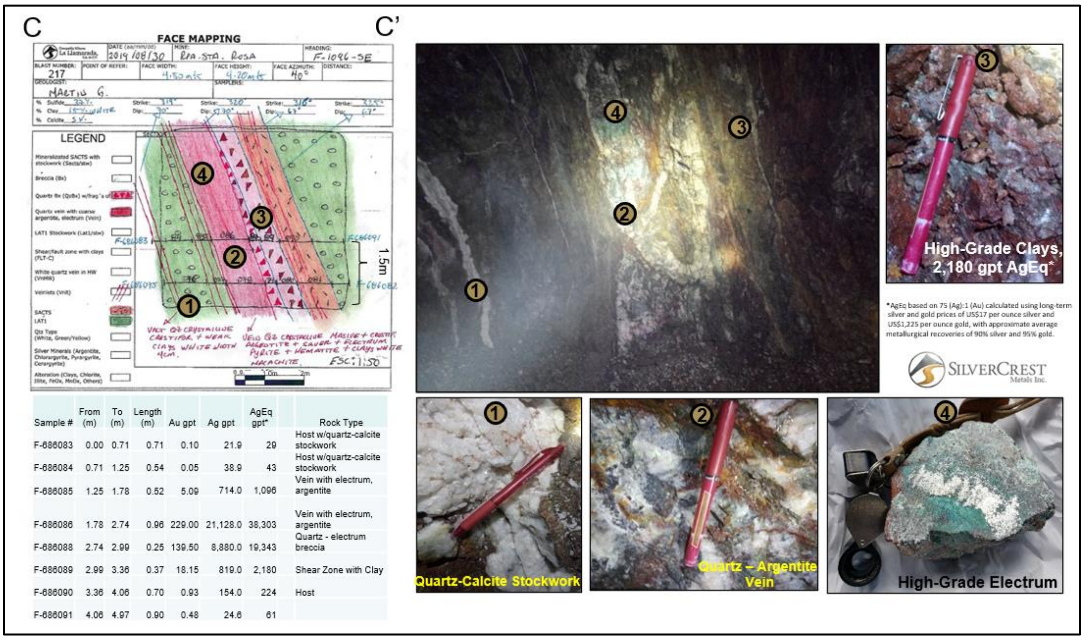
Note: Figure prepared by SilverCrest, 2019.

Figure 9-5: Babicanora Main Vein Plan Map with Information for Reconciliation Estimation



Note: Figure prepared by SilverCrest, 2019.

Figure 9-6: Selective Comparison of Actual Mined Vein with Underground Face Sampling Modelled Grades and February 2019 Vein Model with Block Grade



Sample #	From (m)	To (m)	Length (m)	Au gpt	Ag gpt	AgEq gpt*	Rock Type
F-686083	0.00	0.71	0.71	0.10	21.9	29	Host w/quartz-calcite stockwork
F-686084	0.71	1.25	0.54	0.05	38.9	43	Host w/quartz-calcite stockwork
F-686085	1.25	1.78	0.52	5.09	714.0	1,096	Vein with electrum, argentite
F-686086	1.78	2.74	0.96	229.00	21,128.0	38,303	Vein with electrum, argentite
F-686088	2.74	2.99	0.25	139.50	8,880.0	19,343	Quartz - electrum breccia
F-686089	2.99	3.36	0.37	18.15	819.0	2,180	Shear Zone with Clay
F-686090	3.36	4.06	0.70	0.93	154.0	224	Host
F-686091	4.06	4.97	0.90	0.48	24.6	61	

Note: Figure prepared by SilverCrest, 2019.

Figure 9-7: Example of Underground Mineralized Material Control Methodology for Use in Reconciliation

9.4 Santa Rosa Stockpile

In July 2019, the Santa Rosa Stockpile, located an estimated 100 m north of the Santa Rosa Portal, began to be loaded with in-vein mineralized material. Table 9-7 shows the tracking volumes and grade extracted from exploration in-vein drifting and mined location as of the Report Effective Date.

Table 9-7: Santa Rosa Stockpile (October 2020)

	Stockpile Year-to-date					Stockpile Project-to-date				
Level	Tonnes	Au (g/t)	Ag (g/t)	Ag Eq (g/t)	OZ Ag Eq	Tonnes	Au (g/t)	Ag (g/t)	Ag Eq (g/t)	OZ Ag Eq
BC HISTORIC ORE	5,153.93	3.64	88.91	361.56	59,911.85	8,817.93	2.98	85.58	309.14	87,643.21
LEVEL 1111 SE	2,924.40	11.44	1,325.10	2,182.73	205,223.78	6,588.40	9.26	920.78	1,099.77	232,955.13
LEVEL 1111 NW	2,971.39	5.38	418.39	822.21	78,547.98	7,524.57	5.48	431.33	842.01	203,699.13
LEVEL 1096 SE	1,763.23	5.47	765.49	1,176.04	66,668.65	7,438.82	6.29	596.09	1,068.07	255,443.80
LEVEL 1096 NW	16.00	5.47	1,749.58	2,159.49	1,110.87	5,117.17	7.16	586.16	1,123.05	184,764.37
LEVEL 1081 SE	1,632.00	3.33	326.16	576.12	30,228.83	3,372.02	4.52	458.03	797.38	86,446.62
LEVEL 1081 NW	3,795.31	11.34	808.89	1,659.61	202,508.30	5,211.88	9.80	746.51	1,481.23	248,203.43
LEVEL 1066 SE	748.56	17.73	1,771.07	3,100.96	74,630.27	748.56	17.73	1,771.07	3,100.96	74,630.27
LEVEL 1066 NW	583.36	42.56	4,185.76	7,377.53	138,367.94	583.36	42.56	4,185.76	7,377.53	138,367.94
LEVEL 1051 SE	403.94	29.39	249.59	2,453.85	31,868.14	403.94	29.39	249.59	2,453.85	31,868.14
LEVEL 1051 NW	299.07	18.53	1,463.11	2,853.13	27,433.87	299.07	18.53	1,463.11	2,853.13	27,433.87
TOTALES	***20,291.2***	***8.97***	***732.46***	***1,404.86***	***916,500.47***	***46,105.7***	***7.24***	***590.85***	***1,133.81***	***1,571,455.91***

Note: All numbers are rounded. Composites based on a COG of 150 gpt AgEq. Composites are approximate true widths. Sample results stated in table are only from 1.5 m above the drift floor. Sample results in the block model for reconciliation also include samples from 0.5 m above floor and on the back (roof) of the drift.
*AgEq based on 75 (Ag):1 (Au) calculated using long-term gold and silver prices of $1,225/oz Au and $17.0/oz Ag, with average metallurgical recoveries of 95% Au and 90% Ag.

9.5 Aerial Drone Topographic, Underground Exploration & Drill Hole Surveys

In February 2019, an aerial drone survey to collect light detection and ranging (LiDar) data over the Project was completed by Precision GPS from Hermosillo, Mexico. The survey used a MD4-1000 drone with a LiDAR module.

During the Phase III and Phase III Extended exploration programs, Precision GPS assisted with surveys of underground decline and drifts, and drill hole collar surveys up to October 16, 2020.

10 Drilling

10.1 Program Overview

SilverCrest completed their Phase I and Phase II drilling programs between March 2016 and February 2018. The Phase III drilling program included drilling up to February 2019. The Phase III Extended drilling program, starting in February 2019, focused on in-fill and expansion drilling to support the Feasibility Study and was completed on October 16, 2020 with a total of 309,383.85 m of drilling in 1,137 drill holes. From inception of drilling to October 2020, a total of 1,626 holes were completed for 426,441.50 m drilled with 145,657 samples collected for geochemical analysis. Drilling data to September 1, 2020 was used in the Mineral Resource and Mineral Reserve Estimates.

The Phase I drilling program targeted near-surface mineralization, lateral extensions of previously mined areas, and potential deep extensional mineralization proximal to the historical workings. The Phase II drilling program focused on surface drilling at the Las Chispas, Babicanora, William Tell, and Giovanni veins and on underground drilling at the Las Chispas and Babicanora veins. The Phase III drilling program focused on surface drilling at the Babicanora, Babicanora FW, Babicanora HW, Babicanora Norte, Babicanora Sur, Granaditas, Luigi, and Giovanni veins and underground drilling at the Las Chispas veins. The Phase III Extended drilling program was an infill program to support potential confidence category upgrades, and test for expansion of multiple veins. Table 10-1 summarizes these drilling programs and drill hole locations are represented in Figure 7-5.

Table 10-1: Summary of Sampling Completed by SilverCrest (assay returns to October 16, 2020)

Vein	Drill Location	Number of Drill Holes	Length Drilled (m)	Number of Samples	Length of Samples (m)
Phase I (March 2016 to October 2016)					
Las Chispas(1)	Surface	19	5,461.40	3,516	5,243.10
La Victoria	Surface	3	931.20	711	924.00
Subtotal		**22**	**6,392.60**	**4,227**	**6,167.10**
Phase II (October 2016 to February 2018)					
Las Chispas(1)	Surface	54	14,123.95	10,395	11,233.30
	Underground	21	1,992.90	1,782	1,780.20
Babicanora(2)	Surface	70	21,137.60	8,876	9,781.60
	Underground	14	1,446.70	1,252	1,415.40
Granaditas	Surface	2	653.45	594	653.50
Subtotal		**161**	**39,354.60**	**22,899**	**24,864.00**
Phase III (February 2018 to February 2019)					
Las Chispas(1)	Surface	4	1,176.90	831	907.30
	Underground	7	622.80	526	562.40
Babicanora	Surface	22	9,508.75	1,815	1,930.60
Granaditas	Surface	23	7,144.80	5,978	6,037.20
Babicanora Norte	Surface	40	11,810.70	7,233	7,767.90

Vein	Drill Location	Number of Drill Holes	Length Drilled (m)	Number of Samples	Length of Samples (m)
Babicanora Sur	Surface	7	3,069.30	967	995.30
Ranch	Surface	10	3,305.80	1,856	2,105.30
Test Wells	Surface	12	1,103.00	623	952.90
Subtotal		**125**	**37,742.05**	**19,829**	**21,259.00**
Phase III (February 2018 to February 2019)					
Las Chispas(1)	Underground	12	1,576.80	960	1,008.60
Babicanora(2)	Surface	52	17,075.40	5,328	5,676.10
	Underground	10	1,078.50	770	879.60
Babicanora Norte	Surface	18	3,884.10	1,853	2,241.80
	Underground	3	1,147.20	702	783.80
Babicanora Sur	Surface	32	8,160.40	3,749	4,382.90
Ranch	Surface	4	646.00	360	393.40
Subtotal		**131**	**33,568.40**	**13,722**	**15,366.00**
Phase III Extended (February 2019 to October 2020)					
Las Chispas(1)	Surface	150	44,902.44	16,807	15,294.10
Babicanora(2)	Surface	245	73,949.45	20,557	20,184.33
	Underground	59	3,207.25	2,459	2,038.36
Babicanora Norte	Surface	44	11,350.60	5,018	5,201.09
	Surface	262	63,915.15	17,667	15,626.83
Babicanora Sur	Surface	127	38,008.75	10,407	10,307.18
Babi Vista	Surface	183	57,941.26	5,050	4,062.82
El Muerto Zone	Underground	40	8,498.10	2,898	2,636.21
Ranch	Surface	3	709.00	196	188.41
La Victoria	Surface	8	2,263.30	1,542	1,613.60
Los Chiltepines	Surface	16	4,638.55	2,379	2,408.33
Subtotal		**1,137**	**309,383.85**	**84,980**	**79,561.26**
Total		**1,626**	**426,441.50**	**145,657**	**147,217.36**

Notes: (1) Las Chispas area totals include some re-drilled holes and holes drilled at Las Chispas, William Tell, Giovanni, Giovanni Mini, La Blanquita, La Varela, Luigi, and other unnamed veins in the Las Chispas area.
(2) Babicanora Area totals include holes drilled at Babicanora, Babicanora FW, Babicanora HW, Babicanora Norte, Babicanora Sur, Babi Vista, Amethyst Vein, and other unnamed veins in the Babicanora area.

Surface collar locations were initially surveyed using a handheld GPS unit, then professionally surveyed by a local contractor. A survey was completed by external consultant David Chavez Valenzuela in October 2018. This survey was performed using a GNSS Acnovo GX9 UHF instrument. The most recent surveys were completed by Precision GPS from Hermosillo, Sonora, Mexico using a Trimble VX10 Total Station and a Trimble R8 GNSS GPS RTK system. The survey provided drill collar locations, information on roads, and additional detail on property boundaries.

Underground drill hole collars were surveyed by Precision GPS using the underground control points established for each of the workings. All holes were downhole surveyed as single-shot measurements with a Flex-it® tool starting at 15 m with measurements at every 50 m to determine deviation. The survey measurements were monitoring downhole deviations and for significant magnetic interference from the drill rods that would prevent accurate readings.

10.2 Drilling Results

The information in the following sub-sections is summarized from Barr (2016, 2018), and Barr et al. (2019).

10.2.1 Phase I

During the Phase 1 drilling program, 4,227 core samples totalling 6,167.1 m were collected and assayed. The program targeted the historical Las Chispas Vein to verify the location of the vein and the existence of mineralization along trend of mapped historical workings. All drill holes intercepted quartz stockwork veinlets, veining and/or breccia, along with variable amounts of gold and silver mineralization. The results confirmed the mineralization structure or vein and suggested that relatively unexplored and unmined areas exist proximal to the historical workings.

Additional drilling targeting the William Tell Vein intercepted the mineralization structure or vein in four (4) of seven (7) drill holes.

The 2016 program also included three (3) holes (LV16-01, LV16-02, and LV16-03) in the La Victoria Area, located 800 m southwest of the Babicanora Vein. These holes intersected only low-grade mineralization.

10.2.2 Phase II

During the Phase II drilling program, 22,899 core samples totalling 24,864.0 m were collected and assayed. The program targeted delineation and expansion of known vein targets at the Las Chispas, William Tell, and Babicanora and tested new targets, such as the La Varela, La Blanquita, Granaditas, and Amethyst veins.

10.2.3 Phase III

During the Phase III drilling program, 33,551 core samples totaling 36,625.1 m were collected and assayed. The program targeted delineation and expansion of known vein targets in the Babicanora Area including the Area 51 Zone, Babicanora HW, Babicanora FW, Babicanora Norte, Babicanora Sur veins, in addition to the Giovanni vein in the Las Chispas area. Newly-tested targets for the Phase III program included the Babicanora Norte, Babicanora Sur, Granaditas, Luigi, Amethyst and Ranch veins.

10.2.3.1 Babicanora (Main) Vein

Delineation and expansion drilling in the Babicanora Area during Phase III focused on the southeast portion of the Babicanora Vein, mainly the delineation of the high-grade Area 51 Zone. This drilling was accessed via a high-elevation road from the ridge crest permitting drill access to the vein from the hanging wall side. Numerous high-grade intercepts were encountered in this area.

Drilling established good lithological control on the upper portion of the Area 51 Zone where welded dacitic-rhyodacitic crystal tuff (RDCLF) overlies a more permeable lapilli or lithic tuff, which is host to the high-grade mineralization. Mineralization transects the contact; however, it is reduced in both thickness and grade due to permeability contrasts between the lithic and welded tuff units. The orientation of this lithological contact appears to be a controlling feature on the southeast-directed plunge of mineralization within the Babicanora Vein. A lower boundary is less defined.

10.2.3.2 Babicanora Footwall (FW) Vein

The Babicanora FW Vein is immediately adjacent to the Babicanora Vein and was discovered at the same time as the Babicanora Vein Area 51 Zone, in late 2017. This vein can be observed on surface in select locations and underground in the Babicanora Central Adit and Santa Rosa Decline.

10.2.3.3 Babicanora Norte Vein

Surface drilling commenced on the Babicanora Norte Vein in March 2018 and the vein was discovered with the second drill hole, BAN18-02. The vein is located near the portal of the Babicanora Central Adit and projects under historical mine dumps. Initial drilling was directed 50 m below a shallow shaft where the high-grade vein was intercepted. After discovery, the Babicanora Norte Vein was systematically drilled to the northwest and southeast along vein strike.

In contrast to the Babicanora Vein, the Babicanora Norte Vein is hosted in welded RDCLF as a discordant extensional vein of consistent width and sharp contacts with host rock. Current interpretation of drilling results has identified a flexure in the Babicanora Norte Vein with change in orientation from 160º degrees azimuth in the northwestern portion to 125° azimuth. This flexure may represent a local fault as noted by displacement of local lithologies.

10.2.3.4 Babicanora Sur Vein

The Babicanora Sur Vein is located approximately 300 m southwest and is oriented roughly parallel to the Babicanora Vein. Drilling commenced on Babicanora Sur in the southwest portion of the Project area based on availability and access of surface drill rigs on roads constructed in the Babicanora Area.

10.2.3.5 Granaditas Vein

The Granaditas Vein is parallel to the Babicanora and Babicanora Norte veins and consists of similar southeastward-plunging high-grade precious metal mineralization, however, copper, lead and zinc grades are increasing to the southeast and down plunge confirming that the zonation of mineralization is originating to the southeast of the Babicanora Area. Drilling during Phase III focused on delineating the high-grade footprint.

10.2.3.6 Luigi Vein

The Luigi Vein was discovered in the footwall of the Las Chispas Vein in mid-2017 but remained unnamed until there was sufficient drilling to delineate an actual mineralization vein. The Phase III program focused on delineating the vein through underground drilling on the 550 and 600 Level of the historical Las Chispas workings.

10.2.3.7 Ranch Veins

Surface drilling commenced in the Babicanora Ranch area during Phase III with 13 holes sited as condemnation holes in the surrounding area for potential processing facilities. Results were low-grade, with several wide intersections of calcite veins and veinlets.

10.2.3.8 Espiritu Santo Vein

The Espiritu Santo historical workings are located to the southeast of the Las Chispas and William Tell Veins. Drilling during Phase III targeted two adits and a shaft in this area with three (3) holes completed. Results were negative and the drilling was considered to be below the potential projected plunge of mineralization.

10.2.4 Phase III Extended

10.2.4.1 General

Expansion and in-fill of the Babicanora and Las Chispas Areas during the Phase III Extended program focused on all the main veins (Babicanora, Babicanora FW, Babicanora Norte, Babicanora Sur, Babi Vista and Las Chispas) with adjacent footwall and hanging wall veins or splays to support potential confidence category upgrades.

A systematic drill hole vein piercing pattern of approximately 35 m by 35 m was used to support Inferred Mineral Resources to the Indicated category.

In August 2020, the gold to silver ratio (Au:Ag) for estimating AgEq was changed to 86.9 (Section 14.1.14) to reflect updates in metal prices and metallurgical recoveries.

10.2.4.2 Babicanora (Main) Vein

Drilling on the Babicanora Vein continued to establish lithological controls on the upper and lower precious metal mineralization boundaries where welded rhyodacitic to dacitic crystal lithic tuff (RDCLF1, Figure 7-2) overlies a more permeable lapilli or lithic tuff (LAT1), which is the main host to high-grade mineralization. Underlaying the lapilli tuff and establishing the lower boundary is a similar welded unit of rhyodacitic tuff (RDCLF2). The downward continued plunge of the in-vein high-grade mineralization appears to be limited based on recent drilling that shows vein intersections with low to no grade. This may indicate a barren zone to the southeast toward the source of mineralization, typical of epithermal systems (Figure 7-9).

The Babicanora Vein has a near-parallel hanging wall and footwall vein.

10.2.4.3 Babicanora Footwall (FW) Vein

The Babicanora FW Vein is immediately adjacent to the Babicanora Vein and infill drilling has better defined the limits of high-grade mineralization showing several semi-continuous zones or pods. These zones are associated with near-vertical crosscutting structures and shallower southeast plunging in-vein mineralization related to the lapilli tuff host as seen associated with Babicanora mineralization similar to the adjacent Babicanora Vein.

10.2.4.4 Babicanora Norte (Main) Vein

After its discovery in March 2018, the Babicanora Norte Vein was systematically infill drilled to the northwest and southeast along the vein strike. Infill drilling has better defined the limits of high-grade mineralization showing several semi-continuous zones or pods similar to the nearby

Babicanora FW Vein; however, the lithological relationship of mineralization in this vein is within the RDCLF2 unit, which is considered to be a less favorable host. The southeast plunge of high-grade mineralization is still present and is considered to be controlled by zonation.

Numerous high-grade in-fill intercepts were including several holes in a newly defined zone named Area 200 (Figure 7-9).

The Babicanora Norte Vein has a near-parallel hanging wall and footwall vein.

10.2.4.5 Babicanora Sur (Main) Vein

The Babi Sur Vein is located approximately 300 m southwest of, and is oriented roughly parallel to, the Babicanora Vein. Infill drilling has better defined the limits of high-grade mineralization showing several semi-continuous zones or pods similar to the nearby Babicanora FW Vein.

The Babi Sur Vein has a near-parallel hanging wall and footwall vein (Figure 7-15).

10.2.4.6 Babi Vista (Main) Vein

The Babi Vista Vein is located approximately 250 m northeast of, and is oriented roughly parallel to, the Babicanora Vein. Drilling commenced on Babi Vista using both underground drilling from access on the Santa Rosa Decline, while developing to the Babicanora Vein, and surface drilling. Discovery and infill drilling defined the limits of high-grade mineralization showing several semi-continuous zones or pods similar to the nearby Babicanora Norte Vein (Figure 7-12).

The Babi Vista Vein has a near-parallel footwall vein.

10.2.4.7 Babi Vista Vein Splay

The Babi Vista Vein Splay is located along the southeast strike of the Babi Vista Vein and appears to be a splay off the main Babi Vista vein. The orientation and zones of mineralization are similar to the Babi Vista Vein. Infill drilling defined the limits of high-grade mineralization showing several semi-continuous zones or pods similar to those encountered at the nearby Babicanora Norte Vein.

10.2.4.8 Las Chispas Vein

The Las Chispas vein is located approximately 1 km northeast of the Babicanora area and is an area of historical mining activity. The Phase III Extended drilling focused on delineating and in-filling the newly defined Area 118 that had no previous mining on the vein. Infill drilling better defined the limits of high-grade mineralization showing that not only the LAT1 lithology is a favourable host but also that the FIAT unit (fluvial andesitic tuff, Figure 7-2) appears to be a stratigraphically lower favourable host (Figure 7-15).

The Las Chispas Vein has several hanging wall and footwall veins including the Giovanni, GioMini, Luigi and Luigi FW. Las Chispas Vein is the dominant vein in this area.

11 Sample Preparation, Analyses, and Security

To date, four types of sample collection programs were conducted on the Project:

- Underground and surface sampling as chip samples and/or channel samples;
- Stockpile/backfill sampling as intact historical muck from draw points and (or) placed or remobilized muck within underground development;
- Drill core sampling as hand-split core or wet saw-cut core; and,
- Surface dump trenching and sampling.

The sample collection approaches being conducted by SilverCrest are described in the following subsections. Silver Crest has established a sample processing facility where core samples are logged, bulk density measurements collected, drill core is photographed, sampled, bagged and tagged, and stored on site prior to being transported to the laboratory by SilverCrest staff. Underground chip samples are bagged and tagged at the point of collection and are also stored at the sample processing facility. All coarse reject materials and pulps, are stored in a covered building.

11.1 Underground Chip Sample Collection Approach

This subsection describes SilverCrest's approach to underground rock sample collection for historical underground workings and newly developed in-vein drifting which consisted of the following steps:

- Underground continuous chip and channel samples were marked by a geologist, per lithology or mineralization contacts, using spray paint prior to sample collection;
- Samples were collected using a small sledgehammer, a hand maul/chisel, and a small tarp on the floor to collect the chips, or a power saw for channeling;
- Samples were collected and placed into clear plastic sample bags with a sample tab, secured with a zip tie, labelled, and stored in the semi-secure core storage facility at Las Chispas prior to being transported to ALS Chemex Hermosillo or the Bureau Veritas Minerals Laboratories (Bureau Veritas), also located in Hermosillo;
- Samples were collected along development ribs as longitudinal samples, along backs and overhead stope pillars as transverse samples, and along some crosscuts as transverse samples. The SilverCrest collection program was eventually modified to allow identification of each sample type in the geological database;
- For the historical workings, SilverCrest initiated a follow-up program to collect duplicate and new samples using a power saw to cut a channel along the initial chip path. Saw-cut samples were collected at approximately every five to eight samples, depending on access;
- Each sample path was labelled with a sample number written on a piece of flagging tape, which was anchored to the development wall; and,

- SilverCrest's senior geologist and exploration manager conducted a follow-up review of the sampling program to ensure that all development drifts near the mineralization zone were sampled, that transverse samples were properly collected across veins, and that the samples were clearly and properly labelled.

11.2 Underground Muck/Stockpile Sample Collection Approach

Underground muck and/or stockpile sample collection steps consisted of:

- Samples were collected at random within the existing historical muck and material stockpiles in the Las Chispas, William Tell, and Babicanora workings;
- The average mass of the samples collected was approximately 4 kg;
- Sample spacing along continuous muck piles was approximately 10 m, suggesting that each sample could represent approximately 20 t to 40 t of material, depending on the size of the pile;
- Sample collection was completed by hand or shovel, from near surface material, as non-selective collection to represent both the fine and coarse fragment portions of the muck piles;
- The muck samples were then collected and placed into clear plastic sample bags with a sample tab, secured with a zip tie, labelled, and stored in the semi-secure core storage facility at Las Chispas prior to being transported to ALS Chemex, Hermosillo; and,
- SilverCrest's senior geologist and vice president of exploration and technical services conducted a follow-up review of the sampling program to ensure that all appropriate muck piles were sampled, and that the samples were clearly and properly labelled.

11.3 Drill Core Sample Collection Approach

This subsection describes SilverCrest's approach to drill core sample collection which consisted of the following steps:

- Project geologists logged the drill holes, and the senior geologist reviewed the logs;
- For a newly discovered vein, the first 10 drill holes were completely sampled. Additional drill holes could be entirely sampled, if such sampling was needed to establish a better understanding of geology and mineralization;
- Sample intervals were laid out for mineralization, veining, and structure. Approximately 10 m before and after each mineralization zone was included in the sampling intervals. A minimum of 0.5 m sample lengths of mineralized material was taken up to a maximum of 3 m in non-mineralization rock;
- Each sample interval was either split using a hand splitter or cut using a wet core saw, perpendicular to veining, where possible, to leave representative core in the box and to reduce any potential bias in the sampled mineralization submitted with the sample;
- Half of the core was placed into clear plastic sample bags with a sample tab, secured with a zip tie, labelled, and stored in the semi-secure core storage facility at Las Chispas before being transported to ALS Chemex Hermosillo; and,
- SilverCrest's senior geologist and vice president of exploration and technical services conducted a follow-up review of the core sampling program to ensure that each core sample was properly split/cut, that the sample intervals were clearly marked, that

representative core samples remain in the core box, and that sample tags were stapled to the core boxes in sequential order.

11.4 Bulk Density Determinations

A total of 641 bulk density measurements were collected on site by SilverCrest using the water immersion method. Core fragments greater than 5 cm in length were dried and weighed prior to being suspended and submerged from a scale in a bucket of water using a wire basket. The measurements tested various mineralized and unmineralized material types at approximately 20 m downhole intervals. Where rock material was highly fragmented or strongly clay altered, samples were not collected. The bulk density ranged from 1.53–4.02 t/m^3 with a mean value of 2.52 t/m^3.

Seventy-two (72) samples were tested by ALS Chemex. Hermosillo for wax-coated bulk density to validate the on-site measurements. The samples were collected from non-mineralization hanging wall and footwall materials, and mineralized material free of clay alteration. The overall average bulk density was 2.50 t/m^3, with 2.50 t/m^3 and 2.49 t/m^3 for Las Chispas Area and Babicanora Area, respectively.

In November 2018, two (2) samples were collected and sent by SilverCrest to Geotecnia del Noroeste S.A. de C.V. based in Hermosillo, Sonora, for wax coated dry bulk density testing. Each sample was split into two (2) subsamples. The measured values ranged from 2.48 t/m^3 to 2.60 t/m^3, with an average dry bulk density of 2.56 t/m^3.

A uniform mean bulk density of 2.55 t/m^3 was applied to all rock types in the Mineral Resource estimate based on the results of the bulk density test work completed above by SilverCrest and two laboratories.

11.5 Sample Analytical Methods

All assays were completed by ALS Chemex Hermosillo, ALS Chemex Vancouver, BC, Canada, and Bureau Veritas, Hermosillo.

ALS Chemex has developed and implemented strategically designed processes and a global quality management system at each of its locations that meets all requirements of International Standards ISO/IEC 17025:2017 and ISO 9001:2015. All ALS geochemical hub laboratories are accredited to ISO/IEC 17025:2017 for specific analytical procedures.

Bureau Veritas is a leading provider of laboratory testing, inspection, and certification, operating in 1,430 offices and laboratories in 140 countries. Bureau Veritas is ISO 9001 compliant and for selected methods, ISO 17025 compliant and has an extensive Quality Assurance/Quality Control ("QA/QC" or "QC") program to ensure that clients receive consistently high-quality data.

Both ALS Chemex and Bureau Veritas are independent of SilverCrest.

SilverCrest personnel delivered all of the samples collected from the Las Chispas site to either ALS Chemex, Hermosillo or Bureau Veritas, Hermosillo. Standard analytical procedures were as follows:

- All samples were received, registered, and dried;
- All samples were crushed to 75% (ALS Chemex, Hermosillo) or 70% (Bureau Veritas) less than 2 mm, then mixed and split with a riffle splitter;

- A split from all samples were then pulverized to 80% (ALS Chemex, Hermosillo) or 85% (Bureau Veritas) less than 75 µm;
- All pulverized splits were submitted for multi-element aqua regia digestion with inductively coupled plasma (ICP)-mass spectrometry (MS) detection; and,
- All pulverized splits were submitted for gold fire assay fusion with atomic absorption spectroscopy (AAS) detection (30 g).

Silver analyses were conducted per the following criteria:

- Samples returning grades above the upper detection limit of greater than 100 gpt silver from ICP analysis were re-run using aqua regia digestion and ICP-atomic emission spectroscopy (AES) detection, and diluted to account for grade detection limits (less than 1,500 gpt); and,
- Where silver grades were 1,500 gpt Ag, the sample was re-run using fire assay fusion (FA) with gravimetric detection.

Gold analyses were conducted per the following criteria:

- During Phase I (March 2016 to October 2016) all samples were analyzed for gold by 30 g fire assay with AAS detection;
- During Phase II (October 2016 to February 2018) samples were analyzed by ICP-MS. Where gold values were greater than 1 gpt Au, the samples were re-run using FA with gravimetric detection, and where gold values were greater than 10 gpt Au, the samples were re-run using 30 g FA with AAS detection;
- During Phase III (February 2018 to February 2019), selective metallic screen analysis was completed at SGS Durango;
- During Extended Phase III (February 2019 to October 2020) gold and silver were analyzed using 30 g FA with gravimetric finish; and,
- Samples returning grades of greater than 10,000 ppm of zinc, lead, or copper from ICP-MS analysis were re-run using aqua regia digestion with ICP-AES finish.

11.6 SilverCrest Internal QA/QC Approach

Descriptions of the QA/QC protocol for Phases I through III programs (Sections 11.6.1 through 11.6.3) have been summarized from Barr (2018) and Barr and Huang (2019). The QP has reviewed the QA/QC data for all three phases of drilling and concurs with the following assessment.

11.6.1 Phase I QA/QC Program

At the exploration stage, SilverCrest implemented a program of certified reference material (CRM or standards), blank sample insertions for all sample types being collected, and duplicate samples for some underground chip samples.

For review and assurance of analytical accuracy in the lab, insertion of CRMs is made at an interval of 1:50. The CRMs being used by SilverCrest alternate between CDN-ME-1312 and CDN-ME-1409. A total of 99 CRM samples were reviewed by the QP, as well as a scatter plot showing the analytical results for the CRMs and in relation to their referenced error of two standard deviations.

For monitoring of in situ contamination or contamination of sample crushing, grinding and sorting equipment, SilverCrest inserted a benign rock sample at an interval of 1:50. The material used for blanks was collected from a nearby silica cap. A total of 101 blank insertions were noted in the database was reviewed by the QP. Of these, only one is located adjacent to a sample with >50 gpt Ag.

11.6.2 Phase II QA/QC Program

During the Phase II program, SilverCrest implemented a program of CRM, blank sample insertions for all sample types being collected, and duplicate samples for some underground chip samples, core pulps and coarse rejects.

11.6.2.1 Certified Reference Standards

Commercial standards in 1 kg plastic bottles were sourced from CDN Resource Laboratories Ltd. (CDN Labs). The CRM material is selected to contain Ag/Au grades and a matrix that is consistent with the grades of the known mineralization and with a similar host rock lithology to the host rocks. At the Project's core logging facility, 70 g of the reference material is weighed, placed in a paper envelope, and added to the sample stream as directed by the field geologists. Insertion frequency of the standards is approximately 1:50 samples.

A total of 612 standards were inserted into the sample stream during this phase of drilling. Each standard and corresponding sample number was recorded in a QA/QC sample tracking spreadsheet.

A CRM failure is defined by receipt of a standard greater than three standard deviations above or below the expected value in either gold or silver. In cases where the standard failures occurred in "non-mineralization" rock (generally in zones returning <0.1 gpt Au or <5 gpt Ag), no action is taken.

The protocol for re-assaying the standard failures is to re-analyse the pulps within a range of 10 samples above and 10 samples below the failed standard.

11.6.2.2 Assay confirmation and re-analysis

Assessment of the CRM performance concluded that CDN-ME-1301 had a significant number of failures (47%). While CDN-ME-1505 and CDN-ME-1312 had relatively fewer failures. CDN-ME-1601 had a high failure rate but with a statistically insignificant population. A total of 16 batches (including 306 samples) that were identified as having potential error due to performance of the CRMs CDN-ME-1301 and CDN-ME-1505 were re-submitted to ALS using coarse reject materials. New reference standard sample material was added at the lab by a SilverCrest geologist. At the time of the re-runs, CDN Labs Ltd., had run out of CDN-ME-1301, so CDN-ME-1601 was used instead, as the Ag/Au values are similar.

Comparison of the re-analyses and the original gold and silver assay grades were completed using a relative percent difference (RPD) and scatterplot approach. This approach was selected to assess whether the assays reproduced with a reliable precision and to identify whether high RPD values were associated with high- or low-grade ranges. Since the re-analyses were conducted using coarse reject material, the expected performance threshold would be 90% of the samples with less than 20% RPD. The results of the analysis indicate that approximately 81% of silver assays and 63% of gold assays reproduced with less than 20% RPD. It is noted however, that the sample pairs with anomalous RPD values are in the low-grade range where Ag <45 gpt and Au <0.65 gpt.

A total of 16 CRMs were inserted with the batch re-analyses. CRM performance results show that there were no failures with CRM ME-1505 and 40% failure rate of CRM ME-1601, consistent with the original analyses. The results suggest that analytical method during the fire assay fluxing process may affect the results, however, the likely error may be produced during the manual preparation of the CRM for sample insertion. To resolve the homogeneity issue, the standards are now ordered pre-packaged.

11.6.2.3 Blanks

For monitoring of in situ contamination or contamination of sample crushing, grinding and sorting equipment, SilverCrest is inserting ~1 kg of non-mineralization rock samples at a sample interval of 1:50. The material being used for blanks is collected from a nearby silica cap and the particle size is >2 cm. Blanks were inserted both randomly and at the end of suspected mineralization intervals to check for contamination carry-over between samples.

The failure threshold for the blanks is five (5) times the detection limits of the analytical method, 25 0.25 gpt Au and gpt Ag, for fire assay (gravimetric).

A total of 555 blank samples were inserted during the drill program. Blank failures are indicated by returning results greater than five (5) times the detection limit. No evidence of sample contamination was observed upon review of the analytical results.

11.6.2.4 Duplicate Program

A total of 126 duplicate samples were collected for assessment of sample analytical precision. The samples were collected from drill core (n = 38), underground channel samples (n = 56) and surface stockpile samples (n = 32). Sample rejects were first homogenized, and a subsample was prepared from a 250 g split. This subsample split was then pulverized. Both pulp duplicates (analytical duplicates) were then selected from this split. This method of duplicate preparation allows for assessment of sample preparation at the reject stage (comparison of original assay with assay from the new coarse reject split), in addition to assessment of sample preparation at the pulp stage by comparing the two new pulp splits.

The duplicate sample pairs were assessed using RPD and scatterplot methods. This approach was selected to assess whether the assays reproduced with a reliable precision and to identify whether high RPD values were associated with high- or low-grade ranges. The expected performance threshold for duplicate re-analysis using coarse reject material would be 90% of the samples with less than 20% RPD, and for pulp materials would be 90% of the samples with less than 10% RPD. The results of the analysis indicate that Ag duplicate analysis reproduce successfully above the 90% threshold for both coarse reject and pulp samples. Where the Au duplicate analysis indicated that only 81% of the coarse rejects had RPD of <20%, and only 63% of pulps had <10% which did not meet the expected threshold. Sample pairs with anomalous RPD values are in the low-grade range, where Ag is <45 gpt and Au <0.65 gpt.

11.6.3 Phase III QA/QC Program

11.6.3.1 Certified Reference Standards

Commercial standards in 1 kg plastic bottles were sourced from CDN Labs. The CRM was selected to contain gold/silver grades, a matrix consistent with the grades of the known mineralization, and a similar host rock lithology to the host rocks. At the Project's core logging facility, approximately 100 g of reference material is weighed, placed in a paper envelope, and added to the sample stream as directed by the field geologists. These samples are used to test the precision and accuracy of both gold and silver assays and to monitor the consistency of the laboratory's performance. Insertion frequency of the standards is approximately one in every 50 samples (2.9%).

A total of 389 standards were inserted into the sample stream during this phase of drilling. Each standard and corresponding sample number was recorded in a QA/QC sample tracking spreadsheet. Standard results greater than two standard deviation (SD) and less than three SD are flagged as cautionary for review.

A CRM failure is defined by receipt of analytical results for a standard that is greater than three standard deviations above or below the expected value in either gold or silver. The protocol for re-assaying the standard failures is to re-analyse the pulps within a range of 10 samples above and 10 samples below the failed standard. In cases where the standard failures occurred in a batch of samples comprised of "non-mineralization" rock (generally in zones returning less than 0.1 gpt gold or less than 5 ppm silver), no action is taken.

Assessment of the CRM performance concluded that CDN-ME-1601 had a significant number of failures (33.3% in silver and 25% in gold, respectively) whereas CDN-ME-1505 was better (11.1% for silver, 0% for gold). Both standards were used infrequently (combined only 31 samples, or 8% of standard insertions); however, provided insufficient data to properly validate overall standard performance. Use of the CRM CDN-ME-1601 was discontinued.

Standard CDN-GS-P4A was the primary standard used during the Phase III drill program. This standard had a failure rate of 14.2% for gold and 1.1% for silver. This is a high failure rate for gold that should be investigated further.

SilverCrest purchases its standards in 1 kg plastic bottles and individual standard packages are prepared on site. This leads to a variety of potential issues with standard performance, including contamination of the standard from dust in the air, contamination from a scoop that is not properly cleaned between samples, and a loss of homogeneity from sample settling within the bottle (especially with regard to gold). Purchasing pre-packaged 100 g standards from the standard providing laboratory would help resolve these issues.

It is also worth noting that the gold value of CDN-GS-P4A is 0.738 gpt, which is much lower than the average grade of mineralized material at Las Chispas. Using multiple standards covering a range of gold values, including overlimit values, would provide a more robust QA/QC database.

11.6.3.2 Blanks

To monitor for contamination or contamination of sample crushing, grinding, and sorting equipment, SilverCrest inserted a benign rock sample at an interval of one for every 20 samples. The material used for blanks was collected from a nearby silica cap. A total of 644 blank insertions were noted in the database reviewed by the QP.

The failure threshold for the blanks is five (5) times the detection limits of the analytical equipment: .25 gpt gold and 25 gpt silver 0 for the fire assay (gravimetric) method and 1 gpt silver for the aqua regia (ICP) method. No contamination was identified in the fire assay stream, for high-grade analysis (one gold sample returned a value of 0.23 ppm; however, the previous sample was below the detection limit, therefore contamination was not a factor).

Minor contamination could have been observed in the ICP silver analytical stream, where five (5) of the six (6) failing blanks followed high-grade silver samples; however, the overall failure rate of 1.4% is not considered to indicate any systematic contamination issues.

11.6.3.3 Duplicate Program

A routine duplicate sampling program has not been conducted as part of the Phase III program.

11.6.4 Extended Phase III QA/QC Program

11.6.4.1 Certified Reference Materials

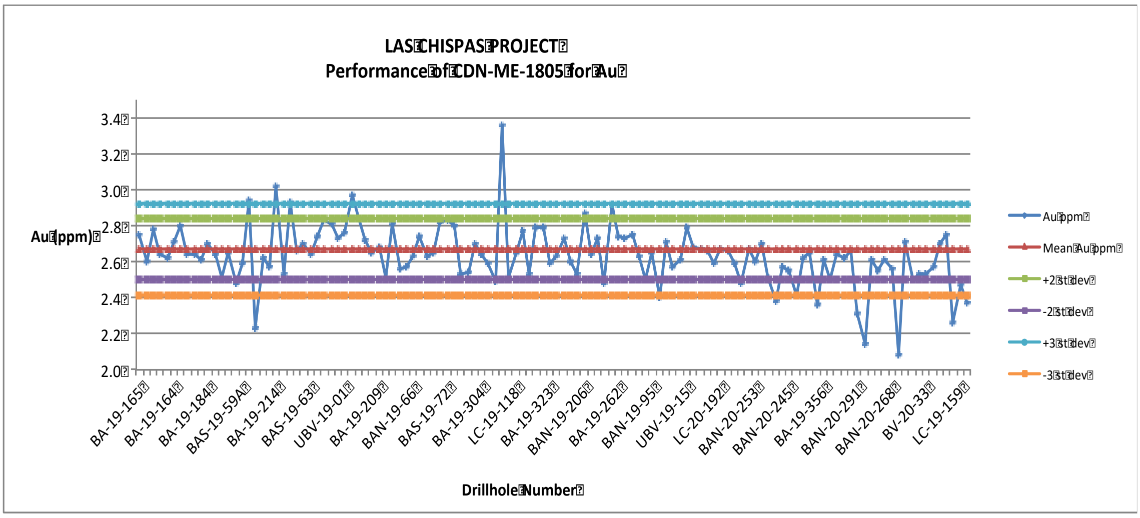
CRMs in 1 kg plastic bottles was sourced from CDN Labs. The CRMs were selected to contain silver/gold grades, a matrix consistent with the grades of the known mineralization, and a similar host rock lithology to the host rocks in the Project area. At the drill core logging facility, approximately 100 g of a CRM was weighed, placed in a paper envelope, and added to the sample stream as directed by the field geologists. These samples were used to test the precision and accuracy of both gold and silver assays and to monitor the consistency of the laboratory's performance. The CRM insertion frequency was approximately one to every 43 samples (2.3%).

A total of 81,262 drill core samples were analyzed during the Extended Phase III program. A total of 1,869 CRMs were inserted into the sample stream during this phase of drilling. Each CRM and corresponding sample number was recorded in a QA/QC sample tracking spreadsheet. Results greater than two (2) SD and less than three (3) SD were flagged as cautionary for review.

A CRM failure was defined by receipt of analytical results for a CRM that was greater than three (3) SD above or below the expected value in either gold or silver. The protocol for re-assaying the CRM failures was to re-analyse the pulps within a range of 10 samples above and 10 samples below the failed CRM. In cases where the CRM failures occurred in a batch of samples consisting of "non-mineralization" rock (generally in zones returning less than 0.1 gpt Au or less than 5 gpt Ag), no action was taken. Table 11-1 shows the CRMs expected values and failure rates. Figure 11-1 through Figure 11-8 chart the results of the CRM performance analysis for sampling conducted during the Extended Phase III program since February 2019.

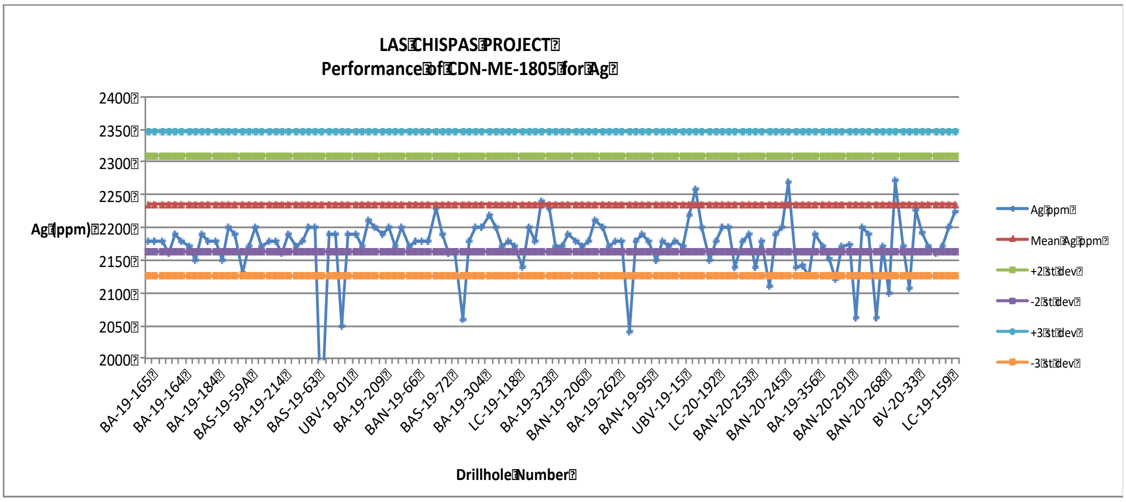
Table 11-1: Standards Expected Au and Ag Values and the Failure Rates for the Extended Phase III Drill Program

Standards	Expected Au Values, ± 3SD (gpt)	Expected Ag Values, ± 3SD (gpt)	Sent	Au Failures (%)	Ag Failures (%)
CDN-ME-1805	2.67, ±0.255	2236, ±111	122	8	7
CDN-ME-1806	3.425, ±0.360	365, ±30	202	2	3
CDN-ME-1901	7.74, ±0.975	371, ±27	1287	1	4
CDN-GS-P6A	0.738, ±0.084	81, ±10.50	258	11	0



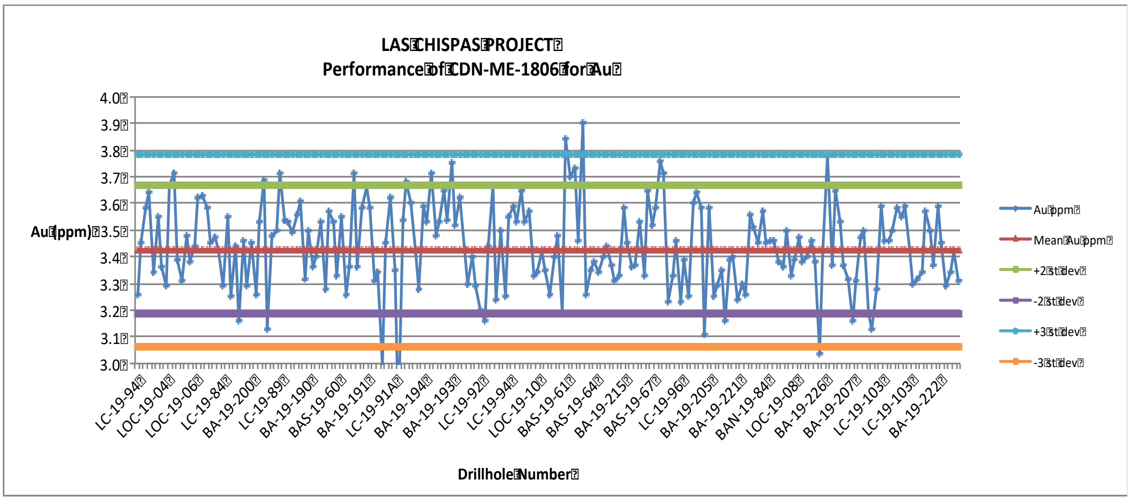
Note: Figure prepared by P&E, 2020.

Figure 11-1: CRM CDN-ME 18015 Analysis, Gold



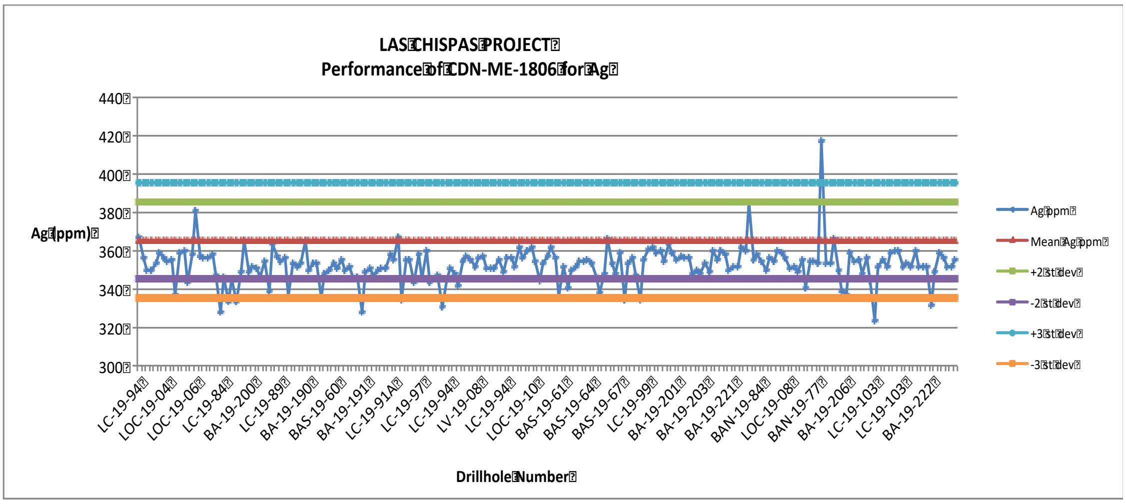
Note: Figure prepared by P&E, 2020.

Figure 11-2: CRM CDN-ME-1805 Analysis, Silver



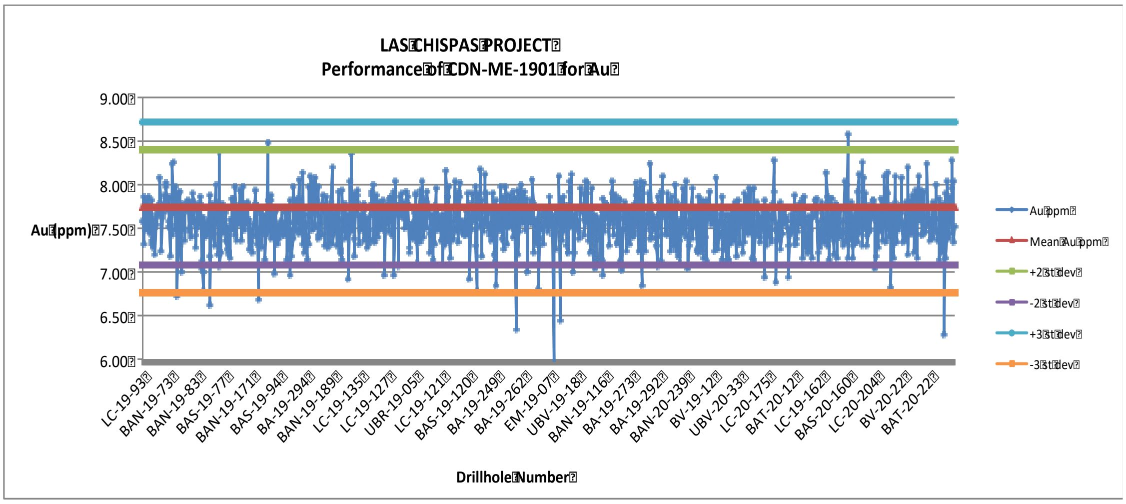
Note: Figure prepared by P&E, 2020.

Figure 11-3: CRM CDN-ME 1806 Analysis, Gold



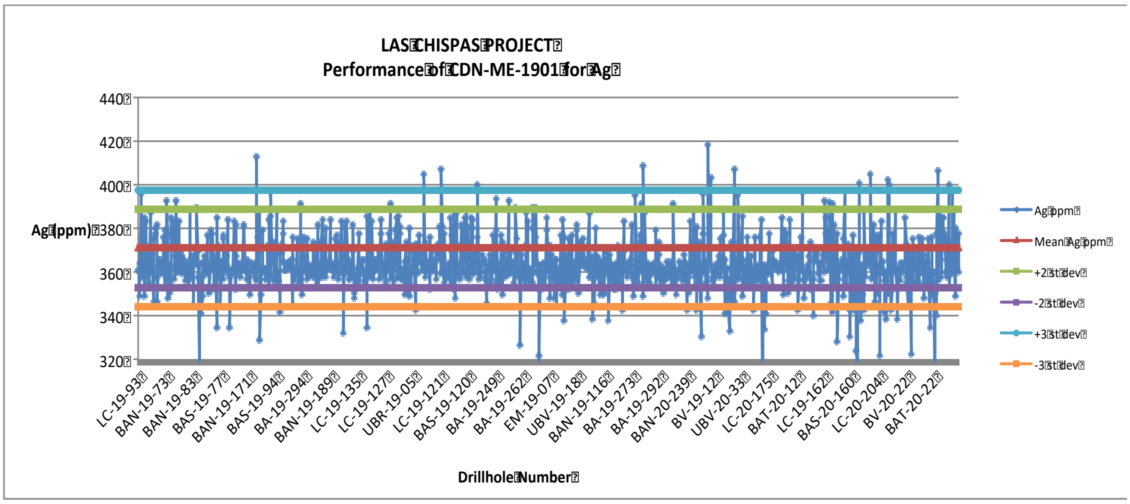
Note: Figure prepared by P&E, 2020.

Figure 11-4: CRM CDN-ME-1806 Analysis, Silver



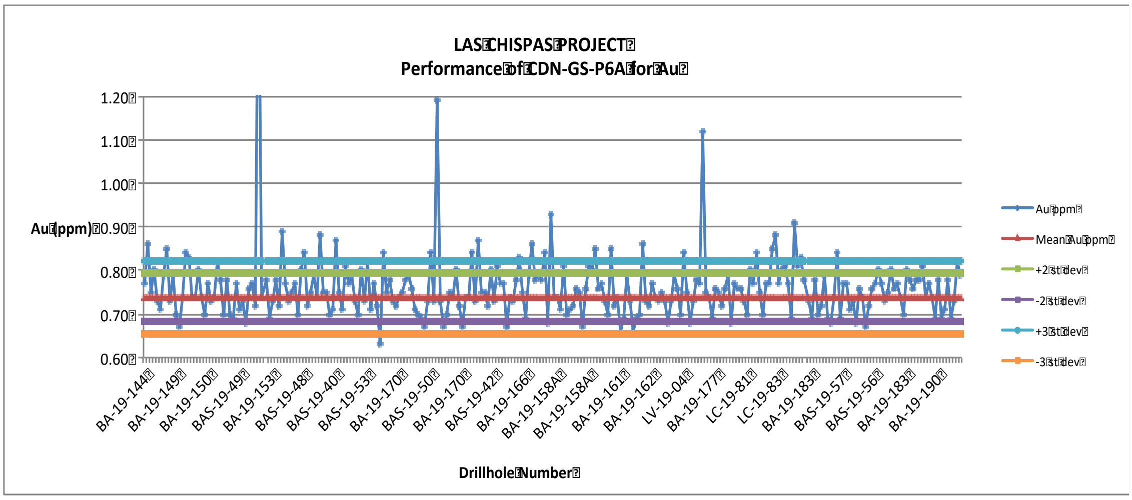
Note: Figure prepared by P&E, 2020.

Figure 11-5: CRM STD CDN-ME 1901Analysis, Gold



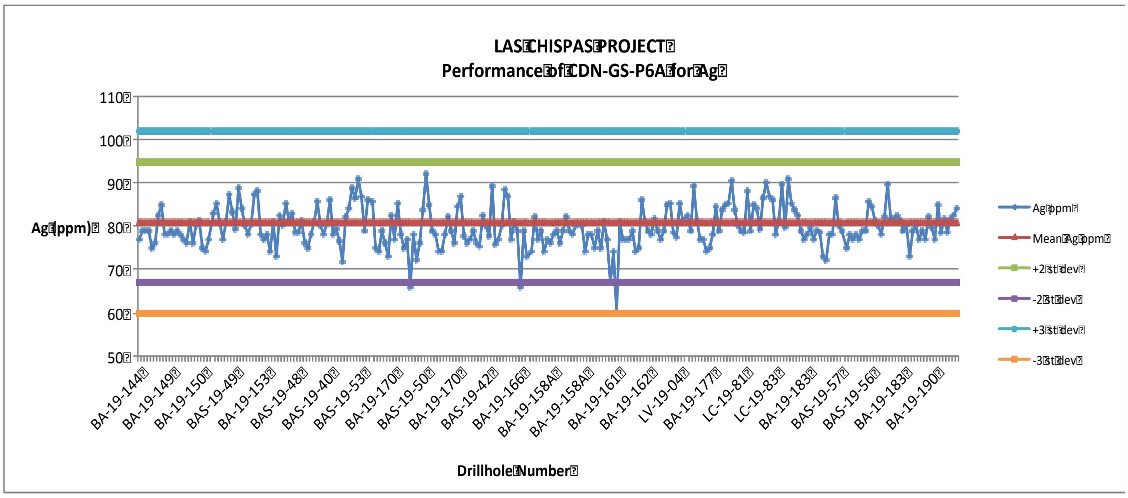
Source: Figure prepared by P&E, 2020.

Figure 11-6: CRM CDN-ME 1901 Analysis, Silver



Note: Figure prepared by P&E, 2020.

Figure 11-7: CRM STD CDN-GS-P6A Analysis, Gold



Note: Figure prepared by P&E, 2020.

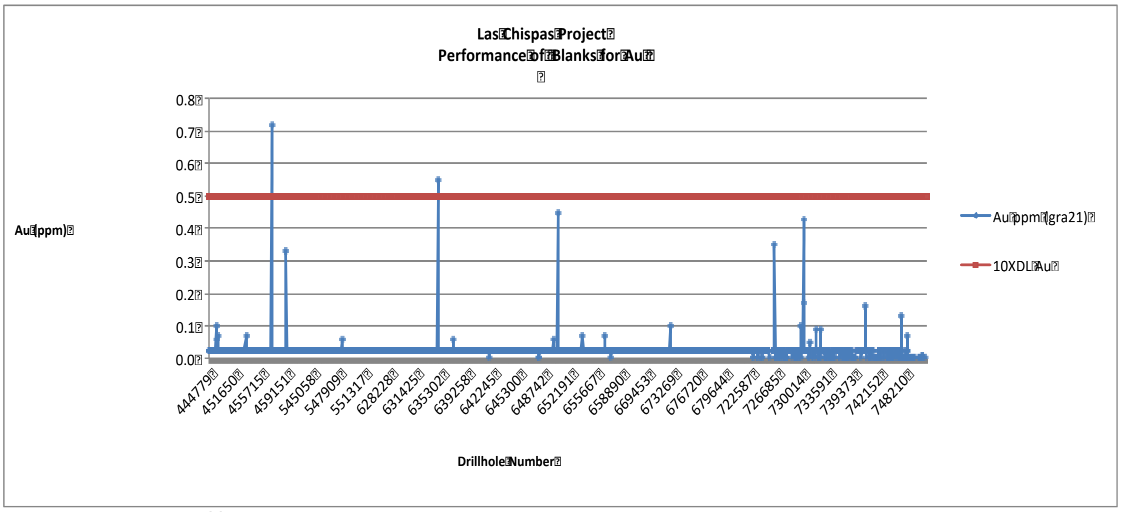
Figure 11-8: CRM STD CDN-GS-P6A Analysis, Silver

SilverCrest uses multiple standards covering a range of gold and silver grades, including overlimit values, which allows for a more robust assessment of the QA/QC database. Assessment of the CRM performance concluded that all four (4) CRMs performed reasonably with an overall failure rate of approximately 3% for both gold and silver. CRMs with the most failures were CDN-ME-1805 for both gold and silver and CDN-GS-P6A for gold only (refer to Table 11-1). Both CRMs were used infrequently. CDN-ME-1901, the primary CRM used throughout Extended Phase III, comprised 69% of the CRM data and returned very few failures (failure rate of 1% for gold and 4% for silver).

11.6.4.2 Blanks

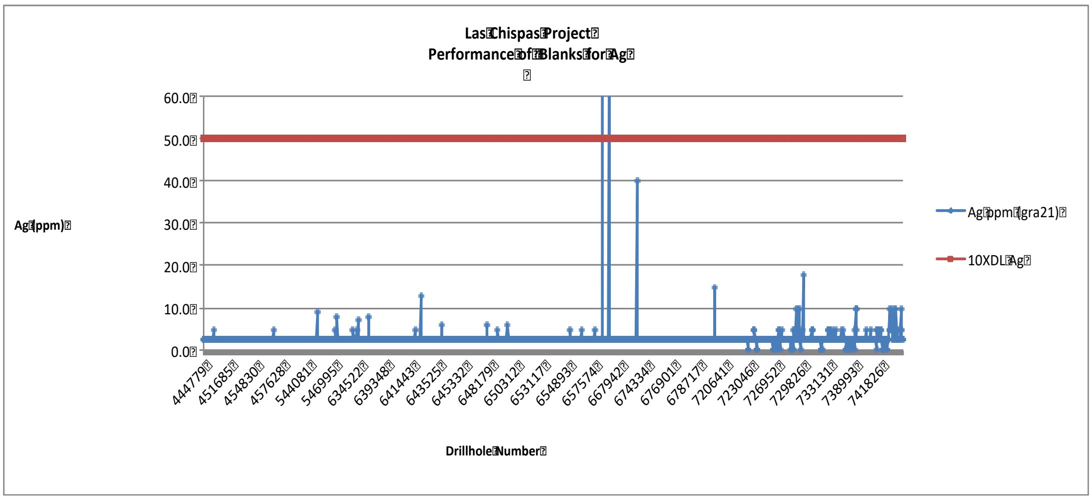
To monitor for contamination or contamination of sample crushing, grinding, and sorting equipment, SilverCrest inserted a non-mineralization rock sample at an interval of one for every 19 samples (5.3%). The material used for blanks was collected from a nearby silica cap. Figure 11-9 and Figure 11-10 show the analytical results for the blank samples. A total of 4,344 blank insertions were noted in the database the QP reviewed.

The failure threshold for the blanks is 10 times the detection limits of the analytical equipment: 50 gpt Ag and 0. 5 gpt Au for the FA (gravimetric) method. No material contamination was identified in the FA stream, and the QP does not consider contamination to be a concern for the Mineral Resource Estimate.



Note: Figure prepared by P&E, 2020.

Figure 11-9: Analytical Results for Gold Grades from QA/QC Blank Sample Insertions



Note: Figure prepared by P&E, 2020.

Figure 11-10: Analytical Results for Silver Grades from QA/QC Blank Sample Insertions

11.6.4.3 Duplicate Program

A routine duplicate sampling program was not conducted as part of the Extended Phase III program.

11.7 QP Opinion on Sample Preparation, Analysis and Security

The sample preparation, analysis, and security program implemented by SilverCrest was designed with the intent to support collection of a large volume of data. Sample collection and handling routines were well documented. The laboratory analytical methods, detection limits, and grade assay limits are suited to the style, grade and distribution of mineralization.

The QA/QC methods implemented by SilverCrest enabled assessment of sample security, assay accuracy, and potential for contamination. There were no duplicate data available to assess the precision of the most recent phase of drilling and the QP recommends a review of the ALS Chemex Vancouver duplicate data for Extended Phase III to be undertaken. There were no other significant concerns related to the integrity of sample collection and analysis.

The QP reviewed sample collection and handling procedures, laboratory analytical methods, QA/QC methods, and QA/QC program results and believes these methods are adequate to support the current Mineral Resource estimate.

12 Data verification

12.1 Database Verification

SilverCrest has developed an extensive dataset for the Project that is stored and managed using a Geospark TM database. The QP has reviewed the data compilation and has audited the GeosparkTM database.

P&E conducted verification of the Las Chispas databases for gold and silver by comparison of the database entries with assay certificates in comma-separated values (csv) file format, obtained directly from ALS WebtrieveTM by P&E.

Assay data were verified for five separate datasets: Las Chispas, Las Chispas Underground, Babicanora Underground, William Tell Underground and Babi Vista.

Assessment of the Las Chispas data was carried out on the constrained data only and involved verification of 95% (4,440 out of 4,662 samples) of the constrained data. A total of four discrepancies were observed.

Assessment of the Las Chispas Underground data was also undertaken on the constrained data only, with verification of 42% (2,821 of 3,884 samples) of the data achieved. No discrepancies were encountered.

Babicanora Underground constrained data involved verification of 13% (132 out of 1,011 samples) of the data and three discrepancies were encountered.

All data for the William Tell Underground data set were included in the verification data, with a total of 42% (128 out of 305 samples) of the data verified. No errors were encountered during the verification process.

Evaluation of the Babi Vista data was undertaken on the constrained data only and comprised verification of 98% (189 out of 192 samples) of the data, with no errors encountered.

The QP believes the databases provided by SilverCrest to be reliable and does not consider the few minor discrepancies encountered during the verification process to be of material impact to the resource data.

12.2 P&E Site Visit and Independent Sampling

Mr. Andrew Turner, P. Geol., of APEX Geoscience Ltd., of Edmonton, Alberta, under subcontract to P&E conducted a site visit to the Las Chispas Property on November 3 through 5, 2020, during which time he received an overview of the Project from SilverCrest geologist, Mr. Ruben Molina. The site visit comprised of:

- Site tour, including a tour of the current underground development within the Babicanora Main Vein (1096, 1066 and 1051 levels);
- Location of several drill pads via GPS; and,
- Discussion of detailed core handling and data management procedures and protocols;
- Completion of drill core sampling from pre-selected intervals.

The details of the Project's exploration procedures and protocols were discussed with Mr. Molina and Mr. Turner found them to meet or exceed industry standard practices.

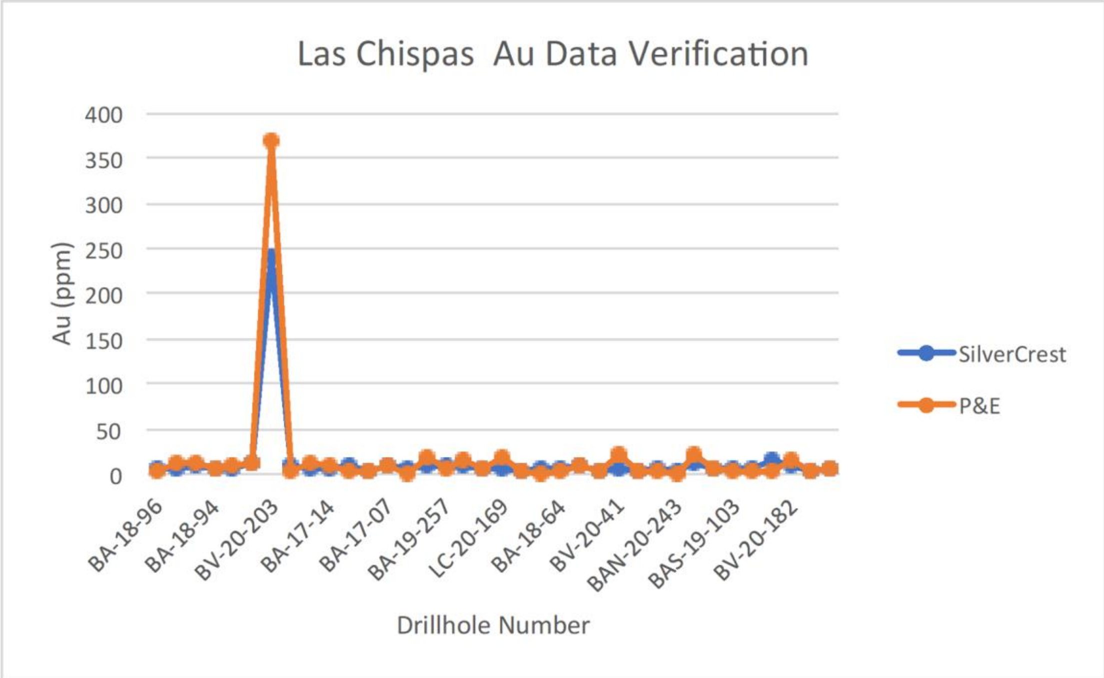
During the site visit, Mr. Turner collected 36 samples from 36 Las Chispas drill holes that were drilled between 2017 and 2020. SilverCrest was provided with a list of preliminary prospective core intervals for the due diligence sampling in advance and a pile of core boxes comprising these intervals was prepared next to the office at the core area. Mr. Turner completed sample collection with assistance from Mr. Molina. Almost all of the sampled intervals, with the exception of one or two more heavily oxidized intervals, were observed to contain quartz veining with variable amounts of grey-black, fine grained disseminated (sooty) to massive (patchy) silver sulphide mineralization, +/- minor amounts of pyrite and chalcopyrite.

The due diligence sampling process involved the quartering of the previously cut/split core. A cut line was marked on each piece of core within each selected sample interval and the box was then taken from the sampling area into the adjacent core cutting area by the cutting assistant and was then returned for sampling. Mr. Turner personally observed the cutting of many of the intervals and observed no irregularities. One half of the quartered core was then placed into a sample bag by Mr. Turner, along with a Tyvek ALS tag, the bag having been marked on both sides with its respective sample number, which was then sealed with a zip tie. At the end of the process, 39 samples (36 core + 3 QC samples) were placed in 3 rice bags, also sealed with zip ties, and were set aside in a secure area. At the end of the visit, on November 5, the samples were driven to Hermosillo and Mr. Turner personally delivered them to personnel at the ALS Minerals (ALS) facility in Hermosillo.

ALS is an independent laboratory that has developed and implemented strategically designed processes and a global quality management system at each of its locations, that meets the requirements of ISO/IEC 17025:2017. All ALS geochemical hub laboratories are accredited to ISO/IEC 17025:2017 for specific analytical procedures.

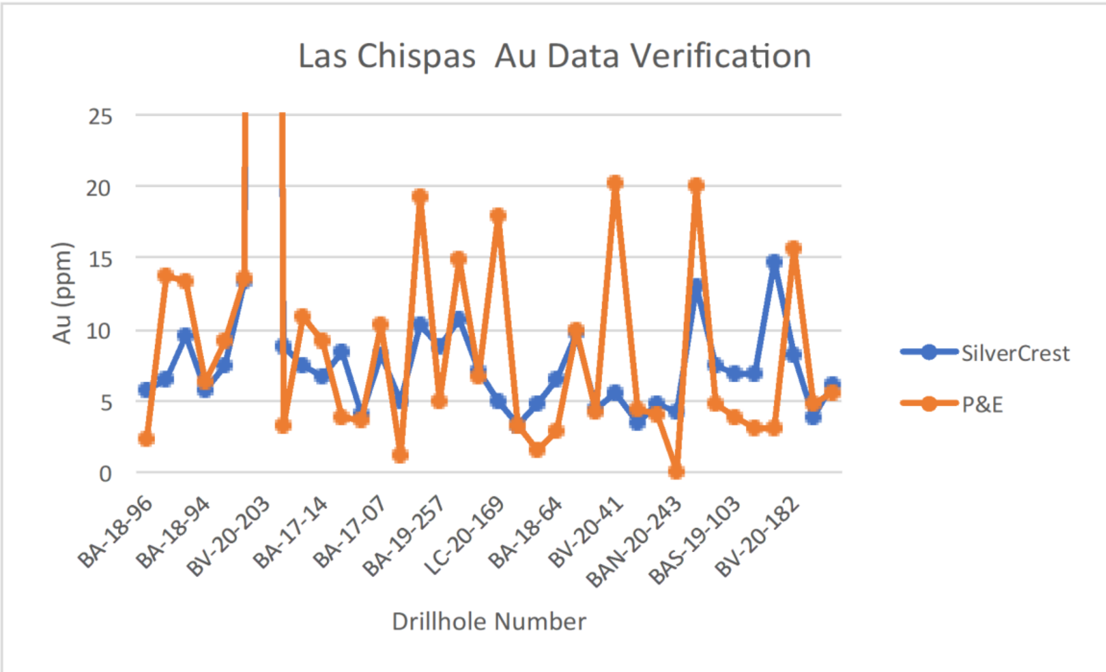
Gold and silver were analyzed using fire assay with gravimetric finish and bulk density by the water displacement method.

Las Chispas site visit sample results are presented in Figure 12-1 to Figure 12-4.



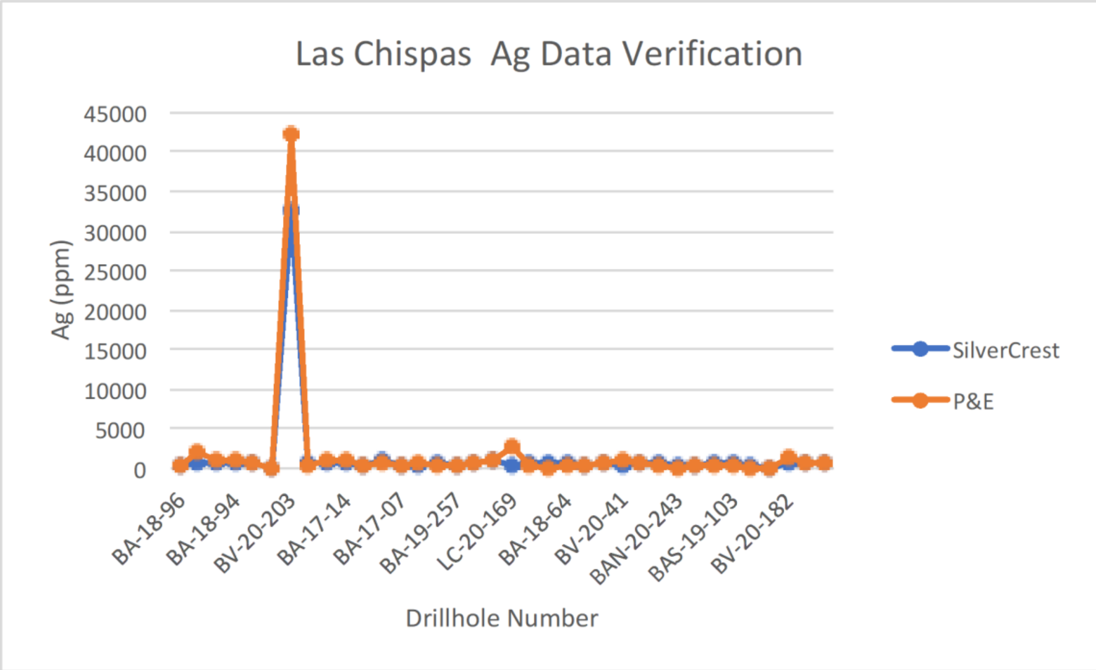
Note: Figure prepared by P&E, 2020.

Figure 12-1: November 2020 Site Visit Sample Comparison for Gold



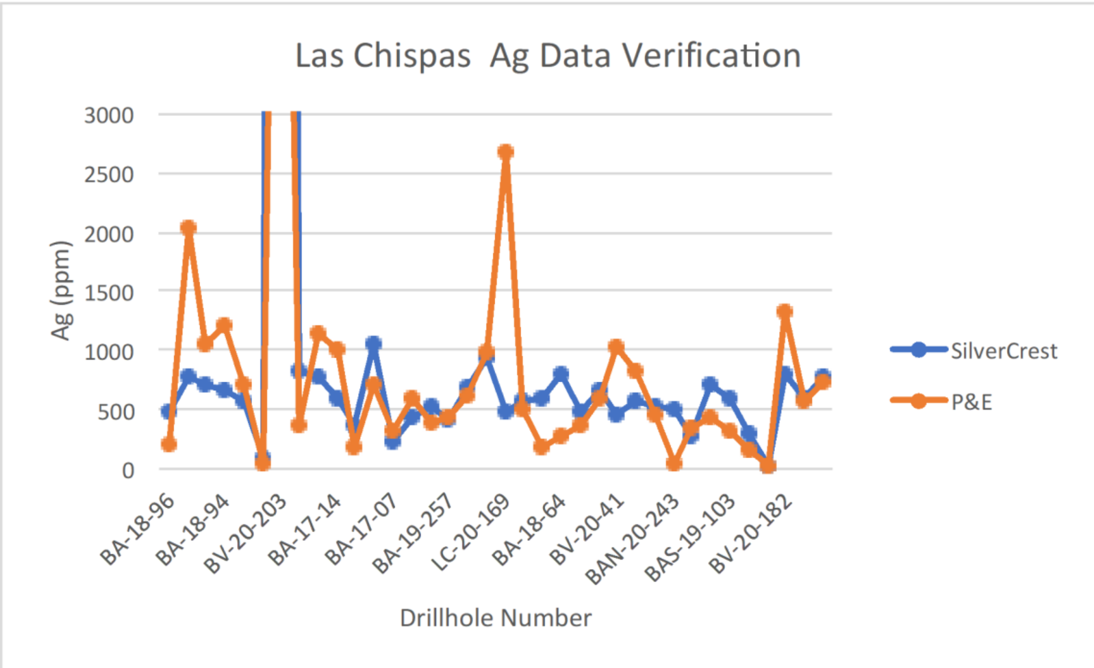
Note: Figure prepared by P&E, 2020.

Figure 12-2: November 2020 Site Visit Sample Comparison for Gold (Zoom to Lower-Grade Samples)



Note: Figure prepared by P&E, 2020.

Figure 12-3: November 2020 Site Visit Sample Comparison for Silver



Note: Figure prepared by P&E, 2020.

Figure 12-4: November 2020 Site Visit Sample Comparison for Silver (Zoom to Lower-Grade Samples)

12.3 Comments

The independent site visit samples match closely to the SilverCrest's data for both gold and silver and the QP considers the due diligence results to be acceptable.

Based upon the evaluation of the QA/QC program undertaken by SilverCrest, as well as P&E's due diligence sampling and database verification, it is the QP's opinion that the data are robust and suitable for use in the current Mineral Resource estimate.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

Two metallurgical testwork programs were undertaken in August 2017 and November 2018 in support of the previous evaluations of the Project prior to start of the Feasibility Study. Both programs were completed at SGS Durango facilities in Mexico.

A series of laboratory testwork completed in 2017 focused on direct cyanide leaching of three (3) mineralization samples including oxide and sulphide components and a blended composite.

A subsequent test program, undertaken in 2018 and completed in early 2019, included testwork to confirm the previous results, and additional testing to improve recoveries. This series of testing was completed using three (3) composite samples which varied in gold and silver grade, and one (1) composite sample from areas of the deposit considered waste. The testwork included mineralogy, evaluation of direct cyanide leaching of the samples (for comparison of the previous work), and investigation of gravity concentration combined with cyanide leaching of the gravity concentration tailings. Preliminary testing was also completed to determine the influence of grind size and reagent additions on the corresponding gold and silver recoveries. This testwork also included a preliminary evaluation of flotation, in combination with cyanide leaching as a potential alternative to gravity concentration.

These earlier test programs highlighted the preferred process options to be evaluated and further defined as part of this Feasibility Study and provided context for selection of drill cores and preparation of composite samples for further, more detailed testing.

In 2019 selected samples of mineralization from the Las Chispas deposit, which had either been used in the previous two series of tests at the SGS Durango facility or were new samples from various drill programs at site, were shipped to SGS Lakefield Research in Ontario, Canada (SGS Lakefield), for further metallurgical testing to support the Feasibility Study.

SGS laboratory facilities in Mexico and in Canada are well respected for their metallurgical testwork and are independent of SilverCrest. SGS holds several internationally recognized quality and technical certification including ISO 9001, however, there is currently no facility that accredits metallurgical testwork methods.

The testwork program undertaken to support the Feasibility Study included:

- Chemical and mineralogical analysis of the feed samples;
- Comminution testwork;
- Investigation of pre-concentration options, including gravity separation and flotation;
- Cyanide leaching of concentrate and tailings fractions from both gravity and flotation pre-concentration options;
- Solid–liquid separation testing;
- Precious metals recovery testing (Merrill Crowe process);
- Cyanide destruction testing; and,
- Variability testing across key unit operations for selected lithologies and veins.

13.2 Feed Materials

13.2.1 Inventory

Samples of mineralization from the Project, either from those remaining from the earlier testwork or those which had been collected from various drilling programs at site, were received at SGS Lakefield between March 2019 and September 2020.

Most of the samples received at SGS Lakefield for the testwork program were whole mineralized samples (core intervals, coarse rocks or core rejects) from various mineralization veins. Mineralized samples from Area 51, an active mining face at the deposit, were also provided, specifically to provide materials with a higher occurrence of clays components within the competent rock structures. Two Area 51 samples were prepared during the testwork: one in 2019 and one in 2020. They were prepared with samples collected from the same active mining face and are considered to be largely the same with respect to component materials, although the gold and silver grades varied slightly. The other muck samples used came from the historical stockpile referenced as Master Composite Historical Stockpile (MC-HS), Babicanora Central (from the open stope area) and the Babicanora Main FW vein (from active mining area).

The medium-grade composite (MED Comp) sample, which was produced during the testwork at SGS Durango, was provided as a gravity tailing sample.

Similarly, the Composite 1 (Comp 1) sample, which represented a blend of multiple intervals of waste, low-, and high-grade intercepts across numerous mineralization veins, was prepared for gravity separation testwork at SGS Durango and was provided as gravity tailings for the current testwork. Comp 1 was largely used for solid-liquid separation testwork.

Master Composite 3 (MC-3), and 4 (MC-4) were prepared to represent high grade materials, relative to the anticipated average grade of the deposit at the time of selection. High-grade composite samples were prepared to test a 'worst case' scenario with respect to response of the materials to pre-concentration and leach testing.

Master Composite 5 (MC-5) was prepared to represent the global average grade material and had been prepared from samples spanning several different production levels based on the preliminary mine plan envisaged at the time.

Master Composite 4/5 (MC-4/5) was a blend of 30% MC-4 and 70% MC-5, which was produced toward the end of the program given sample availability. The MC-4/5 composite was largely considered to be similar to the global average grade material with respect to mineralogy and major components, despite having a slightly higher grade.

The materials tested in support of this Feasibility Study are considered to be representative of the Las Chispas deposit, both with respect to the LOM average materials characteristics, and with respect to specific high-grade, low-grade (waste), and high-clay containing zones within the deposit.

A summary of the various component zone samples which comprised the composite feeds used in the testwork is provided in Table 13-1; the Area 51 sample (from an active mining face) and the MC-HS sample (from the stockpile) are not included as they were both muck samples of those materials as opposed to prescribed composites.

Table 13-1: Summary of Composite Sample Make Up as used in for Gravity Testwork Campaign

Component	Distribution				
	MED Comp*	Comp 1*	MC-3	MC-4	MC-5
Babicanora Central	26	combination of waste, low-grade, and high-grade material	23.4	7.7	8.0
Babicanora Main (Area 51)	30		31.0	46.2	20.0
Babi FW	11		11.4	11.5	8.0
Babicanora Norte	5		9.6	11.5	8.0
Babicanora Sur	–		–	9.6	8.0
Stockpile	–		7.8	13.5	–
Waste Lithic Tuff	–		12.6	–	–
Waste Dyke	–		4.2	–	–
Babi Vista	–		–	–	8.0
Las Chispas	15		–	–	8.0
William Tell	8		–	–	8.0
Comp 1	–		–	–	24.0
Total	**100**	**100**	**100**	**100**	**100**

* Arrived at SGS Lakefield as blended composite.

During the course of the test program, seven (7) additional composite samples were prepared to further evaluate the precious metal recoveries as a function of grade in the bulk flotation tests, and also to determine the deportment of antimony. These additional composites included:

- Sb Composite #1and #2, with varying antimony contents; and,
- Ag Composite #1 to #5, with varying gold and silver grades in the range of 500–2,500 gpt AgEq.

13.2.2 Feed Analysis

Multi-element, ICP spectroscopy analysis was completed on the feed composites. A summary of the head assays and select ICP results for the key composites are presented in Table 13-2.

Table 13-2: Head Assay Summary

Element	Unit	Med Grade Comp	Comp 1	MC-3	MC-4	MC-5	Area 51 (2020)	Area 51 (2019)	MC-HS
Au	gpt	2.76	2.90	11.60	8.86	4.96	16.30	19.80	4.19
Ag	gpt	342	381	872	1004	491	976	1841	515
C_{total}	%	0.12	–	0.25	0.11	0.10	0.05	0.06	0.03
$C_{organic}$	%	<0.05	–	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
S	%	0.17	–	0.19	0.15	0.26	0.10	0.10	0.42
$S^{=}$	%	0.16	–	0.16	0.14	0.20	0.09	0.08	0.37
Al	%	–	–	5.8	3.7	4.7	3.1	2.7	3.9
As	gpt	–	–	<30	<30	<30	<30	<30	<30
Cu	gpt	–	–	265	319	167	472	400	77
Hg	gpt	0.5	–	0.4	0.6	0.4	0.6	2.4	0.8
Fe	%	–	–	1.34	0.91	1.13	0.84	0.66	0.94
Mg	%	–	–	0.33	0.18	0.22	0.16	0.16	0.13
Sb	gpt	–	–	<40	<30	<40	49	<40	54
Se	gpt		–	<30	<30	<30	<30	<80	<30

Note: BAN NW to Babicanora Norte

Total carbon and sulphur contents are typically low, ranging from 0.03–0.25% carbon and from >0.1–0.42% sulphur for the key composites. Individual samples contained sulphur values as high as 1.3% (BAN NW). The testwork did not highlight any preg-robbing behaviour across the samples tested.

The arsenic content was generally below detection limit (<30 gpt As).

The copper content ranged from 0.01–0.05 %for key composites and peaked at 0.12% (Babicanora Area 118).

Some higher levels of antimony were observed in selected samples, although most of the samples contained <40 gpt Sb, with a range of between 40 and 54 gpt Sb. A peak antimony assay of 452 gpt was observed in one sample (Area 51, in Sept 2020) and accordingly the block model was interrogated for possible zonation or antimony trends. A brief series of testwork was completed to show that antimony values as high as 200 gpt Sb did not adversely affect the leaching performance or precious metal recoveries, and therefore the mine plan did not require limitations with regards to antimony.

Mercury assays ranged from <0.3—2.4 gpt, with the highest mercury content in samples from Area 51. Mercury deportment was further assessed in the Merrill-Crowe testwork to determine whether the content in the mineralized material could be considered problematic. Through the Merrill-Crowe testwork, it was demonstrated that there was sufficient mercury present in the silver–gold precipitate (to be refined) to justify the inclusion of a mercury retort to remove mercury prior to smelting, however no adverse effects on recovery were evident.

The samples tested did not contain deleterious elements at the concentrations which would be expected to affect payability of the gold-silver product, or which would necessitate incorporation of additional or unconventional unit operations to mitigate their influence on process performance. In the case of antimony, which was elevated in selected samples, testwork highlighted that it did not adversely affect the metallurgical recoveries.

13.2.3 Mineralogy

Quantitative evaluation of materials by scanning electron microscopy (QEMSCAN) and X-ray diffraction were conducted on to determine the mineralogy of selected samples from the Las Chispas deposit, including major constituents of the bulk of the prepared composites (Babi Vista, Babicanora Sur, Area 51, and Babicanora Central).

Silver is present as native silver, electrum, kustelite, hessite, silver tetrahedrite, argentite, pyrargyrite and stephanite as antimony alloys.

Secondary silver enrichment is indicated by the gradation from chlorargyrite (AgCl) near the surface to pyrargyrite (Ag_3SbS_3) at depth.

Base metal contents are relatively low in the vein structures. Minor zinc and lead contents were principally found in black sphalerite and galena, as blebs and veinlets. Argentite is the principal silver mineral in association with galena, pyrite ± marcasite and chalcopyrite. Some antimony is present. Minor secondary copper minerals as chrysocolla and malachite were noted and are associated with oxidized chalcopyrite.

Gold grains up to 100 µm in size were observed during mineralogical analysis. Increased gold mineralization was noted within the Babicanora area when compared to the Las Chispas Area. Gold and silver are strongly correlated and are likely precipitated together during the crystallization of quartz.

The modes of gold mineralization identified are:

- Gold associated with pyrite and chalcopyrite;
- Gold emplacement with silver sulphides (typically argentite); and,
- Native gold flakes in quartz.

Mineralogical evaluation of the Area 51 sample, and selected components thereof, highlighted the occurrence of clay materials in those samples. The mineral assemblage (modals) indicated that the samples were primarily quartz and potassium feldspar. The major clay fractions for each sample consisted of illite (Babi Vista), montmorillonite (Area 51 and Babicanora Central), and montmorillonite/corresnite in the case of Babicanora Sur.

13.3 Comminution Testing

Comminution testing was completed on selected individual component samples, and included the following tests:

- Crushing work index (Cwi);
- JK drop weight test (JK DWT) to support calibration of SMC (Axb) comminution tests;

- SMC and semi-autogenous grind (SAG) Power Index (SPI);
- Bond ball mill work index (Bwi); and
- Abrasion index (Ai).

The comminution test results are presented in Table 13-3. SMC tests were used as a reference point for variability mapping.

Table 13-3: Grindability Summary

Sample ID	Ai (g)	Cwi (kWh/t)	Bwi (kWh/t)	SPI (min)	DWT (Axb)	Dwi (kWh/m^3)	SMC (Axb)	SG (t/m^3)
Area 51	0.416	5.5	17.5	–	80.5	3.4	77.0	2.65
Waste Dike	0.703	11.2	21.3	–		7.4	33.9	2.52
Waste Lithic Tuff	0.216	8.5	17.0	–		5.6	45.4	2.53
Waste Welded Tuff # 1+2	0.156	10.0	–	–				
Waste WeldedTuff # 1	–	–	13.0	–		5.8	43.6	2.54
Waste WeldedTuff # 2	–	–	16.0	–		5.3	49.1	2.60
William Tell + Las Chispas	0.541	10.6	20.4	–				
Babicanora Central	-	-	20.0	–				
Babicanora Sur	-	-	19.2	–				
Babicanora Norte	-	-	17.8	–				
Babi FW	0.952	13.6	20.4	113		6.5	39.3	2.55
Las Chispas Stockpile	0.888	10.9	18.5	101		5.6	45.5	2.56
Babi Vista	–	–	16.4	–				

The Bwi values ranged from 13–21 kWh/t; the materials tested are considered to have moderate to high hardness. The Axb values, used for SAG milling design, ranged from 34–77; the materials tested are considered to be moderately competent.

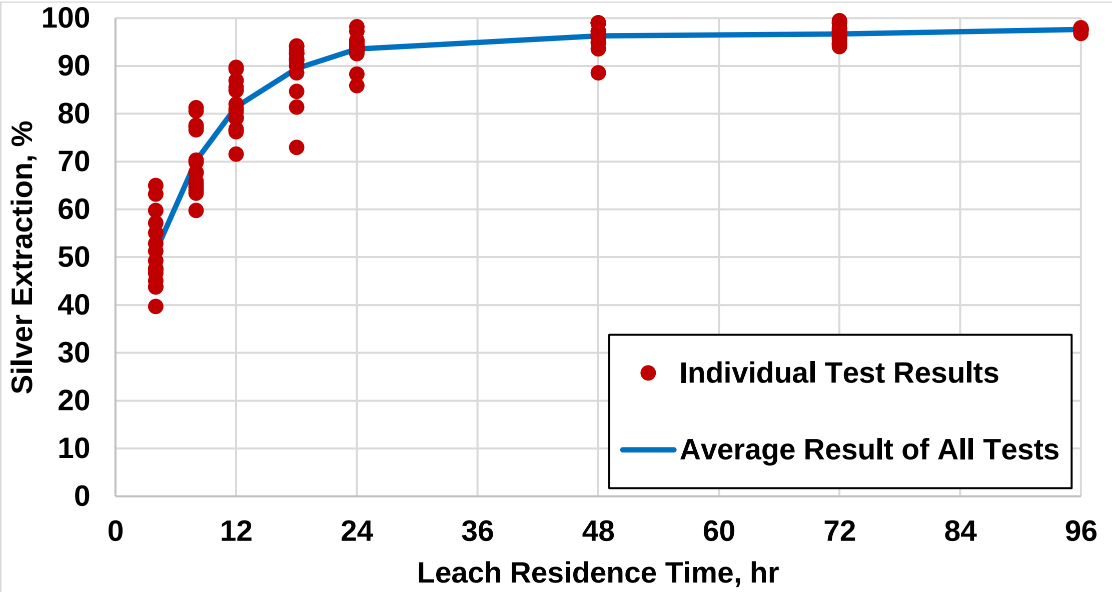
The Ai values varied from 0.156–0.952 g. Material in several of the tests was considered very abrasive, and excessive wear of materials in the plant must be considered.

13.4 'Whole Ore' Cyanide Leaching

A series of tests were completed with the Area 51 composite sample to establish the response of the mineralization to 'whole ore' leaching. 'Whole ore' leaching was simply subjecting the entire feed sample to a cyanide leach test, without any prior pre-concentration. The Area 51 composite was selected for the 'whole ore' leaching testwork as the samples were collected from an active mining face and had not been subject to long-term storage. The Area 51 composite was of particular interest as it was higher grade and expected to contain a higher clay content than the MED Comp, and therefore was expected to present a worst-case scenario for a 'whole ore' leach.

The 'whole ore' leach testwork evaluated gold and silver extractions under a range of conditions, designed largely to determine the influence of particle size, cyanide concentration and addition solids concentration and lead nitrate additions. Also, because many commercial leaching circuits coupled with a Merrill-Crowe circuit feature an interstage leach solution change, selected tests employed repulping the filter cake in barren solution.

Given a 96 hr leach residence time, gold and silver extractions were approximately 97.5% in all of the 'whole ore' leach tests and were largely unaffected by adjustments in leach conditions as tested. A specific concern of the' whole ore' leach flowsheet is the propensity for mineralized feeds with either coarse gold and silver particles or significantly higher grades to yield lower precious metal recoveries because of the residence time and cyanide availability constraints. The testwork did confirm that the relatively long leach residence time was required to achieve high silver extractions, as shown in Figure 13-1 below.



Note: Figure prepared by Ausenco, 2020.

Figure 13-1: Silver Extraction as a Function of Residence Time in 'Whole Ore' Leach Tests

There was considerable variability in silver extraction kinetics amongst the various 'whole ore' leach tests, depending on conditions, and as noted above, there was a consistent trend which highlighted the requirement for the long residence time and the corresponding consistent results for silver.

Slurries in the 'whole ore' leach tests appeared to be more viscous in comparison to other cyanide leach tests that had been completed. Therefore, selected tests were completed to determine the influence of slurry solids concentration on precious metal extractions. There was no apparent benefit in tests completed at solids concentrations of 35%, relative to comparable tests completed at 48% solids.

13.5 Gravity Concentration Testing

The metallurgical testwork program evaluated the response of mineralized samples to a pre-concentration step ahead of cyanide leaching, as a means to accommodate high-grade mineralized material. Earlier testing had demonstrated that a pre-concentration step either via gravity separation or flotation offered an opportunity to provide improved recoveries from treatment of higher-grade mineralization.

An initial series of tests was completed using the MED Comp and Area 51 Composite samples. Preferred conditions were then applied to composites MC-3, MC-4 and MC-5 and all variability samples, to test the response of the range of materials.

Typically, ground mineralization was passed through a Knelson concentrator, and the concentrate was further upgraded on a Mozley table, to produce a Mozley concentrate at a target weight percent of 0.05 to 0.1% of the original feed sample (mass pull). In the current testwork, a 4% mass pull was targeted using multiple Knelson passes, to provide an opportunity for maximum recovery of gold and silver to the gravity concentrate.

For the purposes of the pre-concentration testwork, the samples were ground to a particle size of P_{80} = 95-100 µm. Corresponding cyanide leach tests on the gravity concentrate included a regrind of the concentrate fraction to a target particle size distribution of P_{80} = 35 µm. Knelson tailings were also forwarded for cyanidation.

13.5.1 Medium Grade and Area 51 Composites

The initial gravity separation test results from MED Comp and Area 51 samples showed silver recovery ranged from 45—62%, and gold recovery ranged from 49—51%, at the target 4% mass pull.

13.5.2 MC-3

Four (4) bulk gravity separation tests were completed on the MC-3 Comp sample. The individual tests treated between 10 and 30 kg of feed material each. At mass pulls of 3.8—4.3%, gold and silver recoveries to the gravity concentrate varied from 56—61% for gold and from 65—68% for silver. Mass accountabilities for gold and silver in the testwork was good and provided confidence in the results.

13.5.3 Variability Samples

Gravity concentration testwork on selected grade variability samples was completed with feed samples of between 2 kg and 5 kg, and employed a target mass pull of 4%, as had been done in previous testing. Calculated head grades ranged from 6.5-391 gpt for gold and 945—27,430 gpt for silver. Gravity recoveries for gold ranged from 23–89%, averaging 60%, which was marginally higher than the medium grade composite. Gravity recoveries for silver ranged from 21—70%, with an average 49% and well aligned with the medium grade composite.

13.5.4 MC-4

Two bulk gravity separation tests were completed with the MC-4 Comp, each employing a 20 kg sample of feed. Consistent with the other gravity concentration testing, the MC-4 composite sample showed gold recoveries of 62 and 65%, and silver of 56% in each case, each at a mass pull of about 3.6%.

13.6 Cyanidation of Gravity Concentrates and Tailings

Gravity concentrates were cyanide leached at high cyanide concentration for 96 hr, with the exception of some tests on MC-3, which were performed with 24 hr leach residence time. The high cyanide leach conditions employed for the gravity concentrates were designed to provide the best opportunity for high extractions from high grade materials or coarse gold and silver particles. Gravity tails were leached under conventional cyanidation conditions, with the same residence time of 96 hr to accommodate the relatively slow leach kinetics of the silver.

Gravity concentrate was reground prior to cyanide leaching, whereas the tailings samples were leached without adjustment of particle size.

Cyanidation tests were completed to evaluate the following parameters on overall gold and silver recovery:

- Grind size P_{80};
- Retention time;
- Pre-aeration;
- Pulp density;
- Pulp pH (alkalinity);
- Cyanide concentration;
- Dissolved oxygen concentration;
- Lead nitrate additions;
- Temperature; and,
- Filtering/re-pulping during leach.

13.6.1 Medium Grade Composite

Multiple cyanidation tests were completed on the MED Comp (gravity tails sample) in an effort to identify the preferred conditions. Those test results are summarised in Table 13-4.

The grind size P80 for all of tests was 92 µm (as-received sample size). Free cyanide concentration was maintained at 2 g/L NaCN for all tests. The pulp density was 48% solids for all tests, except one test (CN-73) at 40% solids. All the tests were completed at room temperature, with the exception of CN-33, which was completed at 40°C. Some tests included a filter and re-pulp with fresh solution, to simulate the coupling with a Merrill Crowe metal recovery process downstream. The pH was variable in the testwork, as the bulk of the tests targeted a lime addition, and the pH was allowed to fluctuate. In selected tests, the pH was controlled at 10.5 to 11 by addition of lime as required.

Table 13-4: Medium Grade Composite Cyanidation (Gravity Tails) Test Results

CN Test No.	Reagent Addition kg/t of CN Feed		Reagent Cons. kg/t of CN Feed		Extraction, %															
					Au								Ag							
	NaCN	CaO	NaCN	CaO	4 h	8 h	12 h	18 h	24 h	48 h	72 h	96 h	4 h	8 h	12 h	18 h	24 h	48 h	72 h	96 h
1	2.97	0.87	1.07	0.84	89.4	95.3	…	…	98.2	99.0	99.4	98.6	54.5	73.7	…	…	91.2	93.0	98.5	94.0
2	3.15	1.30	1.13	1.28	90.7	94.9	…	…	98.2	99.0	99.0	98.6	53.3	72.0	…	…	88.8	92.2	96.3	93.9
3	2.75	4.41	0.75	4.10	97.6	98.0	…	…	97.9	95.8	98.7	98.1	73.0	83.9	…	…	93.0	94.4	96.5	95.2
4	3.01	3.82	0.87	3.82	95.3	95.9	87.7	99.0	97.8	…	…	…	49.9	80.1	83.9	97.2	95.1	96.1	95.9	96.9
5	3.39	4.41	1.02	4.17	87.8	99.8	99.0	99.1	92.4	96.0	93.7	96.0	65.5	80.8	87.4	92.4	87.7	95.6	94.2	95.4
6	3.12	4.71	0.93	4.37	99.0	98.4	98.2	93.9	99.0	94.7	91.1	97.1	74.1	85.4	85.1	92.8	99.0	95.3	95.2	95.7
8	3.16	4.63	1.08	4.45	…	…	…	…	…	…	…	92.5	57.5	85.3	92.4	95.6	93.6	95.3	92.5	92.0
9	3.40	5.45	1.32	4.99	…	…	…	…	…	…	…	97.8	64.9	89.0	94.1	~99	~99	~99	99.9	97.0
10	3.30	5.47	0.91	5.21	…	…	…	…	…	…	…	98.4	82.1	95.4	97.4	~99	~99	~99	98.1	96.9
14	3.30	7.51	1.19	6.93	~99	~99	~99	~99	~99	~99	99.8	98.3	74.9	94.2	~99	~99	98.5	98.6	97.9	97.7
15	3.35	4.51	0.44	3.98	~99	~99	92.2	~99	95.3	96.7	…	…	60.3	87.5	85.0	87.8	93.1	95.4	…	…
16	3.11	5.31	0.95	4.70	~99	~99	~99	~99	91.6	96.7	…	…	83.3	89.4	94.4	90.9	94.6	95.2	…	…
28	5.58	7.31	3.33	6.59	…	…	…	…	…	…	…	90.6	…	…	…	…	…	…	…	95.1
29	5.47	6.57	2.24	5.70	…	…	…	…	…	…	93.3	…	…	…	…	…	…	…	95.8	…
30	4.89	6.08	2.28	5.26	…	…	…	…	…	92.9	…	…	…	…	…	…	…	95.0	…	…
31	5.13	9.88	2.20	8.79	…	…	…	…	…	…	94.3	…	…	…	…	…	…	…	95.1	…
32	5.09	9.92	2.04	8.80	…	…	…	…	…	…	91.4	…	…	…	…	…	…	…	93.3	…
33	3.90	7.53	2.06	7.25	…	…	…	…	…	…	…	97.8	82.2	95.4	…	~99	99.8	~99	99.3	95.4
72	3.73	8.43	1.10	7.89	…	…	…	…	…	…	…	95.7	…	…	…	…	…	…	…	93.8
73	3.92	9.94	0.87	9.12	…	…	…	…	…	…	…	95.8	…	…	…	…	…	…	…	93.3
7	2.49	2.05	0.09	1.58	…	…	…	…	…	21.8	…	…	…	…	…	…	…	18.3	…	…

Estimated using Residue / Direct Head

Note: Table prepared by SGS Lakefield, 2020.

High-gold extractions of over 97% were largely achieved given a leach residence time of 24 hr; the corresponding silver extraction exceeded 94%. Silver extractions were shown to largely continue to increase between 24 and 96 hr, from an average of 94.7% to between 95.2 and 96.5%.

Pre-aeration with oxygen had minimal effect on gold and silver recovery. Filtering the pulp in mid leach and re-pulping with fresh cyanide solution was not beneficial. Maintaining a standard pH range of 10.5 – 11.0 (CN 1 and CN2) which resulted in lower lime consumption was favourable in comparison to the remaining tests which were maintaining high free alkalinity at the expense of higher lime consumption.

A single gravity concentrate sample from the previous SGS metallurgical test program was used for a series of cyanide leach tests, to evaluate the influence of concentrate regrinding, pre-aeration, cyanide concentration, lead nitrate addition and alkalinity on the gold and silver extractions. Grind size (P80 = 189 vs 75 µm) and free cyanide concentration (20 vs 2 g/L) were the key variables investigated. As noted previously, the concentrate leach tests were completed using high cyanide concentrations to promote high extractions.

The results of the tests are presented in Table 13-5.

Table 13-5: Gravity Concentrate Cyanidation Test Results (SGS Durango)

CN	Reagent Cons.		Extraction, %									Residue Grade		Calc. Head		Direct Head	
Test	kg/t of CN Feed		Au	Ag								Au	Ag	Au	Ag	Au	Ag
No.	NaCN	CaO	96 h	4 h	8 h	12 h	18 h	24 h	48 h	72 h	96 h	g/t	g/t	g/t	g/t	g/t	g/t
11	64.9	0.07	99.7	19.4	36.5	52.8	71.1	82.9	99.4	~99	99.6	0.29	51.9	88.6	11545	75.0	7273
12	69.9	0.03	99.6	12.6	25.9	38.4	50.1	65.7	~99	~99	99.5	0.31	58.0	87.6	12218		
13	61.2	5.62	99.7	9.8	20.3	30.4	46.6	56.2	91.1	~99	99.3	0.30	84.4	92.2	11655		
17	21.1	12.7	98.7	50.6	82.3	~99	~99	~99	~99	96.9	95.4	0.82	574	64.6	12383		
18	23.5	20.1	98.5	49.7	89.2	95.2	89.7	~99	99.7	97.4	96.7	0.79	499	51.3	15002		
19	24.0	17.5	98.7	47.7	77.4	87.6	79.4	92.6	~99	96.9	97.1	0.41	466	32.5	15871		
20	19.6	26.6	98.8	28.2	61.2	86.3	~99	~99	~99	~99	97.3	1.00	364	86.5	13573		
21	12.8	22.2	98.6	31.2	69.9	87.0	95.6	99.7	99.2	98.9	96.0	0.92	331	63.8	8232		
22	17.6	25.4	99.3	…	…	…	…	…	…	…	96.8	0.54	235	…	…		

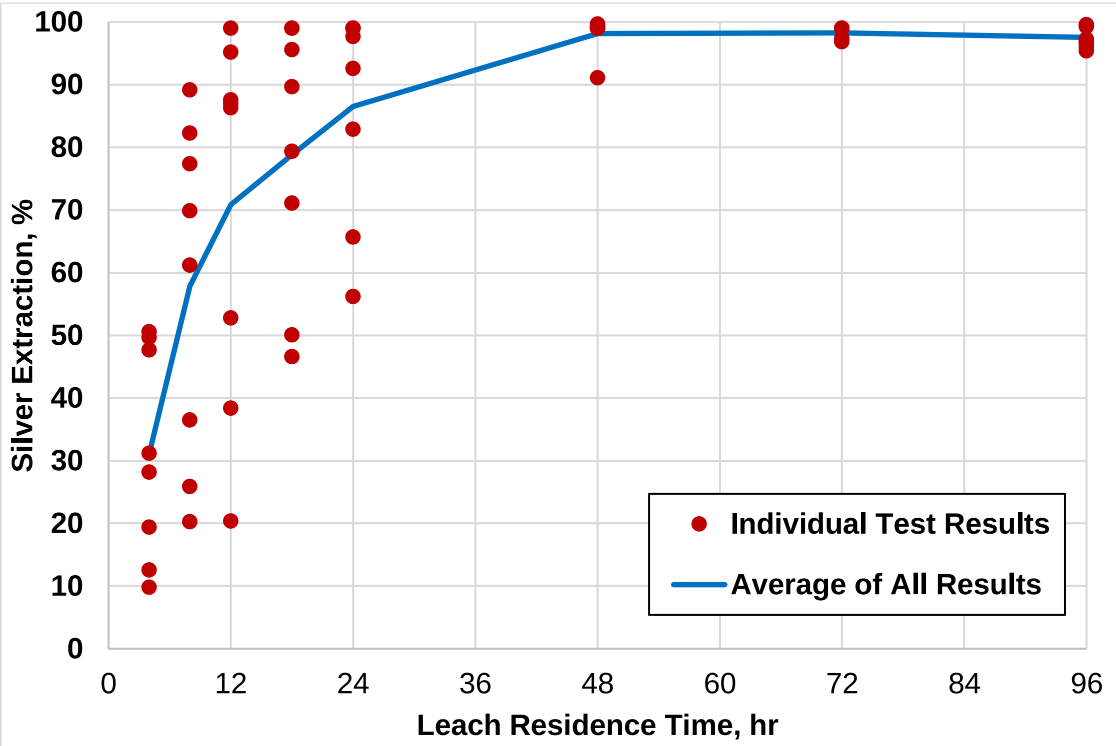
Estimated using Residue / Direct Head   Solution change during test

Note: Table prepared by SGS Lakefield, 2020.

Gold extractions were consistently high in all of the tests, as expected.

The average residue grades at the higher cyanide concentration and coarser grind size were of the order of 0.30 gpt Au and 64.5 gpt Ag. These were markedly lower than the average residue grades for tests with finer particle sizes (P80 = 75 µm) and lower cyanide concentration (0.75 gpt Au and 412 gpt Ag).

Silver extractions were variable given residence times less than 48 hr, as shown in Figure 13-2.



Note: Figure prepared by Ausenco, 2020.

Figure 13-2: Silver Extraction as a Function of Residence Time in Gravity Concentrate Leach Tests

The testwork showed silver extractions broadly increasing across the duration of the tests and achieving the maximum value generally between 72- and 96-hour residence in each case.

Higher cyanide concentrations specifically improved silver extractions kinetics, but not ultimate extractions. Therefore, the preferred conditions for the gravity concentrate leach, despite the high aggressive cyanide concentrations, would include a residence time of 96 hours.

13.6.2 Area 51 Zone

Ten (10) cyanidation tests were completed using tailings solids generated in gravity testing of the Area 51 composite samples. The tests featured testing at varying cyanide concentrations between 1.5 and 5 g/L NaCN, solids concentrations of either 30% or 48% and variable lead nitrate additions. The results are summarized in Table 13-6.

Table 13-6: Area 51 Gravity Tailing Cyanidation Test Results

CN Test	Reagent Cons. kg/t of CN Feed		Extraction, %						Residue Grade						Calc. Head		Direct Head	
			Au			Ag			Au, g/t			Ag, g/t			Au	Ag	Au	Ag
No.	NaCN	CaO	48 h	72 h	96 h	48 h	72 h	96 h	A	B	Avg.	A	B	Avg.	g/t	g/t	g/t	g/t
34	6.26	10.1	91.4	…	…	92.0	…	…	0.70	0.68	0.69	66.3	65.5	65.9	…	826	9.29	720
35	6.32	11.0	86.2	…	…	…	…	…	0.40	0.43	0.42	42.7	42.7	42.7	…	…		
36	6.21	11.1	79.5	…	…	…	…	…	0.41	0.41	0.41	41.7	41.5	41.6	…	…		
37	8.24	9.53	93.0	…	…	…	…	…	0.50	0.52	0.51	49.6	48.8	49.2	…	778		
62	1.77	8.61	…	…	93.9	…	…	94.8	0.54	…	0.54	41.5	…	41.5	…	793	9.54	788
63	1.66	9.56	…	…	95.2	…	…	95.0	0.42	0.40	0.41	39.0	37.2	38.1	…	764		
64	1.67	11.4	…	…	95.6	…	…	95.1	0.40	0.39	0.40	40.1	39.6	39.9	…	808		
65	1.75	11.0	…	…	95.8	…	…	95.5	0.36	0.42	0.39	35.0	37.2	36.1	…	805		
66	1.49	10.7	…	…	95.0	…	…	95.1	0.43	0.47	0.45	40.9	38.1	39.5	…	800		
67	2.06	10.6	…	…	96.7	…	…	96.5	0.39	0.39	0.39	37.3	37.2	37.3	…	1052		

Direct head assays shown (G-1 gravity tailing assays - triplicate average)
Calc. head grades not included for Au because of subsamples taken for Ag only
Calc. heads for not shown for tests which included re-pulp
Note: Table prepared by SGS Lakefield, 2020.

Both gold and silver recoveries ranged from 94—97% and were minimally affected by variation in the solids concentration, cyanide concentration or lead nitrate additions.

In addition, 15 cyanidation tests using gravity concentrate produced from testing the Area 51 sample were completed, principally to evaluate the influence of concentrate particle size, and sodium cyanide concentration. Test results are summarized in Table 13-7.

Table 13-7: Area 51 Gravity Concentrate Cyanidation Test Results

CN Test No.	Reagent Cons. kg/t of CN Feed NaCN	CaO	Extraction, % Au 24 h	Au 48 h	Au 96 h	Ag 24 h	Ag 48 h	Ag 96 h	Residue Grade Au, g/t A	Au, g/t B	Au, g/t Avg.	Ag, g/t A	Ag, g/t B	Ag, g/t Avg.	Calc. Head Au g/t	Calc. Head Ag g/t
38	29.1	4.91	…	79.5	…	47.6	77.0	…	37.0	43.1	40.1	5946	5971	5959	196	25963
39	33.4	11.7	…	57.2	…	…	…	…	29.1	35.3	32.2	2031	2014	2023	…	…
40	34.9	10.7	…	67.6	…	…	…	…	16.9	22.7	19.8	1075	1110	1093	…	…
41	34.9	13.6	…	81.1	…	…	…	…	5.76	3.86	4.81	394	399	397	…	…
42	105	9.82	…	98.0	…	…	…	…	0.96	0.49	0.73	81.3	80.1	80.7	…	…
53	36.8	10.9	…	…	90.2	…	…	99.4	7.56	7.41	7.49	88.0	66.1	77.1	…	…
54	34.1	10.9	…	…	87.3	…	…	99.3	6.81	12.0	9.41	89.4	78.3	83.9	…	…
55	35.7	9.25	…	…	85.1	…	…	99.2	11.6	…	11.6	89.7	…	89.7	…	…
56	30.8	11.0	…	…	94.0	…	…	99.4	3.98	4.51	4.25	96.8	90.4	93.6	…	…
57	46.9	10.7	…	…	99.0	…	…	99.1	0.39	0.68	0.54	62.6	59.4	61.0	…	…
58	29.6	3.19	…	…	94.8	…	…	96.1	11.1	11.4	11.3	1014	989	1002	216	25498
59	38.8	2.32	…	…	99.2	…	…	99.8	1.63	2.04	1.84	53.1	51.4	52.3	216	26329
60	34.5	0.22	…	…	93.9	…	…	99.8	14.8	12.9	13.9	60.6	60.9	60.8	227	25856
61	28.7	1.12	…	…	82.6	…	…	95.8	37.3	42.3	39.8	1035	1020	1028	229	24449
74	19.9	8.23	75.8	…	…	55.1	…	…	…	…	61.8	…	…	12200	255	27160

Note: Table prepared by SGS Lakefield, 2020.

The leach kinetics were relatively slow under most of the conditions tested. Test CN-74 leach residence time was only 24 hr and low recoveries of 76% Au and 55% Ag were attained. A leach residence time of 96 hr was required to maximize the recoveries of both gold and silver in most of the tests. Re-grinding of the gravity concentrate, and maintaining higher cyanide concentrations also improved recoveries initially, but not ultimately. Under the preferred conditions established (CN-57 and CN-59), the recovery of both gold and silver were >99%. Selected concentrate regrind size were tested, and a target regrind P_{80} of ~30–35 µm was selected.

Following identification of the preferred leach conditions for the gravity concentrates, three (3) bulk leach tests were completed using gravity concentrate produced from the Area 51 Composite, chiefly to generate sufficient material for downstream washing (CCD simulation), Merrill-Crowe, cyanide detoxification, and solid/liquid separation testwork (CCD and filtration). A high-level summary of the bulk cyanidation test configuration and conditions is presented in Table 13-8.

Table 13-8: Area 51 Bulk Cyanidation Test Conditions

Sample	Sample Type	CN Test No.	Feed Size P_{80}, µm	NaCN g/L Maintained	% Solids w/w	Re-pulp (Solution Change)	Leach RT	Lead Nitrate g/t	pH	CN Free End of Test, mg/L	Reagent Cons. kg/t of CN Feed NaCN	CaO
AREA 51 G-3	G-3 & G-4 Kc	80	31	2	30	No	24	1000	free 3.5 sol'n	343	19.3	3.02
AREA 51 G-3 Gravity Tailing + CN-80 Residue	G-3 Kr + CNr	81	91	1.5 (0-48h); 1 (48-96h)	48	No	96	300 + 150 (24)	free 2.0 sol'n	662	1.67	2.49
AREA 51 G-3 Tailing Only FOR S/L SEP TESTWORK	G-2 Kr	82	94	1.5 (0-48h); 1 (48-96h)	48	No	96	300 + 150 (24)	free 2.0 sol'n	743	1.03	2.59

Note: Table prepared by SGS Lakefield, 2020.

A summary of the bulk leach test results is presented in Table 13-9.

Table 13-9: Area 51 Bulk Cyanidation Test Results

Sample	Sample Type	CN Test No.	Extraction, % Au 24 h	Extraction, % Au 96 h	Extraction, % Ag 24 h	Extraction, % Ag 96 h	Gravity, % Au	Gravity, % Ag	Grav Mass %	O'All Recovery (Grav + CN), % Au	O'All Recovery (Grav + CN), % Ag	Residue Grade Au, g/t A	Residue Grade Au, g/t B	Residue Grade Au, g/t **Avg.**	Residue Grade Ag, g/t A	Residue Grade Ag, g/t B	Residue Grade Ag, g/t **Avg.**	Calculated Head Au g/t	Calculated Head Ag g/t
AREA 51 G-3	G-3 & G-4 Kc	80	**72.9**	…	**55.4**	…	51	57	3.7	**37.2**	**31.6**	…	…	70.4	…	…	12251	260	27455
AREA 51 G-3 Gravity Tailing + CN-80 Residue	G-3 Kr + CNr	81	…	**95.7**	…	**95.7**			96.3	**97.3**	**97.1**	0.47	0.47	0.47	65.8	65.1	65.5	11.0	1531
AREA 51 G-3 Tailing Only FOR S/L SEP TESTWORK	G-2 Kr	82	…	**93.4**	…	**90.4**	54	59	95.8	96.1	93.5	0.43	0.45	0.44	67.4	68.5	68.0	6.63	705

Estimated using approx. gravity conc recovery and G-2 gravity recoveries

Note: Table prepared by SGS Lakefield, 2020.

Overall gold and silver recoveries for the tests were between 96–97% for gold and between 94–97% for silver. The bulk tests also provided confidence in the cyanidation reagent requirement. For cyanidation of the flotation concentrate, consumption of cyanide and lime were 19.3 kg NaCN/t solids and 3.0 kg Cao/t solids, respectively. For cyanidation of the combined flotation tails and concentrate leach residue, consumption of cyanide and lime were approximately 1.7 kg NaCN/t solids and 2.6 kg Cao/t solids, respectively.

13.6.3 MC-3

MC-3 bulk leach tests were performed at the preferred conditions to provide additional materials, representing more average operation of the plant, for downstream testwork such as Merrill-Crowe, cyanide detoxification and solid/liquid separation testwork. A summary of the results is provided in Table 13-10 below.

Table 13-10: MC-3 Cyanidation Test Results (24 hour concentrate leach + 96 h combined leach)

CN Test No.	Extraction, % Au 12 h	Au 24 h	Au 48 h	Au 72 h	Au 96 h	Ag 12 h	Ag 24 h	Ag 48 h	Ag 72 h	Ag 96 h	Gravity, % Au	Gravity, % Ag	Grav Mass %	O'All Recovery (Grav + CN), % Au	O'All Recovery (Grav + CN), % Ag
89	70.3	72.8				94.8	96.8				56.1	65.1	3.8	40.9	63.1
90	67.0	71.6				94.6	97.1							40.2	63.3
95	…	90.2				…	87.6							50.6	57.1
96	…	89.5				…	89.9							50.2	58.6
91	…	~99	99.7	99.3	95.4	…	99.1	94.0	95.3	94.0	43.9	34.9	96.2	97.2	97.8
92	…	~99	96.2	97.9	95.1	…	~99	98.2	95.5	93.0				97.1	97.4
93	…	~99	98.6	97.5	94.7	…	99.6	96.8	98.7	93.8				96.8	97.7
94	…	95.3	~99	~99	95.3	…	96.4	95.0	96.8	93.5				97.2	97.6

CN-91 to CN-94 overall recovery calculated using CN-90 Conc leach results

Note: Table prepared by SGS Lakefield, 2020.

The results from the combined leach tests indicated that overall gold and silver recoveries of between 97–98% could be expected from a combination of gravity concentration, concentrate leaching with a 24-hr residence time, and tailings + gravity concentrate leach residue leaching with a 96 hr residence time.

13.6.4 Variability Samples (Geo-metallurgy)

Upon completion of the MC-3 Comp testwork, the following preferred conditions were used for all the variability tests:

- Gravity concentrate leach conditions:
  - Re-grind P_{80} of ~35 µm;
  - No pre-aeration;
  - 96-hr leach retention time;
  - Pulp density of 30% solids;
  - Dissolved oxygen concentration target of 20-30 mg/L (controlled with oxygen addition);
  - Lime addition maintained at 1.6 kg CaO/t of feed solids (free titration method), pH was not controlled;
  - Cyanide concentration maintained at 2 g/L NaCN; and,
  - Lead nitrate addition of 1000 gpt at start of test;
- Gravity tailings leach conditions:
  - Feed size as provided;
  - No pre-aeration;
  - 96-hr leach retention time;
  - Pulp density of 48% solids;
  - Dissolved oxygen concentration target of 20–30 mg/L (controlled with oxygen addition);
  - Pulp pH maintained at 0.7 kg CaO/t of feed solids (free titration method), pH was not controlled;
  - Cyanide concentration maintained at 1.5 g/L NaCN for 48 hr, and then 1 g/L for final 48 hr; and,
  - Lead nitrate addition of 300 gpt at start of test, and additional 150 gpt at 24 hr.

Separate gravity concentrate and gravity tailings leaches tests were completed and in the case of each feed sample, “combined” or “overall” gold and silver recoveries were calculated based on the corresponding concentrate and tailings leach results.

Gravity concentrates and tailing streams resulting from the variability samples were leached and assayed separately. Subsamples of select leach residues were combined as required for solid/liquid separation and environmental testing.

Measured extractions of gold and silver from both the concentrate and the tailings streams are reported, along with the overall gold and silver recoveries, which were calculated based on feed to the process and final process tailings, such that they include recovery and losses incurred in gravity concentration, both stages of leaching and washing.

The test results are presented in Table 13-11. As indicated, selected tests included a second stage of leaching (re-leach) because initial results of the concentrate leach test were less than anticipated, and it was determined that the test conditions (insufficient cyanide) had artificially constrained the results.

Table 13-11: Gravity Concentrate and Tailings Cyanide Leach Test Results – Variability Samples

Sample	Concentrate Extraction		Tail Extraction		Overall Recovery (incl. gravity)		Re-leach Final	
	Au %	Ag %	Au %	Ag %	Au %	Ag %	Au %	Ag %
Babicanora Central	99.7	97.6	94.7	84.5	96.8	88.5	97.4	89.5
Babicanora Norte	98.0	80.1	91.3	95.7	96.3	86.6	97.9	98.5
William Tell	99.0	97.7	97.0	94.4	97.6	95.6	-	-
Las Chispas	99.3	98.7	94.4	96.7	97.4	97.9	-	-
Las Chispas stockpile	96.2	86.2	94.7	98.0	96.0	90.6	-	-
Comp 9	85.3	49.1	96.4	97.6	86.5	71.2	99.4	98.7
Comp 10	96.1	76.0	91.8	90.9	95.3	84.1	98.8	97.0
Babi Sur	99.5	98.7	87.4	86.4	94.6	93.7	95.3	94.4
Babi Vista	97.0	92.6	93.8	96.4	95.8	93.9	97.9	99.0
Comp 1 Babicanora Sul	96.6	98.6	95.2	95.6	95.8	97.0	97.5	98.3
Comp 2 Babicanora Sul	99.6	99.2	97.3	90.8	98.3	94.4	-	-
Comp 3 Babicanora Sul	98.7	97.1	96.5	91.1	97.4	93.1		
Comp 4 Babi Mix	96.8	65.4	96.5	95.9	96.7	78.5	97.6	79.2
Comp 5 Babi Mix	99.6	98.0	97.3	88.5	98.4	92.0	-	-
Comp 6 Babi Mix	98.5	96.3	91.9	86.3	93.9	89.1	-	-
Comp 7 Babi Ox	99.3	96.7	96.3	91.1	97.9	92.3	-	-
Comp 8 Babi Ox	99.5	97.7	97.5	83.2	98.4	89.2	-	-
Comp 11 Granaditas	92.3	90.1	94.5	96.2	92.6	93.0	98.7	98.4
Comp 12 Giovanni	96.3	91.3	88.9	91.2	93.4	91.3	97.5	98.1
Comp 13 William Tell	98.6	97.3	95.6	95.0	96.3	95.6	96.8	96.4
Comp 14 Luigi	97.9	94.7	92.2	98.4	96.9	96.1	98.5	98.0
Comp 15 Las Chispas underground	98.2	95.8	95.1	96.8	97.4	96.2	-	-
Babi Vista Feb 2020	86.8	91.1	97.6	97.5	88.7	93	99.2	99.3

Sample	Concentrate Extraction		Tail Extraction		Overall Recovery (incl. gravity)		Re-leach Final	
	Au %	Ag %	Au %	Ag %	Au %	Ag %	Au %	Ag %
Babi FW Feb 2020	99.5	99.3	97.8	96.4	98.8	98.0	-	-
Babicanora Sur Feb 2020	99.5	99.5	96.1	92	98.0	96.5	-	-
Babicanora Norte Feb 2020	86.5	61.5	96	98.7	88.1	77.4	99.4	98.6
							All revised	
Minimum	85.3	49.1	87.4	83.2	86.5	71.2	93.9	79.2
Maximum	99.7	99.5	97.8	98.7	98.8	98.0	99.4	99.3
Average	**97.6**	**90.2**	**94.8**	**93.3**	**95.5**	**91.0**	**97.7**	**94.9**

Overall gold recoveries ranged from 86–99% and averaged 96%. Overall silver recoveries ranged from 71–98% and averaged 91%. The average cyanide and lime consumptions (combined) were 1.00 kgpt NaCN and 3.21 kgpt CaO.

A total of 14 secondary leach tests (re-leach) were completed including using the combined samples and two tests using gravity tailings only. The additional 72 hours of leaching (maintained 1.5 g/L NaCN) increased the overall gold and silver recoveries on average by 7.4% (Ag) and 4.1% (Au). The secondary leach tests indicated that some of the higher-grade samples required more cyanide, particularly in the concentrate leach step. Considering overall recoveries for the 12 samples following secondary treatment, the average overall recoveries from the 26 variability tests increased to 98% for gold and 95% for silver. Given sufficient cyanide addition and adequate residence time in operation, a secondary leach treatment is not required.

13.6.5 MC-4

The MC-4 Composite was used in a series of tests which further evaluated conditions for leaching both the concentrate and tailings fractions, leveraging the understanding from the MC-3 Comp tests and variability tests. The results from the MC-4 test series indicated that overall gold and silver recoveries of about 94% and 97%, respectively can be achieved for the MC-4 composite, which is in agreement with other master composites and variability test results under similar conditions.

13.7 Flotation Testing

The 2019 PEA highlighted occurrences of high-grade zones within the deposit and initial testing indicated that a pre-concentration step either via gravity separation or flotation offered a robust process option to accommodate higher-grade mineralization. Although the gravity pre-concentration method produced acceptable gold and silver recoveries, it was noted that a gravity-based process could be susceptible to losses stemming from fine, high-grade silver materials. It was also apparent that the gravity concentration performance might not scale up from testwork to commercial scale with certainty, given the sample sizes tested and the apparent brittle nature of the silver minerals. For this reason, flotation appeared to offer the most flexible process option for maximizing gold and silver recoveries.

As with the gravity testwork, an initial series of tests was completed using the Medium Grade Composite and Area 51 Composite, and then preferred conditions were applied to composites MC-3, MC-4 and MC-5 and all variability samples, to test the response of the range of materials. Initial baseline flotation testwork was completed on samples including MC-5, Historical Stockpile (MC-HS), MC-4/5, and Area 51 to verify gold and silver recoveries to a high-grade concentrate fraction. The mineralization was ground to a particle size distribution of P_{80} = 95-100 µm for the flotation testwork.

A total of 10 batch tests using different techniques were performed on composite samples MC-HS, MC-4/5, Babicanora Central, MC-4, and Area 51. Five (5) of the tests were performed at a grind size P80 of 95–115 µm and the other five (5) tests were completed at P80 of 38–88 µm on the same composite samples.

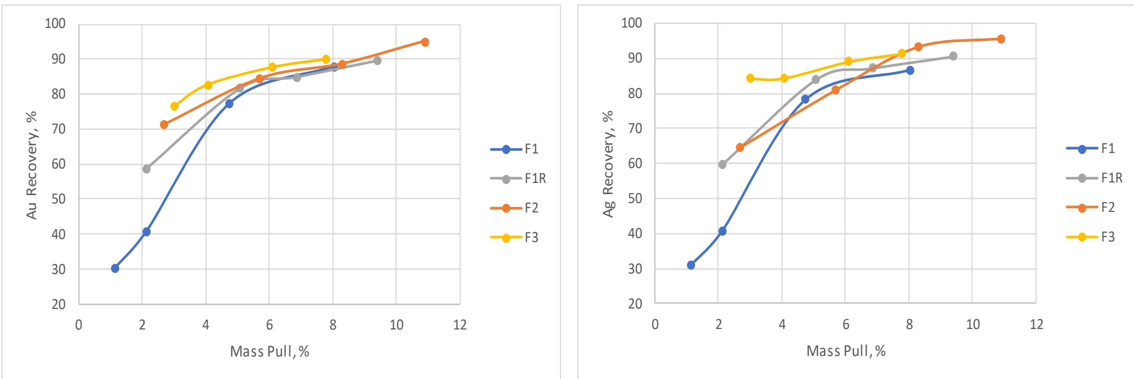
Corresponding cyanidation tests were completed on flotation product streams in similar fashion to those completed on the gravity products. Subsequent tests investigated the effect of different NaCN concentrations and pH levels on the flotation process as well as to establish the preferred flotation conditions.

Nine bulk rougher flotation tests were performed to investigate precious metal value deportment to the flotation concentrate and flotation tails and generated sufficient materials for subsequent cyanide leach testing.

13.7.1 Flotation Tests (MC-5, MC-HS, MC-4/5, Area 51 Zone)

Initial flotation tests (F1/F1R, F2, and F3) were completed with the MC-5, MC-HS, and a 50/50 blend of the MC-4 and MC-5 samples, respectively. The tests were operated at natural pH, with collectors Aero 404 and Aerophine 3418A added to the grinding step prior to conditioning, $CuSO_4$ additions were made to mitigate the influence of any residual cyanide, and potassium amyl xanthate (PAX) collector was added during the tests. Frother Aero 65 was used in all the tests.

Results of the initial flotation tests showed that with an 8 to 11% mass pull, recoveries were about 87 to 95% for both gold and silver, as illustrated in Figure 13-3.

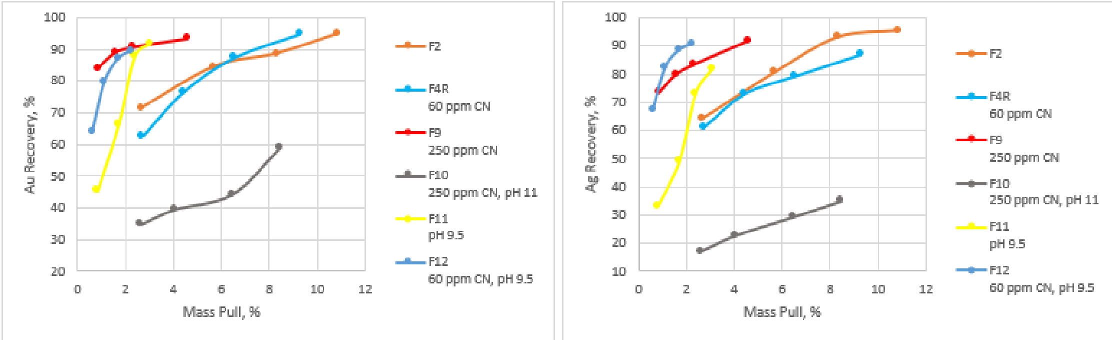


Note: Figure prepared by SGS Lakefield, 2020.

Figure 13-3: Gold and Silver Recovery Performance – Baseline Conditions

Subsequent tests using MC-HS sample evaluated the effect of different cyanide concentrations (between 0 and 500 mg/L NaCN) and pH (between 7 and 11) on the flotation results process, principally to determine whether the use of recycled process water containing cyanide would adversely affect flotation recovery.

Figure 13-4 shows the results of tests on the MC-HS sample at different cyanide concentrations and pH levels, in comparison with baseline test F2, which was completed at pH 7 with no cyanide addition. Note that the figure below reports cyanide concentrate (rather than sodium cyanide concentration).

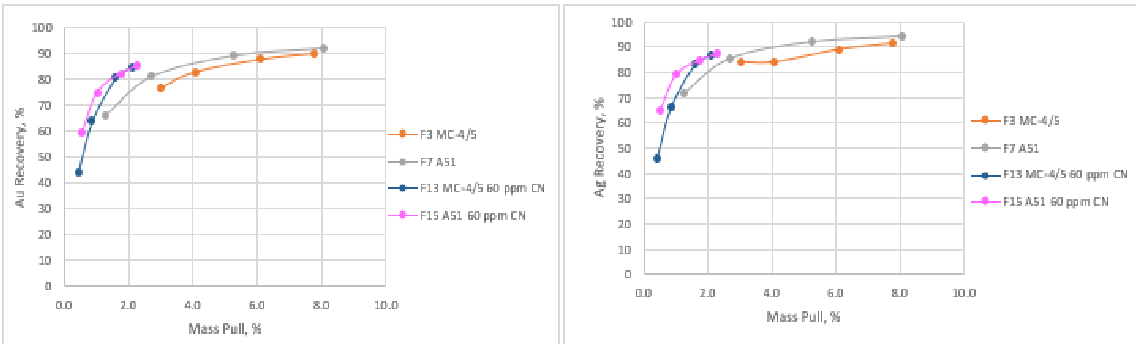


Note: Figure prepared by SGS Lakefield, 2020.

Figure 13-4: Effect of Different Cyanide and pH Levels on MC-HS

The results showed that recoveries were largely unaffected at flotation pH values <11, but at pH 11, for 2% mass pull, recoveries declined to 17% for Ag and 35% for Au. Based on these results, cyanide in the flotation feed can be tolerated or even be beneficial in flotation recovery, especially in case of silver, as observed from comparing F11 and F12, provided the pH can be controlled at around 9.5.

Similarly, results of batch tests (F13 and F15), as shown in Figure 13-5, highlighted the effect of 120 mg/L sodium cyanide in the flotation process solution when treating MC-4/5 and Area 51 samples, at pH 8.5, and showed that residual cyanide in the process solution had no adverse effect on flotation recovery.



Note: Figure prepared by Ausenco, 2020.

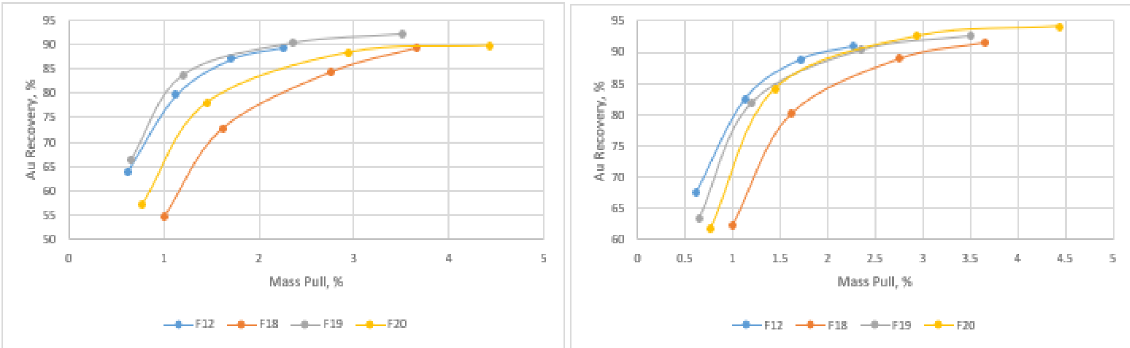
Figure 13-5: Effect of Cyanide on MC-4/5 and Area 51 Samples

Further testwork was performed using MC-HS to determine the preferred flotation reagent scheme. Tests F18–F20 evaluated the effect of different collectors and activators, in comparison to F12, which employed the reagent scheme applied to all of the baseline tests. Table 13-12 shows the reagent scheme used.

Table 13-12: Reagent Scheme Optimization

Test	Aero 3418A (gpt)	Aero 404 (gpt)	Pb $(NO3)_2$ (gpt)	CuSO4	PAX	Aero 208	Aero froth 65	NaCN Con. (mg/L)	pH
F12	5	5		50	25		5	120	9.5
F18		5		50		30	5	120	7.5
F19	5	5			25		5	120	7.5
F20	5	5	50		25		5	120	7.5

Results are illustrated in Figure 13-6.

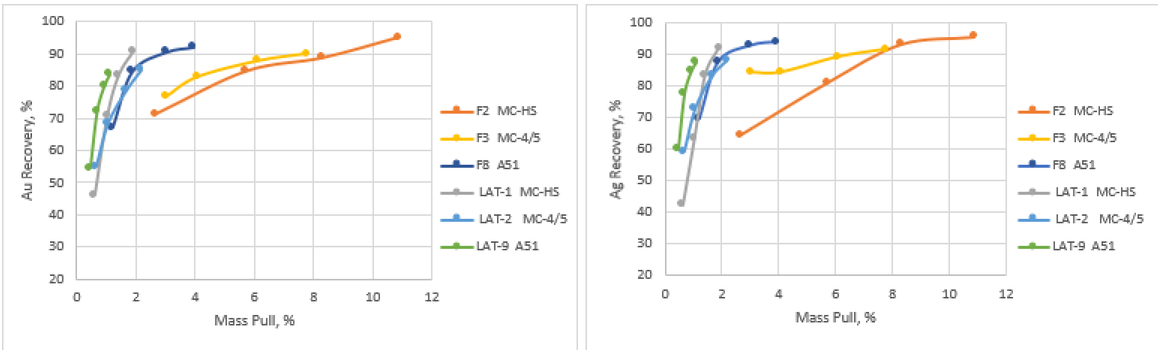


Note: Figure prepared by SGS Lakefield, 2020.

Figure 13-6: Reagent Scheme Optimization Results on MC-HS

Based on these results, a preferred reagent scheme including the following reagents was adopted for the remaining flotation testwork: 20 gpt PAX, 15 gpt Aero 208, and 5 gpt Aerofroth 65. The $CuSO_4$ was removed, as previous testing had suggested that residual cyanide was not detrimental to flotation results.

To improve the selectivity and froth recovery while limiting the concentrate mass pull to about 2%, the13-20baseline flotation tests were repeated using the same reagent scheme but with lower air flow rates and higher skimming frequencies. Two (2) tests were completed for each composite sample including MC-HS, MC-4/5, Babicanora Central, MC-4, and Babicanora Main (Area 51) samples. Figure 13-7 compares the results of the baseline tests (F series) at standard and low air flow rates (LAT series) for MC–HS, MC-4/5 and Area 51 Zone samples.

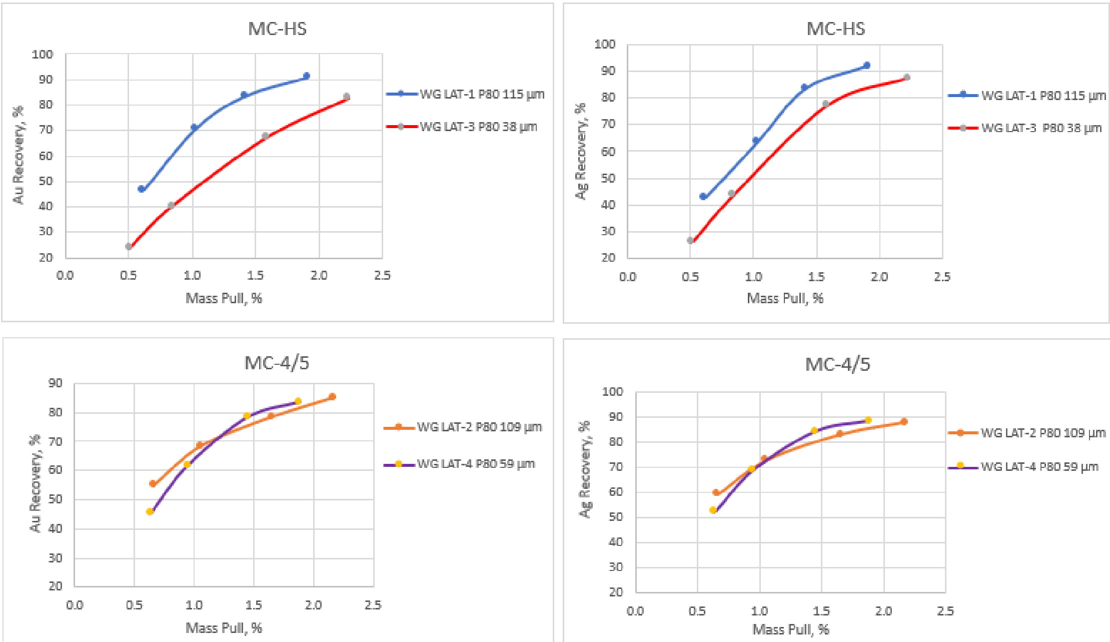


Note: Figure prepared by Ausenco, 2020.

Figure 13-7: Comparison of Baseline Tests – Standard and Low Air Flowrates

The results showed similar or higher recoveries for both gold and silver at a lower mass pull.

Further, directly comparable tests with low air flow rates on finer grind sizes for the same composite samples did not demonstrate any noticeable upgrade in recovery, as shown in Figure 13-8 for MC-HS and MC 4/5. In fact, recoveries were generally worse for both gold and silver for the MC-HS material, and effectively the same for the MC 4/5 material.

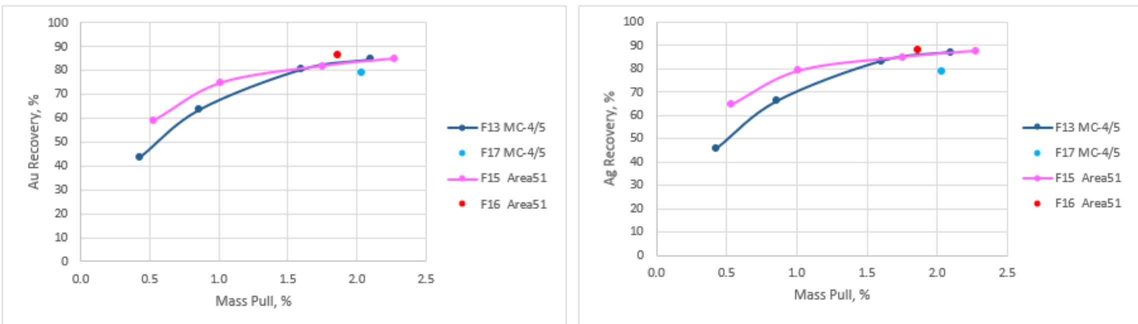


Note: Figure prepared by Ausenco, 2020.

Figure 13-8: Effect of Grind Size on Gold and Silver Recovery for Low Air Flowrates Flotation Tests

13.7.2 Bulk Flotation Tests

Bulk flotation tests with selected samples were completed using the preferred reagent scheme. Each bulk test involved processing either 50 kg or 10 kg of sample, using a 10 kg flotation cell. The concentrates from processing the individual 10 kg charges were combined in the case of larger samples. The results of the batch and bulk production tests for the MC-4/5 and Area 51 samples are presented in Figure 13-9.



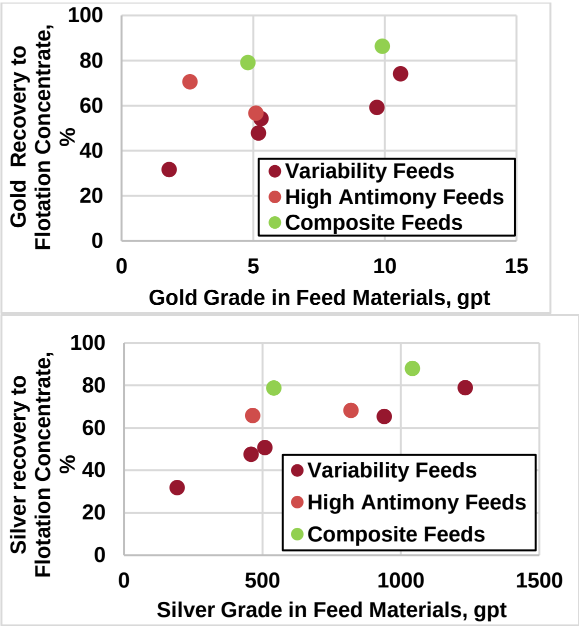
Note: Figure prepared by Ausenco, 2020.

Figure 13-9: MC 4/5 and Area 51 Batch and Bulk Concentrate Production Performance

Additional bulk flotation tests were completed using two (2) composite samples, specifically prepared to represent mineralization with elevated antimony contents (Sb Composite #1 and 2), and five composite samples which were prepared to represent feeds with varying silver grades in the range of 500–2,500 gpt AgEq (Ag Composites #1 to #5). All tests were completed at the natural pH, with 120 mg/L NaCN concentration, and with additions of PAX and Aero 208, along with frother Aero 65.

The objective of these tests was to evaluate the flotation response to mineralization types of varying head grade to gain confidence in recoveries achieved in the batch tests. A second objective was to determine the response of mineralization with elevated antimony and its effect on leach performance of the resulting flotation products.

Figure 13-10 shows the dependence of gold and silver recovery to the flotation concentrate on the grade of the feed materials.



Note: Figure prepared by Ausenco, 2020.

Figure 13-10: Gold and Silver Deportment to Concentrate as a Function of Grade in the Bulk Flotation Tests

Table 13-13 presents the recoveries achieved for the bulk flotation tests with the Area 51 and MC-4/5 composites and the higher antimony composites. Note that the combined gold and silver assays shown in Table 13-3 are the calculated head assays.

Table 13-13: Bulk Flotation Recovery Performance

Sample Composite	Test	Product	Wt %	Assays, gpt		% Distribution	
				Au	Ag	Au	Ag
Area 51	F16	Rougher Conc	1.9	460.0	49283	86.3	88.0
		Rougher Tail	98.1	1.38	128	13.7	12.0
		Combined	100	9.9	1042	100	100
MC -4/5	F17	Rougher Conc	2.0	188.0	20942	79.1	78.8
		Rougher Tail	98.0	1.04	117	20.9	21.2
		Combined	100	4.8	541	100	100
Sb Comp#1	F21	Rougher Conc	3	62.5	10377	70.6	65.8
		Rougher Tail	97	0.79	164	29.4	34.2
		Combined	100	2.6	465	100	100
Sb Comp#2	F22	Rougher Conc	1.2	247.0	47764	56.7	68.3
		Rougher Tail	98.8	2.23	263	43.3	31.7
		Combined	100	5.1	819	100	100

The higher antimony feed materials performed significantly poorer than the composite samples in the bulk flotation testwork, on average recovering about 19% less gold and 16% less silver to the concentrate fraction. Precious metals remaining in the flotation tailings fraction are still available for recovery in a subsequent cyanidation of the tailings, so do not represent overall losses.

13.8 Cyanidation of Flotation Concentrates and Tailings

Flotation concentrate and tailings solids produced from each bulk test underwent individual cyanide leach tests. The objectives of these leach tests were to maximize gold and silver extraction while also generating leach conditions that would be optimized for the plant operation.

13.8.1 Area 51 Zone and MC 4/5

In total, nine (9) flotation concentration cyanidation tests were completed using the Area 51 Zone (F-16) and MC-4/5 (F-17) samples. All cyanidation tests had a duration of 96 hr, with intermediate sampling at 24 hr. The flotation concentrate leach tests evaluated solids concentrations of 15% and 30% and cyanide concentrations between 2–5 g/L. The flotation tails leach tests were performed at solids concentrations of 48% and cyanide concentrations maintained at 2 g/L. Table 13-14 shows results of the tests.

Table 13-14: Area 51 Zone and MC-4/5 Bulk flotation Products Cyanidation Test Results

Sample	CN Test No.	CN Extraction, % Au 24 h	Au 48 h	Au 72 h	Au **96 h**	Ag 24 h	Ag 48 h	Ag 72 h	Ag **96 h**	Residue Grade Au, g/t **Avg.**	Residue Grade Ag, g/t **Avg.**	Calculated Head Au g/t	Calculated Head Ag g/t	Direct Head Au g/t	Direct Head Ag g/t
Area 51 F-16	221 Conc	~99	95.8	92.0	**90.6**	17.1	28.9	41.1	**44.2**	31.1	27910	329	50049	460	49283
	230 Conc	92.9	86.4	96.3	**83.1**	26.6	26.3	27.6	**31.6**	59.8	34545	355	50511		
	236 Conc	...	...	...	**98.8**	97.4	95.8	~99	**98.1**	4.71	1006	388	53409		
	236R Conc	...	...	...	**97.3**	80.9	~99	~99	**96.1**	11.20	1980	423	50885		
	223 Tail	86.8	72.8	75.0	**96.0**	77.7	86.0	85.2	**91.8**	0.10	13.7	2.39	168	1.38	128
	224 Tail	97.7	~99	~99	**92.5**	80.0	86.8	92.4	**90.9**	0.11	13.0	1.40	143		
	234 Tail	84.2	85.4	84.9	**86.2**	70.6	80.8	84.2	**88.1**	0.22	16.0	1.59	134		
MC-4/5 F-17	222 Conc	~99	~99	~99	**98.5**	58.4	79.6	95.8	**93.3**	2.12	1305	141	19500	188	20942
	228 Conc	~99	~99	~99	**98.6**	80.9	97.0	91.1	**92.2**	1.81	1499	125	19134		
	229 Conc	~99	~99	98.7	**98.0**	62.5	86.1	91.9	**95.3**	3.29	957	163	20272		
	232 Conc	~99	~99	98.1	**98.1**	61.2	81.5	92.3	**93.0**	2.34	1456	124	20769		
	237 Conc	...	...	...	**99.1**	51.3	90.2	88.3	**97.1**	1.49	695	162	24209		
	225 Tail	89.1	93.1	96.9	**95.5**	68.3	77.1	85.9	**87.7**	0.06	14.9	1.33	121	1.04	117
	226 Tail	92.7	99.5	95.0	**94.2**	76.7	84.2	86.2	**88.2**	0.06	15.2	1.04	129		
	233 Tail	91.2	95.2	96.4	**94.3**	61.8	78.5	86.8	**89.6**	0.08	13.2	1.31	127		

Note: Table prepared by SGS Lakefield, 2020.

In summary, cyanidation of the flotation concentrates under relatively aggressive conditions gave 96 hr extractions of about 98% for gold (regardless of sample) and between 93–98% for silver (higher in the Area 51 samples). Relative extractions from the tails samples were about 93% for gold with less dependency on the sample and about 89% for silver.

In summary, cyanidation of the flotation concentrates under relatively aggressive conditions gave 96 hr extractions of between 93–98% for silver (higher in the Area 51 samples) and about 98% for gold (regardless of sample). Relative extractions from the tails samples were about 89% for silver and about 93% for gold, with less dependency on the sample.

13.8.1.1 Flotation Concentrate Cyanidation

Two (2) leach tests, CN-221 and CN-230, were completed with the flotation concentrate produced from the Area 51 feed sample, and in both tests, silver extractions were poor. Due to the very high grade of the Area 51 flotation concentrate (~50 000 g/t Ag and ~400 g/t Au), it was concluded that the cyanide additions in the initial two tests (67 and 53 kg/t feed solids) were insufficient. The cyanide additions were increased to target a 4:1 mole ratio of NaCN:Ag to mitigate the potential for the precious metal extractions to be limited by free cyanide availablilty.

CN-221 and CN-230 were repeated (CN-236 and CN-236R, respectively), with significantly higher cyanide additions and at a lower solids concentration, 15% rather than 30% solids. As expected, results were improved; gold extraction increased from 87% to 98% and silver extraction increased from an average of 38% to 97%. This test provided further evidence of the lack of cyanide in initial tests with the variability samples (previously presented in Table 13-11).

Several tests (CN 222,228,229,232) were also completed on the MC 4/5 bulk flotation concentrate at cyanide additions which targeted a 4:1 mole ratio of NaCN:Ag, and at 30% solid density. Silver leach extractions were >92% for all tests. Silver extraction was further improved, to 97%, with increased cyanide additions which targeted a mole ratio of 7:1 NaCN:Ag and was performed at a 15% solid density (CN 237). These tests suggested that there is some scope in operation of the flotation concentrate leach circuit to trial high cyanide additions, relative to the silver grade, to realize a benefit from increased extractions, given the excess cyanide reports to the tailings leach circuit.

13.8.1.2 Flotation Tails Cyanidation

Six (6) cyanidation tests on flotation tails product from the Area 51 and MC-4/5 composites were completed. The cyanide addition and conditions for these tests matched previous gravity tails leach tests. Average extractions of 93% for gold were achieved and 89% for silver, with the best recovery of 96% for gold and 92% for silver in CN 223.

13.8.1.3 Cyanide Requirements

Based on cyanidation testwork completed with flotation products, sodium cyanide consumption ranged from 0.9—2.7 kg/t for flotation concentrate and from 0.6—1.6 kgpt for flotation tails. The testwork highlighted the benefit of ensuring sufficient cyanide is available in the concentrate leach operation, to encourage maximum extraction in this process step.

13.8.2 High Antimony Composites and Variable Silver Grade Composites

Two composite samples with elevated antimony head grades underwent bulk flotation testing (F21–F22) followed by cyanidation of the flotation products to evaluate the effect of high antimony values on the cyanidation performance. Table 13-15 presents the head grades for the antimony composites.

Table 13-15: Antimony Composites Head Grades

Sample	Au gpt	Ag gpt	Sb gpt
Sb Comp 1	3.91	527	129
Sb Comp 2	6.90	1177	157

Five (5) composite samples with varied head grades in the range of 500–2,500 gpt AgEq underwent bulk flotation tests (F23–F27) followed by cyanidation of flotation products. A total of 14 cyanidation tests was completed on the flotation concentrate and flotation tails with the objective of evaluating the overall recovery (in these cases precious metal values extracted in cyanidation were assumed to be quantitatively recoverable from pregnant solution). These tests employed the preferred reagent additions and leach conditions for both leach circuits. All cyanidation tests had a duration of 96 hr with intermediate sampling every 24 hr.

Table 13-16 presents a summary of the cyanidation results for the high-antimony and AgEq variability composites.

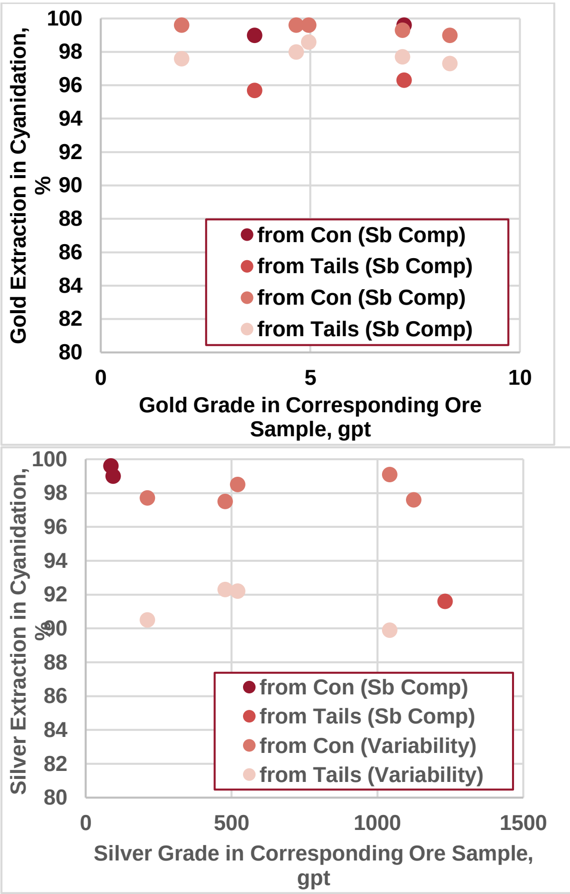
Table 13-16: Antimony Composites and Silver Composites Bulk Flotation Products Cyanidation Test Results

Sample	CN Test No.	CN Extraction, % Au 24 h	Au 48 h	Au 72 h	Au **96 h**	Ag 24 h	Ag 48 h	Ag 72 h	Ag **96 h**	Residue Grade Au, g/t Avg.	Residue Grade Ag, g/t Avg.	Calculated Head Au g/t	Calculated Head Ag g/t	Calculated Head Ag EQ g/t	Direct Head Au g/t	Direct Head Ag g/t
Sb Comp 1	243 Conc	...	...	...	**99.0**	36.9	79.9	96.7	**93.9**	0.83	708.5	80.1	11538		62.50	10377
F-21	241 Tail	97.0	95.2	96.2	**95.7**	47.7	56.1	64.1	**86.2**	0.06	21.7	1.39	170		0.79	164
												3.67	500		3.91	527
Sb Comp 2	244 Conc	...	...	...	**99.6**	~99	~99	99.4	**99.4**	1.63	473.0	390	80514		247	47764
F-22	242 Tail	99.7	98.3	99.0	**96.3**	85.2	87.2	89.4	**91.6**	0.10	22.7	2.59	269		2.23	263
												7.24	1232		6.90	1177
Ag Grade	250 Conc	...	...	...	**99.6**	77.3	76.3	66.1	**97.7**	0.12	76.5	29.1	3325		24.1	2599
Comp 1	245 Tail	94.3	97.0	95.3	**97.6**	67.0	81.6	84.6	**90.5**	0.03	12.7	1.26	134			
F-23												1.93	211	355		
Ag Grade	251 Conc	...	...	...	**99.6**	60.8	76.9	87.7	**97.5**	0.37	265.5	103	10812		114	10076
Comp 2	246 Tail	77.2	77.3	95.7	**98.6**	63.1	62.0	74.4	**92.3**	0.04	19.0	2.75	246			
F-24												4.96	478	850		
Ag Grade	252 Conc	...	...	...	**99.6**	69.6	87.1	88.8	**98.5**	0.53	209.5	118	14229		154.00	13905
Comp 3	247 Tail	93.3	94.7	97.1	**98.0**	63.6	81.5	77.6	**92.2**	0.05	20.0	2.46	255			
F-25												4.66	521	870		
Ag Grade	253 Conc	...	...	...	**99.3**	56.8	...	...	**99.1**	1.19	409.0	178	31301		321.00	39687
Comp 4	248 Tail	91.8	93.2	94.9	**97.7**	49.8	73.3	79.8	**89.9**	0.07	27.0	2.82	266			
F-26												7.20	1042	1582		
Ag Grade	254 Conc	...	...	...	**99.0**	78.5	83.4	93.1	**97.6**	2.19	1024.0	230	41950		305.00	32477
Comp 5	249 Tail	94.4	94.1	95.0	**97.9**	66.1	63.1	78.5	**91.0**	0.09	30.0	4.04	333			
F-27												8.33	1124	1749		

Note: Table prepared by SGS Lakefield, 2020.

In the tests completed, average gold extractions were 99% for the flotation concentrates and 97%, and average silver extractions were 98% for the flotation concentrates and 90% for the flotation tailings for the tailings.

Figure 13-11 below illustrates the effect of head grade on the cyanidation results, following flotation.



Note: Figure prepared by Ausenco, 2020.

Figure 13-11: Extraction of Gold and Silver in Cyanidation as a Function of Grade

It is worthy to note that both gold and silver extractions in cyanidation were relatively consistent across the range of grades tests, regardless of whether in the concentrate or tailings fraction. In the case of silver, calculated extractions are significantly higher in the concentrate leach step largely as a function of the higher silver content in the flotation concentrate.

Results of the testwork showed that the samples with elevated antimony returned comparable leach extractions for both gold and silver, from both concentrate and tailings fractions. This result suggests that elevated levels of antimony in the feed, as may be encountered within specific pockets of the deposit, have no significant adverse effect on leach extraction following flotation.

13.9 Combined Recoveries from Flotation and Cyanidation

Results of the testwork showed that the flotation products responded well to cyanide leaching and high gold and silver extractions were achieved. The overall (flotation + cyanidation) recoveries were then calculated for each test and the results are presented in Table 13-17.

Table 13-17: Overall Recovery from Flotation and Flotation Product Cyanidation

Sample Composite	Test No	Product	Test No	Flotation Recovery %		Cyanide Extractions %		Combined Recovery %	
				Au	Ag	Au	Ag	Au	Ag
Area 51	F-16	Conc	CN 236	86.3	88.0	98.8	98.1	85.3	86.3
		Tail	CN 223	13.7	12.0	96.0	91.8	13.1	11.0
		Combined		100.0	100.0			98.4	97.3
MC 4/5	F-17	Conc	CN 237	79.1	78.8	99.1	97.1	78.4	76.5
		Tail	CN 226	20.9	21.2	94.2	88.2	19.7	18.7
		Combined		100.0	100.0			98.1	95.2
Sb Comp 1	F-21	Conc	CN 243	70.6	65.8	99.0	93.9	69.9	61.7
		Tail	CN 241	29.4	34.2	95.7	86.2	28.1	29.5
		Combined		100.0	100.0			98.0	91.3
Sb Comp 2	F-22	Conc	CN 244	56.7	68.3	99.6	99.4	56.5	67.9
		Tail	CN 242	43.3	31.7	96.3	91.6	41.7	29.1
		Combined		100.0	100.0			98.2	96.9
Ag Comp 1	F-23	Conc	CN 250	31.6	31.9	99.6	97.7	31.5	31.2
		Tail	CN 245	68.4	68.1	97.6	90.5	66.7	61.6
		Combined		100.0	100.0			98.2	92.8
Ag Comp 2	F-24	Conc	CN 251	47.8	47.6	99.6	97.5	47.6	46.4

Sample Composite	Test No	Product	Test No	Flotation Recovery %		Cyanide Extractions %		Combined Recovery %	
				Au	Ag	Au	Ag	Au	Ag
		Tail	CN 246	52.2	52.4	98.6	92.3	51.5	48.4
		Combined		100.0	100.0			99.1	94.8
Ag Comp 3	F-25	Conc	CN 252	54.3	50.8	99.6	98.5	54.0	50.1
		Tail	CN 247	45.7	49.2	98.0	92.2	44.8	45.3
		Combined		100.0	100.0			98.9	95.4
Ag Comp 4	F-26	Conc	CN 253	74.1	78.9	99.3	99.1	73.6	78.2
		Tail	CN 248	25.9	21.1	97.7	89.9	25.3	18.9
		Combined		100.0	100.0			98.9	97.2
Ag Comp 5	F-27	Conc	CN 254	59.2	65.3	99.0	97.6	58.7	63.7
		Tail	CN 249	40.8	34.7	97.9	91.0	39.9	31.6
		Combined		100.0	100.0			98.6	95.3

Overall gold and silver recoveries of approximately 98 and 95%, respectively, were achieved through the combination of flotation and cyanidation of the flotation products. In particular, the overall gold and silver recoveries given this configuration proved relatively insensitive to grade in the feed, with the provision of sufficient cyanide and residence time (particularly in the concentrate leach operation) to accommodate high grade excursions and promote maximum silver recoveries. Additionally, the high antimony samples tested showed comparable results to life of mine average, and high-grade composite materials, and therefore antimony is not anticipated to present any significant metallurgical challenges during operation.

13.10 Liquid–Solid Separation Testing

Outotec completed flocculant screening, dynamic settling, and filtration testwork and SGS Lakefield completed dynamic settling and filtration testwork along with extensive static settling and batch filtration tests on variability samples.

All the liquid–solid separation testwork was completed on gravity tails samples but similar behaviour and results were expected from flotation tails. Three tests were done on gravity tailings prior to leaching, and the remaining tests were all done on gravity tails after leaching.

The samples used for liquid–solid separation testwork, are itemized in Table 13-18.

Table 13-18: Liquid- Solids Separation Test Summary

Sample ID	Test Program
Comp 1 Pre-leach	Flocculant screening; static and dynamic thickening; underflow rheology, CCD modelling
Area 51 Pre-leach (Outotec)	Flocculant screening; static and dynamic thickening; underflow rheology; CCD modelling
MC 3 Comp Pre-leach (Outotec)	Flocculant screening; static and dynamic thickening; underflow rheology; pressure filtration
MC 3 Comp CCD (Outotec)	Flocculant screening; static and dynamic thickening; underflow rheology; pressure filtration
MC3 Comp Cyanide Destruction (CND) (Outotec)	Flocculant screening; static and dynamic thickening; underflow rheology; pressure filtration
Area 51 Cyanide Destruction (CND	Flocculant screening; static and dynamic thickening; underflow rheology; pressure filtration
Variability Samples	Static settling at pH12 and pH 8.5; underflow pressure filtration at pH 8.5

The Area 51 sample was selected to represent the high-grade material with high clay content, and MC-3 was chosen as it was believed to be a representative of the overall mine plan at the time. Later it was determined that the MC-3 Composite was higher grade than the established overall mine plan, however, it has similar mineralogy to most of the material in the current mine plan.

13.10.1 Sample Characterization

For each bulk sample, the particle size distribution, solids SG and slurry pH were measured (Table 13-19).

Table 13-19: Sample Characterization

Sample	P50 µm	P60 µm	P80 µm	SG t/m³	pH
Comp 1 pre-leach	–	–	100	2.65	10.6
Area 51 pre-leach	56	71	103	2.65	8.7
MC-3 pre-leach	46	65	97	2.65	10.3
MC-3 CCD	41	59	97	2.65	12.1
MC-3 CND	36	59	99	2.65	8.9
Area 51 CND	–	–	114	2.73	8.5

Characterisation of the variability samples, and selected major constituents of the composites, is presented in Table 13-20. The P80 grind size and solids SG are comparable to the composites, as expected.

Table 13-20: Variability Sample Characterization

Sample ID	Particle Sizing				SG of Dried Solids
	[1]d_{80}, µm	[1]<20 µm % vol	[1]<1 µm % vol	[2]K_{80}, µm	
Babicanora Central	111	32.8	3.2	90	2.63
Babicanora Norte	119	27.0	2.6	99	2.66
William Tell	110	34.8	3.5	91	2.67
Las Chispas	114	29.3	2.9	90	2.62
Las Chispas Stockpile Bag 41	113	30.7	3.0	93	2.65
Comp 9	116	33.8	3.6	97	2.68
Comp 10	123	33.8	3.4	97	2.65
CN 116-117	122	30.5	2.9	98	2.63
CN 118-119	116	30.5	3.1	90	2.67
CN 130-131	102	36.8	3.5	88	2.67
CN 134-135	129	32.2	3.1	101	2.63
CN 132-133	138	31.8	3.2	116	2.62
CN 128-129	113	36.7	3.6	93	2.63
CN 124-125	129	30.7	3.3	100	2.67
CN 122-123	135	30.6	3.2	113	2.65

[1] Determined using laser diffraction.
[2] Determined using screen (sieve) analysis.

13.10.2 Dynamic Thickening

A summary of results from the dynamic liquid–solid separation tests is presented in Table 13-21.

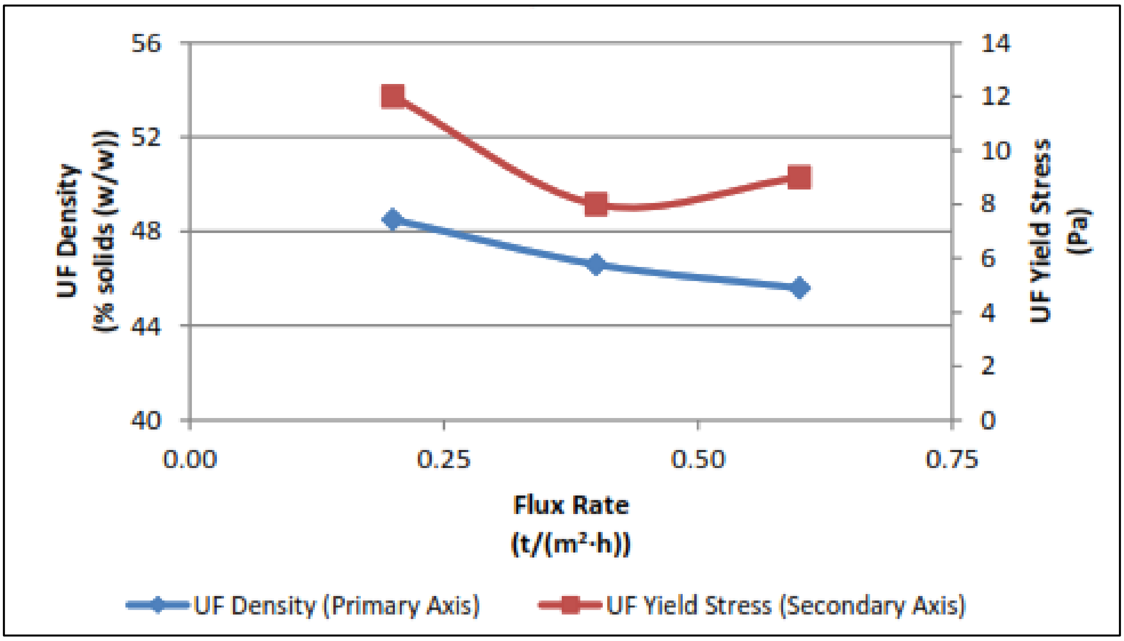
Table 13-21: Summary of Dynamic Settling Results

Sample	Flocc dose (gpt)	Solids loading (t/m².h)	Rise rate (m/hr)	Underflow Density (wt% solids)	Yield Stress (Pa)	Overflow (TSS mg/L)
Comp 1 pre-leach	35	0.5	3.9	44	7	146
Area 51 pre-leach	40	0.8	3.2	50	8	89
MC-3 pre-leach	60	0.6	3.0	46	9	290
MC-3 CCD	20	0.7	4.0	50	48	<100
MC-3 CND	30	0.6	3.0	53	56	<100
Area 51 CND	20	0.5	4.6	55	21	101

Note: TSS = total suspended solids.

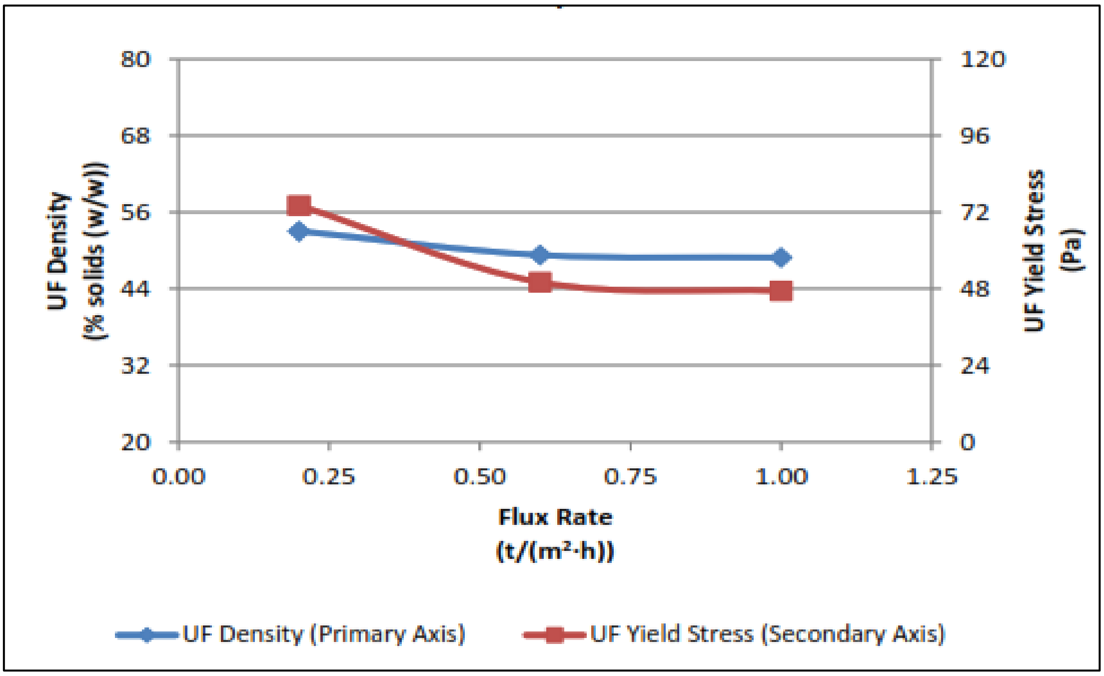
Gravity tailings slurries prior to cyanide leaching (pre-leach) showed poorer settling characteristics than corresponding materials post leaching (CCD) and post cyanide destruction (CND or tails). The addition of a coagulant was recommended for the pre-leach thickener to improve overflow clarity. No precious metal assays were completed on the overflow solids and there is potential that fine gold and silver particles could be lost in supernatant solutions if not recovered.

A comparison of the underflow density and settled solids yield stress as a function of solids loading (flux rate) for MC-3 is presented in Figure 13-12 for the pre-leach slurry, Figure 13-13 for the post-leach slurry and Figure 13-14 for the cyanide detoxification tailings stream.



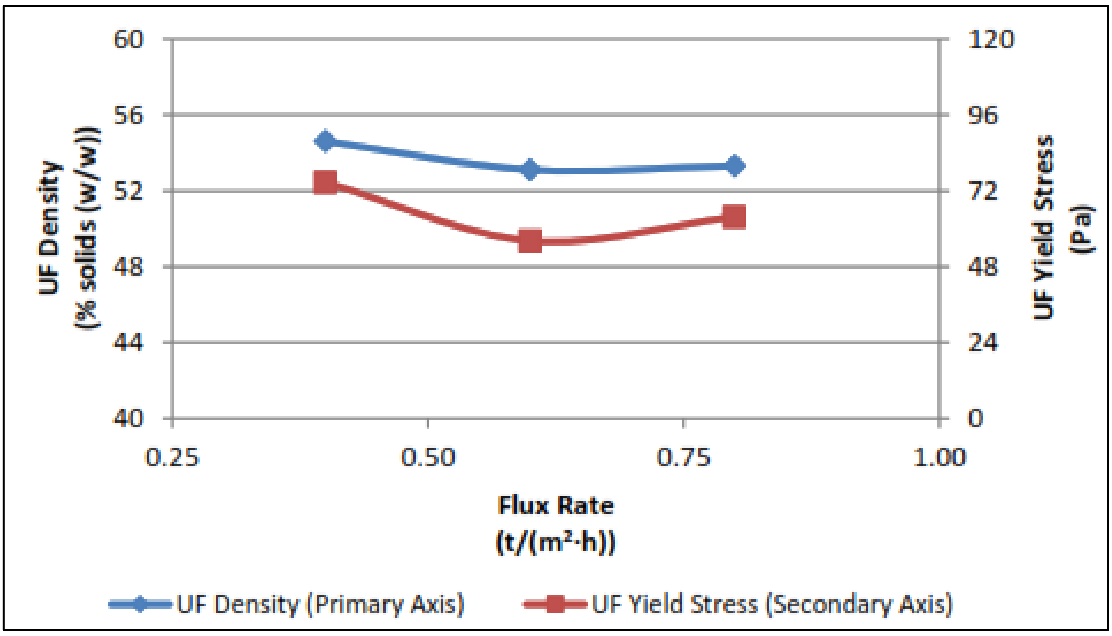
Note: Figure prepared by Outotec, 2020.

Figure 13-12: MC-3 Pre-leach Dynamic Settling and Yield Stress Relationship



Note: Figure prepared by Outotec, 2020.

Figure 13-13: MC-3 CCD Dynamic Settling and Yield Stress Relationship



Note: Figure prepared by Outotec, 2020.

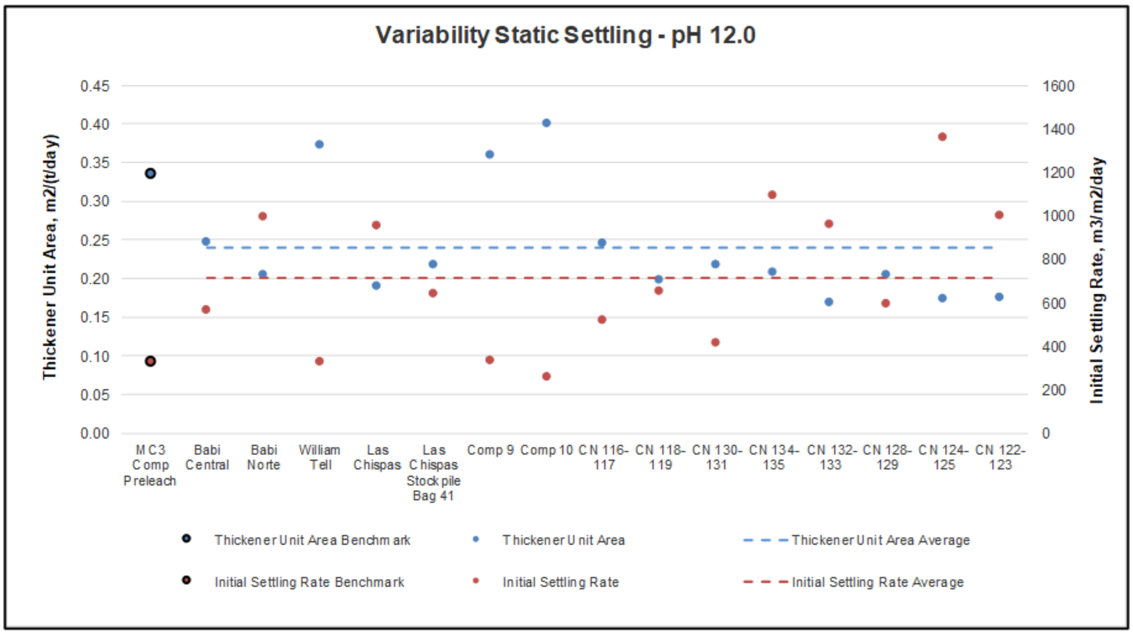
Figure 13-14: MC-3 CND Dynamic Settling and Yield Stress Relationship

Yield stress of the settled slurries was relatively low prior to cyanide leaching, measuring <12ºPa over the range tested and was noted to increase significantly following leaching, measuring up to 80 Pa in both the post leach and detoxified slurries. Design of the commercial thickener mechanisms and underflow slurry systems will need to accommodate these slurries.

13.10.3 Variability Sample Static Settling

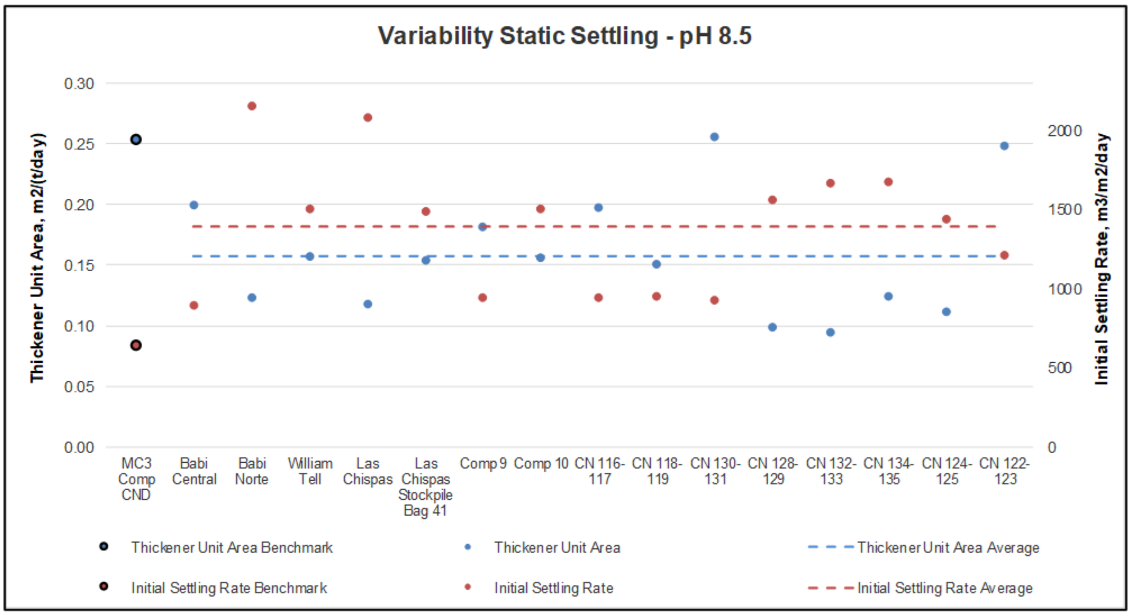
Fifteen (15) variability samples were subjected to static settling tests at pH 12 (post-leach or CCD) and pH 8.5 (post cyanide destruction) to evaluate variability in settling response.

Static settling results for composite samples, MC-3 CCD (pH 12) and MC-3 CND (pH 8.5), are included in Figure 13-15 and Figure 13-16 for comparison with results of individual variability samples.



Note: Figure prepared by SGS Lakefield, 2020.

Figure 13-15: Benchmark and Variability Static Settling Results at pH 12.0 (CCD)



Note: Figure prepared by SGS Lakefield, 2020.

Figure 13-16: Benchmark and Variability Static Setting Results at pH 8.5 (CND)

There was considerable fluctuation in the settling performance of the different samples for both CCD and CND streams; however, in most cases settling parameters of the variability results were better than those for comparable MC-3 samples.

For the tests at pH 12, the thickener unit area was calculated at the target of 50% underflow solid density. However, the range achieved within the time frame of the tests was between 42and 56%. Similarly, for the test performed at pH 8 thickener unit area was calculated at the target of 55% underflow solid density. However, the range achieved within the time frame of the tests was between 54 and 66% for most of the samples with of exception of 72% for two samples.

13.10.4 Rheology

Selected settled slurries produced in the dynamic thickening tests were sampled for rheology measurements. The rheological properties were determined using a rotating viscometer and a summary of the results is presented in Table 13-22.

Table 13-22: Summary of Rheology Parameters for Selected Composite Samples

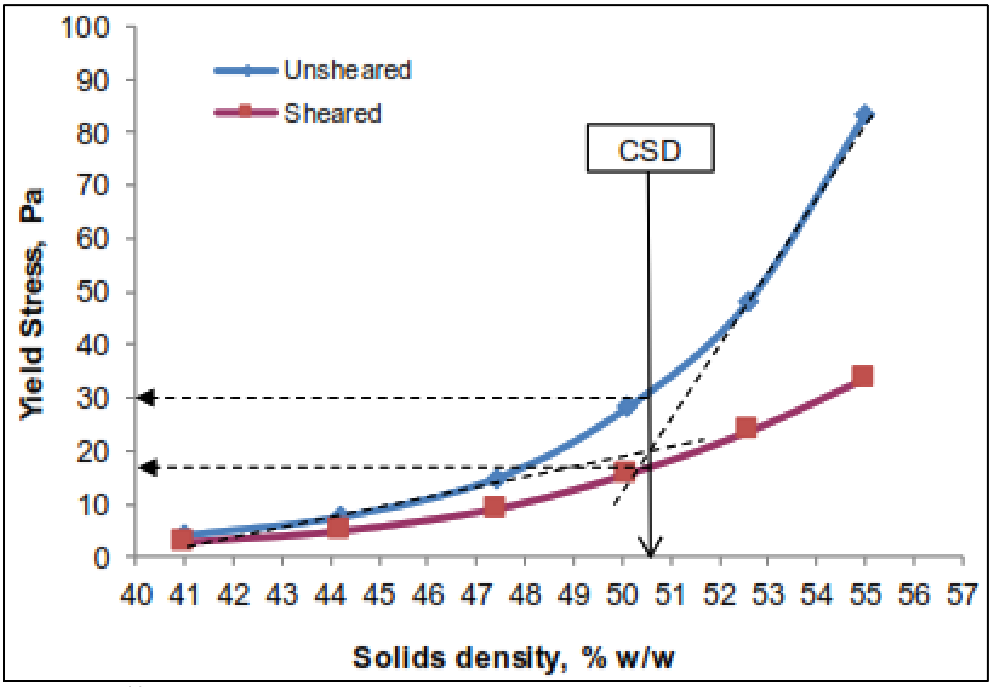
Sample ID	Critical Solids Density (CSD) %	Unsheared Yield Stress Pa	Sheared Viscosity mPa.s
Comp 1 Pre-leach	58	45	20
Area 51 Pre-leach	61	57	32
Area 51 CND	57	34	17
MC-3 CND	50.5	30	15

All of the slurry samples tested exhibited thixotropic (shear thinning) characteristics.

Critical solids content for slurry samples both prior to and following cyanide leaching were between 50–61%. The critical solids content is the slurry solids concentration at which slurry yield stress starts to increase faster than associated solids density and which requires a corresponding increase in pumping power per unit. The MC-3 CND (post-cyanide leaching and detoxification) sample provided the lowest critical solids density and had an unsheared yield stress of 30 Pa. The Area 51 slurry sample, collected prior to cyanide leaching, showed the highest unsheared yield stress at 57 Pa.

None of the samples tested provided a critical solids density less than the solids concentration specified in the process design criteria used for equipment sizing of 46%, ahead of cyanidation. Similarly, a solids concentration of 53% was used for design of post-cyanidation slurries, which provides sufficient margin relative to the measured critical solids density for the Area 51 mineralization. The MC-3 slurry showed a lower critical solids content than the value that was used in the equipment design. There may be some increase in pumping power requirements when processing MC-3 type material after the cyanidation circuits but not enough to affect the current pump design.

Further review of data from measurement of the MC-3 slurry, following cyanidation, as shown in Figure 13-17, highlighted that even though the design solids concentration is above the critical solids density, the unsheared yield stress at 53% solids is <60 Pa. Given the slurries are all shear thinning, design conditions under normal operations are unlikely to be problematic.



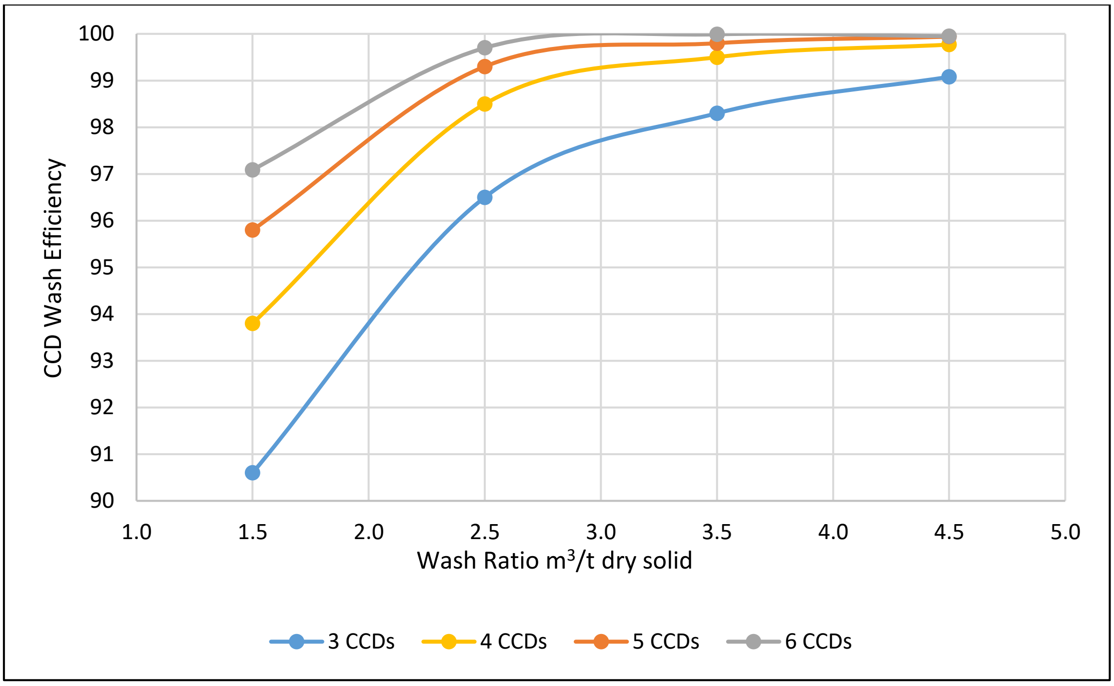
Note: Figure prepared by Ausenco, 2020.

Figure 13-17: Yield Stress Density Relationship for MC3 CND – Unsheared and Sheared

13.10.5 Counter-Current Decantation Modelling

Modelling of the CCD circuit was completed to determine the preferred number of CCD thickeners and wash ratios to provide the lowest economic soluble loss of precious metals. The results of the dynamic thickening tests for the MC-3 sample were used as the basis of the model and to calculate the CCD wash efficiency. CCD parameters including a leach discharge solids content of 48% w/w solids, CCD underflow density of 50% w/w solids in all stages, and a mixing efficiency of 90% were used in the modelling.

Figure 13-18 presents the CCD wash efficiency as a function of wash ratio and the number of CCD thickeners.



Note: Figure prepared by Ausenco, 2020.

Figure 13-18: CCD Modelling Wash Efficiency (wash efficiency vs wash ratio)

At a wash ratio of 3.5 :1 (volume of wash solution: volume of solution in the thickened solids) three stages of CCD washing results in a wash efficiency of 98.3% compared with 99.5% for four stages and 99.7% for five stages. Five stages of CCDs were nominated for the design, which enables bypass of a thickener at any time and provides sufficient flexibility to reduce the wash ratio while limiting soluble losses. Given five stages of CCD thickeners, the effect on overall gold and silver recovery will be only about 0.3%, which represents soluble losses which will be forwarded along with the detoxified slurry to the FTSF. As the bulk of this water will be recovered as filtrate and recycled to the process facility, the actual soluble losses which will be realized in operation may be less.

13.10.6 Pressure Filtration

It is expected that the tailings slurry, post cyanide detoxification, will be thickened and pumped to pressure filtration to generate a tailings filter cake that can be dry stacked for long-term storage. As samples stemming from treatment of the MC-3 mineralization are considered the most representative and proved to have the poorest liquid–solid separation characteristics, this composite was used for filtration testing to provide sizing criteria.

13.10.6.1.1 MC-3 CND Thickened Tailings

Outotec completed batch filtration tests with MC-3 CND thickened slurry, pH 8.9 and P80 of 99 mm, using a filter with 0.02 m^2 filtration area and equipped with a membrane squeeze.

Four (4) tests were completed at ambient temperature, using a filter feed slurry solids concentration of 47%. Results of the tests showed slow filtration rates, ranging from 110–135 kg/m^2•hr (excluding cycle time), and moisture contents of about 18% w/w after drying.

13.10.6.1.2 Area 51 CND Thickened Tailings

SGS Lakefield completed a series of pressure filtration tests on Area 51 CND thickened slurry at 57 wt% solids, pH 8.9 and P80 of 99 µm. A similar batch laboratory filter equipped with a membrane squeeze was used. Four ambient temperature tests were completed using a filter feed slurry solids concentration of 57 wt%.

The Area 51 sample exhibited superior filtration results in comparison to the MC-3 CND sample, with filtrations rates ranging from 563–818 kg/m^2•hr (excluding cycle time) and had a residual moisture content of about 15% w/w.

13.10.6.1.3 MC3 Comp CND Underflow

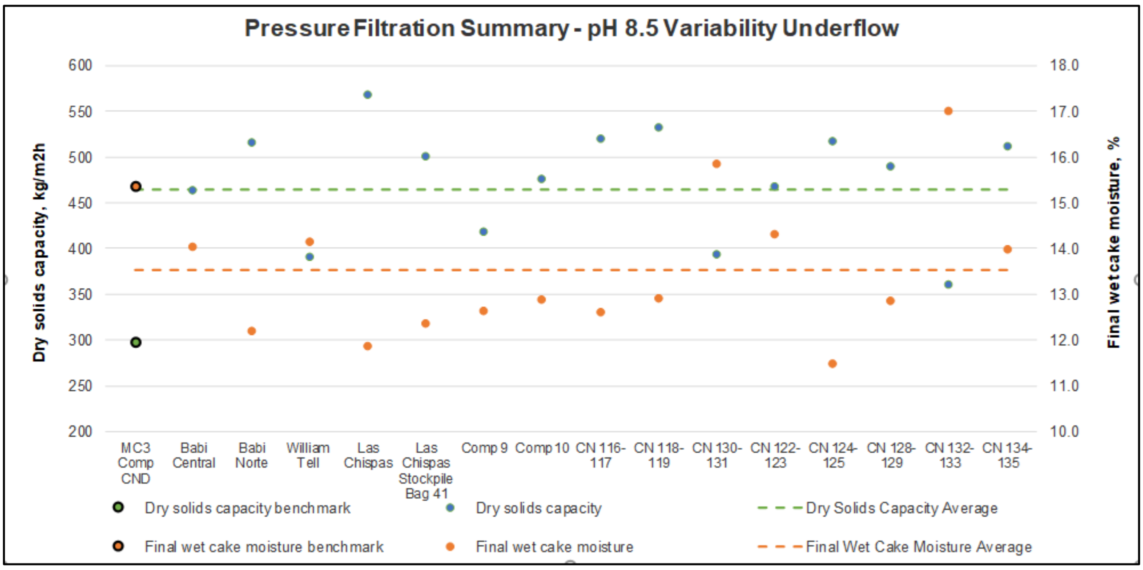
A further series of pressure filtration tests was completed by SGS Lakefield with MC-3 CND thickened slurry at 50.0% w/w solids and using mostly the same conditions as for the Area 51 samples (as described above).

MC-3 filtration rates ranged from 183–299 kg/ m^2•hr (excluding cycle time) with a residual moisture content ranging from 15–19% w/w, which is in excellent agreement with the test results achieved by Outotec on the same material.

13.10.7 Pressure Filtration – Tailings Variability Samples

Thickened tailings samples from the variability leach testwork were subjected to small-scale batch pressure filtration tests (using the same equipment at SGS Lakefield as described above) to evaluate the relative response.

Results of comparable filtration tests with the MC-3 CND tails served as the benchmark for comparison. All tests on the variability samples were completed with a feed slurry solids concentration of 50% w/w and with a target cake thickness of 15 mm. The results are summarized in Figure 13-19.



Note: Figure prepared by SGS Lakefield, 2020.

Figure 13-19: Variability Pressure Filtration Overall Plot – pH 8.5 Underflow

The resulting filtration rates ranged from 299–568 kg/m^2·h with discharge cake residual moisture contents ranging from 12–16% w/w.

In general, slurries produced from treatment of MC-3 mineralized material exhibited the poorest filtration results in terms of filtration rate and at the upper level of moisture rate. Therefore MC-3 results would provide a conservative design for all the material for the pressure filters.

13.11 Precious Metals Recovery (Merrill Crowe)

The Merrill Crowe process is historically well established, and widely applied in modern commercial practice for high silver-bearing, precious metal ores. The process uses zinc dust precipitation, including complete clarification and deaeration of the pregnant solution before the addition of zinc dust, with subsequent separation of the precipitate from the solution by filtration. The objectives of the Merrill-Crowe tests completed as part of the current program were to confirm the efficacy of gold and silver recovery in the presence of free cyanide, and to develop a basis for the zinc dust requirement. An additional objective was to generate sufficient solution for subsequent use in cyanide detoxification testing.

The precious metal recovery testwork, was completed using the pregnant leach solution (PLS) from the bulk cyanidation test (CN-81). CN-81 was the bulk leach of gravity concentrate leach residue (from CN-80) combined with G3 gravity tailings produced by treatment of the Area 51 Composite, using the preferred leach conditions. Upon completion of test CN-81, the pulp was filtered and washed using a 3:1 wash ratio. The PLS and wash liquor were then combined and used for 2 L and 30 L Merrill-Crowe precipitation tests.

A summary of the Merrill-Crowe solution analyses, including analyses of the feed solution (and its constituent streams) and barren solution is presented in Table 13-23 below. The results for the larger scale test are also presented for comparison as they are considered to represent a relative worst case.

Table 13-23: Summary of Merrill Crowe Feed and Merrill Crowe Barren Solution Analysis

Element	Unit	PLS CN-81	Wash CN-81	MC Feed Solution (PLS+Wash)	Barren Solution 2L Test	Barren Solution 30L Test
Au	mg/L	8.20	1.24	2.98	<0.01	0.31
Ag	mg/L	1140	131	383	0.08	2.2
CN_{total}	mg/L	1270	183	455	399	399
C_{WAD}	mg/L	1070	183	405	399	412
CNS	mg/L	260	28	86	81	82
CNO	mg/L	18	2.1	6	6.0	6.6
CN_{free}	mg/L	662	99.8	240	—	—
As	mg/L	<30	<3	—	—	—
Cu	mg/L	112	33.6	—	40.7	46.9
Fe	mg/L	<2	0.8	—	—	—

Element	Unit	PLS CN-81	Wash CN-81	MC Feed Solution (PLS+Wash)	Barren Solution 2L Test	Barren Solution 30L Test
Ni	mg/L	<6	<0.6	—	—	—
Zn	mg/L	8	<2	—	194	204

The Merrill-Crowe feed solution contained sufficient free cyanide (>250 mg/L NaCN) and the pH (>11) was high enough that there was no requirement to add either cyanide or lime prior to treatment with zinc dust. The sample was filtered deaerated prior to testing. Zinc was added at five times the stoichiometric requirement for gold and silver precipitation to promote quantitative recovery of the gold and silver values. Lead nitrate was added at 25% of the zinc addition. Results showed the gold and silver values in the barren solution were low, which indicated a high precipitation efficiency. In the worst case, approximately 10% of the gold and 0.6% of the silver were left in the barren solution following precipitation. Under more typical circumstances, as illustrated in the smaller scale tests, <0.5% of the gold and 0.1% of the silver can be expected to be left in solution.

13.11.1 Merrill-Crowe Optimization Tests

A series of Merrill Crowe tests were completed to investigate the effects of zinc additions on the recovery of gold and silver. Cyanide solution from bulk leaching of MC-3 material was used for the tests. Following combination of filtrate and wash solution, the Merrill-Crowe feed solution contained 3.05 mg/L Au and 239 mg/L Ag. The feed solution was representative of the expected Merrill-Crowe feed solution under commercial conditions.

Results from the Merrill-Crowe optimization testwork, completed at a constant pH of 11.8 and NaCN concentration of 250 mg/L NaCN, showed that there was very little benefit in precious metal recovery with zinc dust additions beyond 1.5 the stoichiometric requirement. Average residual gold and silver in the barren solution was about 0.5% and 0.1%, respectively, relative their concentration in the feed solution regardless of increased zinc additions.

13.11.2 Mercury Content of the Zinc Precipitate

With the presence of appreciable mercury in the high-grade samples sourced from Area 51 (up to 2.4 gpt Hg) and given that mercury could report to the zinc precipitate in the Merrill Crowe process, selected precipitate samples from the Merrill Crowe tests were assayed for mercury.

Table 13-24 shows the deportment of mercury from selected mineralization types to the corresponding zinc precipitates across the unit operations tested.

Table 13-24: Mercury Deportment to Zn Precipitate

Sample	Test	Hg in Mineralized Material (gpt)	Hg in Zn Precipitate (gpt)	% of Feed Hg Reporting to Zn Precipitate
Area 51	MC-2 Precipitate	2.4	70	8.5
MC-3	MC-3 Precipitate	0.4	31	16.7

These tests were performed with a zinc addition of five times the stoichiometric requirement, to present a condition which would simulate a maximum collection of mercury in the zinc precipitate. On the basis of analyses collected throughout the test program, about 30% of the mercury in the mineralized material may dissolve and report to the pregnant solution. In the worst case about half of the dissolved mercury would report to the zinc precipitate, representing up to 17% of the amount in the feed to the mill. The remaining mercury in the feed, of about 70%, is expected to remain undissolved, and will report to the final tails.

The deportment of mercury appeared to be relatively independent of its concentration in the feed.

The mercury content of the zinc precipitate produced in the above tests suggest that there will be sufficient mercury present in the zinc/precious metals precipitate to require the inclusion of a mercury retort to remove mercury prior to smelting.

13.11.3 Deleterious Elements

Based on the testwork performed to date, two deleterious elements, antimony and mercury, were identified. Antimony showed no adverse effect on gold and silver recoveries, as described herein. Mercury is anticipated to be extracted to a small extent and that dissolved is expected to report directly to the zinc precipitate. Therefore, a mercury retort facility is recommended for the commercial facility to recovery the mercury attendant with the zinc precipitate and minimize its deportment to the doré product.

There are no other deleterious elements in the mineralization which would require special processing techniques or compromise doré marketability.

Area 51 mineralization with high clay content did not present any problematic issues on gold and silver recovery or material handling.

13.12 Cyanide Detoxification

Combined leach slurries (post CCD) from treatment of Area 51 and MC-3 composites were individually submitted for cyanide detoxification testwork. Cyanide detoxification using a combination of sulphur dioxide (SO_2) and air is a well-established process that has been widely adopted in commercial operations. The testwork objectives were to confirm the nominal condition for producing treated pulp containing <1 mg/L weakly acid dissociable cyanide (CN_{WAD}) and to provide indicative reagent requirements for use in the commercial design.

In all cases copper was added as $CuSO_4.5H_2O$ and SO_2 was added as sodium metabisulphite (SMBS) solution. Oxygen was used in preference to air.

Five (5) continuous cyanide detoxification tests were completed to test various SMBS and/or copper additions on the efficiency of cyanide detoxification, and one test was operated to evaluate residence time. Each continuous test was operated for sufficient duration to provide a minimum of three displacements for the detoxification tank. The tests were completed at solids concentration of 48% w/w, and with a pH target of approximately 8.5, using SMBS. All tests were operated at room temperature and with a retention time of 60 minutes.

Cyanide detoxification test conditions and results for solutions corresponding to Area 51 mineralization are shown in Table 13-25, and results corresponding to the MC-3 composites are presented in Table 13-26 (MC-3 Comp).

Table 13-25: Cyanide Destruction Test Results – Area 51

Test	Ret'n Time Min	pH	CN_T	Product Solution Phase mg/L				Reagent Addition					
								g/g CN_{WAD}			kgpt Solids		
				CN_{WAD} Analytical	CN_{WAD} Picric	Cu	Fe	SO_2 Equiv.	Lime CaO	Cu#	SO_2 Equiv	Lime (CaO)	Cu#
Area 51 FEED (CN82/MC-2C)		10.6	374	362	-	42.6	1.48	-	-	-	-	-	-
CND 1 Batch Test (17L)	270	8.5	0.6	<0.1	<0.1	0.4	0.6	10.2	8.1	0.04	4.0	3.2	0.02
Area 51 FEED (CN81/MC-2C)		11.0	324	337	-	36.9	0.94	-	-	-	-	-	-
CND 2 Batch (1L)	156	8.7	-	-	0.68	-	-	5.1	5.9	0.04	1.9	2.2	0.02
Continuous Tests													
2-1	62	8.5	<0.1	<0.1	0.62	9.7	<0.1	5.7	1.8	0.03	2.1	0.65	0.01
2-2	60	8.5	0.20	<0.1	2.4	10.0	0.2	5.9	2.1	0.03	2.2	0.77	0.01
2-3	59	8.5	-	-	4.6	-	-	6.5	2.2	0.00	2.4	0.79	0.00
2-4	57	8.5	0.40	<0.1	2.6	11.6	0.2	5.8	2.3	0.09	2.1	0.74	0.03
2-5	65	8.5	0.22	0.17	4.5	18.4	0.1	5.6	1.4	0.04	2.0	0.51	0.02

Table 13-26: Cyanide Destruction Test Results – MC-3 Comp

Test	Ret'n Time Min	pH	CN_T	Product Solution Phase mg/L				Reagent Addition							
								g/g CN_{WAD}				kgpt Solids			
				CN_{WAD} Analytical	CN_{WAD} Picric	Cu	Fe	H_2SO_4	SO_2 Equiv.	Lime CaO	Cu#	H_2SO_4	SO_2 Equiv	Lime (CaO)	Cu#
MC-3 Composite FEED (CN101/MC-3)		11.4	289	283	-	22.5	2.2	-	-	-	-	-	-	-	-
CND 3 Batch Test (17 L)	210	8.6	<0.1	<0.1	0.31	3.2	<0.1	-	3.8	0.93	0.05	-	1.2	0.3	0.02
MC-3 Composite FEED (CN100/MC-3)		10.8	218	231	-	32.7	<0.2	-	-	-	-	-	-	-	-
CND 4 Batch Test (1 L)	180	8.6	-	-	<0.1	-	-	-	7.2	10.2	0.06	-	1.8	2.6	0.02
Continuous Tests															
4-1	60	9.6	-	-	7.3	-	-	-	7.2	0.4	0.05	-	1.8	0.1	0.01
4-2	58	9.4	-	-	2.8	-	-	-	8.2	2.3	0.04	-	2.1	0.6	0.01
4-3	56	8.5	0.93	0.06	<0.1	0.4	0.3	6.4	7.9	-	0.04	1.6	2.0	0.0	0.01
4-4	55	8.5	1.78	<0.1	<0.1	0.2	0.4	0.4	7.9	-	0.00	0.1	2.0	0.0	0.0
4-5	58	8.5	0.95	0.2	<0.1	1.3	0.2	3.7	6.8	-	0.00	0.9	1.7	0.0	0.0
4-6	57	8.5	1.00	0.5	1.1	3.4	0.2	8.5	4.5	-	0.00	2.1	1.1	0.0	0.0
4-7	59	9.6	-	-	35.4	-	-	-	4.5	0.0	0.00	-	1.1	0.0	0.0

CN_{WAD} was shown to be effectively reduced to <1 mg/L within 60 minutes for both Area 51 and MC-3 Composites.

In the tailings streams generated from both composites, cyanide detoxification under the following conditions proved to be adequate in achieving <1 mg/L CN_{WAD} in the continuous tests:

- Feed slurry solids concentration of 48% w/w;
- >7.0 g SO2: g CN_{WAD};
- 10 mg/L copper sulfate in solution;
- pH 8.5; and,
- 60-minute retention time.

These conditions are well aligned with industry norms.

13.13 Recovery Projection

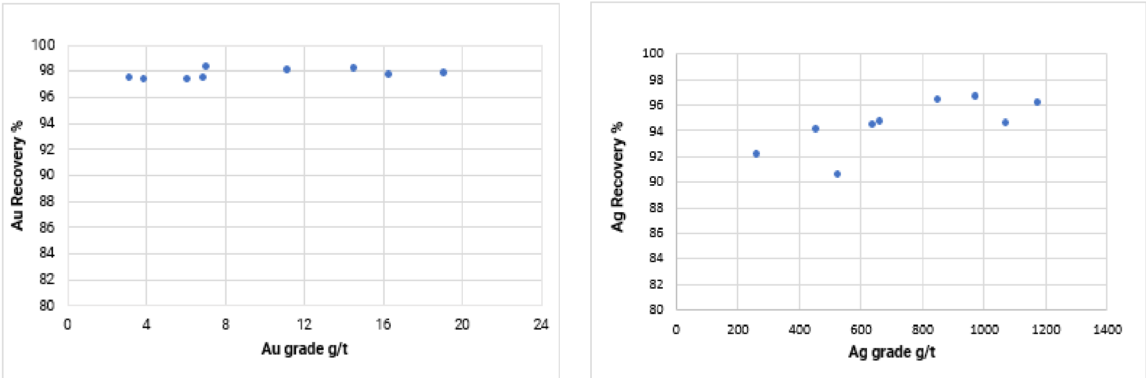
Testwork indicated that the preferred flowsheet for the Las Chispas deposit should include flotation, cyanide leaching of both the concentrate and tails fraction, CCD washing, and precious metal recovery by the Merrill Crowe process.

Table 13-27 presents LOM average gold and silver recovery forecasts.

Table 13-27: Forecast Life of Mine Average Au and Ag Recovery

Product	Flotation Recovery %		Cyanide Extractions %		CCD Recovery %	Merrill Crowe Recovery %	Overall Recovery	
	Au	Ag	Au	Ag			Au	Ag
Conc	62.2	63.9	99.3	97.7	99.7	99.5	61.3	61.9
Tail	37.8	36.1	96.9	90.4	99.7	99.5	36.3	32.4
Combined							**97.6**	**94.3**

Based on the selected flowsheet for the plant operation, over the range of grades (variability samples) tested, the recoveries for gold and silver varied from 97.2–98.3% and 90.5–96.6% and respectively, as presented in Figure 13-20.



Note: Figure prepared by Ausenco, 2020.

Figure 13-20: Au and Ag Recoveries for Variable Grades

For the purposes of economic evaluation, gold recoveries were shown to be relatively independent of grade over the ranges tested. Use of an average recovery value of 97.6% would be appropriate. Conversely, overall silver recovery was shown to vary almost linearly with grade over the ranges tested. The silver recovery is relatively predictable therefore and can be estimated at 94.3% on the basis of the LOM average grade determined in the mine plan.

13.14 Conclusions

Over 200 variability and composite leach tests were conducted, and a gravity pre-concentration step was included. The proposed flowsheet incorporates the following steps:

- Comminution;
- Gravity pre-concentration;
- Cyanide leaching of the gravity concentrate and tailings fractions;

- CCD washing of the leach residue, and ultimately cyanide detoxification of the tailings slurry prior to filtration and long-term storage; and,
- Merrill Crowe recovery of precious metals and conventional smelting to generate a doré.

Overall, high recoveries were achieved at laboratory scale by application of conventional, commercially proven processes. However, due to the brittle nature of the silver sulphide mineral, argentite, there is a concern that the mineral could overgrind in practice and result in reduced gravity recovery. Therefore, pre-concentration using flotation was pursued with favourable results from 40 flotation tests:

- Gravity concentration: 40–50% gold and silver recovery and 4% mass pull; and,
- Flotation concentration: 60–80% gold and silver recovery and 2% mass pull.

An extensive campaign was commissioned late in the Feasibility Study to provide confidence in gold and silver extractions post flotation. This included evaluation of the response of high-grade mineralization and samples with elevated antimony values. Overall recovery, under the most favourable conditions tested, ranged from 98–99% for gold and 91–97% for silver.

It was determined that the flowsheet incorporating flotation followed by separate cyanidation of the concentrate and tailings was the most appropriate design for Las Chispas mineralized. Over the range of samples tested, overall gold recovery was determined to be relatively insensitive to grade, whereas silver varied linearly with increasing grade.

14 Mineral Resource Estimates

14.1 Introduction

The purpose of this Technical Report section is to update the 2019 Mineral Resource Estimate with the Extended Phase III 2019-2020 drilling program. The data cut-off date for data supporting Mineral Resource estimation is October 16, 2020. The Mineral Resource Estimate includes in-situ narrow vein gold and silver mineralization at the Babicanora and Las Chispas areas, and gold and silver mineralization contained within surface stockpiles that resulted from historical operations.

The Mineral Resources Estimate presented herein is reported in accordance with the Canadian Securities Administrators' National Instrument 43-101 and were estimated in conformity with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Estimation of Mineral Resource and Mineral Reserves Best Practice Guidelines" (November, 2019) and reported using the definitions set out in the 2014 CIM Definition Standards on Mineral Resources and Mineral Reserves. Mineral Resources that are not converted to Mineral Reserves do not have demonstrated economic viability. Confidence in the estimate of Inferred Mineral Resource is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Mineral Resources may be affected by further infill and exploration drilling that may result in increases or decreases in subsequent Mineral Resource Estimates. Mineral Resources in this estimate are inclusive of the Mineral Reserves stated in Section 0.

This Mineral Resource Estimate was based on information and data supplied by SilverCrest, and was undertaken by Yungang Wu, P.Geo. and Eugene Puritch, P.Eng., FEC, CET of P&E Mining Consultants Inc. of Brampton, Ontario. Both QPs are independent of SilverCrest as independence is defined in NI 43-101.

The effective date of this Mineral Resource Estimate is October 16, 2020.

14.2 Previous Mineral Resource Estimate

The most recent previous public Mineral Resource Estimate for the Las Chispas Deposit was prepared by Tetra Tech Canada Inc. (Tetra Tech) with an effective date of February 8, 2019 and amended on May 15, 2019. The Mineral Resource Estimate at a cut-off value of 150 gpt AgEq is presented in Table 14.1. This previous Mineral Resource Estimate is superseded by the Mineral Resource estimate reported herein.

Table 14-1: Las Chispas Mineral Resource Estimate by Tetra Tech, Effective Date February 8, 2019

Type	Cut-off AgEq[1, 2] (gpt)	Classification	Tonnes[3]	Au (gpt)	Ag (gpt)	AgEq (gpt)	Contained Au (oz)	Contained Ag (oz)	Contained AgEq[2] (oz)
Vein	150	Indicated	1,002,200	6.98	711	1,234	224,900	22,894,800	39,763,600
Vein	150	Inferred	3,464,700	3.42	343	600	380,700	38,241,400	66,823,700
Stockpile	100	Inferred	174,500	1.38	119	222	7,600	664,600	1,246,100
Overall	-	Indicated	1,002,200	6.98	711	1,234	224,900	22,894,800	39,763,600
Overall	-	Inferred	3,639,000	3.32	333	582	388,300	38,906,000	68,069,800

Notes:

(1)Vein Mineral Resource was reported using a 150 gpt AgEq COG and minimum 0.5 m true width; the Babicanora Norte, Babicanora Sur, Babicanora FW, and Babicanora HW Veins were modelled to a minimum undiluted thickness of 0.5 m; the Babicanora Main Vein was modelled to a minimum undiluted thickness of 1.5 m.

(2)AgEq is based on a gold to silver of 75:1. This was calculated using respective long-term gold and silver prices of $1,225/oz gold and $17.0/oz silver, with approximate average metallurgical recoveries of 95% gold and 90% silver.

(3)Bulk density has been applied to all materials at 2.55 t/m^3.

All numbers are rounded. Overall numbers may not be exact due to rounding.

14.3 Database

All drilling and assay data were provided by SilverCrest in the form of Excel data files up to and including a data cut-off date of October 16, 2020. The GEOVIA GEMS™ V6.8.2 database for this Mineral Resource Estimate consisted of surface drill holes, underground drill holes and underground channel and chip samples for the in-situ narrow veins and surface channel and RC samples for the historical stockpiles (Table 14-2). Drill hole plans are shown in Appendix A.

Table 14-2: Database Summary

Area	Data Type	Number	Total Metres
Babicanora	Surface drill holes	1,158	337,507
	Underground drill holes	131	15,474
	Total drill holes	1,289	352,981
	Underground channel/chip	646	2,671
Las Chispas	Surface drill holes	229	66,134
	Underground drill holes	40	4,157
	Total drill holes	269	70,291
	Underground channel/chip sample points	6,739	
Other*	Surface drill holes	68	13,888
Stockpiles	Sample points	2,185	

Note: * drill holes named with LOC, LV, WELL, POZO, RA, VNT, GTP including re-drilled holes and assays pending as of October 16, 2020 are not included in the Mineral Resource Estimate.

The database contains assays for gold and silver and other elements of non-economic importance. The basic gold and silver raw assay statistics and sample lengths are presented in Table 14-3.

Table 14-3: Las Chispas Assay Database Summary

Dataset	Variable	Au	Ag	Length
Babicanora Drill Holes	Number of samples	103,455	103,455	104,455
	Minimum value *	0.001	0.20	0.18
	Maximum value *	3,366.00	114,814	7.70
	Mean *	0.28	22.7	0.99
	Median *	0.05	5.0	0.90
	Variance	149.57	261,778	0.22
	Standard deviation	12.23	511	0.47
	Coefficient of variation	44.36	22.54	0.47

Dataset	Variable	Au	Ag	Length
	Skewness	224.99	139.83	1.21
	Kurtosis	58,314.70	27,525.87	6.40
Babicanora UG Channel / Chip samples	Number of samples	4,640	4,640	4,640
	Minimum value *	0.01	0.20	0.17
	Maximum value *	469.00	39,331	2.59
	Mean *	6.88	574	0.58
	Median *	0.14	69.3	0.55
	Variance	640.80	3,242,872	0.03
	Standard deviation	25.31	1,800	0.17
	Coefficient of variation	3.68	3.14	0.29
	Skewness	7.94	8.37	1.46
	Kurtosis	91.92	109.86	11.83
Las Chispas Drill Holes	Number of samples	35,304	35,304	35,304
	Minimum value *	0.01	0.20	0.25
	Maximum value *	513.00	42,322	5.80
	Mean *	0.13	13.3	1.03
	Median *	0.05	1.8	1.00
	Variance	11.00	103,185	0.23
	Standard deviation	3.32	321.22	0.48
	Coefficient of variation	25.90	24.19	0.47
	Skewness	120.73	98.54	1.01
	Kurtosis	17,192.20	11,485.45	3.78
Las Chispas UG Channel / Chip Samples	Number of samples	6,739	6,739	N/A
	Minimum value *	0.001	0.20	N/A
	Maximum value *	136.00	10,000.00	N/A
	Mean *	0.81	120	N/A
	Median *	0.05	10.0	N/A
	Variance	11.14	167,041	N/A
	Standard deviation	3.34	408.74	N/A
	Coefficient of variation	4.13	3.40	N/A
	Skewness	16.44	10.22	N/A
	Kurtosis	484.11	170.25	N/A

Note: UG = underground. N/A = not applicable.*Au and Ag units are gpt and units are metres.

All drill hole survey and assay values are expressed in metric units, with grid coordinates reported using the WGS84, zone 12N UTM system.

14.4 Data Verification

P&E validated the Mineral Resource database in GEMS™ by checking for inconsistencies in analytical units, duplicate entries, interval, length or distance values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals or distances greater than the reported drill hole length, inappropriate collar locations, survey and missing interval and coordinate fields. A few errors were identified and corrected in the database. The QPs are of the opinion that the supplied database is suitable for Mineral Resource estimation.

14.5 Domain Interpretation

The mineralization vein wireframes were interpreted and constructed by SilverCrest using Seequent Limited Leapfrog® and the P&E QPs who reviewed the vein models. Some adjustments to the wireframes were made as a result of the reviews, and P&E considers the wireframes reasonable and suitable for Mineral Resource estimation.

Vein models were developed for each vein by manually tagging drilling intercepts using the drill core field logs and assays. The vein models represent the continuous zone of structurally hosted gold and silver mineralization and the structural extensions of the veins. All veins were constrained to a minimum thickness of 0.5 m true width, which were named as “unclip” solids. The “unclip” solids were manually clipped to include mineralization areas with ≥150 gpt AgEq (where AgEq = Ag gpt + Au gpt * 75), and renamed as “clip” solids. The clipping boundary was placed at the midpoint between drill hole intercepts and was extended a maximum distance of 80 m from drill intercepts into untested areas along strike and down dip. In some cases, samples <150 gpt AgEq were included to maintain the mineralization continuity and minimum width.

Zones of internal waste were delineated within the mineralization veins where a minimum true thickness of 1.5 m of <150 gpt AgEq across two or more adjacent drill holes, and where the remaining vein model maintained a 0.5 m minimum true thickness. These zones were triangulated, and volumes clipped from the vein wireframe model.

Areas with excavations from historical mining in the Las Chispas area were wireframed and block volumes were excluded from the Mineral Resource Estimate.

A topographical surface was provided by SilverCrest. All mineralization veins were clipped above that surface. The historically mined areas and internal waste zones created by SilverCrest were clipped from the related vein wireframes.

Both “clip” and “unclip” wireframes were utilized as constraining boundaries during Mineral Resource estimation, for rock coding, statistical analysis, and compositing limits. The 3-D domains are presented in Appendix B.

14.6 Model Rock Code Determination

A unique model rock code was assigned to blocks contained within each rock type unclipped vein in the Mineral Resource model as presented in (Table 14-4). Material outside the veins was treated as external waste and not accounted for in the Mineral Resource Estimate.

Table 14-4: Model Rock Codes Used for the Mineral Resource Estimate

Area	Vein	Model Rock Code	Average True Width (m)
Babicanora	Babicanora Main (BAM)	1000	1.96
	Babicanora FW(BAM_FW)	1005	0.73
	Babicanora HW (BAM_HW)	1010	0.80
	El Muerto (EM)	1050	0.83
	Babicanora Norte (BAN)	1101	0.91
	Babicanora Norte SE (BAN_SE)	1102	1.28
	Babicanora Norte NW (BAN_NW)	1103	0.71
	Babicanora Norte Central (BAN_CEN)	1104	0.61
	Babicanora Norte HW SE (BAN_HW_SE)	1105	0.94
	Babicanora Norte HW NW (BAN_HW_NW)	1106	0.72
	Babicanora Sur Main (BAS)	1200	1.04
	Babicanora Sur FW (BAS_FW)	1205	0.63
	Babicanora Sur HW (BAS_HW)	1210	0.65
	Granaditas1 (GRAN1)	1300	0.56
	Granaditas2 (GRAN2)	1305	0.72
	Babi Vista (BAV)	1400	0.74
	Babi Vista Vein Splay (BAV_Splay)	1410	1.27
	Babi Vista FW (BAV_FW)	1450	0.62
Las Chispas	Giovanni (GIO)	2001	0.75
	Giovanni Mini (GIO_MINI)	2002	0.71
	Las Chispas Main (LC)	2003	1.24
	William Tell (WT)	3001	1.60
	William Tell HW (WT_HW)	3002	1.00
	William Tell Mini (WT_MINI)	3003	0.78
	Luigi	4001	0.68
	Luigi_FW	4002	0.84

Note: True width was determined from drill hole intersections within the clipped wireframes.

14.7 Wireframe Constrained Assays

Wireframe constrained assays were back coded in the assay database with model rock codes that were derived from intersections of the clipped mineralization solids and drill holes. The basic statistics of clipped mineralization wireframe constrained assays are presented in Table 14-5.

Table 14-5: Basic Statistics of All Assays Constrained Within Vein Wireframes

Veins	Variable	Au	Ag	Length
BAM Drill Holes (BAM, BAM_FW, BAM_HW & EM)	Number of samples	1,172	1,172	1,172
	Minimum value *	0.01	0.30	0.40
	Maximum value *	479.00	21,858	3.70
	Mean *	7.35	681	0.82
	Median *	1.27	211	0.69
	Geometric mean *	0.91	192	0.75
	Variance	626.39	2,824,590	0.16
	Standard deviation	25.03	1,680.65	0.40
	Coefficient of variation	3.41	2.47	0.49
BAM UG Channel/Chip Samples	Number of samples	2,263	2,263	2,263
	Minimum value *	0.01	1.00	0.22
	Maximum value *	469.00	39,331	2.59
	Mean *	13.74	1,120	0.57
	Median *	1.87	343	0.54
	Geometric mean *	1.69	330	0.54
	Variance	1,171.06	5,746,544	0.03
	Standard deviation	34.22	2,397.20	0.17
	Coefficient of variation	2.49	2.14	0.30
BAN (BAN, BAN_SE, BAN_NW, BAN_CEN & BAN_HW,)	Number of samples	349	349	349
	Minimum value *	0.01	0.20	0.49
	Maximum value *	305.00	18,048	2.38
	Mean *	8.57	1,051	0.65
	Median *	1.28	195	0.55
	Geometric mean *	0.81	112	0.62
	Variance	654.66	5,263,567	0.06
	Standard deviation	25.59	2,294.25	0.24
	Coefficient of variation	2.98	2.18	0.36
BAS (BAS, BAS_FW & BAS_HW)	Number of samples	241	241	241
	Minimum value *	0.05	0.90	0.50
	Maximum value *	152.50	4,890	1.84
	Mean *	6.70	352	0.74
	Median *	1.75	136	0.60
	Geometric mean *	1.62	117	0.69
	Variance	291.66	450,398	0.08
	Standard deviation	17.08	671.12	0.29
	Coefficient of variation	2.55	1.91	0.39
	Number of samples	274	274	274

Veins	Variable	Au	Ag	Length
BAV (BAV, BAV_Splay & BAV_FW)	Minimum value *	0.00	0.20	0.48
	Maximum value *	3,366.00	114,814	2.44
	Mean *	30.20	1,689	0.64
	Median *	1.62	159	0.57
	Geometric mean *	Not Calculated	108	0.62
	Variance	51,287.26	72,653,662	0.04
	Standard deviation	226.47	8,523.71	0.21
	Coefficient of variation	7.50	5.05	0.33
GRAN1&2	Number of samples	28	28	28
	Minimum value *	0.05	0.50	0.49
	Maximum value *	47.50	5,620	1.20
	Mean *	3.71	403	0.63
	Median *	0.85	127	0.50
	Geometric mean *	0.73	51.7	0.60
	Variance	78.82	1,153,583	0.04
	Standard deviation	8.88	1,074.05	0.20
	Coefficient of variation	2.39	2.66	0.32
Giovanni (Main & MINI)	Number of samples	160	160	160
	Minimum value *	0.01	0.20	0.35
	Maximum value *	92.60	7,240	3.12
	Mean *	3.39	425	0.71
	Median *	0.64	115	0.57
	Geometric mean *	0.50	52.7	0.66
	Variance	81.51	697,291	0.11
	Standard deviation	9.03	835.04	0.34
	Coefficient of variation	2.67	1.96	0.48
Las Chispas Drill Holes	Number of samples	229.00	229.00	229.00
	Minimum value *	0.01	0.20	0.35
	Maximum value *	513.00	42,322	2.97
	Mean *	7.10	847	0.76
	Median *	0.72	138	0.60
	Geometric mean *	0.53	78.6	0.69
	Variance	1,553.77	14,091,266	0.16
	Standard deviation	39.42	3,753.83	0.40
	Coefficient of variation	5.55	4.43	0.52
	Number of samples	1,864	1,864	N/A

Veins	Variable	Au	Ag	Length
Las Chispas UG Channel/Chip Samples	Minimum value *	0.01	0.30	N/A
	Maximum value *	70.30	10,000	N/A
	Mean *	1.83	298	N/A
	Median *	0.43	82.0	N/A
	Geometric mean *	0.42	72.6	N/A
	Variance	19.25	413,653	N/A
	Standard deviation	4.39	643.16	N/A
	Coefficient of variation	2.39	2.16	N/A
WT (WT, WT_HW & MINI)	Number of samples	75	75	75
	Minimum value *	0.01	0.20	0.35
	Maximum value *	13.10	1,445.00	2.00
	Mean *	1.64	190	0.84
	Median *	0.43	63.6	0.87
	Geometric mean *	0.37	21.6	0.78
	Variance	6.52	85,365	0.11
	Standard deviation	2.55	292.17	0.33
	Coefficient of variation	1.56	1.54	0.39
WT UG Channel/Chip Samples	Number of samples	111	111	N/A
	Minimum value *	0.01	0.60	N/A
	Maximum value *	52.20	2,730	N/A
	Mean *	2.66	240	N/A
	Median *	0.35	75.6	N/A
	Geometric mean *	0.39	65.3	N/A
	Variance	48.07	195,098	N/A
	Standard deviation	6.93	441.70	N/A
	Coefficient of variation	2.61	1.84	N/A
Luigi (Main & FW)	Number of samples	121	121	121
	Minimum value *	0.01	0.30	0.40
	Maximum value *	26.30	3,720	2.35
	Mean *	2.27	291	0.70
	Median *	0.79	114	0.58
	Geometric mean	0.48	59.1	0.66
	Variance	21.97	346,039	0.10

Veins	Variable	Au	Ag	Length
	Standard deviation	4.69	588.25	0.31
	Coefficient of variation	2.06	2.02	0.44

Note: N/A = not applicable.
* Au & Ag units are gpt and length units are metres.

14.8 Compositing

Due to the nature of the narrow veins, in order to regularize the assay sampling intervals for grade interpolation, a 0.5 m compositing length was selected for the drill hole intervals that fell within the constraints of the above-mentioned Mineral Resource wireframe domains. The composites were calculated for gold and silver over 0.5 m lengths starting at the first point of intersection between assay data hole and hanging wall of the 3-D zonal constraint. The compositing process was halted upon exit from the footwall of the 3-D wireframe constraint. Un-assayed intervals and below detection limit assays were set to 0.001 gpt. If the last composite interval was less than 0.25 m, the composite length was adjusted to make all composite intervals of the vein intercept equal. This process would not introduce any short sample bias in the grade interpolation process. Two sets of composites were calculated for each unclipped and clipped vein wireframe. The constrained composite data were extracted to a point area file for grade capping analysis. The composite statistics of the clipped wireframes are summarized in Table 14-6.

Table 14-6: Basic Statistics of Composites Constrained Within Clipped Vein Wireframes

Veins	Variable	Au Composite	Au Cap	Ag Composite	Ag Cap	Length
BAM Drill Holes (BAM, BAM FW, BAM HW & EM)	Number of samples	2,156	2,156	2,156	2,156	2,156
	Minimum value *	0.00	0.00	0.00	0.00	0.33
	Maximum value *	476.80	195.00	21,297	14,750.00	0.75
	Mean *	6.62	6.39	618	609.47	0.50
	Median *	1.58	1.58	211	211.14	0.50
	Variance	389.87	284.12	2,187,210	1,926,973	0.00
	Standard deviation	19.75	16.86	1,478.92	1,388.15	0.05
	Coefficient of variation	2.98	2.64	2.39	2.28	0.10
BAM UG Channel/ Chip Samples	Number of samples	1,134	1,134	1,134	1,134	1,134
	Minimum value *	0.02	0.02	1.50	1.50	0.35
	Maximum value *	365.00	296.00	31,166	19,051.12	0.74
	Mean *	13.16	13.10	1,076	1,065.49	0.50
	Median *	2.84	2.84	396	395.73	0.49
	Geometric mean	2.55	2.55	426	425.57	0.50
	Variance	844.66	806.04	4,544,116	4,030,507	0.00

Veins	Variable	Au Composite	Au Cap	Ag Composite	Ag Cap	Length
	Standard deviation	29.06	28.39	2,131.69	2,007.61	0.04
	Coefficient of variation	2.21	2.17	1.98	1.88	0.08
BAN (BAN, BAN SE, BAN NW, BAN CEN & BAN HW)	Number of samples	433	433	433	433	433
	Minimum value*	0.00	0.00	0.00	0.00	0.38
	Maximum value *	303.27	115.00	18,009	12,850.00	0.75
	Mean *	9.39	8.49	1,166	1,120.55	0.50
	Median *	1.72	1.72	255	255.00	0.50
	Geometric mean	610.69	332.58	5,123,079	4,240,357	0.00
	Variance	24.71	18.24	2,263	2,059	0.06
	Standard deviation	2.63	2.15	1.94	1.84	0.12
	Coefficient of variation	433	433	433	433	433
BAS	Number of samples	284	284	284	284	284
	Minimum value *	0.05	0.05	2.40	2.40	0.27
	Maximum value *	152.50	119.00	4229	3280.00	0.50
	Mean *	8.60	8.25	394	389.07	0.48
	Median *	1.95	1.95	181	181.33	0.50
	Geometric mean	2.15	2.14	165	164.73	0.48
	Variance	449.55	359.43	442823	407115	0.00
	Standard deviation	21.20	18.96	665.45	638.06	0.05
	Coefficient of variation	2.46	2.30	1.69	1.64	0.10
BAV (BAV, BAV SPLAY & BAV FW)	Number of samples	444	444	444	444	444
	Minimum value *	0.00	0.00	0.00	0.00	0.38
	Maximum value *	2,775.91	380.00	94,867	34,355.00	0.74
	Mean *	21.55	11.84	1,280	1,027.81	0.50
	Median *	1.59	1.59	138	137.84	0.50
	Geometric mean	0.89	0.88	48.0	47.67	0.50
	Variance	24,378.78	1,331.17	36,919,610	10,676,156	0.00
	Standard deviation	156.14	36.49	6,076.15	3,267.44	0.06

Veins	Variable	Au Composite	Au Cap	Ag Composite	Ag Cap	Length
	Coefficient of variation	7.24	3.08	4.75	3.18	0.12
GRAN 1 & 2	Number of samples	38.00	38.00	38.00	38.00	38.00
	Minimum value *	0.05	0.05	0.40	0.40	0.25
	Maximum value *	47.34	20.00	5,601	2,000.00	0.50
	Mean *	3.29	2.57	328	232.30	0.48
	Median *	0.95	0.95	101.80	101.80	0.50
	Geometric mean	0.76	0.74	39.80	38.71	0.47
	Variance	59.60	15.36	872,063	201,207	0.00
	Standard deviation	7.72	3.92	933.84	448.56	0.06
	Coefficient of variation	2.35	1.53	2.85	1.93	0.12
Giovanni (Main & MINI)	Number of samples	222	222	222	222	222
	Minimum value *	0.01	0.01	0.20	0.20	0.38
	Maximum value *	92.60	73.00	7,240	2,890.00	0.73
	Mean *	3.47	3.25	419	398.35	0.50
	Median *	1.23	1.23	181	181.24	0.50
	Geometric mean	0.69	0.69	77.4	77.06	0.50
	Variance	82.33	60.33	564,175	375,794	0.00
	Standard deviation	9.07	7.77	751.12	613.02	0.06
	Coefficient of variation	2.62	2.39	1.79	1.54	0.12
Las Chispas Drill Holes	Number of samples	310	310	310	310	310
	Minimum value *	0.00	0.00	0.00	0.00	0.37
	Maximum value *	417.69	241.00	35,654	13,515.00	0.74
	Mean *	6.20	5.63	757	628.42	0.50
	Median *	0.85	0.85	154	153.51	0.50
	Variance	921.33	552.64	8,954,831	3,077,991	0.00
	Standard deviation	30.35	23.51	2,992.46	1,754.42	0.05
	Coefficient of variation	4.90	4.18	3.95	2.79	0.09
Las Chispas	Number of samples	1,864	1,864	1,864	1,864	N/A

Veins	Variable	Au Composite	Au Cap	Ag Composite	Ag Cap	Length
UG Channel/ Chip Samples	Minimum value *	0.01	0.01	0.30	0.30	N/A
	Maximum value *	70.30	65.00	10,000	8,340.00	N/A
	Mean *	1.83	1.83	298	296.89	N/A
	Median *	0.43	0.43	82.0	82.00	N/A
	Geometric mean	0.42	0.42	72.6	72.60	N/A
	Variance	19.25	18.87	413,653	397,850	N/A
	Standard deviation	4.39	4.34	643.16	630.75	N/A
	Coefficient of variation	2.39	2.37	2.16	2.12	N/A
WT (WT, WT HW & MINI)	Number of samples	151	151	151	151	151
	Minimum value *	0.00	0.00	0.00	0.00	0.27
	Maximum value *	13.08	13.08	1,436	1,370.00	0.75
	Mean *	1.21	1.21	150	149.41	0.50
	Median *	0.15	0.15	21.0	20.98	0.50
	Geometric mean	0.15	0.15	5.1	5.04	0.50
	Variance	4.44	4.44	66,906	65,805	0.01
	Standard deviation	2.11	2.11	258.66	256.53	0.07
	Coefficient of variation	1.75	1.75	1.73	1.72	0.14
WT UG Channel/ Chip Samples	Number of samples	111	111	111	111	N/A
	Minimum value *	0.01	0.01	0.60	0.60	N/A
	Maximum value *	52.20	33.00	2,730	1,910.00	N/A
	Mean *	2.66	2.49	240	232.78	N/A
	Median *	0.35	0.35	75.6	75.60	N/A
	Geometric mean	0.39	0.38	65.3	65.12	N/A
	Variance	48.07	34.22	195,098	164,315	N/A
	Standard deviation	6.93	5.85	441.70	405.36	N/A
	Coefficient of variation	2.61	2.35	1.84	1.74	N/A
Luigi (Main & FW)	Number of samples	178	178	178	178	178

Veins	Variable	Au Composite	Au Cap	Ag Composite	Ag Cap	Length
	Minimum value *	0.00	0.00	0.00	0.00	0.38
	Maximum value *	26.30	26.30	3,720	3,280.00	0.72
	Mean *	2.10	2.10	274	271.02	0.49
	Median *	0.81	0.81	122	121.71	0.50
	Geometric mean	0.48	0.48	56.7	56.65	0.49
	Variance	18.74	18.74	302,217	286,260	0.00
	Standard deviation	4.33	4.33	549.74	535.03	0.06
	Coefficient of variation	2.06	2.06	2.01	1.97	0.12

Note: N/A = not applicable.
* Au & Ag units are gpt and length units are metres

14.9 Grade Capping

Grade capping and high-grade transition analyses were undertaken on the 0.5 m composite values in the database within the constraining domains to control the possible bias resulting from erratic high-grade composite values in the database, as well as to maintain the high-grade local variation. The high-grade transition consists of a restrictive search ellipse as well as a maximum limiting composite value. Log-normal histograms and log-probability plots for gold and silver composites were generated for each mineralization vein (Appendix C). The grade capping and high-grade transition values for gold and silver are detailed in Table 14-7. The capped composites were utilized to develop variograms and for block model grade interpolation.

Table 14-7: Gold Grade Capping Values

Au Grade Capping									
Vein	Maximum Au Uncapped Value (gpt)	Au Cap (gpt)	Percentile (%)	Number Samples Capped	CoV of Composites	Au Metal Loss (%)	Au HG Transition Value (gpt)	Number Samples for HG Transition	Au HG Transition Percentile (%)
Babicanora Main (UG	365	296	99.9	1	2.21	0.5	202.3	6	99.5
Babicanora Main (drill	248	195	99.9	1	2.52	0.4	130.0	6	99.6
Babicanora Main FW	477	179	99.6	1	4.70	17.6	46.0	5	97.8
Babicanora Main HW	90.6	18	99.2	1	2.79	19.9	7.5	6	95.0
El Muerto	21.8	No	100.0	0	1.54	0.0	11.0	3	92.7
Babicanora Norte	91.2	No	100.0	0	1.72	0.0	NA	0	100.0
Babicanora NorteSE	181	115	98.9	2	2.04	5.5	95.0	3	98.3
Babicanora Norte NW	66.6	35	98.5	1	1.73	8.5	20.0	2	96.9

Au Grade Capping									
Babicanora Norte	303	75	97.4	1	3.95	47.6	25.0	2	94.9
Babicanora Norte HW	7.8	No	100.0	0	0.82	0.0	NA	0	100.0
Babicanora Norte HW	14.5	No	100.0	0	0.99	0.0	5.0	2	97.1
Babicanora Sur	152	119	98.6	4	2.46	4.1	48.9	8	97.2
Babicanora Sur HW	61.8	50	94.3	2	2.59	12.4	NA	0	100.0
Babicanora Sur FW	9.2	No	100.0	0	1.18	0.0	NA	0	100.0
Granaditas1	47.3	20	95.5	1	1.93	24.6	NA	0	100.0
Granaditas2	3.2	No	100.0	0	1.20	0.0	NA	0	100.0
Babi Vista	1,598	160	99.1	2	5.13	33.7	93.0	9	96.1
Babi Vista Splay	2,776	380	96.6	2	5.63	66.8	NA	0	100.0
Babi Vista FW	111	68	98.5	1	3.02	11.2	15.0	5	92.6
Giovanni	48.6	20	99.3	1	2.08	8.1	NA	0	100.0
Gio Mini	92.6	73	98.7	1	2.53	4.8	15.0	4	94.7
Las Chispas Main (Drill	418	241	99.7	1	4.90	9.2	65.0	5	98.4
Las Chispas Main (UG	70.3	65	99.9	1	2.39	0.2	NA	0	100.0
William Tell (drill holes)	7.9	No	100.0	0	1.37	0.0	NA	0	100.0
William Tell (UG	52.2	33	99.7	1	3.86	5.1	NA	0	100.0
William Tell HW	5.7	No	100.0	0	1.04	0.0	NA	0	100.0
William Tell Mini	13.1	No	100.0	0	2.72	0.0	NA	0	100.0
Luigi	26.3	No	100.0	0	2.27	0.0	NA	0	100.0
Luigi FW	15.3	No	100.0	0	1.35	0.0	NA	0	100.0

Note: CoV = coefficient of variation.

Table 14-8: Silver Grade Capping Values

Ag Grade Capping									
Vein	Maximum Ag Uncapped Value (gpt)	Ag Cap (gpt)	Percentile (%)	Number Samples Capped	CoV of Composites	Au Metal Loss (%)	Au HG Transition Value (gpt)	Number Samples for HG Transition	Au HG Transition Percentile (%)
Babicanora Main (UG samples)	31,166	19,051	99.9	1	1.98	1.0	11,141.0	10	99.1
Babicanora Main (drill holes)	16,315	14,750	99.9	2	2.15	0.3	11,380.0	7	99.5
Babicanora Main FW	21,297	14,426	99.6	1	2.93	4.7	2,365.0	9	96.0
Babicanora Main HW	1,040	686	98.3	2	0.97	3.1	425.8	17	85.7
El Muerto	2,178	2,071	97.6	1	2.12	0.7	920.0	3	92.7
Babicanora Norte	12,845	No Cap	100.0	0	1.66	-	7,550.0	1	98.6
Babicanora NorteSE	18,009	12,850	98.9	2	1.54	2.4	7,550.0	8	95.4
Babicanora Norte NW	6,916	2,800	98.5	1	1.55	9.4	2,000.0	4	93.8
Babicanora Norte Central	13,811	6,210	97.4	1	2.85	23.4	2,760.0	2	94.9
Babicanora Norte HW SE	589	530	93.8	1	0.76	1.7	210.0	4	75.0
Babicanora Norte HW NW	1,774	No Cap	100.0	0	0.82	0	590.0	5	92.6
Babicanora Sur	4,229	3,280	98.9	3	1.69	1.4	NA	0	100.0
Babicanora Sur HW	839	No Cap	100.0	0	3.05	0	NA	0	100.0
Babicanora Sur FW	1,107	800	95.5	1	1.62	9.6	NA	0	100.0
Granaditas1	5,601	2,000	90.9	2	2.54	35.0	NA	0	100.0

Ag Grade Capping									
Granaditas2	462	No Cap	100.0	0	1.14	0	NA	0	100.0
Babi Vista	65,154	34,355	99.6	1	3.61	9.5	6,665.0	8	96.6
Babi Vista Splay	94,867	25,270	96.6	2	4.33	43.4	NA	0	100.0
Babi Vista FW	8,933	6,631	98.5	1	2.92	6.6	950.0	7	89.7
Giovanni	7,240	2,800	99.3	1	2.06	8.3	815.0	12	91.8
Gio Mini	3,060	2,890	98.7	1	1.41	0.4	1,824.0	6	92.1
Las Chispas Main (Drill holes)	35,654	13,515	99.4	2	3.95	17.0	10,265.0	5	98.4
Las Chispas Main (UG samples)	10,000	8,340	100.0	1	2.16	0.3	NA	0	100.0
William Tell (drill holes)	1,436	1,370	98.6	1	1.34	0.4	610.0	7	90.3
William Tell (UG samples)	2,730	1,910	99.7	1	2.60	2.4	NA	0	100.0
William Tell HW	352	No Cap	100.0	0	1.00	0	NA	0	100.0
William Tell Mini	392	No Cap	100.0	0	2.12	0	NA	0	100.0
Luigi	3,720	3,280	99.3	1	2.20	1.2	1,253.0	6	95.6
Luigi FW	1,397	No Cap	100.0	0	1.02	0	NA	0	100.0

Note: CoV = coefficient of variation. HG = high grade.

14.10 Variography

A variography analysis was performed using the gold and silver composites within each individual vein wireframe as a guide to determining a grade interpolation search distance and ellipse orientation strategy (Appendix D).

Continuity ellipses based on the observed ranges were subsequently generated and utilized as the basis for estimation search ranges, distance weighting calculations and Mineral Resource classification criteria.

14.11 Bulk Density

A total of 641 bulk density measurements were collected on site by SilverCrest using the water immersion method. Core fragments greater than 5 cm in length were dried and weighed prior to being suspended and submerged from a scale in a bucket of water using a wire basket. The measurements tested various mineralization and non-mineralized material types at approximately 20 m downhole intervals. Where rock material was highly fragmented or strongly clay altered, samples were not collected. The bulk density ranged from 1.53 to 4.02 t/m^3 with a mean value of 2.52 t/m^3.

Seventy-two (72) samples were tested by ALS Chemex in Hermosillo, Mexico, for wax coated bulk density to validate the on-site measurements. The samples were collected from non-mineralization HW and FW materials, and mineralized material free of clay alteration. The overall average bulk density was 2.50 t/m^3, with 2.50 t/m^3 and 2.49 t/m^3 for Las Chispas and Babicanora areas respectively.

In November 2018, two samples were collected and sent by SilverCrest to Geotecnia del Noroeste S.A. de C.V. based in Hermosillo, Sonora, for wax coated dry bulk density testing. Each sample was split into two subsamples. The measured values ranged from 2.48 t/m^3 to 2.60 t/m^3, with an average dry bulk density of 2.56 t/m^3.

A uniform mean bulk density of 2.55 t/m^3 was applied to all rock types in the Mineral Resource Estimate based on the results of the bulk density test work completed above by SilverCrest and two laboratories.

14.12 Block Modelling

The block models for Babicanora Sur, Babicanora SurHW, Babicanora Sur FW and Las Chispas veins were SilverCrest reviewed and verified by P&E. All other block models were independently created and reviewed by P&E. All models have been discussed among SilverCrest and P&E during the course of this Mineral Resource Estimate.

The Las Chispas block model was constructed using GEOVIA GEMS™ V6.8.2 modelling software. The block model origin and block size are presented in Table 14-9. The block model consists of separate model attributes for estimated gold and silver grades, rock type (mineralization domains), volume percent, bulk density, AgEq value, and classification.

Table 14-9: Las Chispas Block Model Definition

Area	Direction	Origin	No. of Blocks	Block Size (m)
Babicanora	X	580,040.676	600	2.5
	Y	3,342,025.777	524	5
	Z	1,400	200	5
	Rotation	Counter clockwise 27°		
Las Chispas	X	580,398.801	240	2.5
	Y	3,343,924.556	388	5
	Z	1,240	94	5
	Rotation	Counter clockwise 25°		

*Origin for a block model in GEMS™ represents the coordinate of the outer edge of the block with min X and Y, and max Z, before rotation is applied.

All blocks in the rock type block model were initially assigned a waste rock code of 99, corresponding to the surrounding country rocks. The mineralization domain was used to code all blocks within the rock type block model that contain 0.01% or greater volume within the wireframe domain. These blocks were assigned individual model rock codes as presented in Table 14-4.

A volume percent block model was set up to accurately represent the volume and subsequent tonnage that was occupied by each block inside the constraining wireframe domain. As a result, the domain boundary was properly represented by the volume percent model ability to measure individual infinitely variable block inclusion percentages within that domain. The minimum percentage of the mineralization block was set to 0.01%.

The gold and silver grade values were interpolated into the blocks using inverse distance weighting to the third power (ID^3). Multiple passes were executed for the grade interpolation to progressively capture the sample points, to avoid over-smoothing and preserve local grade variability. Pass 0 was interpolated with underground samples, when available; Pass 1 and 2 were interpolated with capped composites derived from clipped wireframes for blocks coded with clipped solid; whereas Pass 3 was interpolated with composites derived from unclipped solid for blocks coded with unclipped solids. Grade blocks were interpolated using the parameters in Table 14-10.

Table 14-10: Las Chispas Block Model Grade Interpolation Parameters

Rock Code	Pass	No. of Composites			Search Range (m)			High-Grade Transition Range (m)		
		Min	Max	Max per Hole	Major	Semi-Major	Minor	Major	Semi-Major	Minor
BAM 1000	Pass 0	5	10	3	35	25	15	20	15	10
BAM 1000	Pass 1	7	12	3	45	30	15	45	30	15
BAM 1000	Pass 2	4	12	3	60	40	20	45	30	15
BAM 1000	Pass 3	2	12	3	80	60	30	25	15	10
BAMFW 1005	Pass 1	7	12	3	45	30	15	45	30	15
BAMFW 1005	Pass 2	4	12	3	60	40	20	45	30	15
BAMFW 1005	Pass 3	2	4	3	80	80	30	25	15	10
BAMHW 1010	Pass 1	7	12	3	45	30	15	45	30	15
BAMHW 1010	Pass 2	4	12	3	60	40	20	45	30	15
BAMHW 1010	Pass 3	2	12	3	80	80	30	25	15	10
EM1050	Pass 2	4	12	3	60	40	20	45	30	15
EM1050	Pass 3	2	12	3	180	120	60	25	15	10
BAS Main	Pass 1	7	12	3	45	30	15	NA	NA	NA
BAS Main	Pass 2	4	12	3	60	40	20	NA	NA	NA
BAS Main	Pass 3	2	12	3	80	80	30	NA	NA	NA
BAS Main	Pass 4	1	12	3	80	80	30	NA	NA	NA
BAS FW	Pass 1	6	12	3	45	30	15	NA	NA	NA

Rock Code	Pass	No. of Composites			Search Range (m)			High-Grade Transition Range (m)		
		Min	Max	Max per Hole	Major	Semi-Major	Minor	Major	Semi-Major	Minor
BAS FW	Pass 2	4	12	3	60	40	20	NA	NA	NA
BAS FW	Pass 3	2	12	3	80	80	30	NA	NA	NA
BAS HW	Pass 1	6	12	3	45	30	15	NA	NA	NA
BAS HW	Pass 2	4	12	3	60	40	20	NA	NA	NA
BAS HW	Pass 3	2	12	3	80	80	30	NA	NA	NA
BAN 1101	Pass 1	6	9	2	45	30	15	45	30	15
BAN 1101	Pass 2	4	10	2	60	40	20	45	30	15
BAN 1101	Pass 3	2	12	2	80	80	30	45	30	15
BAN 1102	Pass 1	6	9	2	45	30	15	45	30	15
BAN 1102	Pass 2	4	10	2	60	40	20	45	30	15
BAN 1102	Pass 3	2	12	2	80	80	30	45	30	15
BAN 1103	Pass 1	6	9	2	45	30	15	45	30	15
BAN 1103	Pass 2	4	10	2	60	40	20	45	30	15
BAN 1103	Pass 3	2	12	2	80	80	30	45	30	15
BAN 1104	Pass 1	6	9	2	45	30	15	45	30	15
BAN 1104	Pass 2	4	10	2	60	40	20	45	30	15
BAN 1104	Pass 3	2	12	2	80	80	30	45	30	15
BAN HW1105	Pass 1	6	9	2	45	30	15	NA	NA	NA

Rock Code	Pass	No. of Composites			Search Range (m)			High-Grade Transition Range (m)		
		Min	Max	Max per Hole	Major	Semi-Major	Minor	Major	Semi-Major	Minor
BAN HW1105	Pass 2	4	10	2	60	40	20	NA	NA	NA
BAN HW1105	Pass 3	2	12	2	80	80	30	NA	NA	NA
BAN HW1106	Pass 1	6	9	2	45	30	15	45	30	15
BAN HW1106	Pass 2	4	10	2	60	40	20	45	30	15
BAN HW1106	Pass 3	2	12	2	80	80	30	45	30	15
BAV 1400	Pass 1	7	12	3	45	30	15	45	30	15
BAV 1400	Pass 2	4	12	3	60	40	20	45	30	15
BAV 1400	Pass 3	2	12	3	80	60	30	25	15	10
BAV_SPLAY1410	Pass 1	1	3	3	5	5	5	5	5	5
BAV_SPLAY1410	Pass 2	4	12	3	60	40	20	NA	NA	NA
BAV_SPLAY1410	Pass 3	2	12	3	80	60	30	NA	NA	NA
BAVFW 1450	Pass 1	5	10	2	45	30	15	45	30	15
BAVFW 1450	Pass 2	3	10	2	60	40	20	45	30	15
BAVFW 1450	Pass 3	2	10	2	80	80	30	25	15	10
GRAN1 1300	Pass 1	6	12	3	45	35	15	NA	NA	NA
GRAN1 1300	Pass 2	4	12	3	60	50	20	45	35	15
GRAN1 1300	Pass 3	2	12	3	120	100	40	45	35	15
GRAN2 1305	Pass 1	6	12	3	45	35	15	NA	NA	NA

Rock Code	Pass	No. of Composites			Search Range (m)			High-Grade Transition Range (m)		
		Min	Max	Max per Hole	Major	Semi-Major	Minor	Major	Semi-Major	Minor
GRAN2 1305	Pass 2	4	12	3	60	50	20	NA	NA	NA
GRAN2 1305	Pass 3	2	12	3	80	80	40	NA	NA	NA
GIO 2001	Pass 1	5	10	2	45	30	15	45	30	15
GIO 2001	Pass 2	3	10	2	60	40	20	45	30	15
GIO 2001	Pass 3	2	10	2	80	80	30	25	15	10
GIO 2002	Pass 1	6	10	2	45	30	15	45	30	15
GIO 2002	Pass 2	4	10	2	60	40	20	45	30	15
GIO 2002	Pass 3	2	10	2	80	80	30	25	15	10
Luigi 4001	Pass 1	6	10	2	45	30	15	45	30	15
Luigi 4001	Pass 2	4	10	2	60	40	20	45	30	15
Luigi 4001	Pass 3	2	10	2	80	80	30	25	15	10
LuigiFW 4002	Pass 1	6	10	2	45	30	15	45	30	15
LuigiFW 4002	Pass 2	4	10	2	60	40	20	45	30	15
LuigiFW 4002	Pass 3	2	10	2	80	80	30	25	15	10
WT 3001	Pass 0	4	8	2	35	25	15	NA	NA	NA
WT 3001	Pass 1	6	10	2	45	30	15	45	30	15
WT 3001	Pass 2	4	10	2	60	40	20	45	30	15
WT 3001	Pass 3	2	10	2	80	80	30	25	15	10

Rock Code	Pass	No. of Composites			Search Range (m)			High-Grade Transition Range (m)		
		Min	Max	Max per Hole	Major	Semi-Major	Minor	Major	Semi-Major	Minor
WT HW 3002	Pass 1	6	10	2	45	30	15	NA	NA	NA
WT HW 3002	Pass 2	4	10	2	60	40	20	NA	NA	NA
WT HW 3002	Pass 3	2	10	2	80	80	30	NA	NA	NA
WT Mini 3003	Pass 1	6	8	2	45	30	15	NA	NA	NA
WT Mini 3003	Pass 2	4	8	2	60	40	20	NA	NA	NA
WT Mini 3003	Pass 3	2	8	2	80	60	30	NA	NA	NA
LC 2003	Pass 0	4	12	3	10	10	5	NA	NA	NA
LC 2003	Pass 1	7	12	3	45	30	15	45	30	15
LC 2003	Pass 2	5	12	3	60	40	20	45	30	15
LC 2003	Pass 3	3	12	3	80	60	30	25	15	10

Note: NA = not applicable.

Selected longitudinal projections of gold, silver, and AgEq blocks are presented in Appendix E to Appendix G.

14.13 Mineral Resource Classification

In the QP's opinion, all the drilling, assaying and exploration work on the Project supports this Mineral Resource Estimate and based on spatial continuity of the mineralization within a potentially mineable shape are sufficient to indicate a reasonable potential for economic extraction, thus qualifying it as a Mineral Resource under the 2014 CIM Definition Standards. The Mineral Resource was classified as Measured, Indicated, and Inferred based on the geological interpretation, variogram performance and drill hole spacing.

A Measured Mineral Resource was classified for the Babicanora Main Vein underground sampled area only with a 10 m range extended up and down from the underground samples and interpolated with both underground channel and chip samples and drill holes within this area.

Indicated Mineral Resources were classified for the blocks interpolated with the Pass 1 and 2 in Table 14.9, which used at least two holes within a 50 m of mean distance.

Inferred Mineral Resources were classified for all remaining grade blocks within the mineralization veins.

The classifications were manually adjusted on a longitudinal projection to reasonably reflect the distribution of each classification.

Selected classification block longitudinal projections are attached in Appendix H.

The Las Chispas Mineral Resource Estimate was derived from applying AgEq cut-off values to the block models and reporting the resulting tonnes and grades for potentially mineable areas.

14.14 AgEq Cut-off Value Calculation

The following parameters were used to calculate the AgEq cut-off values that determine the underground mining potentially economic portions of the constrained mineralization:

- Ag price: $18.50/oz (approximate two-year trailing average as of October 31, 2020);
- Ag process recovery: 95%;
- Marginal Mining cost: $40/t;
- Processing cost: $30/t; and
- G&A: $15/t.

The AgEq cut-off value of the underground Mineral Resource is calculated as follows:

- ($40+ $30+$15) / ($18.50/31.1035 x 95%) = 150 gpt AgEq

14.15 Stockpiles Mineral Resource Estimate

A total of 38 stockpiles (historical dumps and tailing deposits) were sampled with trench and RC drill hole cuttings. The perimeters and surfaces of the stockpiles were surveyed between December 14, 2017 and January 26, 2018 using a Trimble Spectra Total Station Model TS-415. The surveyed outlines and sample points were imported into GOVIA GEMS^{TM} 6.8.2. Generally, the sample points matched the perimeters well with a few sample points relabelled to the appropriate stockpiles.

The area of each stockpile was calculated using its perimeter in GEMS. The average thickness of the stockpiles was estimated based on the trench profiles and RC drill holes. The bulk density of the stockpiles was estimated at 1.7 t/m^3. Thus, the estimated tonnage of each stockpile was calculated using the average thickness, estimated bulk density, and its measured surface area.

Average grades were estimated for each stockpile area based on the samples collected. Grade capping was applied as presented in Table 14-11. The tonnage and average grades for stockpiles at a cut-off of ≥110 gpt AqEq were tabulated in Table 14-12. The cut-off was derived from the Mineral Resource cut-off with a deduction for no mining costs applied. A gold and silver ratio of 86.9:1 was used for the AgEq calculation. The gold and silver grades from the assay results were averaged for each stockpile, which can have a significant standard deviation and difference between the minimum and maximum values as well as spatial distribution bias.

14.16 Mineral Resource Estimate

The Mineral Resource Estimate is reported with an effective date of October 16, 2020 and is tabulated in Table 14-13. The QP considers the vein mineralization of the Project to be potentially amenable to underground mining methods.

Table 14-11: Las Chispas Stockpile Grade Capping

Stockpiles	Au Capping				Ag Capping			
	Au Uncapped Max (gpt)	Au Cap Value (gpt)	Percentile (%)	No of Samples Capped	Ag Uncapped Max (gpt)	Ag Cap Value (gpt)	Percentile (%)	No. of Samples Capped
La Capilla	138.0	20	97.5	4	1,675	425	97.5	4
San Gotardo	74.1	35	99.9	2	10,000	2,535	99.7	4
Lupena	12.2	12	100	0	651	530	99.5	1
Las Chispas 1	17.9	7	99.7	1	1,240	550	99.2	3

Table 14-12: Las Chispas Stockpile Indicated Mineral Resource Estimate at 110 gpt AgEq Cut-off

Stockpiles	Area (m^2)	Ave. Height (m)	Density (t/m^3)	Tonnes	Au (gpt)	Ag (gpt)	AgEq (gpt)	Contained Au (oz)	Contained Ag (oz)	Contained AgEq (oz)
North Chispas 1	344	2.00	1.7	1,170	0.54	70.5	117	20	2,651	4,416
La Capilla	3,686	2.00	1.7	12,532	3.66	123	440	1,475	49,459	177,611
San Gotardo	23,349	2.00	1.7	79,387	0.71	105	166	1,812	268,327	425,804
Lupena	3,435	3.00	1.7	17,519	1.62	80.6	221	912	45,408	124,699
Las Chispas 1 (LCH)	5,932	2.40	1.7	24,203	0.75	120	185	584	93,461	144,176
Las Chispas 2	274	2.30	1.7	1,071	1.23	236	343	43	8,129	11,822
Las Chispas 3 (San Judas)	233	2.50	1.7	990	2.05	703	882	65	22,391	28,074
La Central	750	3.00	1.7	3,825	0.75	116	181	92	14,298	22,260

Stockpiles	Area (m^2)	Ave. Height (m)	Density (t/m^3)	Tonnes	Au (gpt)	Ag (gpt)	AgEq (gpt)	Contained Au (oz)	Contained Ag (oz)	Contained AgEq (oz)
Chiltepines 1	88	1.00	1.7	150	0.87	1745	250	4	841	1,203
Espiritu Santo	486	2.00	1.7	1,652	0.52	93.9	139	28	4,986	7,387
La Blanquita 2	1,083	2.50	1.7	4,603	0.53	118	164	78	17,477	24,292
El Muerto	1,370	2.50	1.7	5,823	2.52	79.4	298	472	14,860	55,854
Sementales	236	2.00	1.7	802	4.38	47.0	428	113	1,213	11,032
Buena Vista	153	1.50	1.7	390	4.62	56.9	458	58	713	5,749
Babicanora	1,509	4.00	1.7	10,261	1.91	58.4	224	630	19,263	74,020
Babicanora 2	269	2.20	1.7	1,006	2.63	275	504	85	8,911	16,304
Total Stockpile >110 gpt AgEq	**43,197**	**2.31**	**1.7**	**164,214**	**1.23**	**108**	**215**	**6,471**	**572,389**	**1,134,702**

Table 14-13: Las Chispas Mineral Resource Estimate (1-7) at 150 gpt AgEq Cut-off

Vein	Classification	Tonnes (k)	Au (gpt)	Ag (gpt)	AgEq (gpt)	Contained Au (koz)	Contained Ag (koz)	Contained AgEq (koz)
Babicanora Main	Measured	143.3	13.52	1,192	2,366	62.3	5,490	10,901
Babicanora Main	Indicated	919.0	5.29	532	992	156.3	15,720	29,302
Babicanora Main	**M+I**	**1,062.3**	**6.40**	**621**	**1,177**	**218.6**	**21,210**	**40,204**
Babicanora Main FW	Indicated	162.7	6.60	610	1,184	34.5	3,190	6,191
Babicanora Main HW	Indicated	119.3	2.48	151	366	9.5	579	1,406
Babicanora Norte	Indicated	351.5	9.03	1,067	1,851	102.0	12,051	20,919
Babicanora Norte HW	Indicated	66.9	2.87	236	486	6.2	507	1,045
Babicanora Sur	Indicated	233.4	7.09	372	988	53.2	2,791	7,412
Babicanora Sur HW	Indicated	18.4	2.62	97.5	325	1.5	57	191
Babi Vista	Indicated	179.9	15.81	1,293	2,668	91.5	7,480	15,482
Babicanora Sur FW	Indicated	20.2	9.53	928	1,756	6.2	603	1,141
Babicanora Area Total	**M+I**	**2,214.5**	**7.35**	**681**	**1,319**	**523.2**	**48,471**	**93,939**
Las Chispas	Indicated	208.2	5.74	748	1,246	38.4	5,007	8,344
Giovanni	Indicated	70.8	2.76	394	634	6.3	896	1,443
Gio Mini	Indicated	54.9	3.70	466	787	6.5	821	1,388
William Tell Main	Indicated	17.3	1.99	283	456	1.1	157	253

Vein	Classification	Tonnes (k)	Au (gpt)	Ag (gpt)	AgEq (gpt)	Contained Au (koz)	Contained Ag (koz)	Contained AgEq (koz)
Luigi	Indicated	61.9	2.48	338	553	4.9	672	1,101
Luigi FW	Indicated	31.9	2.74	281	520	2.8	288	533
Las Chispas Area Total	**Indicated**	**445.1**	**4.20**	**548**	**913**	**60.1**	**7,844**	**13,064**
Total Undiluted Veins	**M+I**	**2,659.6**	**6.82**	**659**	**1,251**	**583.3**	**56,315**	**107,004**
Stockpiles	Indicated	164.2	1.23	108	215	6.5	572	1,134
Total (Veins+stockpiles)	**M+I**	**2,823.8**	**6.50**	**627**	**1,191**	**589.8**	**56,888**	**108,139**
Babicanora Main (Inc. El Muerto)	Inferred	342.0	3.02	256	519	33.2	2,819	5,706
Babicanora Main FW	Inferred	5.4	1.39	154	275	0.2	27	48
Babicanora Main HW	Inferred	6.0	1.97	79	250	0.4	15	48
Babicanora Norte	Inferred	53.1	2.09	317	499	3.6	541	851
Babicanora Norte HW	Inferred	27.2	1.77	172	326	1.6	151	286
Babicanora Sur	Inferred	79.4	4.94	251	681	12.6	641	1,737
Babicanora Sur HW	Inferred	2.8	2.53	6	226	0.2	1	21
Babicanora Sur FW	Inferred	42.0	1.77	162	316	2.4	219	426
Babi Vista	Inferred	14.1	3.05	222	488	1.4	101	221
Babi Vista Splay	Inferred	211.4	13.00	909	2,039	88.3	6,180	13,857

Vein	Classification	Tonnes (k)	Au (gpt)	Ag (gpt)	AgEq (gpt)	Contained Au (koz)	Contained Ag (koz)	Contained AgEq (koz)
Babi Vista FW	Inferred	15.1	2.36	214	419	1.1	104	204
Granaditas 1	Inferred	43.5	4.11	295	653	5.8	413	913
Granaditas 2	Inferred	19.7	1.19	182	285	0.8	115	180
Babicanora Area Total	**Inferred**	**861.6**	**5.47**	**409**	**884**	**151.6**	**11,325**	**24,496**
Las Chispas	Inferred	71.7	3.27	469	753	7.5	1,082	1,736
Gio Mini	Inferred	6.8	2.20	535	726	0.5	118	160
William Tell Main	Inferred	155.5	1.49	233	363	7.4	1,166	1,813
William Tell HW	Inferred	55.9	2.00	237	412	3.6	427	740
William Tell Mini	Inferred	33.5	1.60	172	311	1.7	185	334
Luigi	Inferred	19.7	1.14	161	260	0.7	102	165
Luigi FW	Inferred	35.2	0.33	202	230	0.4	229	261
Las Chispas Area Total	**Inferred**	**378.4**	**1.80**	272	**428**	**21.9**	**3,308**	**5,209**
Total Undiluted Veins	**Inferred**	**1,240.0**	**4.35**	367	**745**	**173.4**	**14,633**	**29,705**

1. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
3. The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It can be reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.
4. The Mineral Resources in the Report were estimated using the 2014 CIM Definition Standards on Mineral Resources and Mineral Reserves.
5. Historical mined areas were removed from the wireframes and block model.
6. AgEq is based on Au: Ag ratio of 86.9:1 calculated using $1,410/oz Au and $16.60/oz Ag, with average metallurgical recoveries of 96% Au and 94% Ag.

7. Mineral Resources are inclusive of the Mineral Reserves stated in Section 15.
8. All numbers are rounded.

14.17 Model Validation

The block model was validated using a number of industry standard methods including visual and statistical methods.

Visual examination of composites and block grades on successive plans and sections were performed on-screen to confirm that the block models correctly reflect the distribution of composite grades, see Appendix E to G. The review of estimation parameters included:

- Number of composites used for estimation;
- Number of drill holes used for estimation;
- Mean distance to sample used;
- Number of passes used to estimate grade;
- Actual distance to closest point;
- Grade of true closest point; and,
- Mean value of the composites used.

The drill hole composites were clipped with solids, constrained and the blocks were interpolated with Pass 1 and 2 using the capped composites. The ID3 estimate was compared to a nearest-neighbour (NN) estimate. A comparison of mean composite grades of four largest veins with the block model at zero grade are presented in Table 14-14. For the Babicanora Main Vein, estimates were generated and compared between ID^3 and ordinary kriging (OK) grade interpolation methods. The results from the comparison showed OK interpolation with more tonnes, less grade, and approximately 4% more AgEq ounces.

Table 14-14: Average Grade Comparison of Composites with Block Model

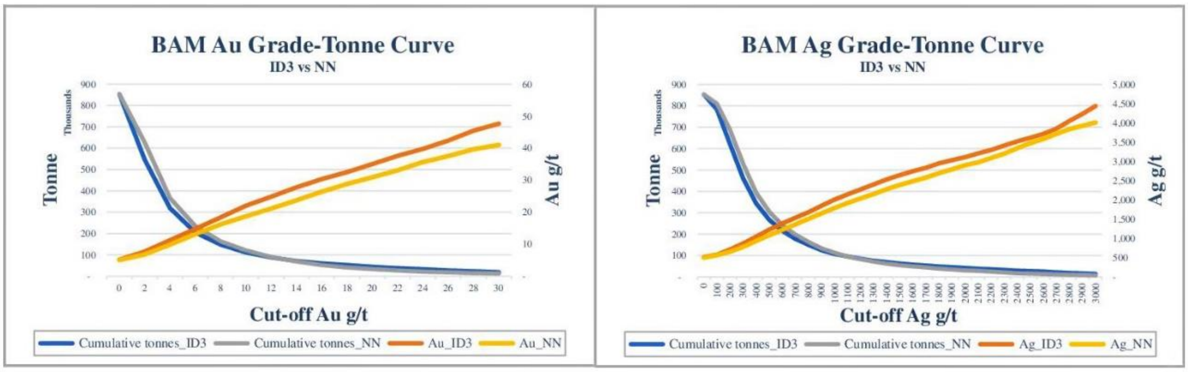
Vein	Data Type	Au (gpt)	Ag (gpt)
Babicanora Main	Composites	7.32	697
	Capped composites	7.29	695
	Block model ID^3	5.22	519
	Block model NN	5.08	507
Babicanora Notre	Composites	11.32	1,402
	Capped composites	10.21	1,346
	Block model ID^3	8.10	923
	Block model NN	9.10	1,063
Babi Vista	Composites	16.96	1,128
	Capped composites	11.52	1,023
	Block model ID^3	13.99	1107
	Block model NN	14.33	1135
Las Chispas	Composites	6.20	757
	Capped composites	5.63	628

Vein	Data Type	Au (gpt)	Ag (gpt)
	Block model ID3	4.23	552
	Block model NN	4.27	549

Notes: ID3 = Au and Ag interpolated with Inverse Distance Cubed.
NN = Au and Ag interpolated using Nearest Neighbour.

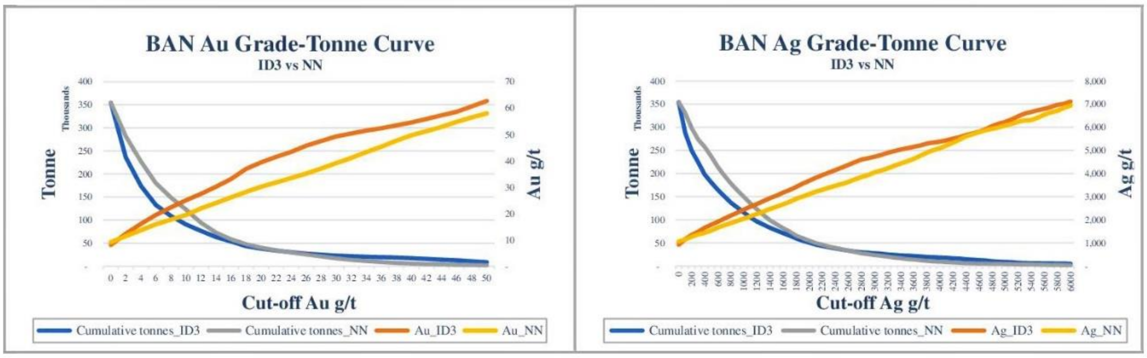
The comparison shows the average grades of block models were somewhat different to that of the composites used for the grade estimations. These were most likely due to grade declustering and interpolation process. The block model values will be more representative than the composites due to 3-D spatial distribution characteristics of the block models.

A comparison of the grade-tonnage curves of the gold and silver grade model interpolated with ID3 and NN on a global mineralization basis which interpolated by Pass 1 and Pass 2 using the drill hole composites. The selected grade-tonnage curves for the four largest veins (Babicanora Main, Babicanora Norte, Babi Vista and Las Chispas) are presented in Figure 14-1 through Figure 14-4.



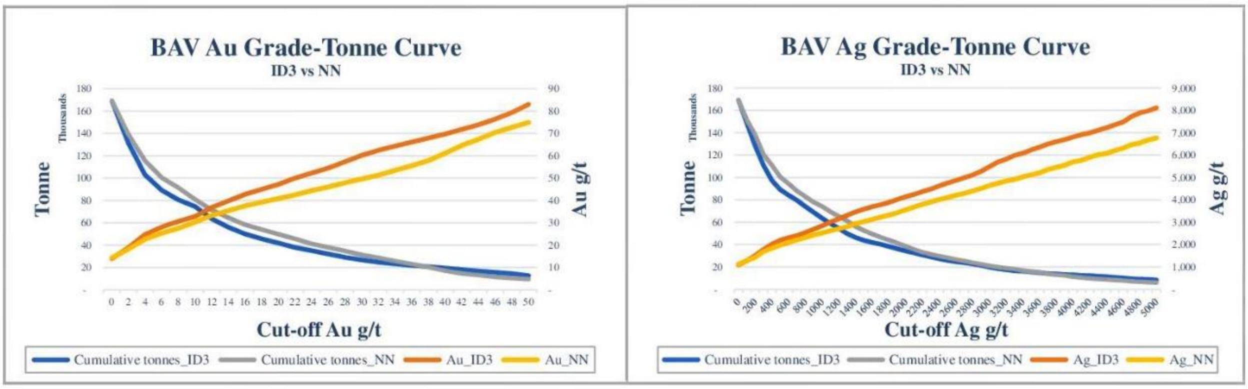
Note: Figure prepared by P&E, 2020.

Figure 14-1: Grade–Tonnage Curve of Babicanora Main



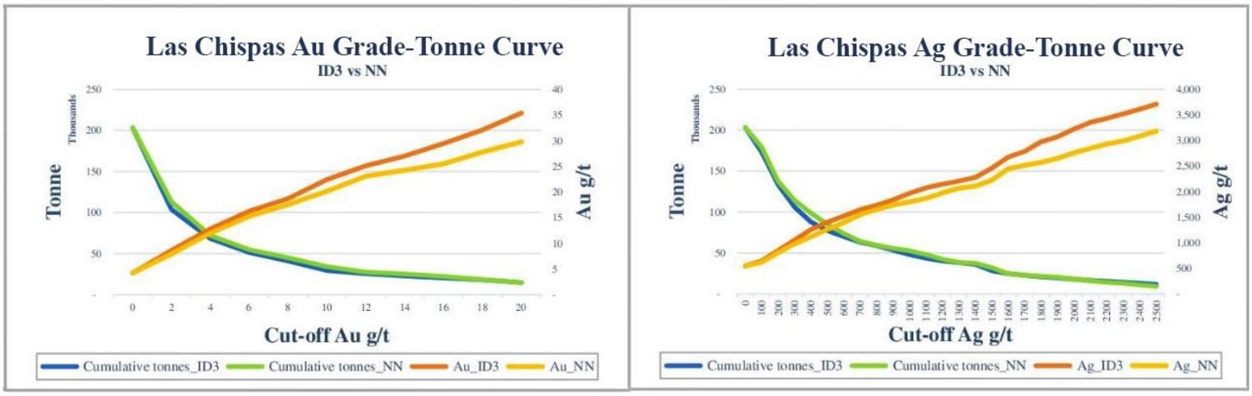
Note: Figure prepared by P&E, 2020.

Figure 14-2: Grade–Tonnage Curve of Babicanora Norte Vein



Note: Figure prepared by P&E, 2020.

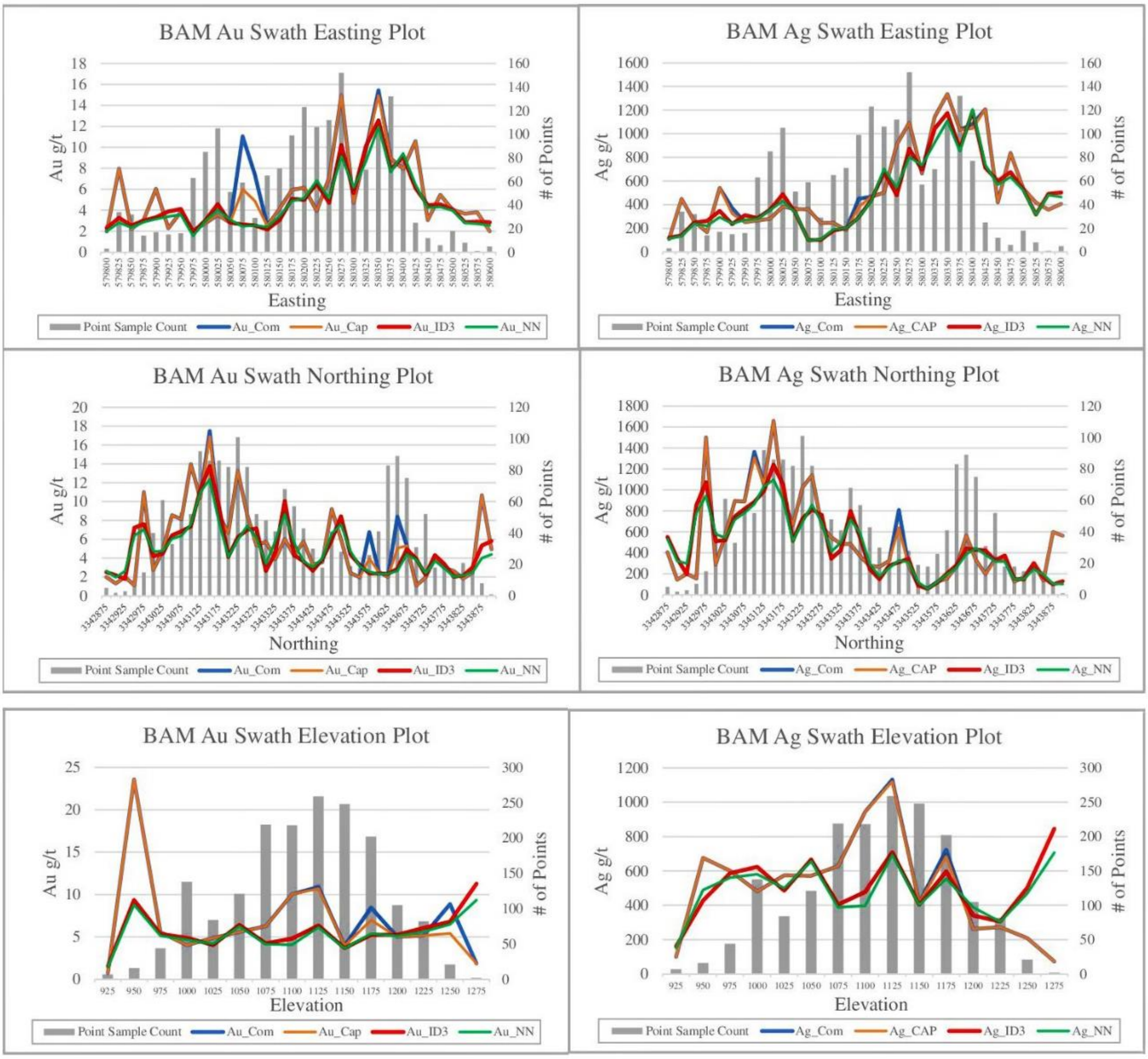
Figure 14-3: Grade–Tonnage Curve of Babi Vista Vein



Note: Figure by P&E, 2020.

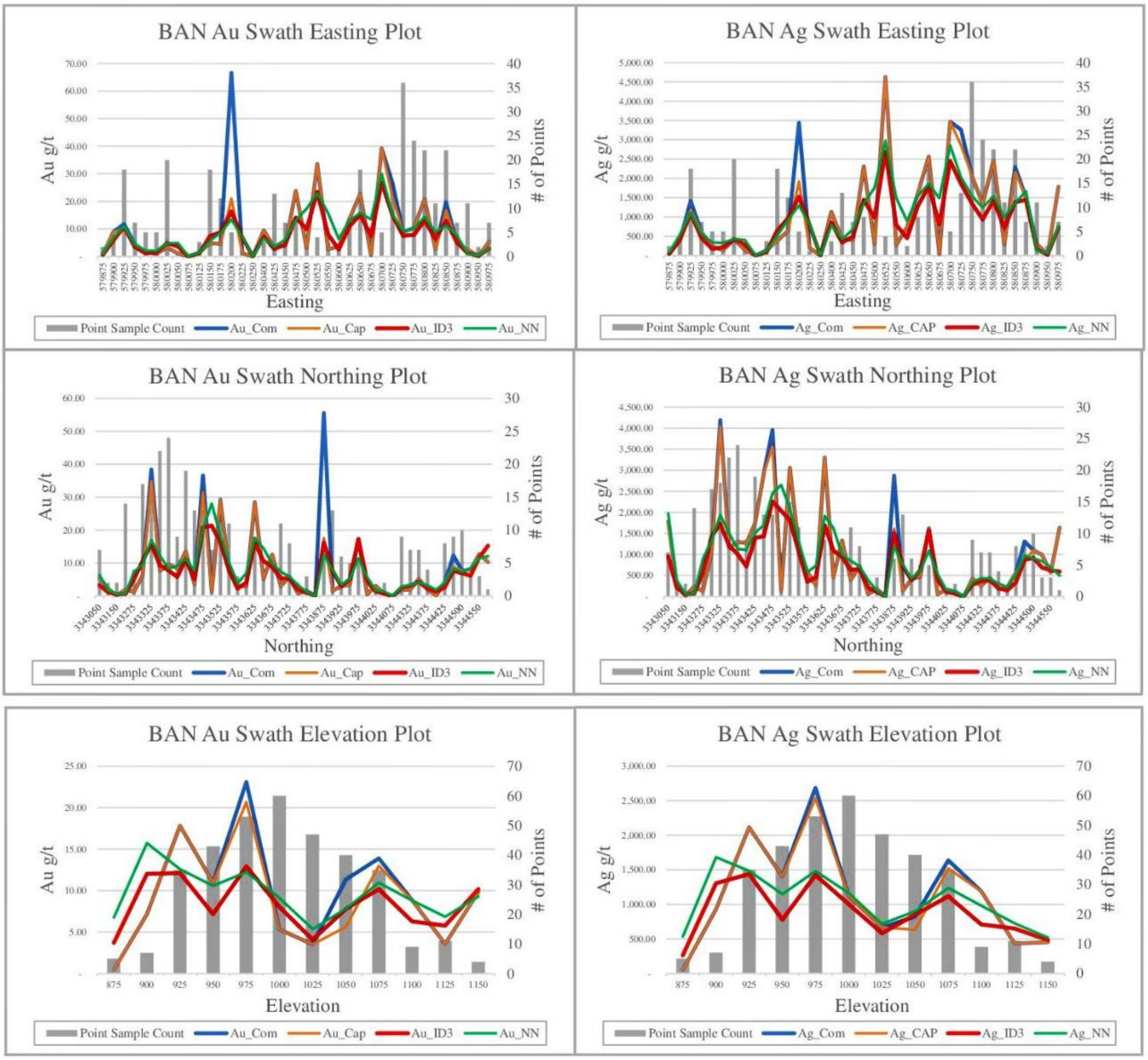
Figure 14-4: Grade–Tonnage Curve of Las Chispas Vein

Local trends for gold and silver were evaluated by comparing the ID^3 and NN estimate against the composites. The selected special swath plots of the four largest veins are shown in Figure 14-5 through Figure 14-8. The plots were generated with drill hole composites constrained in the clipped wireframes and the first two passes of the grade interpolations.



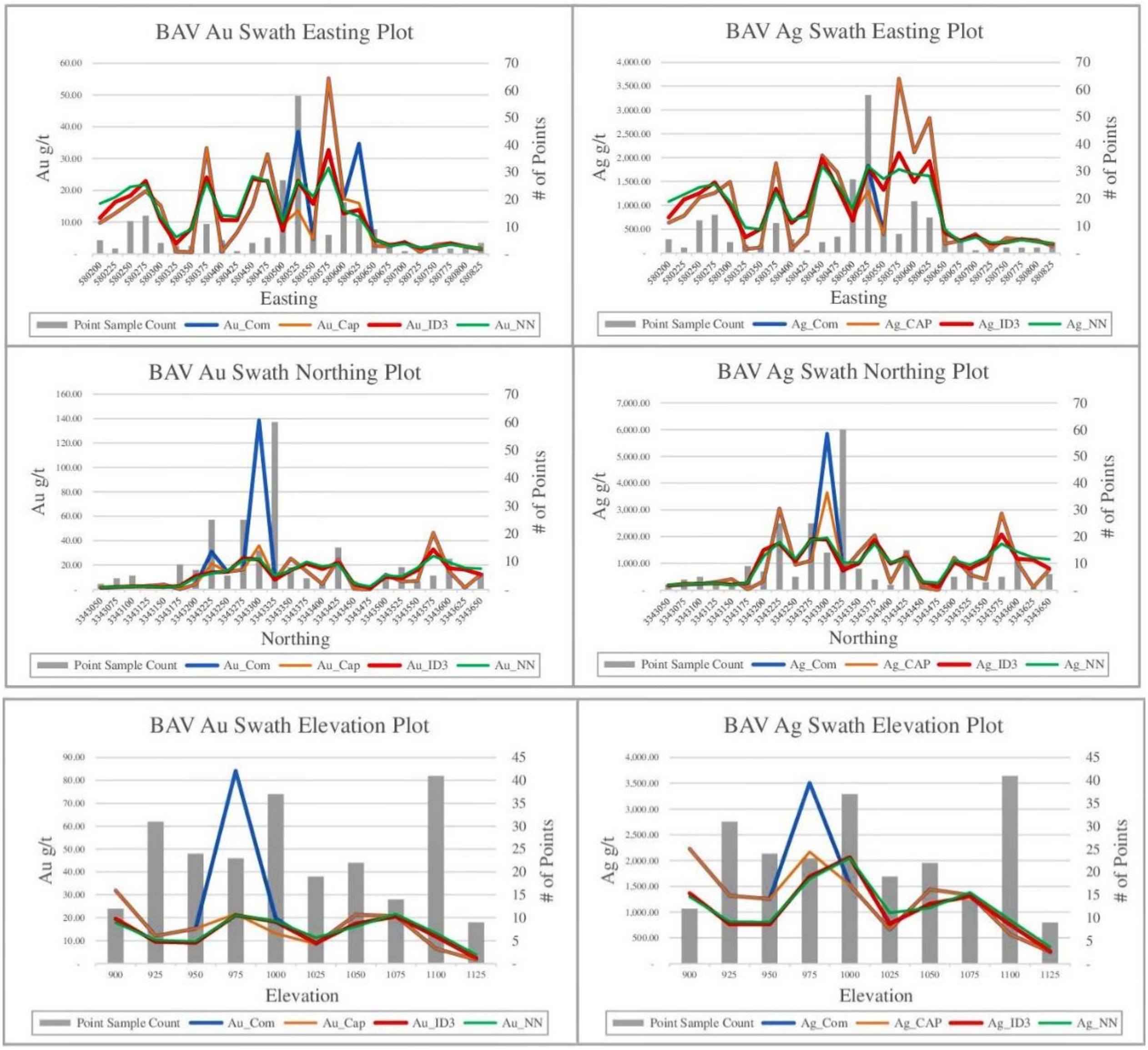
Note: Figure prepared by P&E, 2020.

Figure 14-5: Babicanora Main Vein Ag and Au Grade Swath Plot



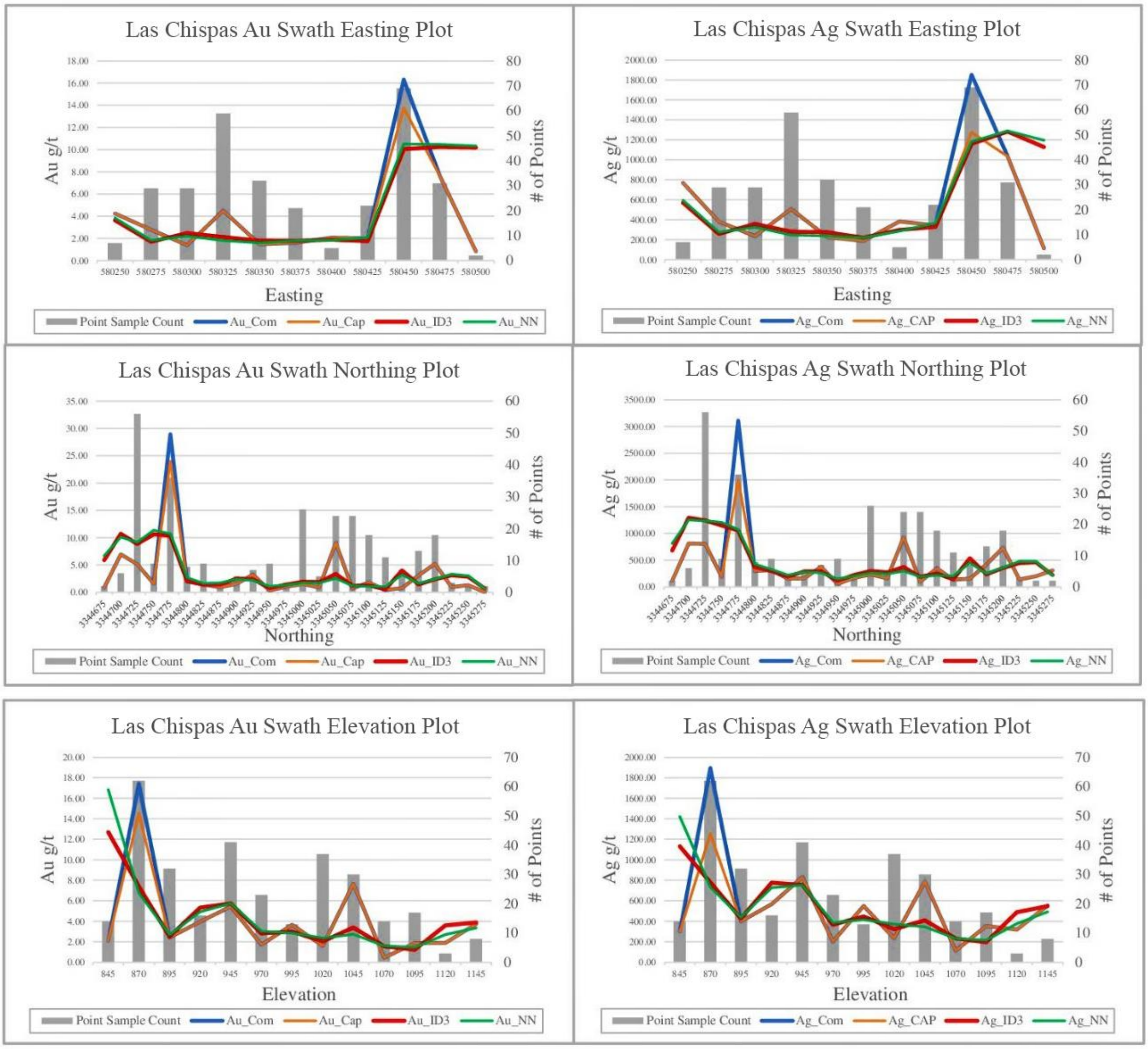
Note: Figure prepared by P&E, 2020.

Figure 14-6: Babicanora Norte Vein Ag and Au Grade Swath Plot



Note: Figure prepared by P&E, 2020.

Figure 14-7: Babi Vista Vein Ag and Au Grade Swath Plot



Note: Figure by P&E, 2021.

Figure 14-8: Las Chispas Vein Ag and Au Grade Swath Plot

15 Mineral Reserve Estimates

15.1 Mineral Reserve Estimate

The Proven and Probable Mineral Reserve for the Project is estimated at 3.35 Mt, at an average grade of 4.8 gpt Au and 461 gpt Ag or 879 gpt AgEq, as summarized in Table 15-1 and Table 15-2.

Table 15-1: Mineral Reserve Estimate

Classification	Tonnes (k)	Au (gpt)	Ag (gpt)	AgEq (gpt)	Contained Au (koz)	Contained Ag (koz)	Contained AgEq (koz)
Proven	336.5	6.21	552	1,091	67.1	5,971	11, 806
Probable	3,014.7	4.65	451	855	451.0	43,707	82,898
Proven + Probable	**3,351.2**	**4.81**	**461**	**879**	**518.1**	**49,679**	**94,740**

Note:

1. The Mineral Reserve is estimated using the 2019 CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines and 2014 CIM Definition Standards for Mineral Resources & Mineral Reserves.
2. The Mineral Reserve is estimated with a variable COG which was calculated by vein width and economic and operating parameters. Refer to Subsection 15.2 for COG estimation details.
3. The Mineral Reserve is estimated using long-term prices of $1,410/oz for gold and $16.60/oz for silver.
4. A government gold royalty of 0.5% is included in the Mineral Reserve estimates.
5. The Mineral Reserve is estimated with a mining recovery of 95%.
6. The Mineral Reserve presented includes both internal and external dilution. The external dilution included a mining dilution of 0.5 m width on the hanging wall and footwall for the long hole mining method and a 0.2 m width on the hanging wall and footwall for the cut-and-fill and resue mining methods. Backfill dilution is also included and represents 7% for the long hole mining method and 10% for cut-and-fill and resue mining methods.
7. A minimum mining width of 1.5 m was used for the long hole and cut-and-fill mining methods. A minimum mining width of 0.5 m was used for the resue mining method.
8. The economic viability of the Mineral Reserve has been demonstrated.
9. AgEq is based on gold to silver ratio of 86.9:1 calculated using US$1,410/oz Au and US$16.60/oz Ag, with average metallurgical recoveries of 96% Au and 94% Ag.
10. The Qualified Person for the estimate is Mr. Carl Michaud, P.Eng., Underground Engineering Manager for GMS. The estimate has an effective date of January 4, 2021.
11. Totals may not add due to rounding

Table 15-2: Mineral Reserve Estimate by Vein (effective date: January 4, 2021)

Vein	Classification	Tonnes (k)	Au (gpt)	Ag (gpt)	AgEq (gpt)	Contained Au (koz)	Contained Ag (koz)	Contained AgEq (koz)
Babicanora Main	Proven	119.4	13.11	1,168	2,307	50.3	4,486	8,860
Babicanora Main	Probable	1,474.8	3.47	337	638	164.6	15,965	30,273
Babicanora Norte	Probable	514.8	5.87	682	1,192	97.1	11,289	19,732
Babi Vista	Probable	220.7	11.71	955	1,972	83.1	6,774	13,994
Babi Vista FW	Probable	18.8	8.64	867	1,618	5.2	524	978
Babicanora Sur	Probable	305.3	4.98	262	695	48.9	2,569	6,818

Vein	Classification	Tonnes (k)	Au (gpt)	Ag (gpt)	AgEq (gpt)	Contained Au (koz)	Contained Ag (koz)	Contained AgEq (koz)
Babicanora Sur HW	Probable	6.1	3.12	102	373	0.6	20	73
Total Babicanora Area	**Proven**	**119.4**	**13.11**	**1,168**	**2,307**	**50.3**	**4,486**	**8,860**
Total Babicanora Area	**Probable**	**2,540.6**	**4.89**	**455**	**880**	**399.6**	**37,142**	**71,867**
Las Chispas	Probable	180.6	5.21	661	1,114	30.2	3,841	6,470
Giovanni	Probable	65.8	2.02	301	476	4.3	637	1,007
Gio Mini	Probable	64.6	2.51	318	536	5.2	660	1,112
Luigi	Probable	37.7	2.27	308.5	506	2.8	374	613
Luigi FW	Probable	38.7	1.67	178.1	323	2.1	222	402
William Tell	Probable	8.9	1.94	273	441	0.6	78	126
Total Las Chispas Area	**Probable**	**396.3**	**3.54**	**456**	**764**	**45.1**	**5,812**	**9,731**
Historical Stockpiles	Proven	162.6	1.23	108	215	6.4	565	1,123
Babicanora Stockpile + Open stope	Proven	54.5	5.93	525	1040	10.4	920	1,823
Babicanora Stockpile + Open stope	Probable	77.8	2.51	301	519	6.3	754	1,300
Total Mineral Reserve Estimate	**Proven**	**336.5**	**6.21**	**552**	**1,091**	**67.1**	**5,971**	**11,806**
	Probable	**3,014.7**	**4.65**	**451**	**855**	**451.0**	**43,707**	**82,898**
	Proven+ Probable	**3,351.2**	**4.81**	**461**	**879**	**518.1**	**49,679**	**94,704**

Note:

1. Footnotes to Table 15-1 also apply to this table.
2. Babicanora Main Vein includes Babicanora Central Zone, Babicanora FW Vein and Babicanora HW Vein.
3. Babicanora Norte Vein includes Babicanora Norte HW Vein.

The Mineral Reserve Estimate was prepared by GMS. Modifying factors were applied to the 12 Mineral Resource block models, prepared by P&E and summarized in Table 15-3 to determine the mineable portion of the Las Chispas deposit.

Table 15-3: Resource Block Models from P&E Mining Consultants Inc.

Vein	Block Model Name
Babicanora Main Babicanora Main FW and Babicanora Main HW	BAM – Interpolated YZ Very Fine Combined – July 21, 2020.dm
Babi Vista FW	BAV FW – Interpolated YZ Very Fine Combined – July 11, 2020.dm
Babi Vista	BAV – Interpolated repetitive from YZ Very Fine Combined – Sep 5, 2020.dm
Babicanora Sur	BAS – YZ Very Fine Interpolated Combined – July 11, 2020.dm
Babicanora Sur HW	BAS HW – YZ Very Fine Interpolated Combined – July 12, 2020.dm
Babicanora Norte and Babicanora Notre HW	BAN – Interpolated YZ Very Fine Combined – June 25, 2020.dm
Las Chispas	Las Chispas – Interpolated YZ Very Fine Combined – July 26, 2020.dm
Luigi	Luigi – Interpolated YZ Very Fine Combined – July 18, 2020.dm
Luigi FW	Luigi FW – Interpolated YZ Very Fine Combined – July 22, 2020.dm
Gio Mini	Gio Mini – Interpolated YZ Very Fine Combined – July 18, 2020.dm
Giovanni	Giovanni (Trimmed) – Interpolated YZ Very Fine Combined – July 15, 2020.dm
William Tell	William Tell – Interpolated YZ Very Fine Combined – July 25, 2020.dm

The mine design and Mineral Reserve Estimate were completed to a level appropriate to support a Feasibility Study. The Mineral Reserve Estimate stated herein is consistent with the 2014 CIM Definition Standards and is suitable for public reporting. As such, the Mineral Reserve is based on Measured and Indicated Mineral Resources and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the Mineral Resource block models are treated as waste at zero grade.

The factors that may affect the Mineral Reserve include the following:

- Geological complexity, geological interpretation, and Mineral Resource block modelling.
- COG estimations;
- Commodity prices, market conditions and foreign exchange rate assumptions;
- Operating cost assumptions;
- Sustaining capital costs to develop;
- Rock quality and geotechnical constraints, dilution and mining recovery factors;
- Hydrogeological assumptions; and,

- Metallurgical process recoveries.

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

15.2 Cut-off Grade

Gold and silver are the only payable commodities recovered and the COG is defined as AgEq content in gpt. COG are provided in Table 15-5, Table 15-6, Table 15-7 and Table 15-8, and were calculated using input parameters, such as commodity recovery, processing costs, general and administrative (G&A) costs, commodity prices, exchange rate, and marketing costs as shown in Table 15-4. Operating costs were based on an extraction and processing rate of 1,250 t/d. Marginal COG were also calculated without the costs associated with the development.

Table 15-4: COG Input Parameters

Input Parameters		
Processing Costs	$/t processed	35.00
General & Administrative costs	$/t processed	15.00
Silver Price	$/oz	16.60
Gold Price	$/oz	1,410
Conversion	g/oz	31.10
Government Gold Royalty	%	0.5
Gold Recovery	%	96.0
Silver Recovery	%	94.0
Recovered Gold/Silver Ratio	Au/Ag	86.9
Gold Payable	%	99.85
Silver Payable	%	99.85
Transport	$/oz	0.01
Treatment and Refining	$/oz	0.22
Net Value (Ag)	$/oz	15.25
Net Value (Ag)	$/g	0.49

Table 15-5: COG Long Hole and Avoca Mining Methods

Stope Dimensions		Long Hole / Avoca
Width	m	1.5–10.0
Diluted width	m	2.5-11
Total mining unit costs	$/t	35.58
Processing costs	$/t milled	35.00
General & administrative costs	$/t milled	15.00
Total costs	$/t	85.58
COG with dilution	gpt AgEq	190
COG marginal with dilution	gpt AgEq	170

Table 15-6: COG, Cut and Fill Mining Methods

Stope Dimensions		Cut-and-Fill Mining Methods		
Width	m	1.5	2.5	4.5
Diluted width	m	1.9	2.9	4.9
Total mining unit costs	$/t	62.82	49.86	39.92
Processing costs	$/t milled	35.00	35.00	35.00
General & administration	$/t milled	15.00	15.00	15.00
Total costs	$/t	112.82	99.86	89.92
COG with dilution	gpt AgEq	250	220	200
Marginal COG with dilution	gpt AgEq	210	200	190

Table 15-7: COG Resue Mining Method

Stope Dimensions		Resue		
Width	m	0.5	1.0	1.5
Diluted width	m	0.9	1.4	1.9
Total mining unit costs	$/t	141.53	96.55	78.73
Processing costs	$/t milled	35.00	35.00	35.00
General & administration costs	$/t milled	15.00	15.00	15.00
Total costs	$/t	191.53	146.55	128.73
COG with dilution	gpt AgEq	430	330	290
Marginal COG with dilution	gpt AgEq	300	240	220

Table 15-8: COG, Surface Stockpile

		Stockpile
Total Mining Unit Costs	$/t	3.00
Total Costs	$/t	53.00
COG with dilution	gpt AgEq	110

15.3 Mine Dilution and Recovery

15.3.1 Mining Dilution

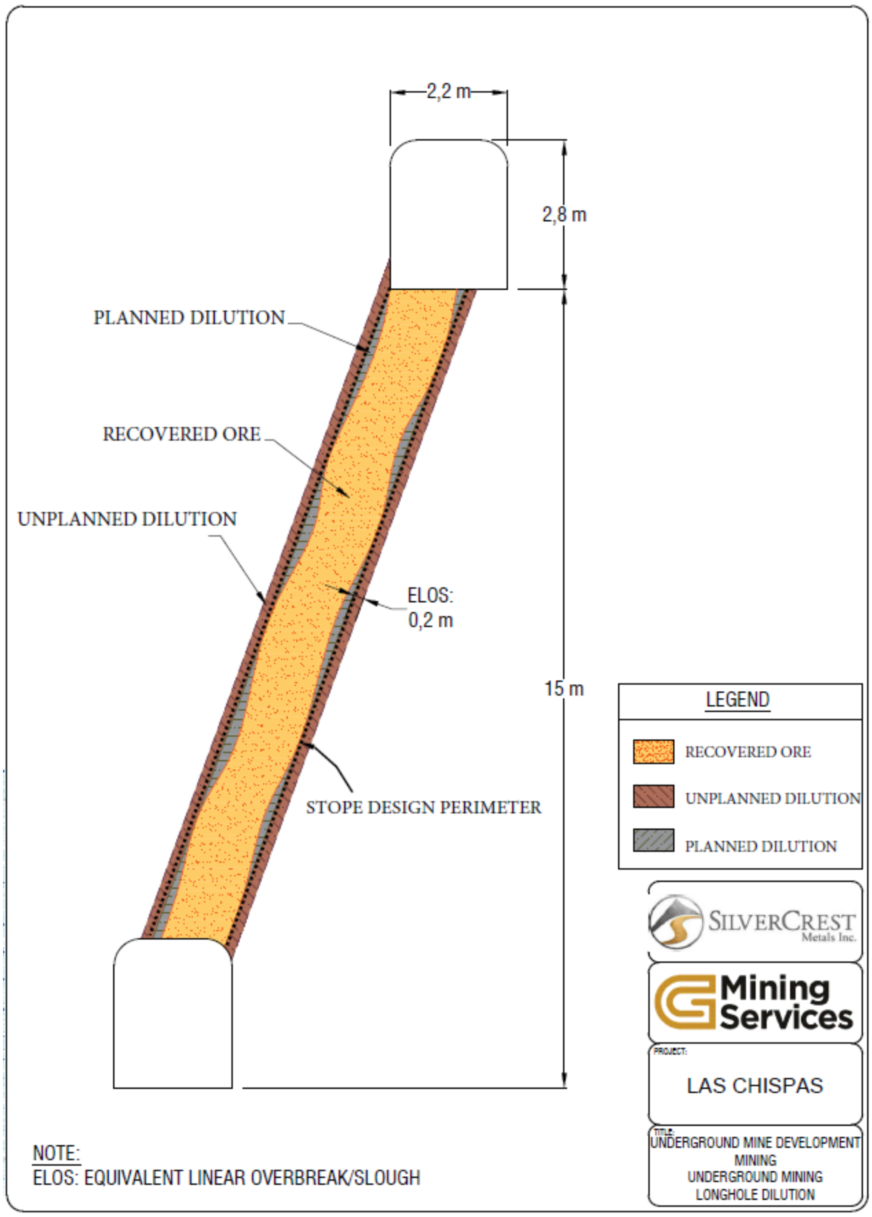
Mining dilution comes from three principal sources: planned (or design, or internal) dilution, unplanned (or external) dilution, and backfill dilution. Dilution grades for dilution were estimated from the Mineral Resource block model for the planned and unplanned dilution. Table 15-9 presents the average expected unplanned dilution by area.

Table 15-9: Average Dilution by Mining Area

Areas	Average Dilution
Babicanora Main	48%
Babi Vista	90%
Babicanora Norte	56%
Babicanora Central	24%
Babicanora Sur	60%
Las Chispas	65%
Average All Areas	52%

15.3.1.1 Planned Dilution

The planned dilution is material outside the mineralization vein which is included in the stope shape. Planned dilution includes the waste present in the shape of the stope that cannot be separated, and therefore is extracted with the mining block. The factors that influence this dilution are minimum mining width and geology of mineralization (continuity and shape). The minimum mining widths are primarily a function of the geometry of the deposit and the selection of the mining equipment. Regarding the long hole mining method, the minimum mining width was estimated at 1.5 m. For the cut and fill methods, small equipment will be used which also allows a minimum width of 1.5 m. The resue mining method will permit for significantly reduced minimum mining width of 0.5 m. The mining methods are detailed and explained in Section 16. Figure 15-1 shows the origin of the planned and unplanned dilution. Table 15-10 shows the minimum width depending on the mining method.



Note: Figure prepared by GMS, 2020.

Figure 15-1: Diagram for Planned Dilution and Unplanned Dilution

Table 15-10: Minimum Mining Width

Vein	Minimum Mining Width (m)
Long Hole	1.5
Cut and Fill	1.5
Resue	0.5

15.3.1.2 Unplanned Dilution

Unplanned dilution comes from the breakage outside the profile of the stope in the hanging wall and footwall. Unplanned mining dilution was applied through the process of Deswik Stope Optimizer (DSO) by allowing for dilution on the hanging wall and footwall. Different thicknesses of dilution were selected depending on the mining method, this "thickness" is defined as equivalent linear overbreak / slough (ELOS) and was calculated from Rockland (2020a). Table 15-11 shows the ELOS used according to the mining method.

Table 15-11: ELOS by Mining Method

Mining Method	Hanging Wall ELOS (m)	Footwall ELOS (m)
Long Hole	0.5	0.5
Cut and fill	0.2	0.2
Resue	0.2	0.2

A 0.2 m shape surrounding the mineralization vein was created in the Mineral Resource block model using drill holes assays to evaluate the dilution grade outside the mineralization zone. The diluted shapes and grade estimates surrounding veins was prepared by P&E. Apart from these shapes, the grade of the diluting material has been assumed at zero.

15.3.1.3 Backfill Dilution

In terms of backfill dilution, some of the cemented rock fill (CRF) is expected to fall into the stope and be removed from an adjacent stope and/or be inadvertently scraped off the stope floors during the mineralized material loading. To ensure optimal mining recovery, it is planned to withdraw about 0.3 m of backfill from the floor when mining the stope. In the event that a backfill wall is exposed during mining, the backfill dilution of this wall is expected to be 0.6 m thick. All backfill dilution grade was assumed to be zero. Table 15-12 shows the backfill dilution used according to the mining method.

Table 15-12: Backfill Dilution vs Mining Method

Mining Method	Floor Dilution (m)	Wall Dilution (m)	CRF Dilution
Long Hole	0.3	0.6	7%
Cut and fill	0.3	N/A	10%
Resue	0.3	N/A	10%

15.3.2 Mining Recovery

Mining recovery represents material loss in the mining process and is defined as the percentage of actual mineable material extracted from the planned mining shape. It is important to have a good balance between dilution and mining recovery in order to optimize the profitability of the deposit. Since the deposit contains very high-grade material in narrow veins, mining recovery is a priority. A 95% mining recovery was used for the various mining methods to account for the under-break and for the mineralized material left in place. As the mining methods used are largely very selective, this percentage also included some of the rock left in place in the sill pillars.

15.4 Stope Optimization

The DSO software was used to determine the mineable portion of the Mineral Resource. The DSO is a strategic mine planning tool that automates and optimizes the design of stope shapes for a range of stoping methods for underground mines. The goal is to maximize the value of a deposit within existing constraints using a specified mining method and design parameters. Multiple DSO scenarios were run to obtain the best results in terms of tonnage and grade. The DSO software was first programmed to estimate the economic stopes with the applicable COG. A second set of economic calculations were then completed with the use of the applicable marginal COG. The stopes calculated from the marginal COG that were continuous to the first stopes, were considered for reserve inclusion.

The optimized stopes were built from DSO CAD 2020.2 with the Stope Optimized V4 plugin. The outputs (stope wireframes, section strings, and reports) are suitable for use in strategic and tactical planning. To generate stope shapes and optimize the value of the deposit, GMS used the 12 Mineral Resource block models prepared by P&E in Datamine format as listed in Table 15-3. Table 15-13 presents the final parameters used for the stope optimization scenario in the Babicanora Area and Table 15-14 presents the final parameters used for the stope optimization scenario in the Las Chispas Area.

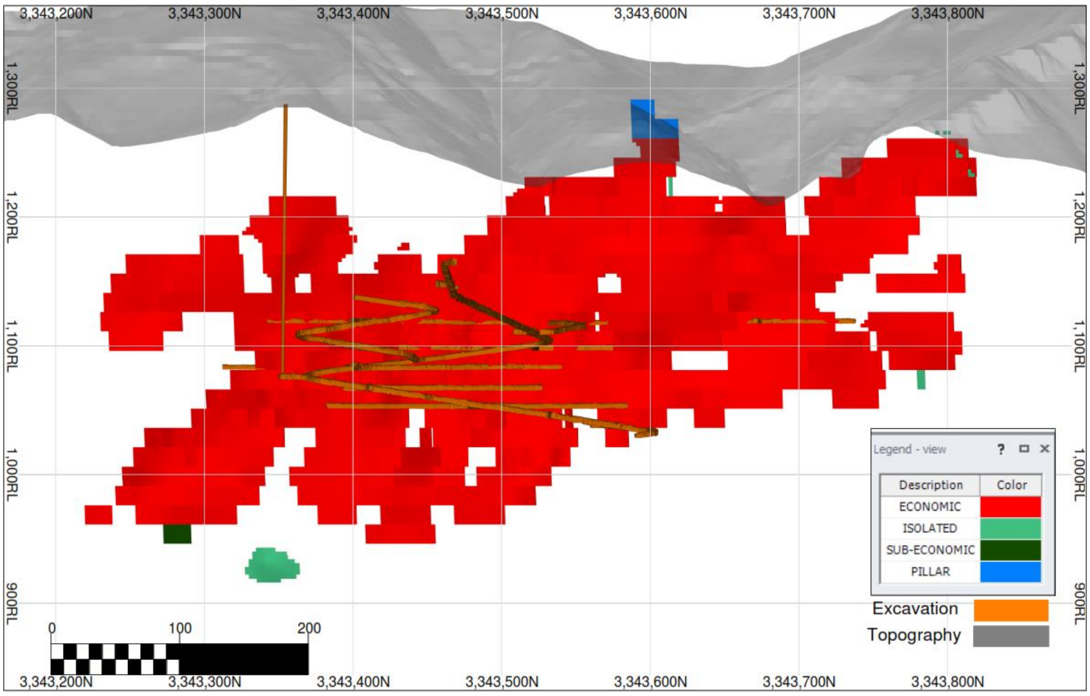
Table 15-13: Babicanora Area DSO Parameters

		Babicanora						
		Main	Central	Sur	Sur HW	Norte	Vista	Vista FW
Level interval	m	15	15	18	18	18	18	18
Minimum stope length	m	12	12	12	12	12	12	12
Mining method		Long-hole	C&F	C&F	Resue	C&F or Resue	Resue	Resue
Minimum mining Width	m	1.5	1.5	1.5- C&F	0.5 – Resue	1.5 – C&F 0.5 – Resue	0.5 – Resue	0.5 – Resue
ELOS hanging wall	M	0.5	0.2	0.2	0.2	0.2	0.2	0.2
ELOS footwall	m	0.5	0.2	0.2	0.2	0.2	0.2	0.2
Back fill dilution	%	7	10	10	10	10	10	10
Mining recovery	%	95	95	95	95	95	95	95

Table 15-14: Las Chispas Area DSO Parameters

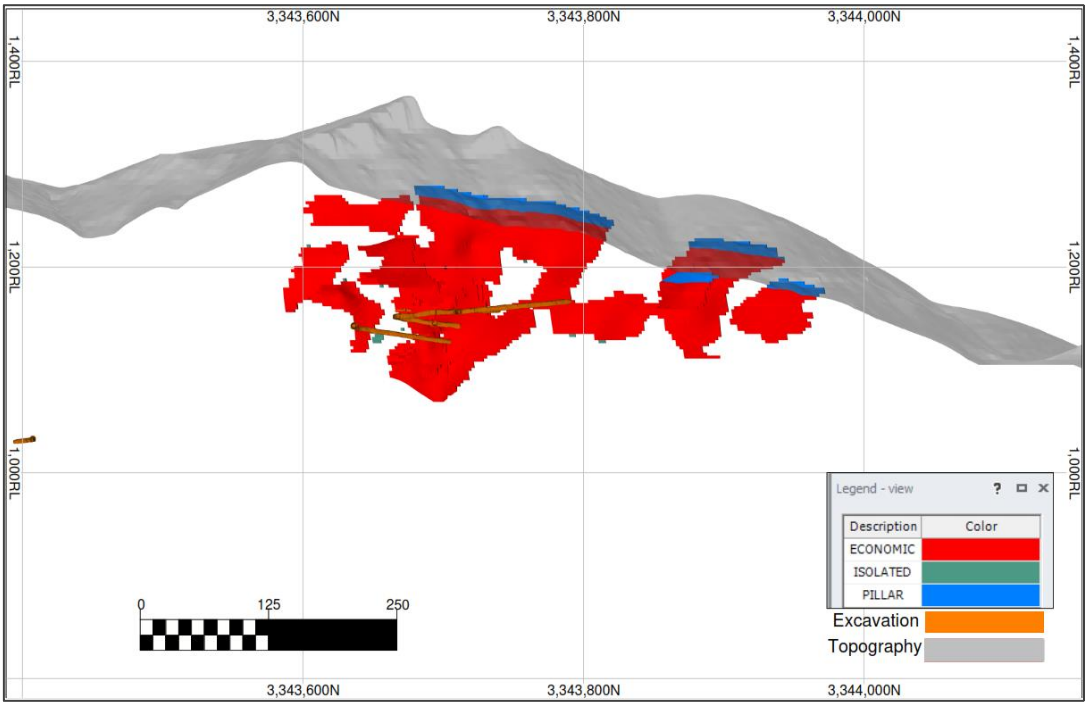
		Las Chispas					
		Las Chispas	Luigi FW	Luigi	Giovanni	Giovanni Mini	William Tell
Level Interval	m	18	18	18	18	18	18
Minimum Stope Length	m	12	12	12	12	12	12
Mining Method		C&F	C&F	C&F	Resue	C&F or Resue	Resue
Minimum mining width	m	1.5	1.5	1.5 – C&F	0.5 – Resue	1.5 – C&F 0.5 – Resue	0.5 – Resue
ELOS hanging wall	m	0.2	0.2	0.2	0.2	0.2	0.2
ELOS footwall	m	0.2	0.2	0.2	0.2	0.2	0.2
Back fill dilution	%	10	10	10	10	10	10
Mining Recovery	%	95	95	95	95	95	95

Figure 15-2 to Figure 15-12 present the DSO stope design for the different veins. The red colour represents the economic stopes. The blue colour represents the areas where pillars were needed. These stopes (blue colour) were removed from the Mineral Reserve estimate regardless of their grade due to the absence of appropriate geotechnical information. The dark green colour represents the stopes that contain grade above the applicable COG but cannot support the costs of development. As such these stopes were also removed from the Mineral Reserve estimate. Finally, the light green colour is used for stopes that contain grade above the applicable COG but cannot support the additional development costs.



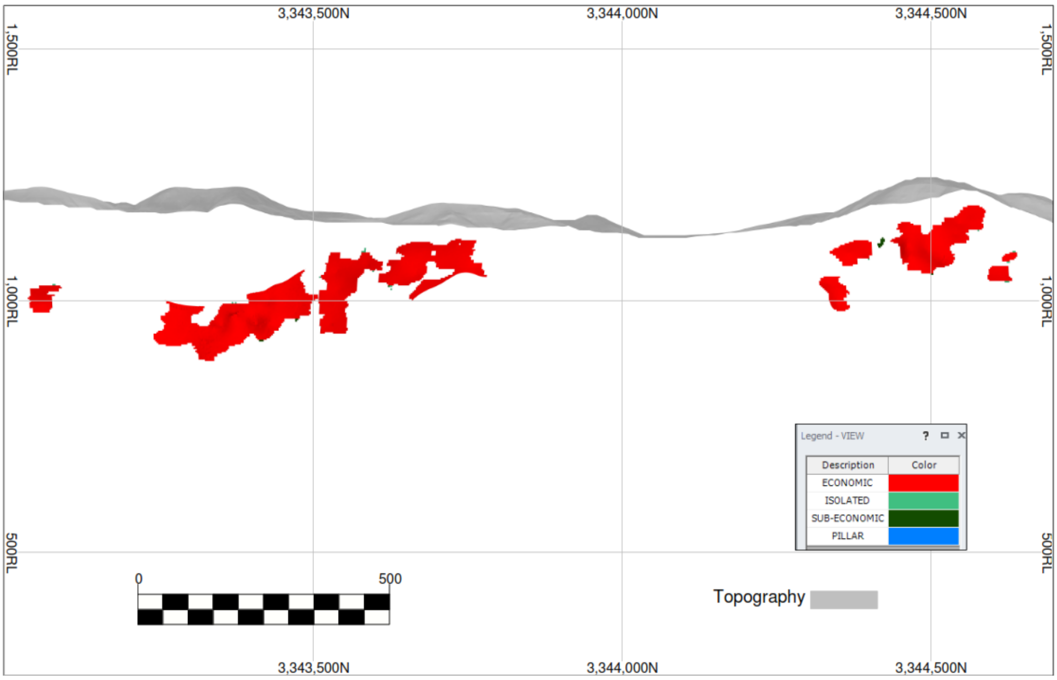
Note: Figure prepared by GMS, 2020.

Figure 15-2: Babicanora Main DSO Results (looking southwest)



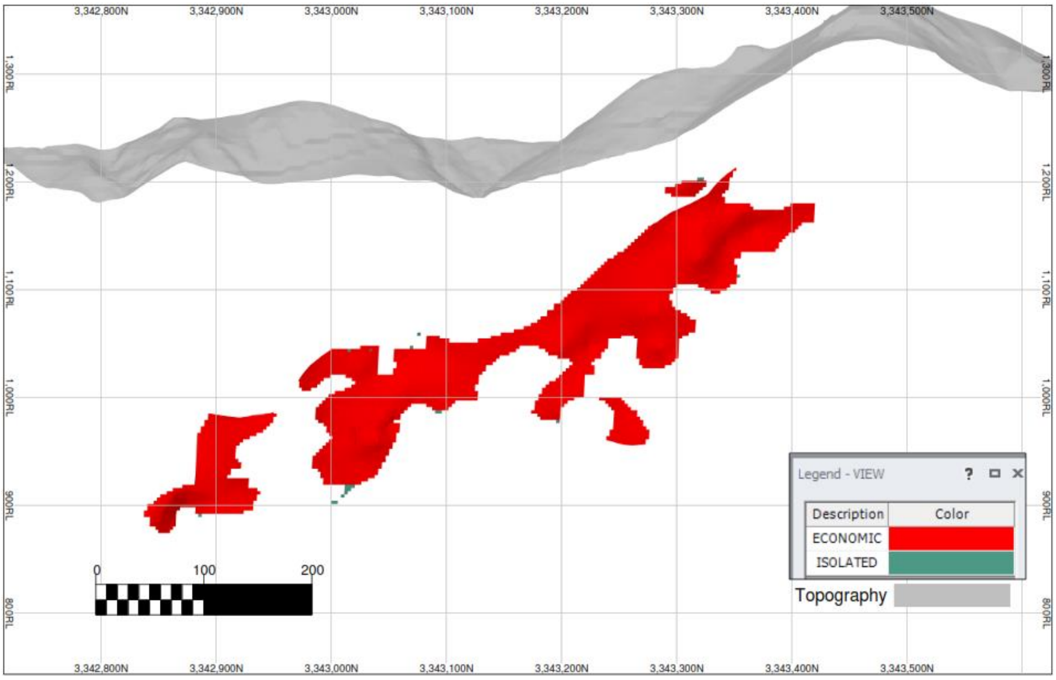
Note: Figure prepared by GMS, 2020.

Figure 15-3: Babicanora Central DSO Results (looking southwest)



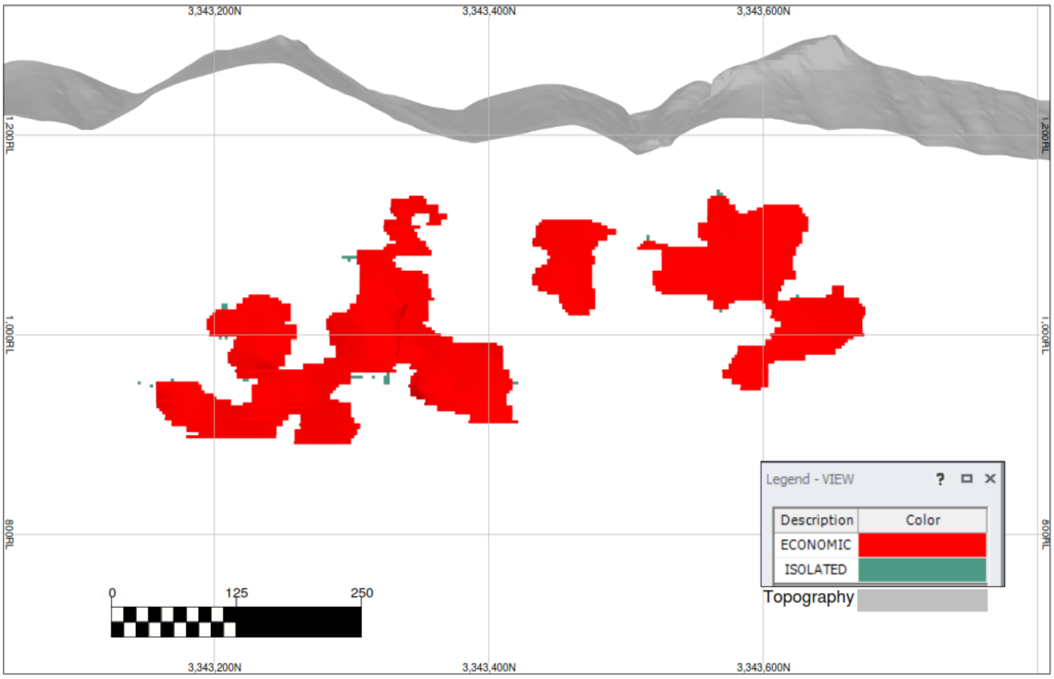
Note: Figure prepared by GMS, 2020.

Figure 15-4: Babicanora Norte DSO Results (looking southwest)



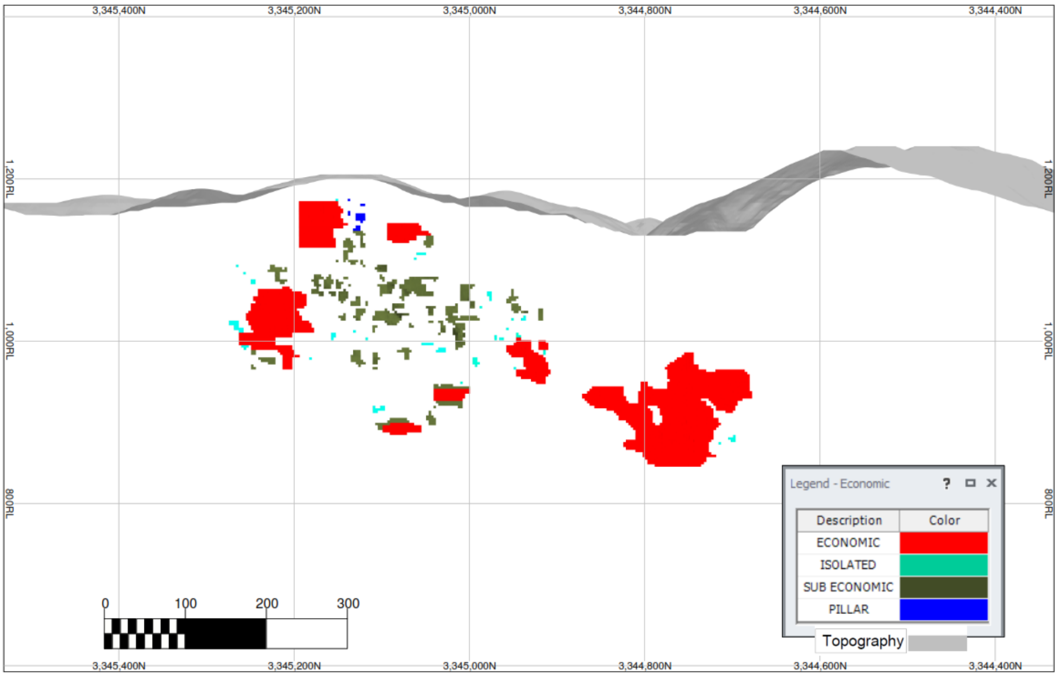
Note: Figure prepared by GMS, 2020.

Figure 15-5: Babicanora Sur DSO Results (looking southwest)



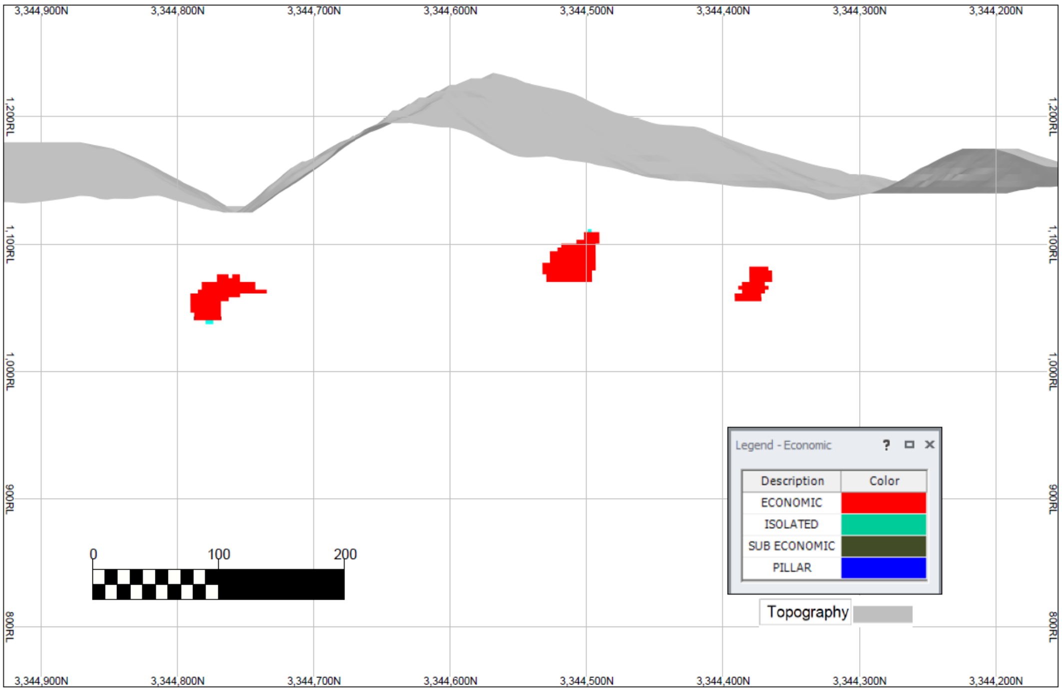
Note: Figure prepared by GMS, 2020.

Figure 15-6: Babi Vista DSO Results (looking northeast)



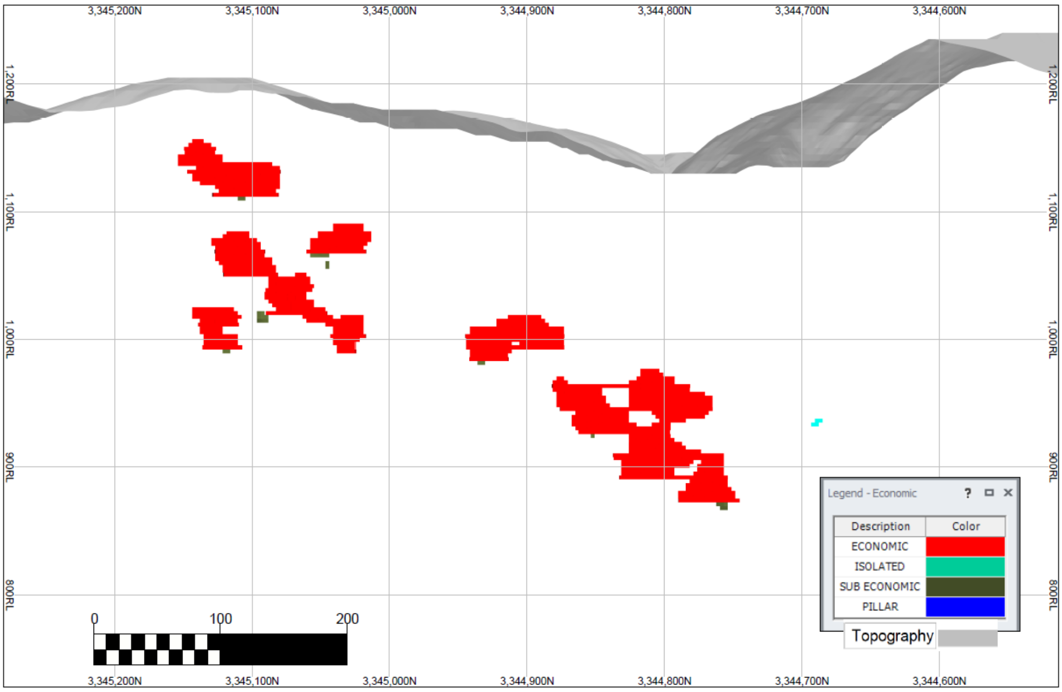
Note: Figure prepared by GMS, 2020.

Figure 15-7: Las Chispas DSO Results (looking northeast)



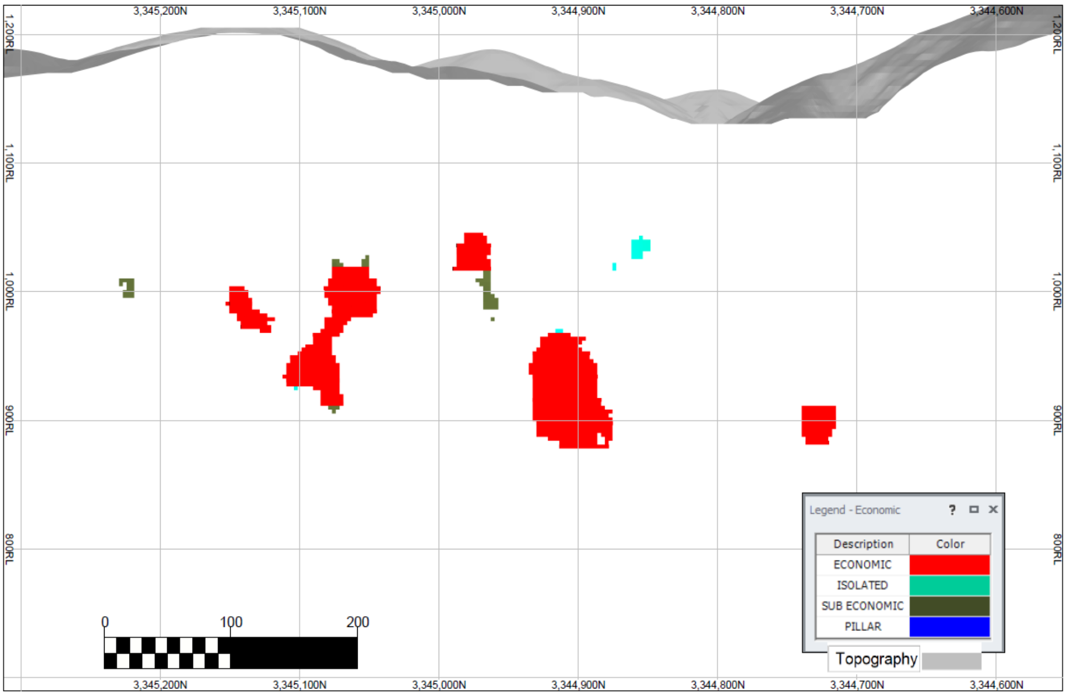
Note: Figure prepared by GMS, 2020.

Figure 15-8: William Tell DSO Results (looking northeast)



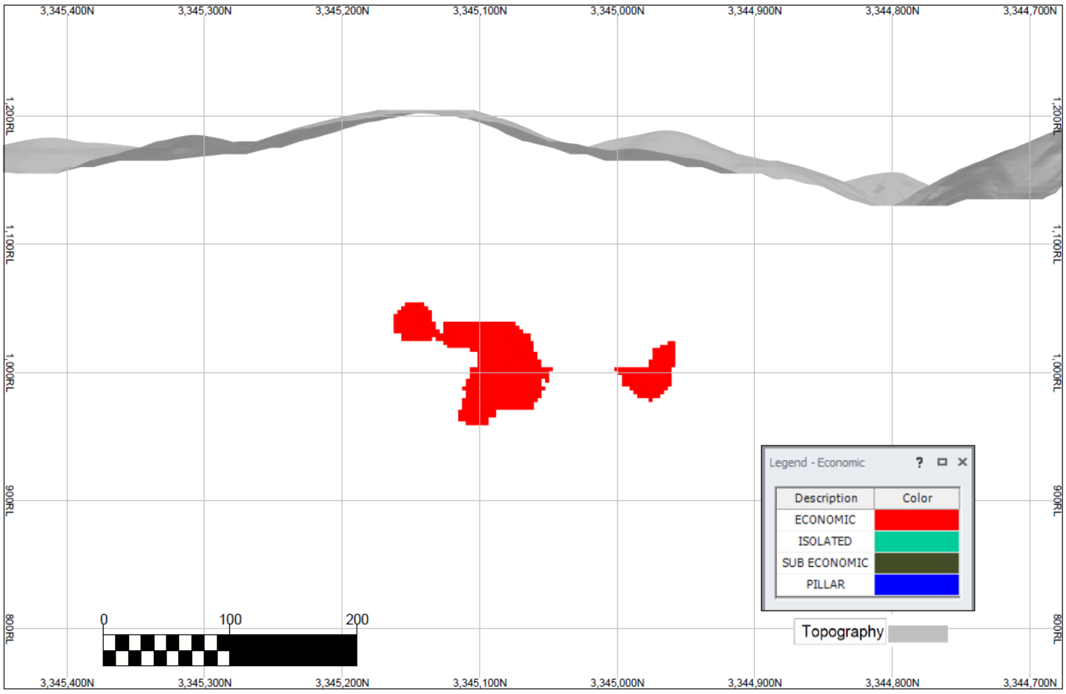
Note: Figure prepared by GMS, 2020.

Figure 15-9: Giovanni DSO Results (looking northeast)



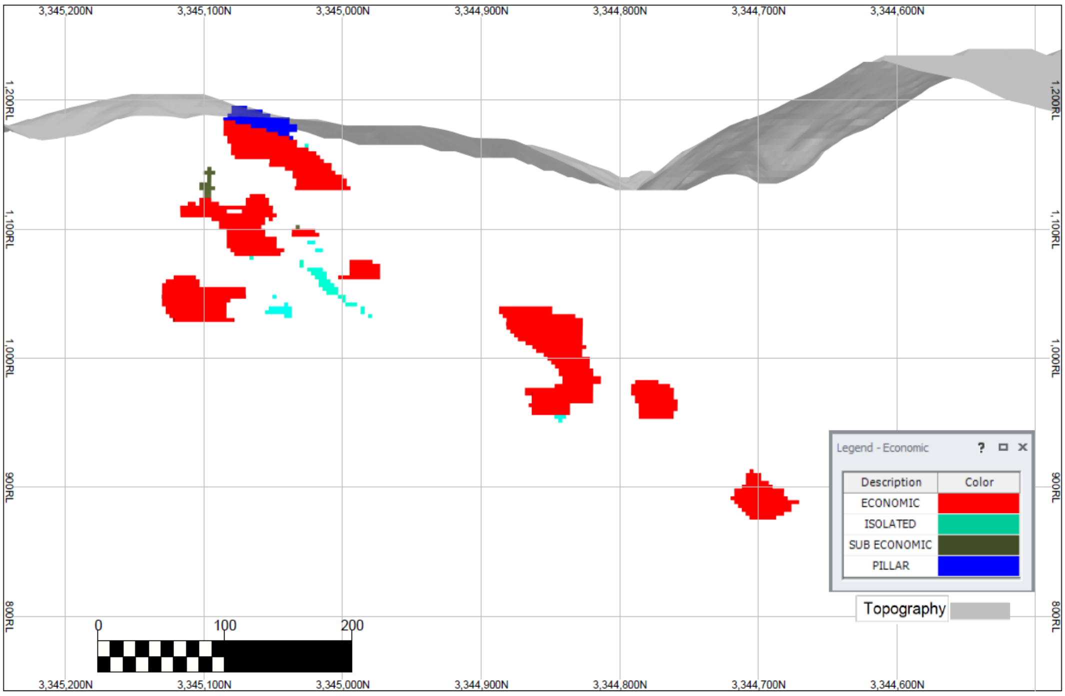
Note: Figure prepared by GMS, 2020.

Figure 15-10: Luigi DSO Results (looking northeast)



Note: Figure prepared by GMS, 2020.

Figure 15-11: Luigi FW DSO Results (looking northeast)



Note: Figure prepared by GMS, 2020.

Figure 15-12: Gio Mini DSO Results (looking northeast)

16 Mining methods

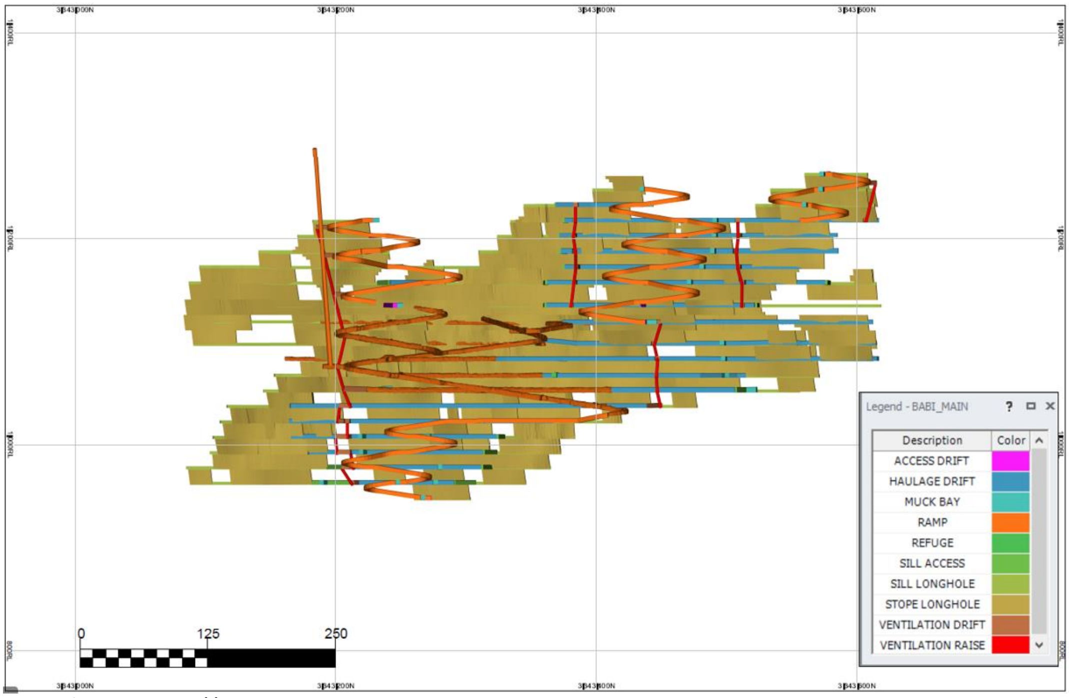
16.1 Introduction

The proposed mining approach will use variations of long-hole stoping and cut-and-fill mining methods via several access drifts and ramps. These methods are appropriate to the sub-vertical geometry of the veins that have thicknesses ranging from 0.5–10 m.

The long-hole stope mining methods will include long hole longitudinal retreat stoping and Avoca. These methods will be used in mining areas where vein thicknesses are >1.5 m, and when the good ground allow it. Avoca requires multiple accesses to the veins, whereas long-hole longitudinal retreat typically requires only one access.

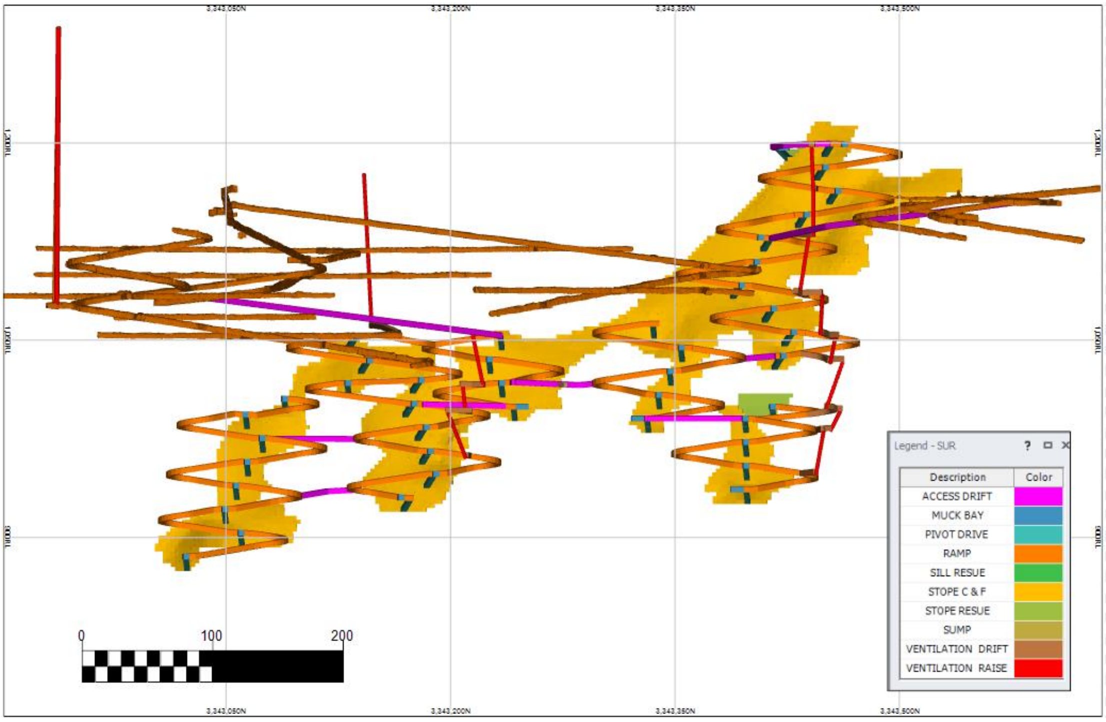
Variations of cut-and-fill mining methods will include cut-and-fill with uppers, cut-and-fill with breasting and resuing. Cut-and-fill with uppers will be used in mining areas with “Fair” ground conditions and where the vein thickness is >1.5 m. Cut-and-fill with breasting will be used in mining areas with adverse ground conditions, and where the vein thickness is >1.5 m. Resuing is used in mining areas where the vein thickness is <1.5 m, independent of ground conditions.

Mining operations will extract from 15 principal veins: Babicanora Main, Babicanora FW, Babicanora HW, Babicanora Sur, Babicanora Sur HW, Babicanora Norte, Babicanora Norte HW, Babi Vista, Babi Vista FW, Luigi, Luigi FW, William Tell, Giovanni, Gio Mini and Las Chispas. These veins are grouped into six (6) mining areas: Babicanora Main, Babicanora Sur, Babi Vista, Babicanora Norte, Babicanora Central and Las Chispas. Each of these mining areas will be serviced by supporting infrastructure including power distribution, compressed air distribution, water supply, ventilation, dewatering and communications. Figure 16-1 to Figure 16-6 show the planned infrastructure for each area and Figure 16-7 shows the overall mine layout.



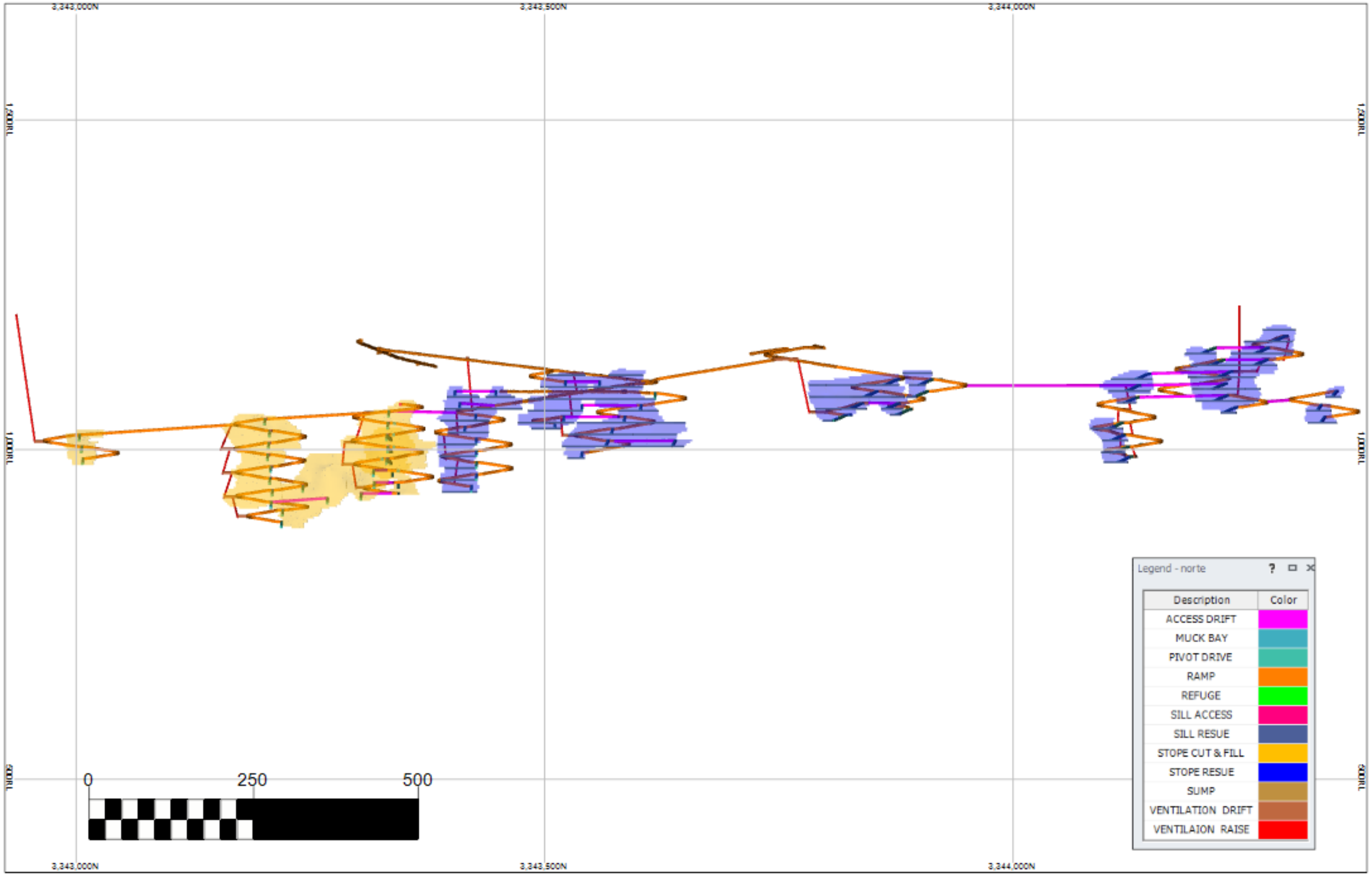
Note: Figure prepared by GMS, 2020.

Figure 16-1: Babicanora Main, FW and HW Longitudinal View (looking southwest)



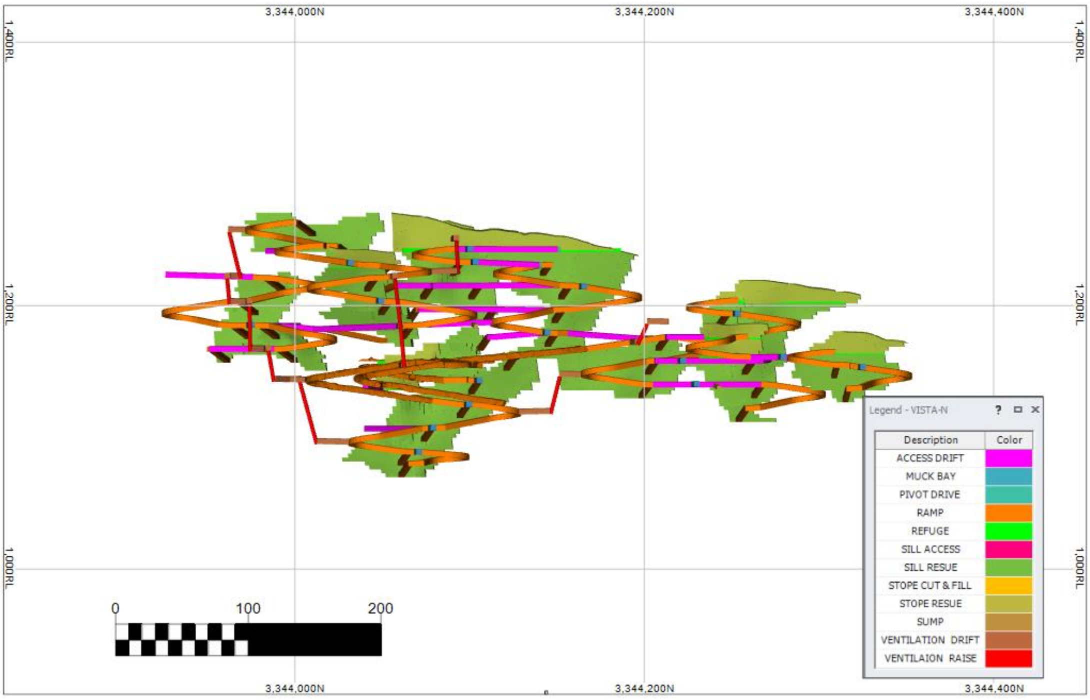
Note: Figure prepared by GMS, 2020.

Figure 16-2: Babicanora Sur and HW Longitudinal View (looking southwest)



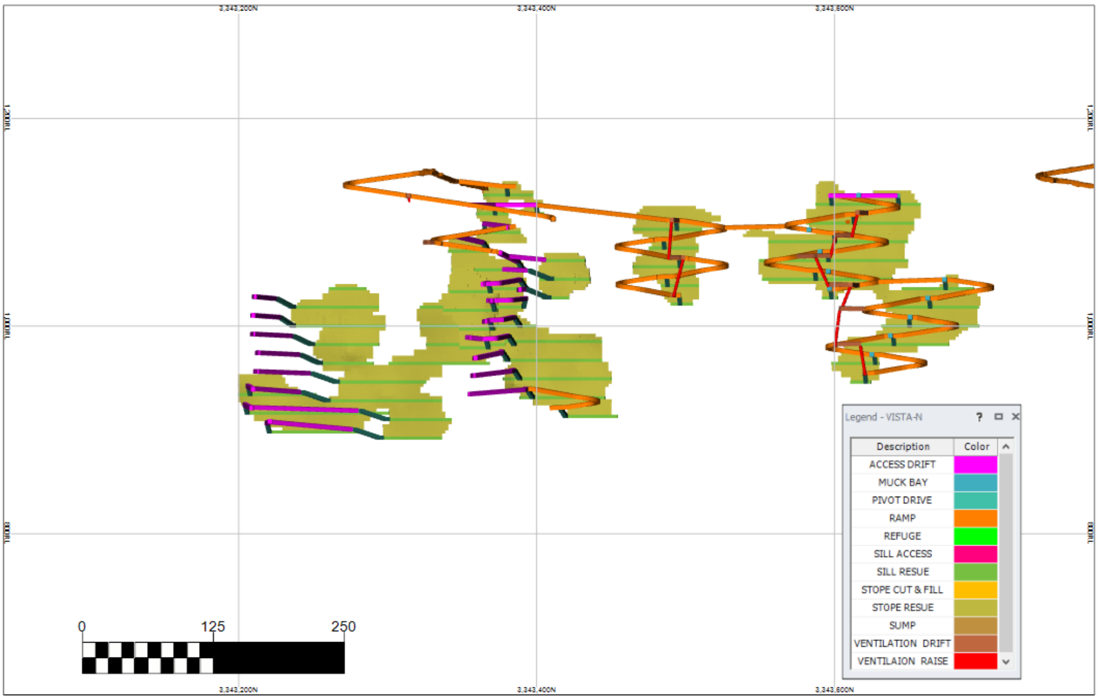
Note: Figure prepared by GMS, 2020.

Figure 16-3: Babicanora Norte Longitudinal View (looking southwest)



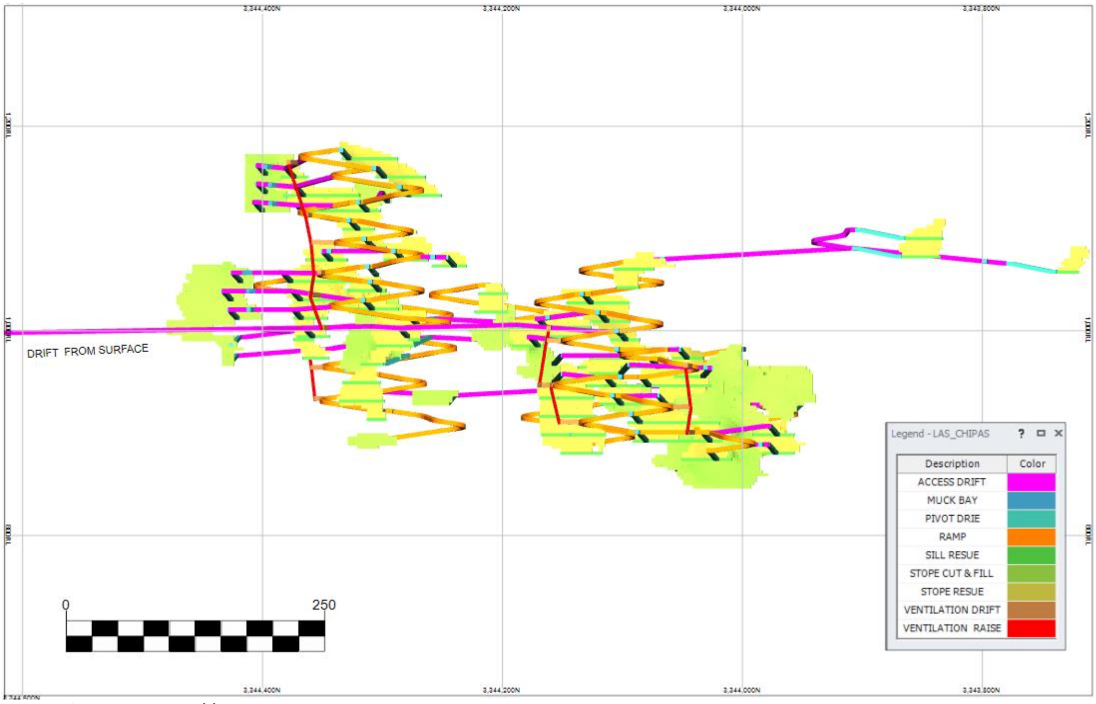
Note: Figure prepared by GMS, 2020.

Figure 16-4: Babicanora Central Longitudinal View (looking southwest)



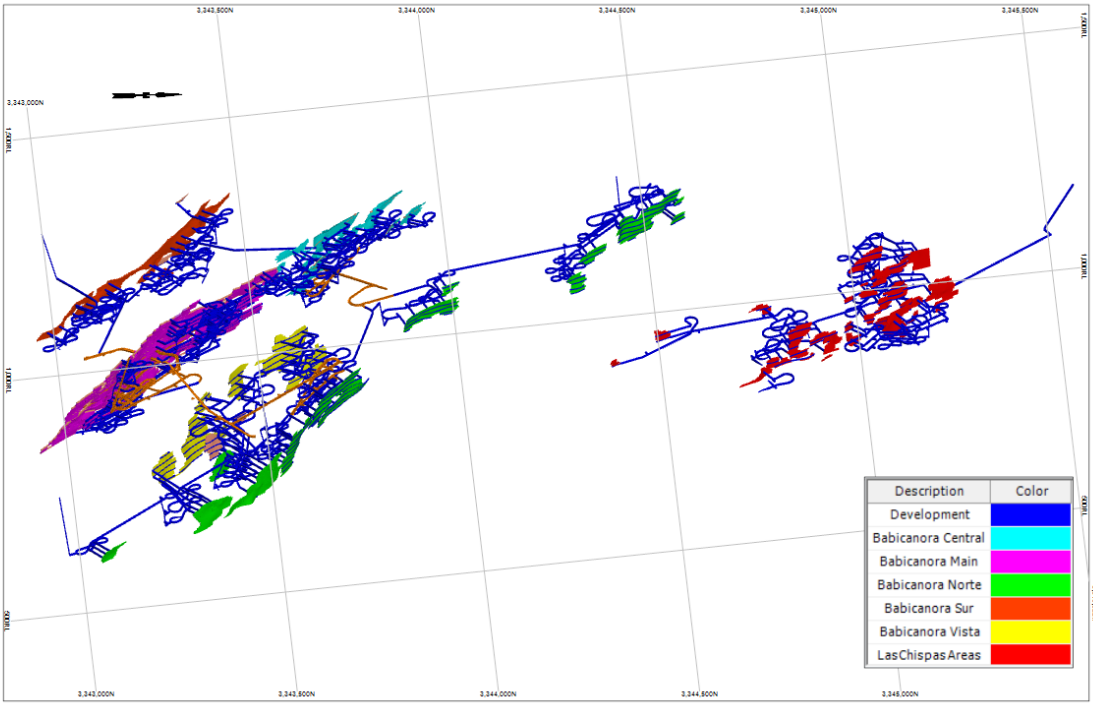
Note: Figure prepared by GMS, 2020.

Figure 16-5: Babi Vista and FW Longitudinal View (looking southwest)



Note: Figure prepared by GMS, 2020.

Figure 16-6: Las Chispas, Gio Mini, Luigi, Luigi FW and William Tell Longitudinal View (looking northeast)



Note: Figure prepared by GMS, 2020.

Figure 16-7: Babicanora and Las Chispas areas Oblique View (looking west)

The mining areas will be accessed by ramps leading either to the surface or to other mining areas. Three portals are planned: the Santa Rosa, Babicanora Central and San Gotardo portals. The Santa Rosa portal will provide access to the Babicanora Main, Babi Vista, Babicanora Norte and Babicanora Sur mining areas. The Babicanora Central portal will access the Babicanora Central, Babi Vista, Babicanora Norte and Babicanora Sur areas, and the San Gotardo Portal will access the Las Chispas mining area. A total of 85,773 m of lateral development and 3,428 m of vertical development is required over the LOM as shown in Table 16-1.

Table 16-1: LOM Development (m)

	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	Total
Lateral development	6,567	7,591	9,846	11,053	13,578	13,459	10,115	7,957	5,390	217	85,773
Vertical development	607	471	259	456	436	468	493	181	57	0	3,428

16.2 Geotechnical Considerations

A detailed geotechnical program was carried out in 2019–2020 by Rockland Ltd., which provided the geomechanical mine design criteria (Rockland, 2020a). The program consisted of field data collection, laboratory tests, analytical, and numerical modeling investigations. Initially, a training program for geotechnical data collection from drill hole cores and underground joint mapping was implemented. The majority of resource drill holes were geotechnically logged. Representative rock cores were selected and tested at EPGC in Mexico which is an independent commercial geotechnical l laboratory. This laboratory is accredited under the NMX-EC-17025-IMNC-2018/ISO / EC 17025: 2017 standard.

16.2.1 Geotechnical Characterization Data

The following information was used as the basis for the geotechnical assessments:

- Lithological and geological structural information prepared by SilverCrest geologists;
- A long section of each vein that showed the location of geotechnical drill holes;
- Detailed geotechnical logs from 167 drill holes;
- Field point load test from 167 drill holes;
- Core photographs for most resource and geotechnical drill holes;
- Site visits and geotechnical face mapping from the existing underground developments; and,
- Commercial laboratory strength tests including uniaxial compressive strength (UCS) and Young's modulus.

The geotechnical program was planned, directed, and monitored regularly by Rockland Ltd, and data were collected by SilverCrest's rock mechanics engineers and geo-technicians. The veins considered for the geomechanical assessments included Babicanora Main, Babicanora Central, Babicanora Norte, Babicanora Norte Northwest, Babi Vista, Babicanora Sur, Babicanora FW, Las Chispas Vein, La Blanquita zone, and Giovanni Veins including Area 118 zone.

16.2.2 Rock Mass Properties and Geotechnical Domain

The UCS results are provided in Table 16-2.

Table 16-2: UCS Test Results Summary

Rock Type Waste	No. of Tests	Average UCS MPa	Vein Name	No. of Tests	UCS MPa
LAT1	4	38.1	Babicanora FW	1	28.1
FIAT1	4	41.7	Babicanora Sur	1	36.2
RDCLF	4	35.3	Babicanora Central Zone	1	36.3
SACTS	2	39.1	Babicanora Norte Northwest	1	29.5
MADT3	4	55.5	Giovanni	1	28.5
			Babi Vista	1	21.3
			Babicanora Norte	1	21.3

As part of the geotechnical core logging program, point load tests were also conducted on core samples from most of the resource drill holes, and point load indices were established for immediate hanging wall, vein, and the immediate footwall cores.

Two rock mass classification systems; RMR (Bieniawski, 1976) and Q, Rock Tunneling Index, (Barton et al, 1974), were employed for the rock quality data collection program. Geotechnical data were collected for the immediate hanging wall, vein domain, and immediate footwall domains. The dry condition was assumed and the range of rock quality for each domain was established. Table 16-3, summarizes the results.

Table 16-3: Rock Quality Ranges Based on the RMR_{76} Rock Mass Classification

Vein Name	Hanging wall Domain Range 25–75% Percentile	Vein Domain Range 25–75% Percentile	Footwall Domain Range 25–75% Percentile
Babicanora Main (upper part) and Babicanora Central Zone	26–41 Poor–Fair	24–45 Poor–Fair	30–62 Poor–Good
Babicanora Main (lower part)	40-51 Poor–Fair	40–47 Poor–Fair	47–60 Fair
Babicanora Norte, Babicanora Norte Northwest, Babi Vista, Babicanora Sur, Babicanora FW, Las Chispas Vein, La Blanquita zone, and Giovanni Veins	51-84 Fair-V. Good	41–76 Fair–Good	47–84 Fair–V. Good

Therefore, the rock quality of the various Las Chispas veins can be broadly divided into two main domains, "Poor–Fair/Good" and "Fair–Good/V. Good". The lower part of the Babicanora Main vein has a better rock quality than the upper portion. The Babicanora Norte, Babicanora Norte Northwest, Babi Vista, Babicanora Sur, Babicanora FW, Las Chispas, La Blanquita zone, and Giovanni Veins have a range of "Fair-Good/V. Good" rock quality.

16.2.3 Discontinuity Sets

Joint data were collected from several underground locations including Babicanora Main Vein, Babicanora Central Zone, and Las Chispas Vein. The results showed three to four main joint sets were present. This information was used for the empirical stope design analysis and ground support recommendations. As underground headings are excavated in the veins, additional data should be collected and used for geomechanical analyses.

16.2.4 Stress Regime

The magnitude and orientation of in-situ stresses influence ground behaviour. There have been no in-situ stress measurements conducted at Las Chispas or in the vicinity of the planned operation. Based on a review of the World Stress Map, the maximum horizontal stress may be oriented perpendicular to the vein strike. Considering the relatively shallow mining depth, the rock types hosting mineralization, the planned use of backfill, and current underground observations, the main stress regime will not likely impact the size of the stope openings. However, reduced mining stresses around excavations will affect the stability of large, open span areas. Gravity-induced unraveling is expected as the main failure type in unsupported areas. The pre-mining stress field should be further evaluated during the pre-construction and early construction phases, and if required, the design should be adjusted.

In the absence of stress measurements, it was initially assumed that the magnitude of the major horizontal stress was approximately two times the vertical stress and the minimum principal stress was approximately equal to vertical stress. Subsequently, sensitivity analyses were carried out to evaluate the impact of stress variations. The anticipated range of in-situ stresses was considered in the numerical modeling and calculation of stress parameters during stability analysis and stope design as discussed in the following sub-sections. The accuracy of these assumptions should be reviewed as more information becomes available during the mining operation.

16.2.5 Empirical Stope Design Analysis

The stope dimension analysis was conducted for all veins using the Mathew-Potvin or stability graph analysis method (Potvin, 1988). The design procedure is based on the calculation of two factors, the stability number (N') and hydraulic radius (HR). N' is calculated based on the rock mass quality (Q'), and three empirical factors: A (induced stress conditions), B (geologic structure orientations), and C (dip angle of the stope face). Underground joint mapping and numerical modeling were conducted to establish empirical parameters. The range of rock mass qualities was selected, and upper and lower bounds of the hydraulic radius were established. This information was used in mining method selection.

Long-hole mining methods were planned for Babicanora Main. Empirical and numerical modeling methods were conducted to establish the hydraulic radius range. An underground stope with a 70 degrees dip angle was used for the analyses. Two-dimensional numerical modeling was conducted to estimate the maximum induced stresses around the stope. The calculated Q' value ranges were based on drill hole data. The HR values are summarized in Table 16-3.

Table 16-3: Stope Stability Analyses

<table>
<tr><th>Vein</th><th>Walls</th><th>Dip (°)</th><th>Q'</th><th>A</th><th>B</th><th>C</th><th>N'</th><th>HR (m)</th><th>Mining Method</th></tr>
<tr><td rowspan="2">Babicanora Main</td><td>HW Domain / FW Domain</td><td>70</td><td>2-8</td><td>0.15</td><td>0.3</td><td>6</td><td>0.54-2.2</td><td>~2.5-3.5</td><td rowspan="2">long hole (Avoca)</td></tr>
<tr><td>Back</td><td>0</td><td>1-2.8</td><td>0.15</td><td>0.4</td><td>2</td><td>0.12-0.34</td><td>~1.5-2</td></tr>
</table>

Vein	Walls	Dip (°)	Q'	A	B	C	N'	HR (m)	Mining Method
	End	90	1-2.8	1	1	8	8-22.4	~5.5-8	

Additional numerical modeling is recommended to optimize the mining sequence of various veins, confirm pillar dimensions, and for stope stability purposes.

16.2.6 Estimates of Unplanned Dilution

The potential for unplanned dilution was estimated for stope hanging walls and footwalls using the equivalent linear overbreak/slough (ELOS) method developed by Clark (1988). The method is similar to the empirical stope stability charts. The estimates are approximated in terms of average thickness spread over the entire hanging wall or footwall.

The ELOS graph does not apply to small hydraulic radii less than approximately 4 m. Where hydraulic radii are small, the blasting quality will be the primary dilution control. The unplanned dilution, where hydraulic radii and N' are 4 and 2.2 respectively, will range from 0.5–1 m. The ELOS method does not apply to areas that will be mined using cut-and-fill mining methods as the ground is supported in those areas.

16.2.7 Ground Support

Ground support selection considered industry-standard empirical design guidelines and Rockland's experience with variable ground conditions. Ground support was recommended based on rock quality, the period in use (long-term or short-term), and the size of headings (excavation). For headings in waste rocks with "Fair" and "Good" rock qualities, resin rebar and welded wire mesh were recommended. For headings in veins with "Fair" to "Good" rock quality, inflatable bolts (e.g., Swellex) and split set were specified. In areas with "Poor" rock quality, inflatable ground support and split sets with reduced spacings and mesh coverage to the sill were specified. The installation of spiling (pre-support), and shotcrete, are recommended for "Poor" areas. Rock mass damage should be expected at excavation intersections and where fault zones are present. Additional ground support was specified for these areas.

A kinematic assessment was carried out to evaluate the potential for wedge-type failures in the underground excavations. Unwedge (Rocscience, 2019) software was used to identify potential wedge combinations. The analysis assumes that joints are planar and that wedges will behave as rigid bodies. The program calculates the size, shape, weight, and factor of safety for wedges based on the size and orientation of excavation, in this instance assuming 5 x 5 m excavations. The underground joint mapping data were used. An angle of friction of 14º and cohesion of 98 KPa were assumed.

Results indicate that where the headings are bolted, wedges formed will be stable. The excavation axes were rotated to examine the instability of potential wedges around the excavation. Small and large wedges were formed which are stable if the recommended ground support is installed. The analyses confirmed the importance of scaling as well as the adequacy of the recommended ground support.

16.2.8 Crown Pillar

Several of the Babicanora Central stopes will come close to the ground surface; crown pillars are required for these stopes. A stability analysis was carried out to recommend the minimum crown pillar thickness (Rockland, 2020b). In general, for a surface crown pillar design, both empirical and numerical modeling assessments are required. The empirical assessment is used to provide the "first pass" design approach. Other methods are used to verify the suggested design. The scaled crown span empirical method (Carter, 2014) was used to estimate the crown pillar thickness. The limit equilibrium analysis using CPillar software (Rocscience, 2019) and finite element analysis using RS2 (Rocscience, 2019) were employed to verify the suggested design.

The crown pillar geometries were based on the three-dimensional solid model, and the rock quality was based on drill hole information from the crown pillar area. Using the scaled crown span method, a minimum factor of safety of 1.5 with a probability of failure of 5–10% was chosen. The crown pillar thicknesses were subsequently calculated. The results show that a crown pillar with strike lengths of 25 m, 50 m, and 100 m should have a minimum thickness of 12 m, 12.5 m, and 13 m, respectively.

A CPillar analysis was carried out with the same geometry as used for the scaled crown span method and strike lengths of 25m, 50m, and 100 m. The rigid plate analysis was employed for the Babicanora Central crown pillar assessment. A rock mass quality of RMR_{76}=27 was selected based on the collected information. An angle of internal friction and cohesion values were established for the analyses. Since the total stress ratios at the crown pillar area are unknown, a range of 0.7–1 was used. The factor of safety was >2 for the selected strike lengths, and therefore the crown pillars are stable.

For finite element analyses, a model consisting of backfilled stopes, the final excavated stope, and the crown pillar was constructed. As RS2 is a two-dimensional numerical model, it is more representative of a stope with a 100m strike length. The stress regime data for the Babicanora Central vein is unknown, and a range of 0.7–1 for the total stress ratios was selected for the analyses. Material properties for the host rock and vein were based on the rock quality in the area and generalized Hoek and Brown failure criteria were used.

The modeled crown pillar was 13 m thick, had a 2 m span, and a stress ratio of one. The crown pillar was stable, and the factor of safety was generally >2 except for a limited area close to the back of the stope that had a factor of safety of <2. As the stress ratio was reduced to 0.7 the factor of safety was reduced in the back of the stope, but it was still >2.

A cut-and-fill mining method will be used for mining at Babicanora Central. The rock quality is expected to be "Poor" within the crown pillar area. Ground support installation with good QA/QC (such as pull test program and visual monitoring) and immediate backfilling after mining will be required as mining advances in the crown pillar area. As underground development advances in the mining block close to the crown pillars, detailed information on geology/structural geology (in particular presence of fault zones), hydrogeology, and geotechnical data should be collected. This would allow the recommended crown pillar thicknesses to be optimized using three-dimensional numerical modeling and risk assessment measures.

16.3 Hydrological Considerations

16.3.1 Introduction

A hydrological and hydrogeological study was completed by HRI. The general physiography of the Project area and the prevailing climate were discussed in Sections 5.4 and 5.2 respectively.

16.3.2 Field Investigations

The following field investigations were completed:

- Installation of six (6) pressure probes to measure surface flow elevations, with measurements taken every 12 hr;
- Water elevation measurements taken for quality control purposes in three (3) piezometers to provide a cross-check on measurements taken by SilverCrest;
- Slug testing in three (3) piezometers to determine the hydraulic conductivity; and.
- Pump test in a stope at the base of the historical workings at Las Chispas that is filled with groundwater, and which is the only known location in the historical operations that has groundwater.

16.3.3 Field Investigation Results

There was insufficient rainfall during the monitoring period to generate any pressure variation between the six pressure probes.

Water elevation measurements indicated the presence of a perched phreatic surface considerably above the natural water table. The water table is at approximately 900 masl elevation, and the perched phreatic surface is at about 1,032 m elevation. The perched phreatic surface does not impact the historical workings, and for the purposes of the mine plan, will not require dewatering.

Results of the slug tests are summarized in Table 16-4.

Table 16-4: Slug Test Result Summary

Piezometer	From (m)	To (m)	K (m/s)
3B	4.4	100.7	1.17e – 05
3A	4.8	100.7	1.30e – 06
6	15.25	36	1.81e – 06

Values in Table 16-4 apply to the upper, saturated, portion of the rock formations and are not suitable for use in underground dewatering calculations.

The pump test had two objectives, firstly to see if there was sufficient water to supply future mining operations, and secondly, to assess the potential groundwater inflow.

The water-filled stope volume was estimated at approximately 500 m^3, and the total flow pumped during the pump test was measured at 3,297 m^3. The pump operated approximately 8 hr/day, at a flow rate of 6.3 L/s. The average flow rate was 1.7 L/s for the 24-day duration of the pump test. A drawdown of almost 9 m was measured for a flow rate of 1.7 L/s, giving an approximate specific yield of 0.19 L/s. The result indicated that the host rocks had low permeability. Two data trends were noted, with drawdowns at <3 m and >3 m. As a result, the theoretical flow rate was recalculated to ensure that the drawdown was <3 m. After 15 years, to maintain the drawdown at <3 m, the sustainable flow rate would be 0.31 L/s. This theoretical flow rate agrees with the pump-test results and supports the interpretation of a low-permeability medium.

Evaluation of the data collected from the pump test indicated that the flow rate used during testing was too low (on its own) to provide a reliable and sustainable water source for operations (see Section 18.14 for the site water balance).

16.3.4 Structural Analysis

A review of SilverCrest's structural model indicated that only one fault, which could be intersected in the Babicanora area (Figure 16-8), could be associated with potential water inflows. However, as the majority of the workings will be above 900 masl (water table elevation), groundwater inflows are not expected to be a concern to mining operations.

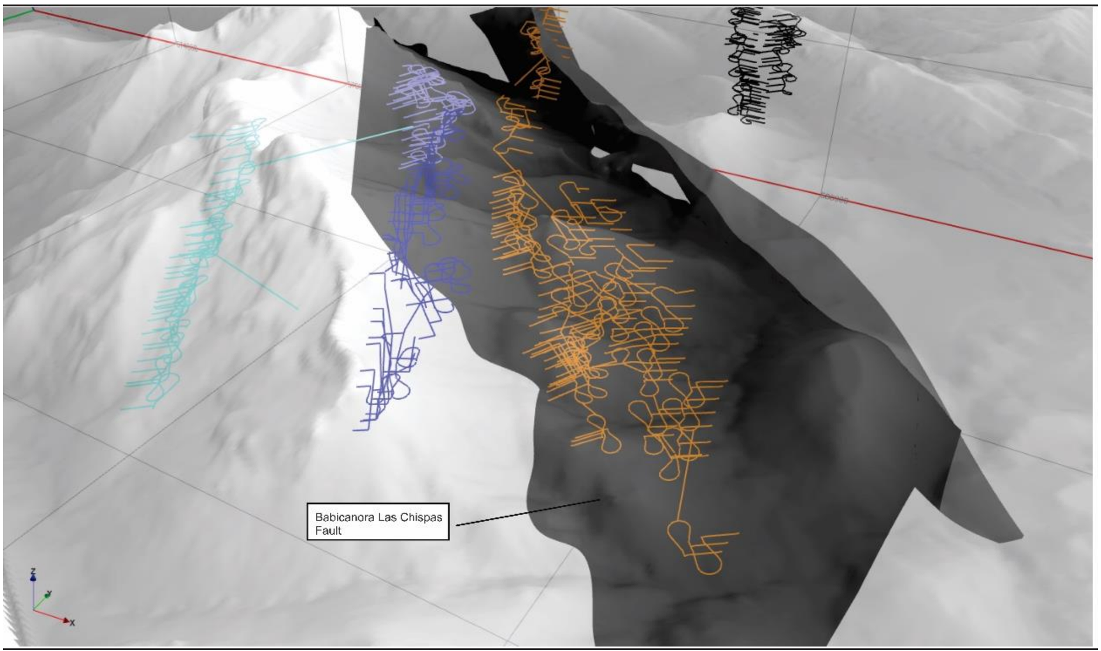


Figure 16-8: Faults Interpreted by SilverCrest

HRI performed a review of drill core logging data to identify areas of discontinuity. As the rock quality designation (RQD) data were not considered reliable, core recovery data were used instead. Data were filtered to remove all but the lowest 25% of the recovery values. Two trends were noted in Figure 16-9.

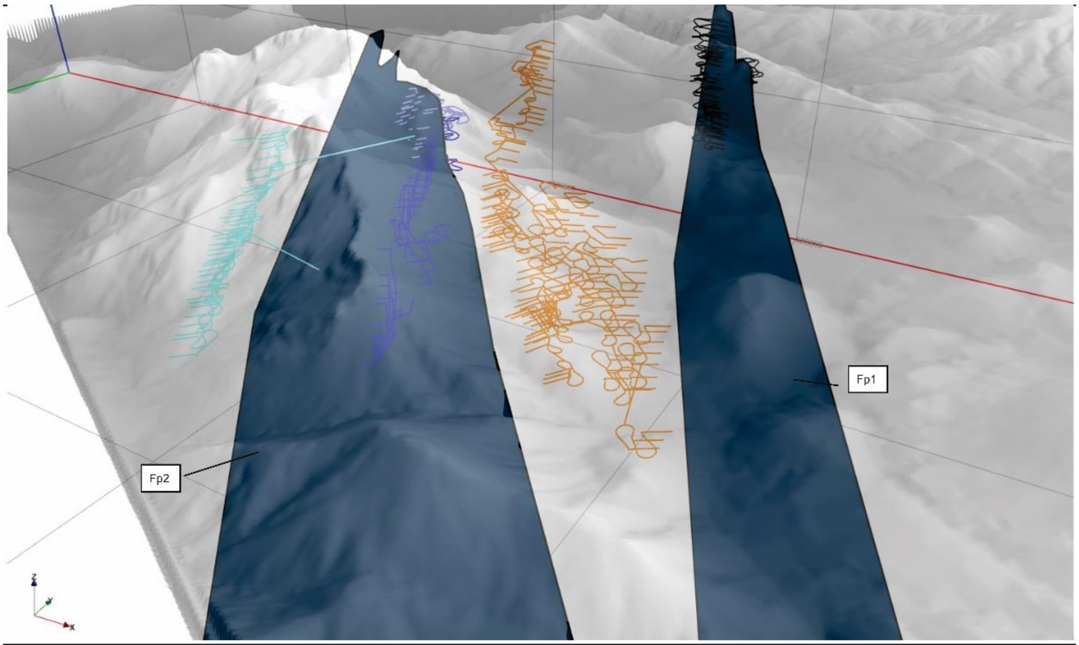


Figure 16-9: Discontinuities Interpreted by HRI

The first discontinuity (FP1) is aligned northwest and intersects the historical Las Chispas mine at a sub-vertical inclination. This discontinuity is likely associated with the fracturing of the vein system in the mine. The FP1 is expected to cause some inflow into the Las Chispas mining area when the mine plan is below the 900 m level. The lowest planned mining elevation is 850 m, 50 m beneath the water table elevation of 900 m.

Based on the pump test results, a maximum flow of about 9.4 L/s could be expected in the deepest mining level. There is insufficient data to determine if this flow rate will be sustained in the long-term. As a result, the mine plan in this area has been designed with a dewatering system in the lower levels of the deposit with a pumping capacity of 9.4 L/s; however, this pumping system will not be required until late in the mine life.

The second discontinuity (FP2) also trends northwest, has a subvertical inclination, and dips slightly to the northeast. It is interpreted to be associated with the fracturing of the vein system in the Babicanora Central zone. The Babicanora Central zone will be mined above the water table; and will not be impacted by FP2.

16.3.5 Potential Environmental Impacts

No impacts to surrounding perennial streams or valley bottoms are expected from mine dewatering activities, since these are typically dry other than during short-term, low precipitation rainfall events.

The Rio Sonora, located 7 km west of the future operation, is considered to be too distant to be affected by any future mine-related pumping.

16.4 Selection of Mining Methods

The mine design was based on a production rate of 1,250 t/d and will be reached by maintaining a proper balance between productive and selective mining methods.

The underground access and infrastructure development were designed to support the mining method and sized based on mining equipment and production rate requirements.

16.4.1 Longitudinal Long-hole Retreat

The longitudinal long-hole retreat method will be used in mining areas where the vein thickness is >1.5 m and the rock quality is “Fair” to “Good”. A minimum mining width of 1.5 m was applied to stope optimization to allow the use of mechanized equipment. Longitudinal long-hole retreat was applied to mining areas with a single access. Figure 16-10 is a schematic showing the longitudinal long-hole retreat mining method.

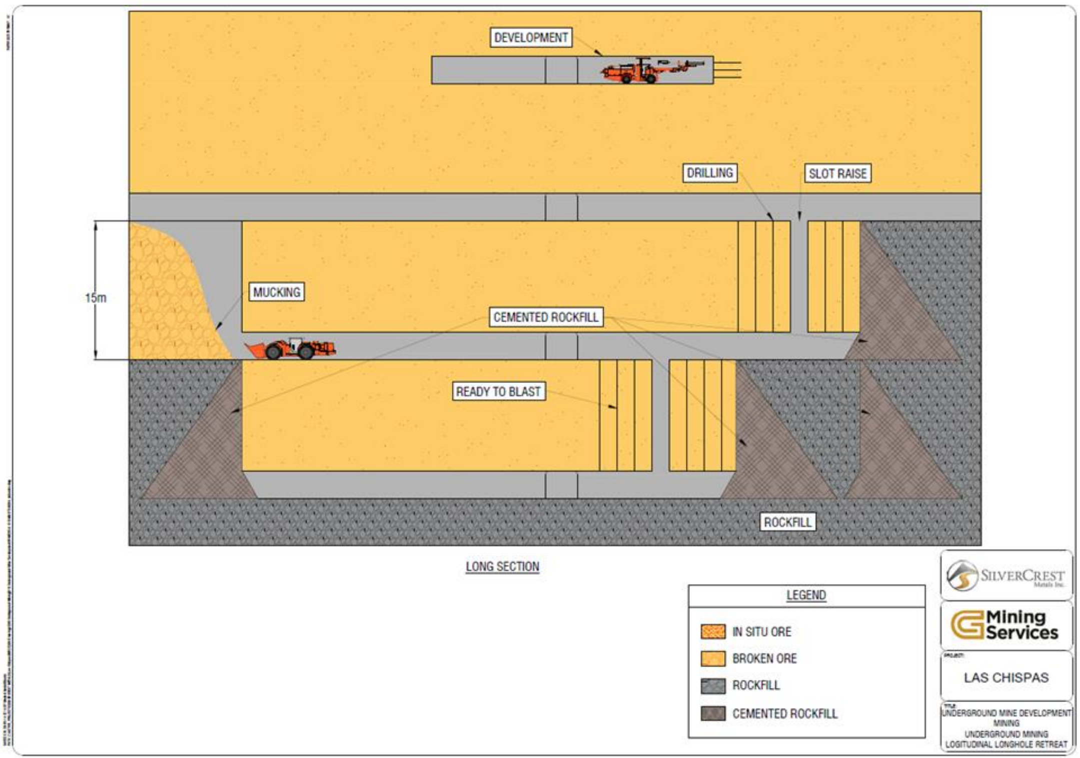


Figure 16-10: Longitudinal Long-hole Retreat

The mining area will be accessed by driving sill drifts below and above the stoping area using a single-boom jumbo and a 3 t scoop. Once the sills are developed to the extremity of the mining area, a drop raise will be developed between the two sill drifts to create sufficient open stope space to allow blasting. Down holes will be drilled by a narrow production drill between the two levels, loaded, and blasted. The stope will be mucked from the lower level before cemented rockfill is placed from the upper level to create a plug. Rockfill will be placed in the remainder of the stope. Subsequent stopes will be mined in the exact same cycle while retreating towards the access. Stopes will be sequenced in an overhand fashion.

16.4.2 Avoca

The Avoca method will be used in mining areas with “Fair” ground conditions where vein thicknesses are >1.5 m. A minimum mining width of 1.5 m was applied to stope optimization to allow the use of mechanized equipment. Avoca requires access from both extremities, allows increased productivities, and maintains better ground conditions. Figure 16-11 is a schematic showing the Avoca long-hole mining method.

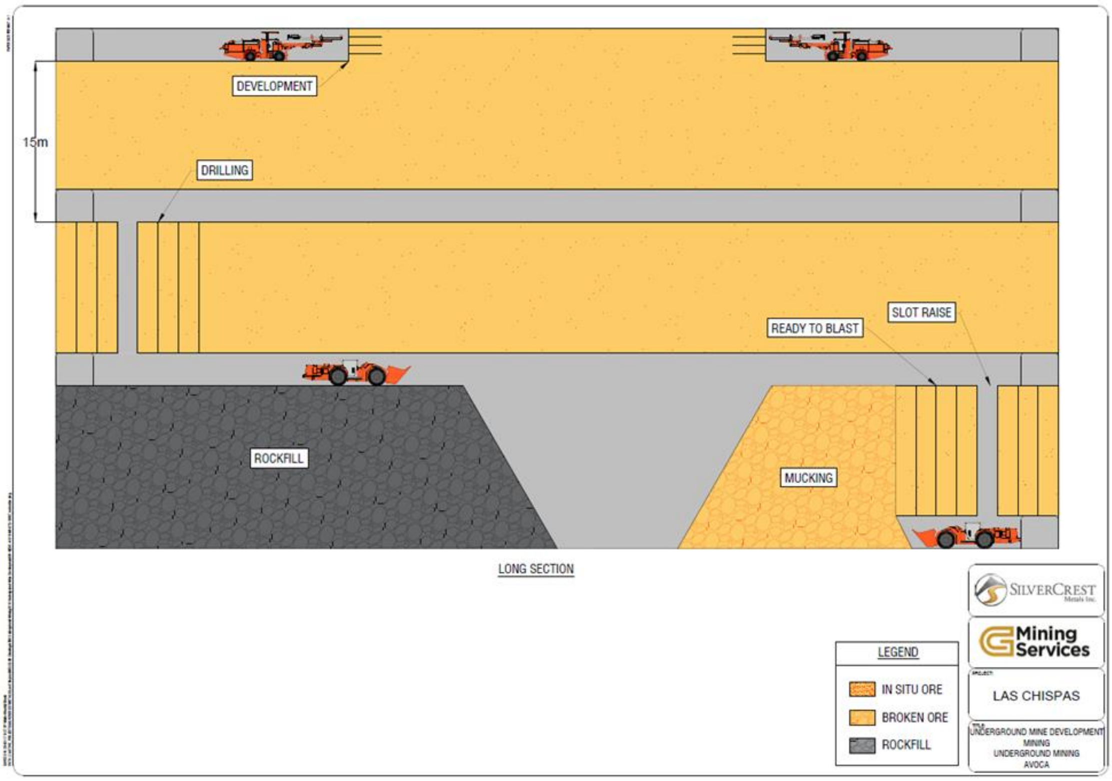


Figure 16-11: Avoca

The mining area will be accessed by driving sill drifts both below and above the stoping area using a single-boom jumbo and a 3 t scoop. Once the sills are developed with accesses at both extremities of the mining area, a drop raise will be developed between the two sill drifts to create sufficient open stope space to blast the first stope. Vertical blastholes will be drilled by a narrow production drill between the two levels, loaded, and blasted. The stope will be mucked from the lower level while the adjacent stope is being drilled from the upper level. Once the stope is mucked out, the adjacent stope will be blasted into the open stope created by the mined-out stope. Rockfill will be placed from the upper level at the opposite end. The cycle will be continued while mucking in a retreating fashion from one end and advancing with rockfill from the opposite end until the entire area is mined out. Stopes will be mined in an overhand fashion. The stope span will be kept at about 12 m between the rockfill and the intact rock. In the event that the dilution becomes too great, the stope span may be reduced, or the stope can be completely backfilled before restarting a new stope.

16.4.3 Cut-and-Fill Uppers

The cut-and-fill using uppers method will be applied in mining areas with "Fair" ground conditions where the vein thickness is >1.5 m. A minimum mining width of 1.5 m was applied to stope optimization to allow the use of mechanized equipment. Figure 16-12 is a schematic showing the cut-and-fill uppers mining method.

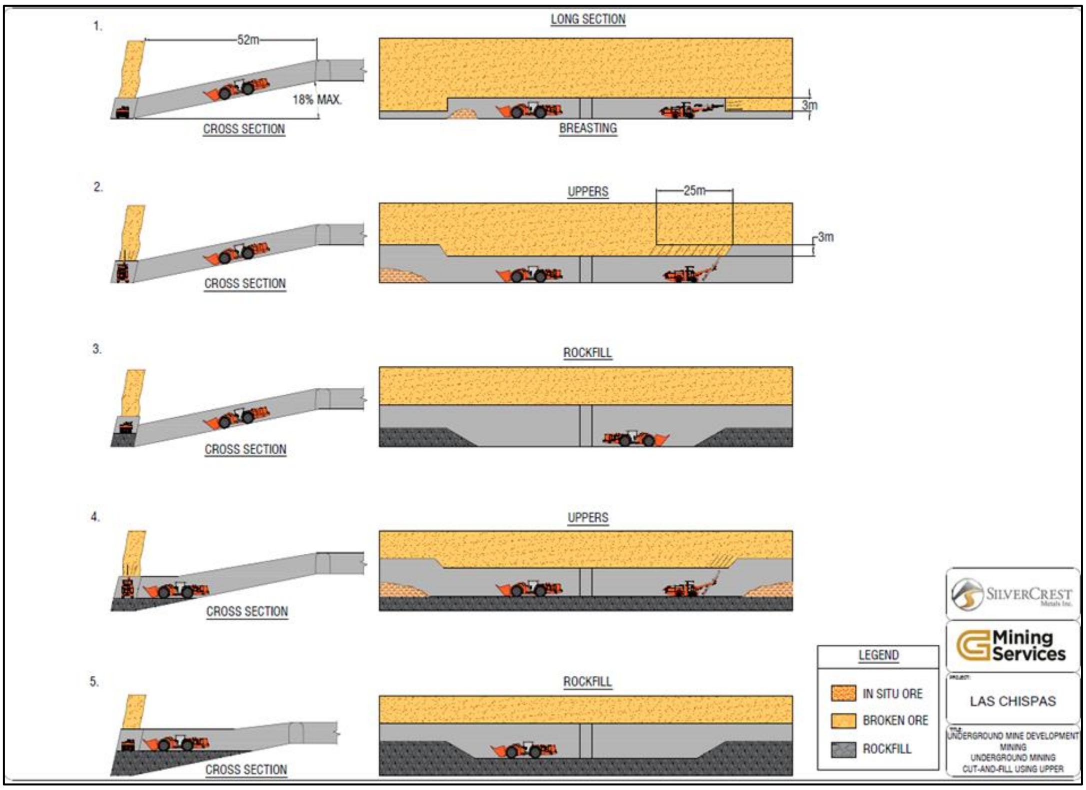


Figure 16-12: Cut-and-Fill Uppers

The mining area will be accessed by developing a pivot drive. An initial sill will be developed through the lower portion of the mining area with a single-boom jumbo and either a 1.2 t or 3 t scoop, depending on stope width. Once the sill has been developed to the extremity of the mining area, upper holes will be drilled with a single-boom jumbo, loaded, and blasted. The mineralized material will then be mucked. Subsequent rounds will be mined in the same cycle while retreating towards the access. Rockfill will be placed to create a new working floor for the next cut to be mined in an overhand fashion. The sill development was not included in the development metres but is included in the stope operating costs.

16.4.4 Cut-and-Fill Breasting

The cut-and-fill using breasting method will be applied in mining areas with adverse ground conditions where the vein thickness is >1.5 m. A minimum mining width of 1.5 m was applied to stope optimization to allow the use of mechanized equipment. Figure 16-13 is a schematic showing the cut-and-fill breasting method.

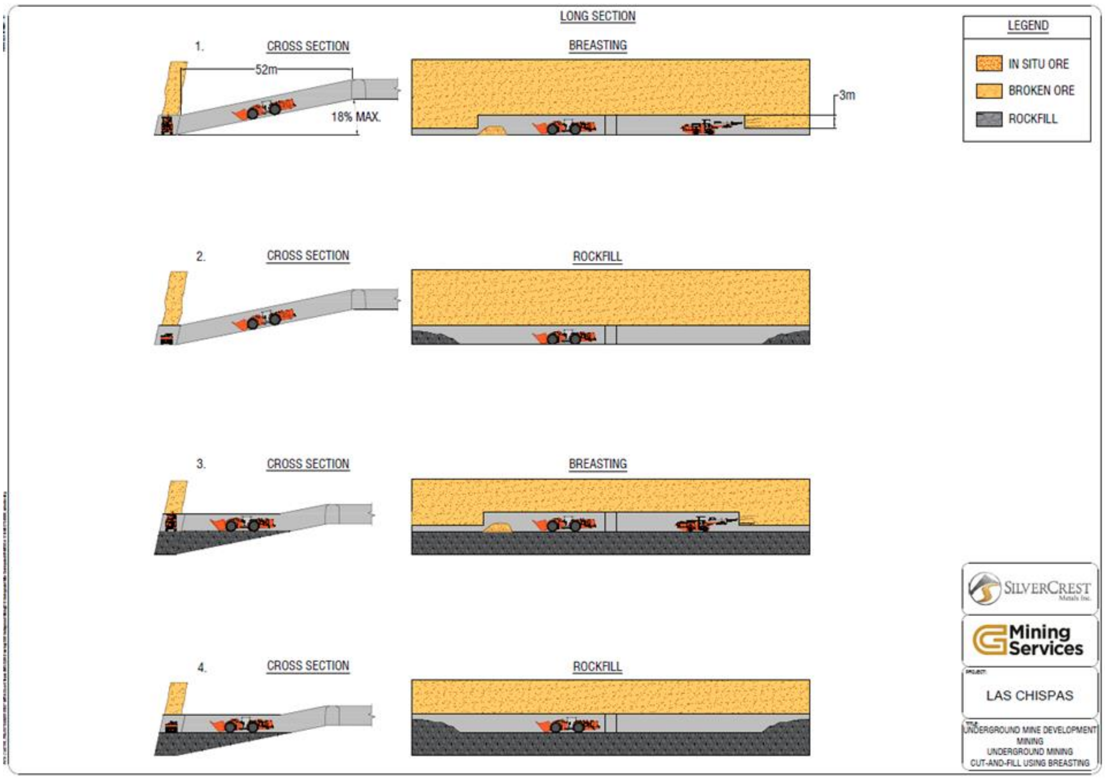


Figure 16-13: Cut-and-Fill Breasting

The mining area will be accessed by developing a pivot drive. An initial sill will be developed through the lower portion of the mining area with a single-boom jumbo and either a 1.2 t or 3 t scoop, depending on stope width. Once the sill has been developed to the extremity of the mining area, rockfill will be placed while allowing a 0.3 m gap to the sill back. The next cuts will be driven by slashing the breast holes rounds into this gap to enhance productivity. Rockfill will be placed to create a new working floor for the next cut to be mined in an overhand fashion. The sill development was not included in the development metres but is included in the stope operating costs.

16.4.5 Resue

Resuing will be used in mining areas where the vein thickness is <1.5 m and the rock quality is "Fair". A minimum mining width of 1.0 m was applied to stope optimization to allow for proper blasting of the mineralization. This technique allows for the use of mechanized equipment in narrow veins while minimizing dilution, although productivity is somewhat hindered. Figure 16-14 is a schematic showing the resue method.

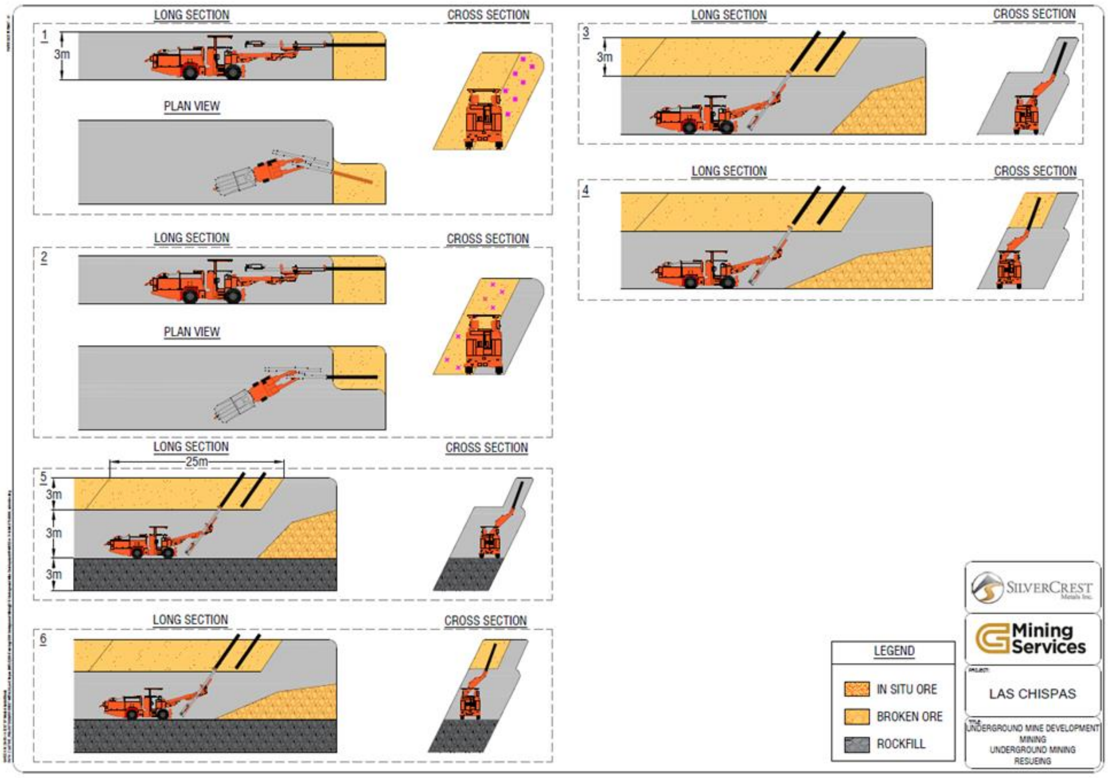
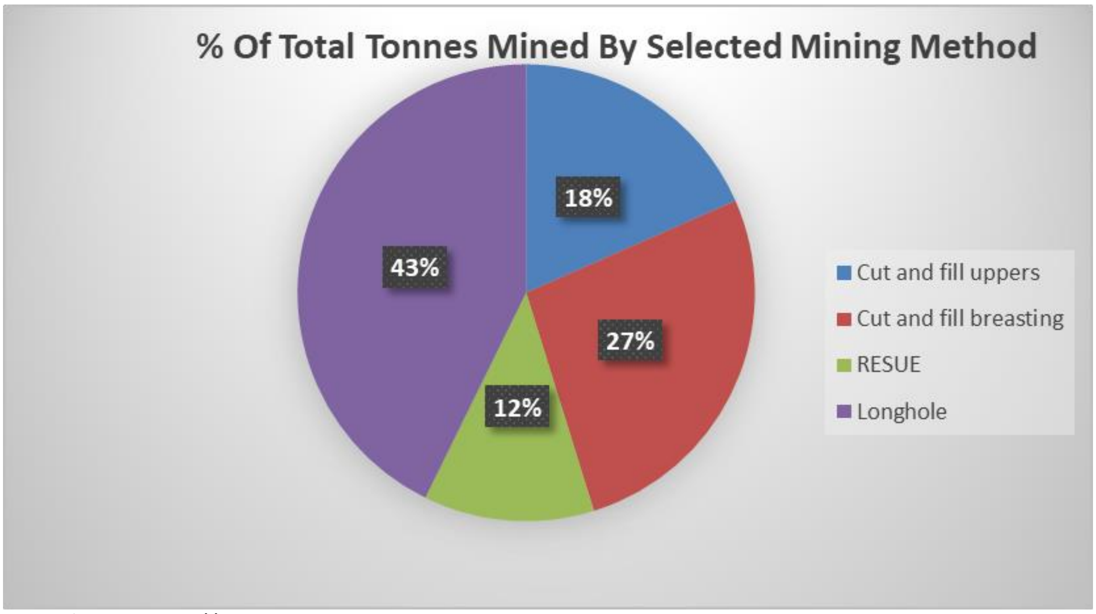


Figure 16-14: Resuing

The mining area will be accessed by developing a pivot drive. An initial sill will be developed through the lower portion of the mining area with a single-boom jumbo and a 3 t scoop. This sill will be developed using split face blasting techniques where the mineralized vein is drilled, loaded, blasted, and mucked before the same cycle is applied to the waste portion of the round. Once the sill has been developed to the extremity of the mining area, a resuing blasting technique will be used for the subsequent cuts. A single-boom jumbo will be used to drill upper holes in the mineralized vein portion of the round, which will then be loaded and blasted. The mineralized material will be mucked before the waste portion of the round is drilled, loaded, and blasted. The waste will be left in place to create a new working floor for the next cut to be mined. Subsequent rounds will be mined in the same cycle while retreating towards the access. The cuts will be mined in an overhand fashion.



Note: Figure prepared by GMS, 2020.

Figure 16-15: Total Tonnes Mined by Mining Method

16.5 Mine Design

The planned mine is divided into six (6) main mining areas:

- Babicanora Sur: Babicanora SUR and Babicanora SUR HW stopes;
- Babicanora Main;
- Babicanora Central;
- Babicanora Norte: Babicanora Norte stopes
- Babi Vista: Babi Vista FW stopes; and,
- Las Chispas: Las Chispas, Giovanni, Luigi, Luigi FW Giovanni mini and William Tell stopes.

The mineralized zones will be accessed via three portals: the Santa Rosa, Babicanora Central and San Gotardo portals. The San Gotardo portal will be oriented toward the southeast from the north side of the William Tell Vein towards the Las Chispas Vein. Portals will open onto an underground decline. Mineralized material will be transported by truck from the underground workings to the surface via the same declines. While shaft access was evaluated, that option was discarded because based on shallow vein mineralization and favourable topography for declines to access veins.

All mining activities will be completed via a contractor. SilverCrest will supply fuel, electricity, explosives, explosives accessories, ground support consumables (e.g. rebars, wire mesh, etc.), construction consumables (e.g. steel), and services consumables (e.g. piping, rigid ventilation ducts, etc.). The contractor will supply adequate underground mining equipment for the different mining activities.

The lateral and vertical development dimensions are summarized in Table 16-5.

Table 16-5: Excavations Dimensions

Excavation Type	Width (m)	Height (m)
Sill Resue	3.0	3.0
Sill LH	3.0	3.0
Sill Access	4.5	4.5
Haulage Drift	4.5	4.5
Access	4.5	4.5
Raise Access	4.5	4.5
Muck Bay	4.5	4.5
Safety Bay	2.0	2.0
Sump	3.0	3.0
Main Ventilation Raise	3.0 m diameter	
Internal Ventilation Raise	3.0	3.0
Vent Drift	4.5	4.5
Electric Bay	4.5	4.5
Refuge	4.5	4.5
Escape Way	3.0 m diameter	
Pivot	4.0	4.0
Ramp	4.5	4.5

16.5.1 Level Spacing

The level spacing was selected based on the mining method chosen and the efficiency of long hole drilling. The level distance is generally 18 m.

A 15 m level spacing was used for the Babicanora Main to reduce the length of the long hole drilling, and thus reduce the deviation of the drilling. This allows for reduced dilution and better mineralized material recovery.

The level distance for the cut-and-fill and resuing was set at 18 m to reduce the total development required.

16.5.2 Main Access Drifts and Ramps

Declines will provide access to mining levels for all zones. Depending on the mining methods selected, drifts may be required to access themineralized zones. Excavation dimensions are set at 4.5 m x 4.5 m. Rigid ventilation ducts, freshwater, compressed air, and dewatering pipes and power cable will be installed in declines and main access drifts. The ramp incline is set at -15% grade. Loading and hauling will be carried out to the nearest muck bay that will be located at a maximum distance of 200 m. Dimensions will allow the use of 10 t scoops, 30 t truck and two-boom jumbos. This equipment will be sufficient to achieve the daily development productivity target.

The priority of the main horizontal development will be to access the fresh air ventilation raise as soon as possible. The principal objective is to establish a first ventilation circuit and thus reduce the need for ventilation ducts. The second objective of the declines and main access drifts is to reach the bottom of the first mining zone so that it may be brought into production as quickly as possible. Figure 16-16 illustrates the proposed decline and main access drift dimensions.

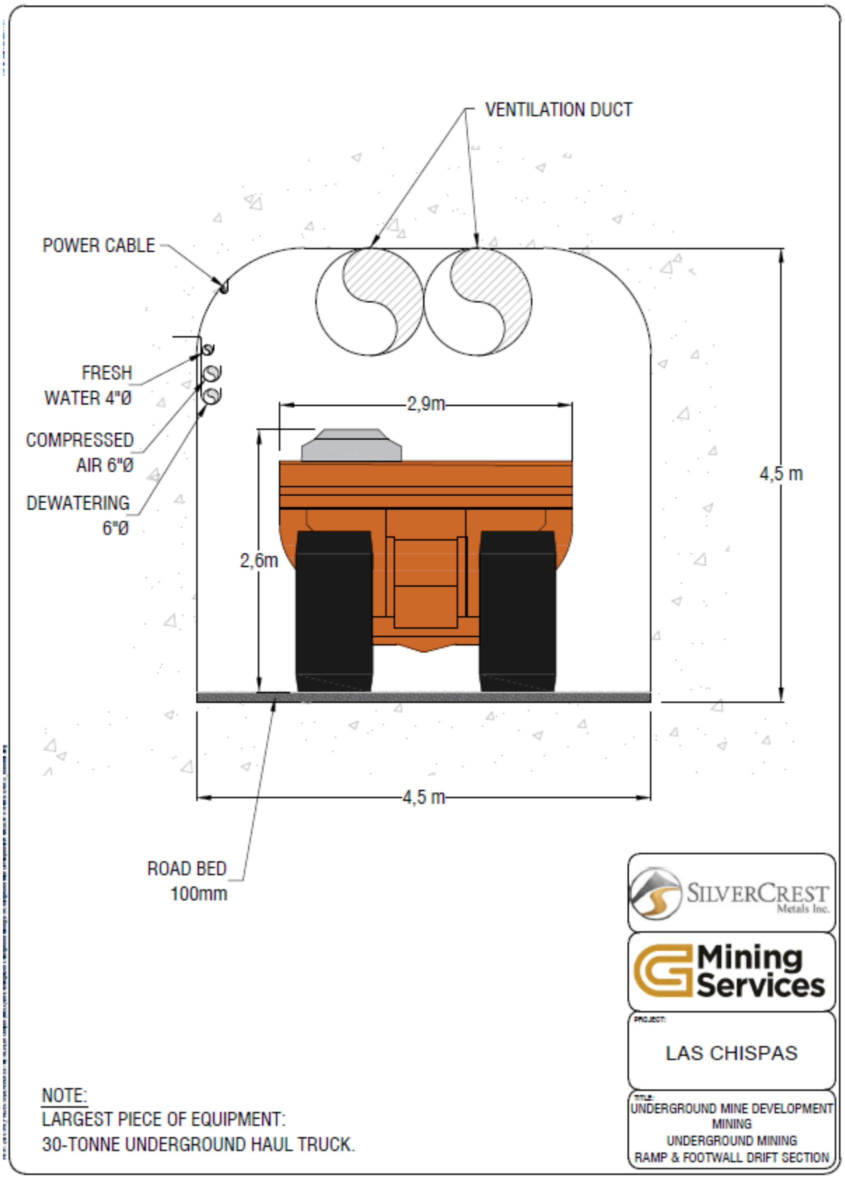
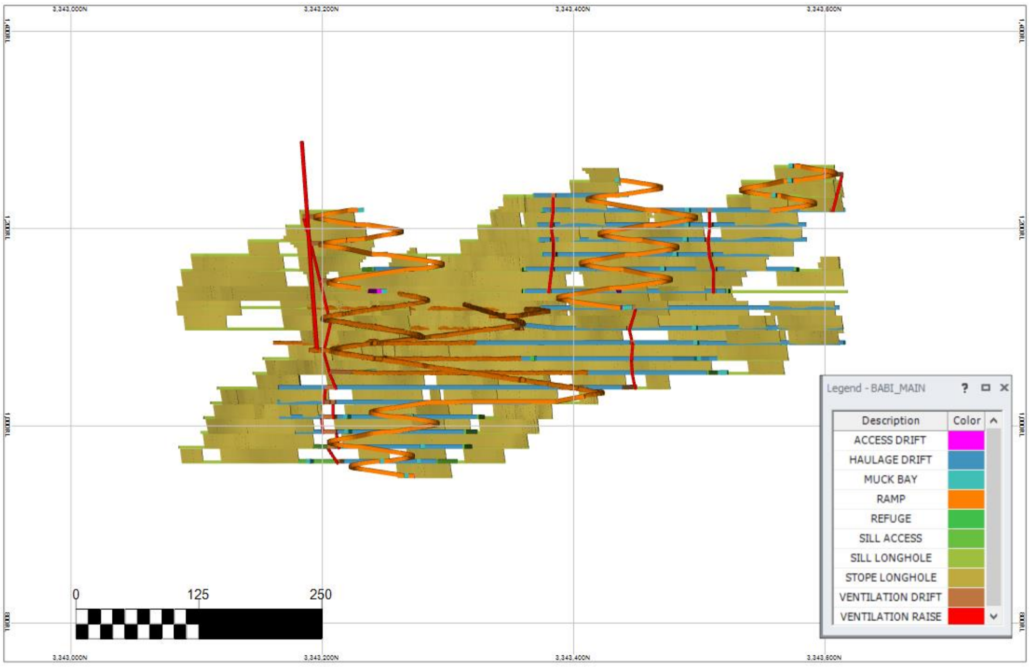


Figure 16-16: Decline and Main Access Drift Development Profile

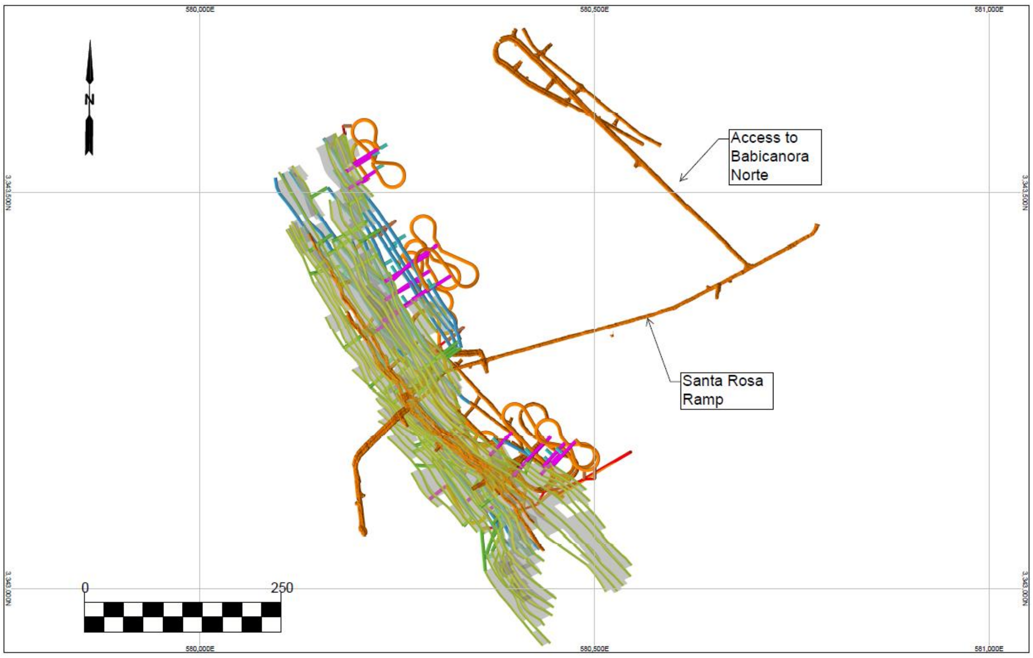
16.5.2.1 Babicanora Main

Babicanora Main will be the first and the highest tonnage area mined and is accessed via the Santa Rosa decline. A single decline with has been driven at 15% dip from surface down to level 1,096 m where it separates into a decline and a ramp. The ramp will serve as the access for the upper part of the vein, and the decline will allow mining of the lower part of the mineralized zone. Other declines will provide access to the lowest and upper northwest parts of the Babicanora Main deposit. Re-muck bays and safety bays will be excavated in the declines and used throughout the LOM. Declines will provide access to the mining levels every 15 vertical metres. Each production level will be connected to the main decline via a level access drift. A footwall drift will be excavated to link the deposit with the decline on almost all levels. Figure 16-17 and Figure 16-18 illustrate the main access and decline proposed for Babicanora Main.



Note: Figure prepared by GMS, 2020.

Figure 16-17: Main Access Drift and Decline for Babicanora Main (looking southwest)

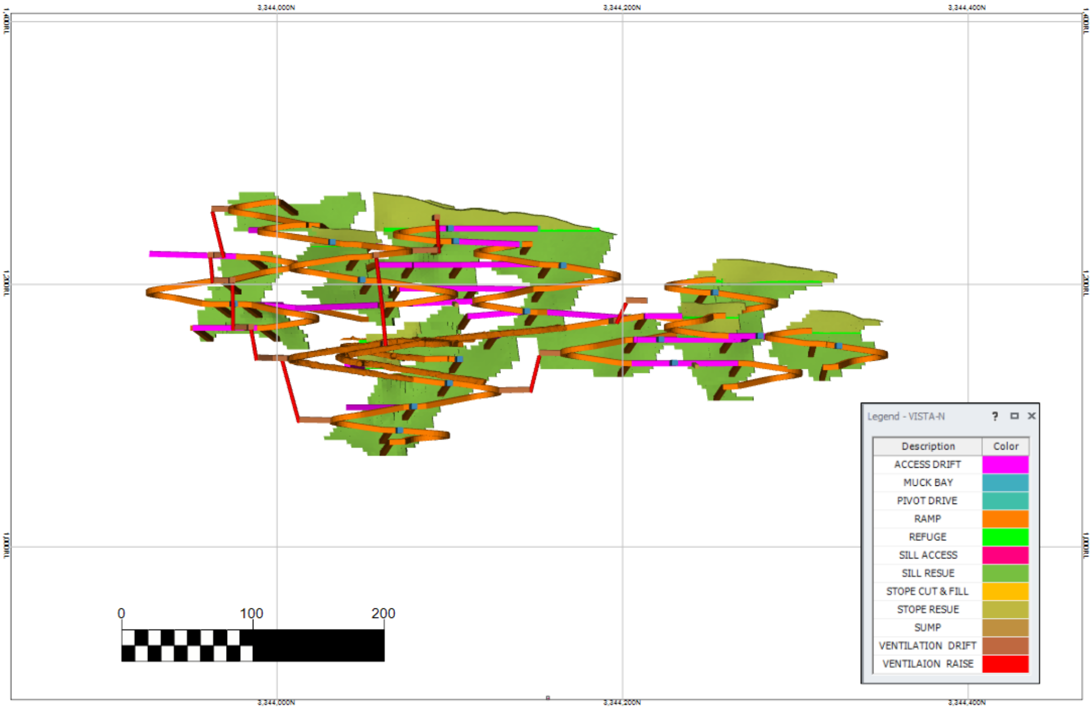


Note: Figure prepared by GMS, 2020.

Figure 16-18: Main Access Drift and Decline for the Babicanora Main – Plan View

16.5.2.2 Babicanora Central

Babicanora Central has an existing historical decline that provides access to the mineralized zone. This ramp serves as access to Level 1156. This main decline will split off, providing access to the upper and lower levels. Isolated groups of stopes will have their own decline connected to the main ramp. Declines will provide access to the mining level every 18 vertical metres. Babicanora Central will be connected to Babicanora Main via a drift at 1,192 m elevation. Figure 16-19 and Figure 16-20 illustrate the main access and decline for Babicanora Central.



Note: Figure prepared by GMS, 2020.

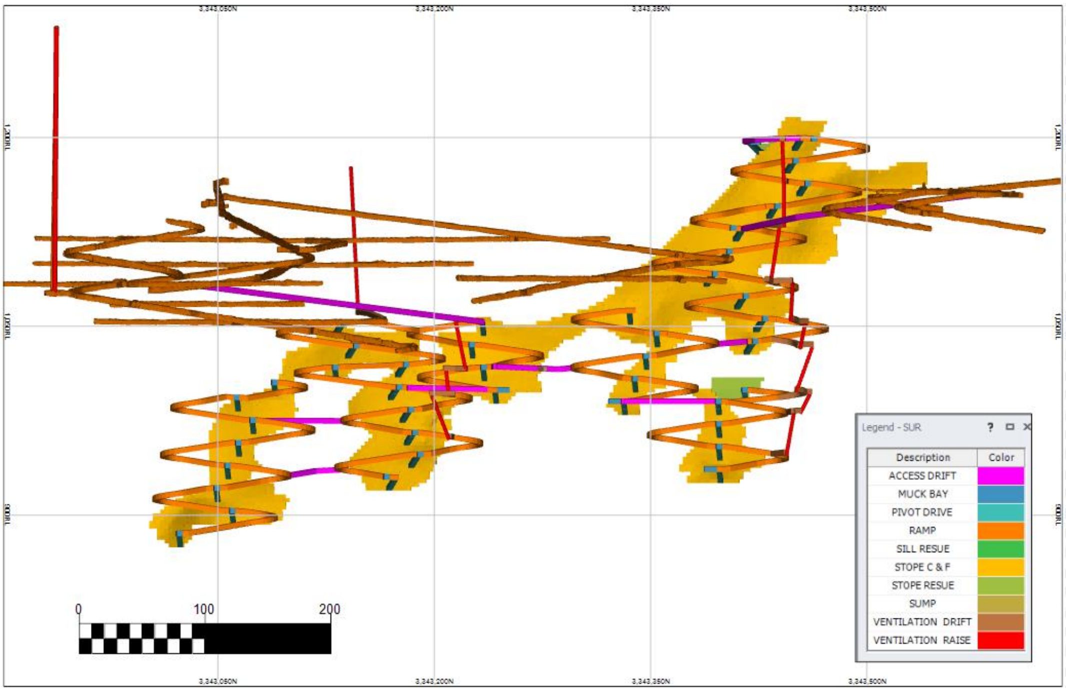
Figure 16-19: Main Access Drift and Decline for Babicanora Central (looking southwest)

Note: Figure prepared by GMS, 2020.

Figure 16-20: Main Access and Decline for Babicanora Central – Plan View

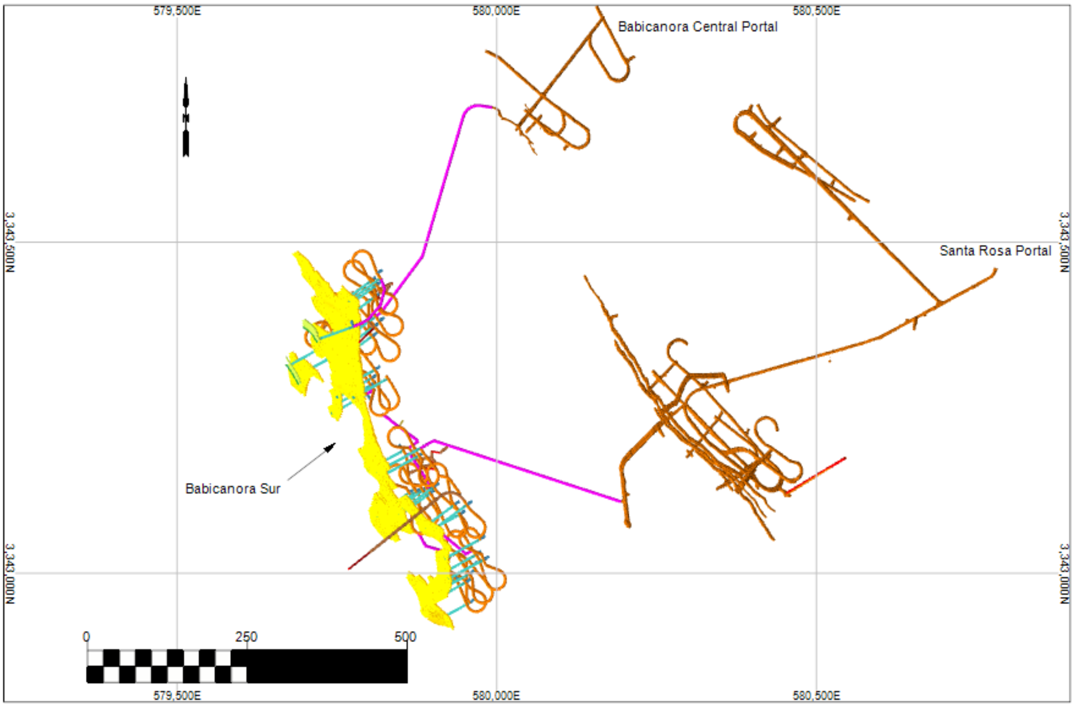
16.5.2.3 Babicanora Sur

The southwest part of the Babicanora Sur deposit will be accessed via a decline connected to the Santa Rosa decline extension. A decline will drive towards Babicanora Sur from the Santa Rosa extension to Level 1042. The rest of the deposit will be accessed via a decline from the existing northwest drift at Level 1156 of the Babicanora Central area to Level 1350 of Babicanora Sur. These main declines will split off, providing access to the upper and lower levels. Isolated groups of stopes will have their own decline connected to the main ramp. Declines will provide access to the mining level every 18 vertical metres. Figure 16-21 and Figure 16-22 show the main access and decline for Babicanora Sur.



Note: Figure prepared by GMS, 2020.

Figure 16-21: Main Access Drift and Decline for Babicanora Sur (looking southwest)



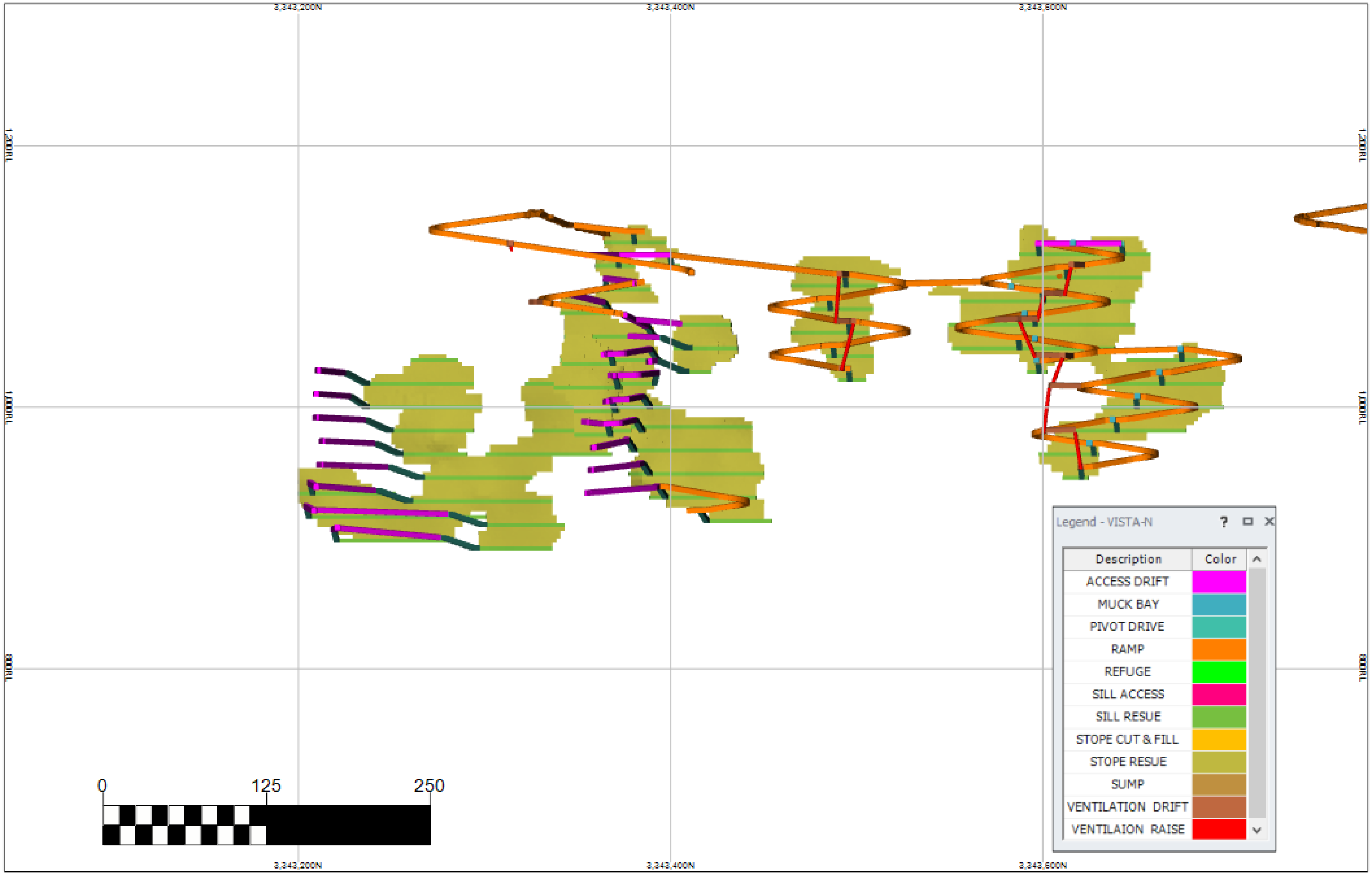
Note: Figure prepared by GMS, 2020.

Figure 16-22: Main Access Drift and Decline for Babicanora Sur – Plan View (historical workings shown in brown)

16.5.2.4 Babi Vista and Babicanora Norte

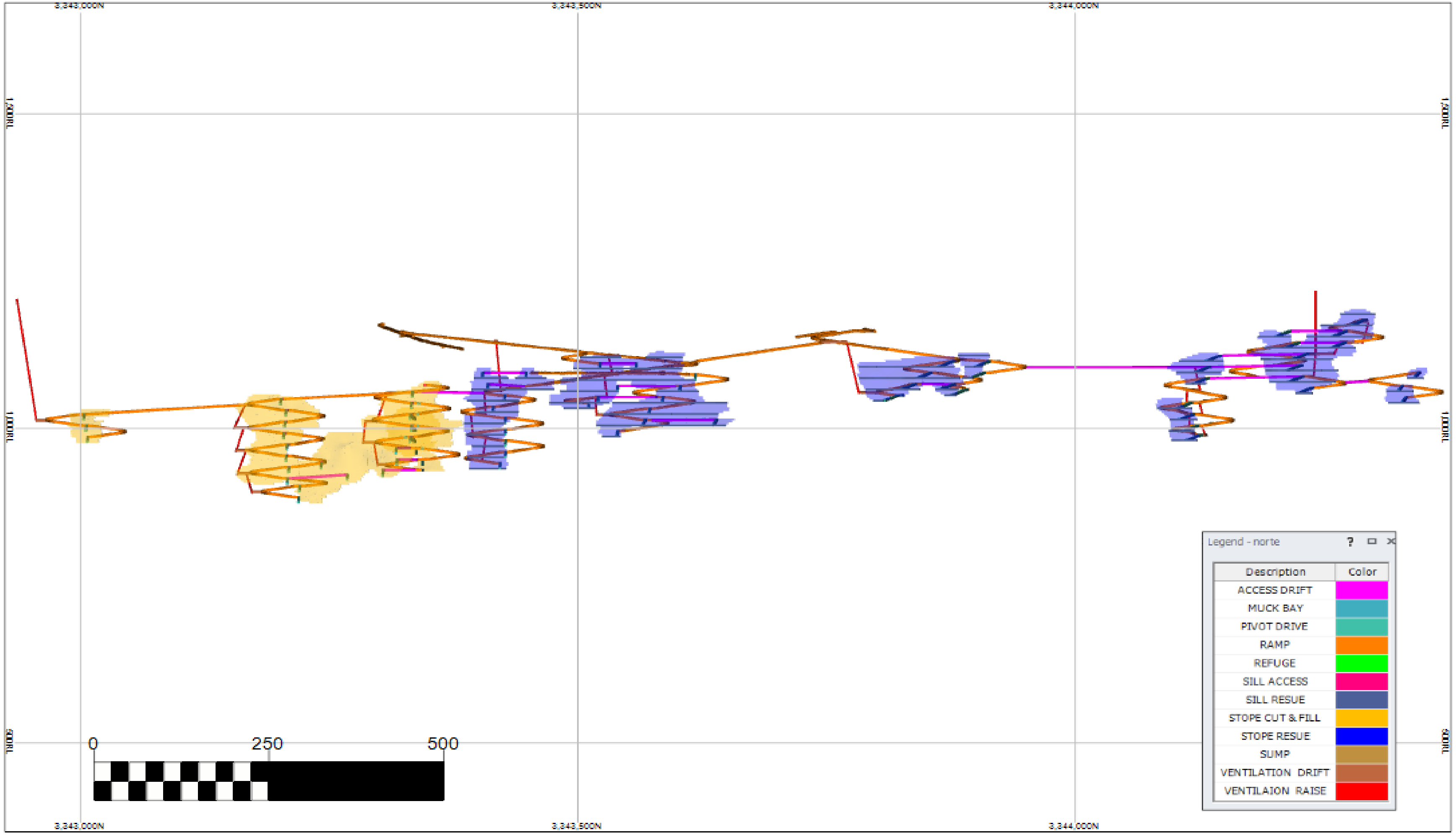
Babi Vista will be accessed via the Santa Rosa decline. The Babicanora Norte Ramp will be excavated perpendicular to the Santa Rosa decline. One part of the Babicanora Norte deposit will be accessed via the Babicanora Norte ramp and the other sectors will be accessed via an extension to be constructed perpendicular to the Babicanora Central ramp. The development for Babicanora Main and Babicanora Norte has limited restrictions due to isolated groups of stopes. Each group will have its own ramp. All declines will be connected to each other. Some drifts are designed to backfill the existing open stope left from historical mining before mining around and near that open stope. Declines will provide access to the mining level every 18 vertical metres.

Figure 16-23, Figure 16-24, and Figure 16-25 show the main access and decline.



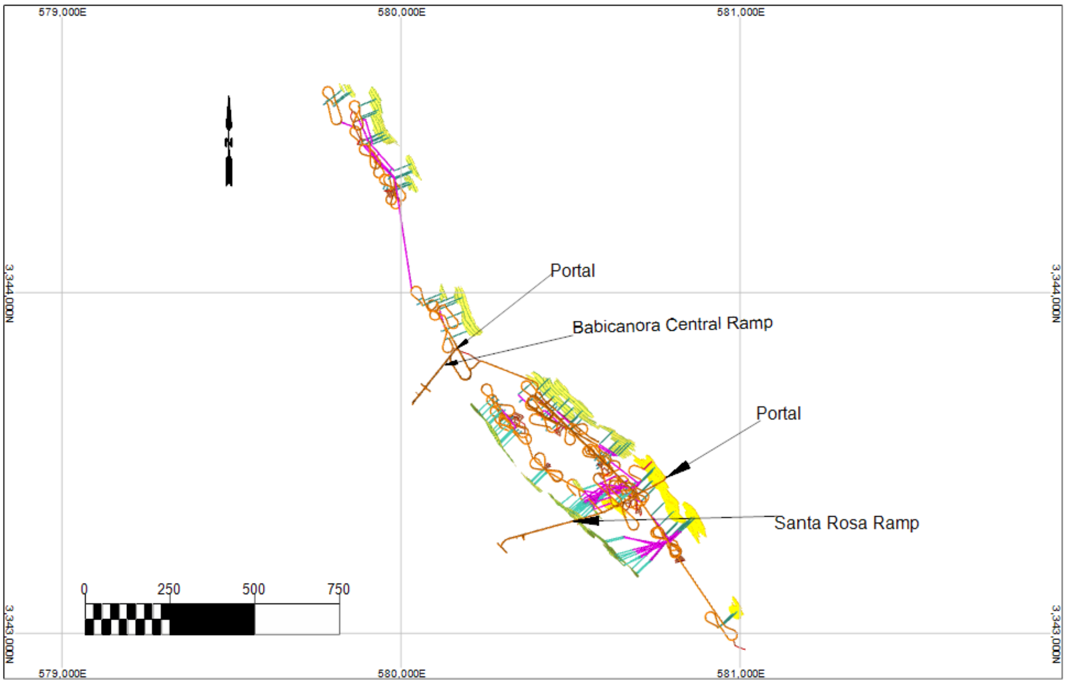
Note: Figure prepared by GMS, 2020.

Figure 16-23: Main Access Drift and Decline for Babi Vista (looking southwest)



Note: Figure prepared by GMS, 2020.

Figure 16-24: Main Access Drift and Decline for Babicanora Norte (looking southwest)

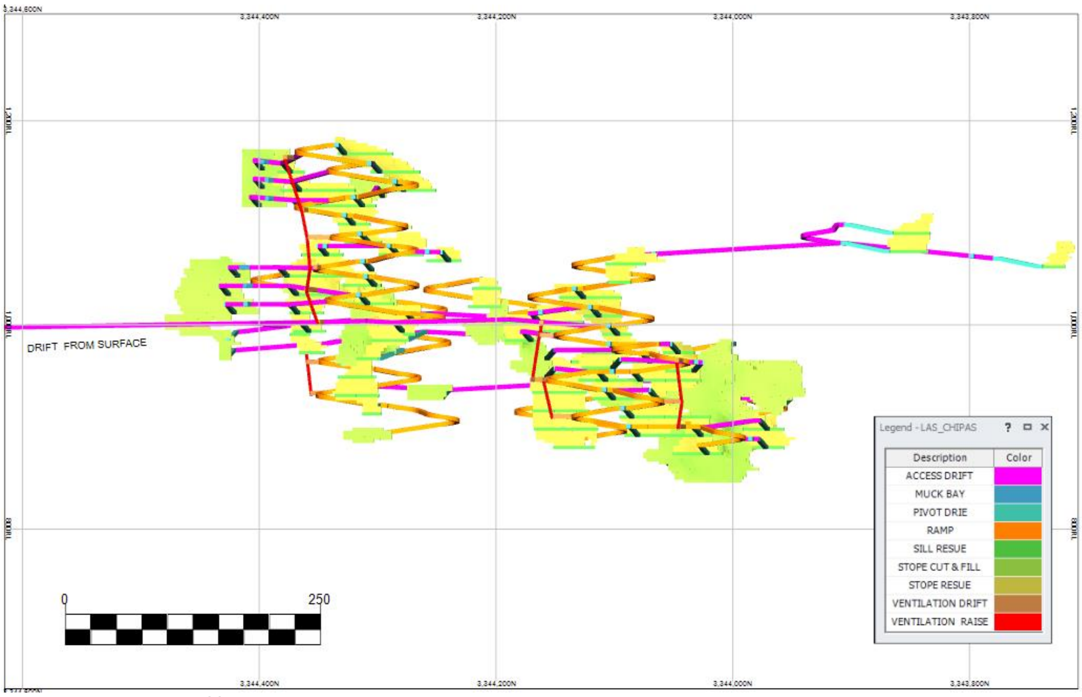


Note: Figure prepared by GMS, 2020.

Figure 16-25: Main Access Drift and Decline for Babicanora Norte – Plan View

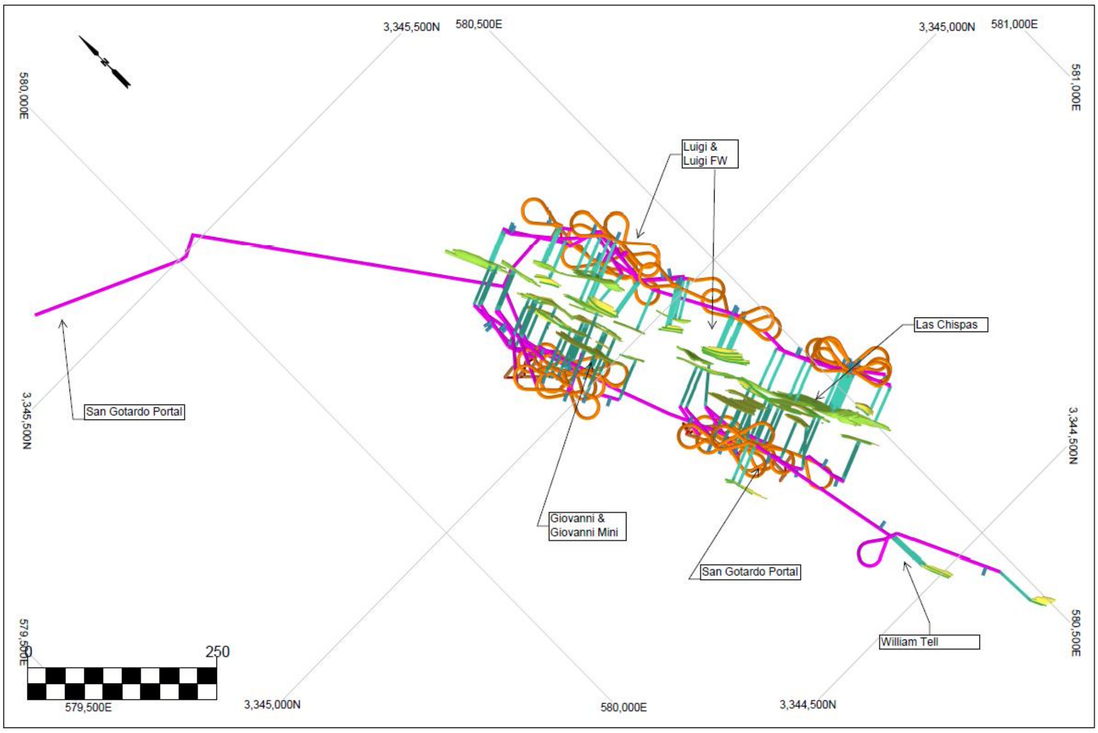
16.5.2.5 Las Chispas

The San Gotardo portal will provide access from the surface. This drift heads southeast from the north side of the William Tell zone, intercepts the William Tell Vein, and then continues towards the Las Chispas Vein. This drift will serve as access to the veins. Declines will provide access to the mining level every 18 vertical metres. Configuration of the mine development and access is shown in Figure 16-26 and Figure 16-27.



Note: Figure prepared by GMS, 2020.

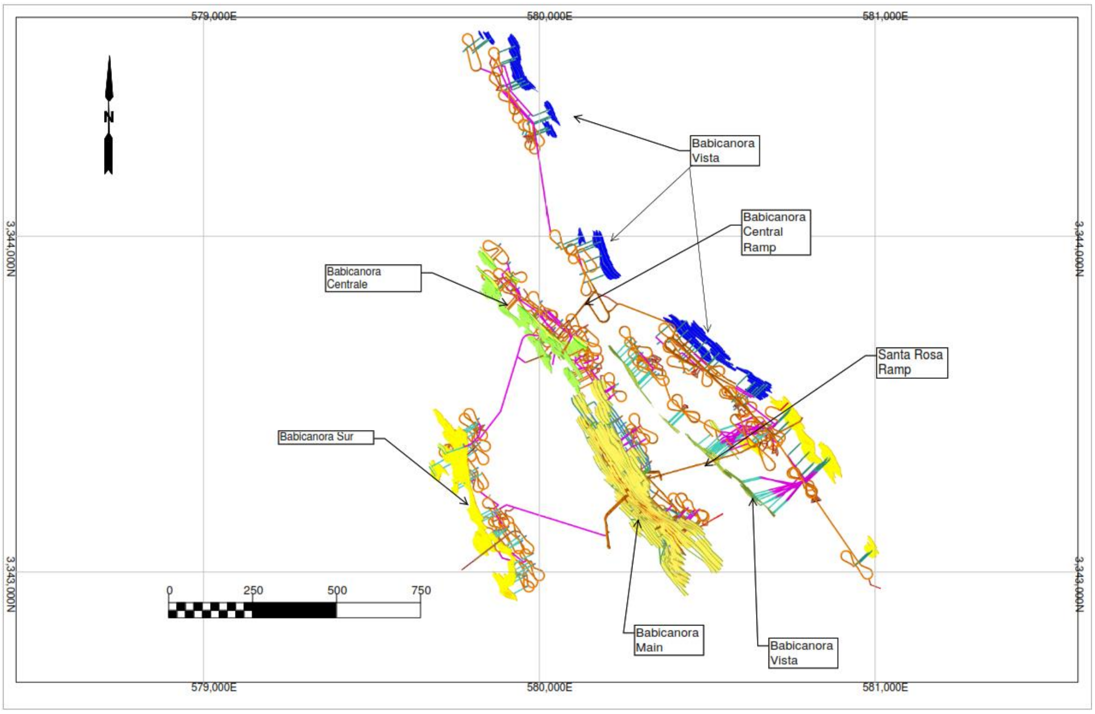
Figure 16-26: Main Access and Decline for Las Chispas (looking northeast)



Note: Figure prepared by GMS, 2020.

Figure 16-27: Main Access Drift and Decline for Las Chispas – Plan View.

Figure 16-28 is a plan view of the combined Babicanora Main, Central, Norte, Sur and Vista areas.



Note: Figure prepared by GMS, 2020.

Figure 16-28: Babicanora Main, Central, Norte, Sur and Vista Combined – Plan View

16.5.3 Stope Access

16.5.3.1 Pivot Drives

Pivot drives will be excavated for zones mined using cut-and-fill methods (breasting, uppers and resue). Only Babicanora Main, which will be mined by long-hole method, will not have pivot drives. The pivot drive dimensions are set at 4 m x 4 m. Fan, freshwater and compressed air pipes will be installed. The pivot drive incline will range from -18% to +18% grade. Loading and hauling will be carried out to the nearest muck bay, which will be located at a maximum distance of 60 m. The first pass advance will be conventional development and will be followed by five backslash passes. Dimensions will allow the use of 10 t scoops and two-boom jumbos. Figure 16-29 shows the pivot drive development profile and Figure 16-30 is a section view of a pivot drive.

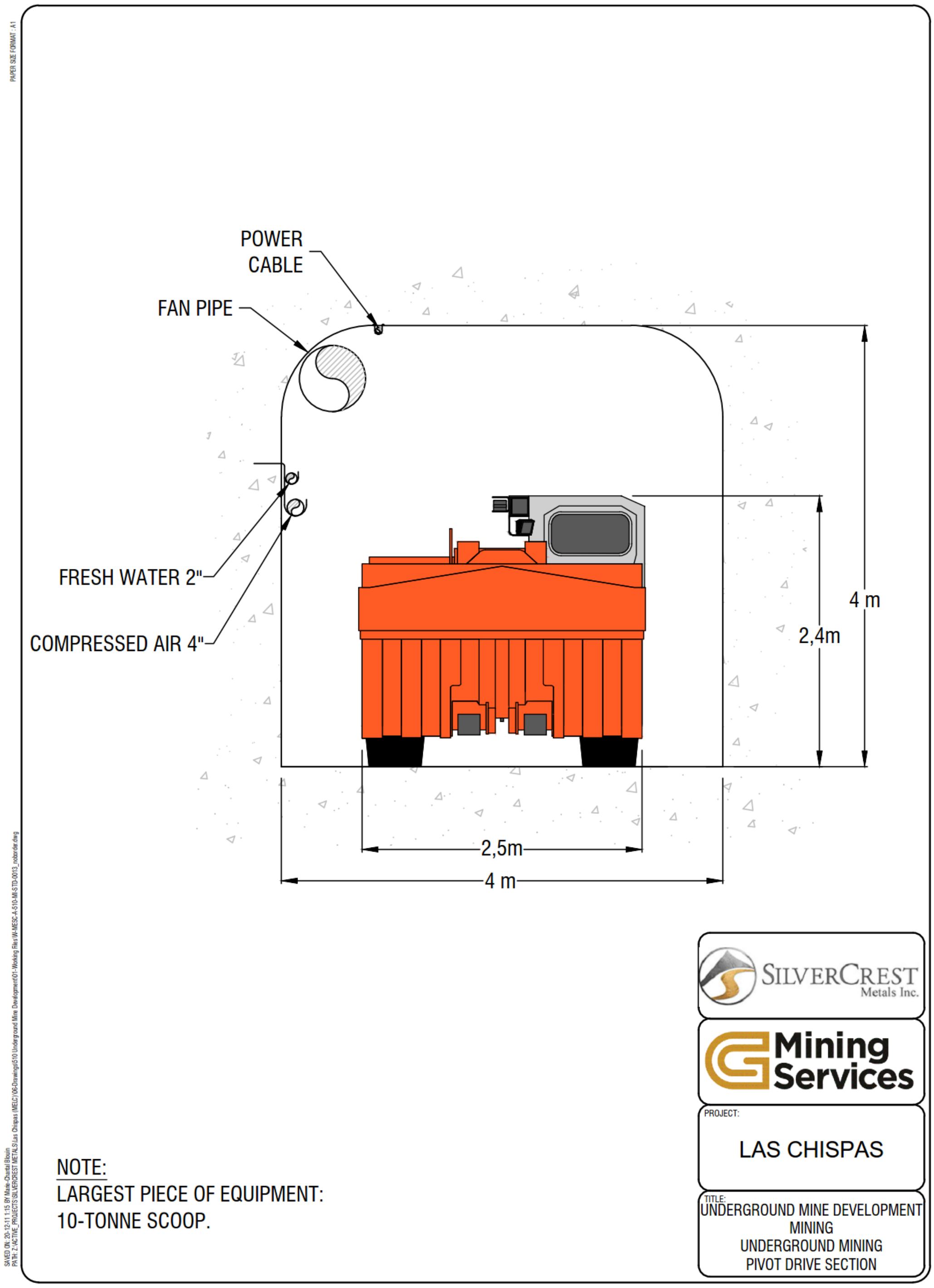


Figure 16-29: Development Profile and Pivot Drive

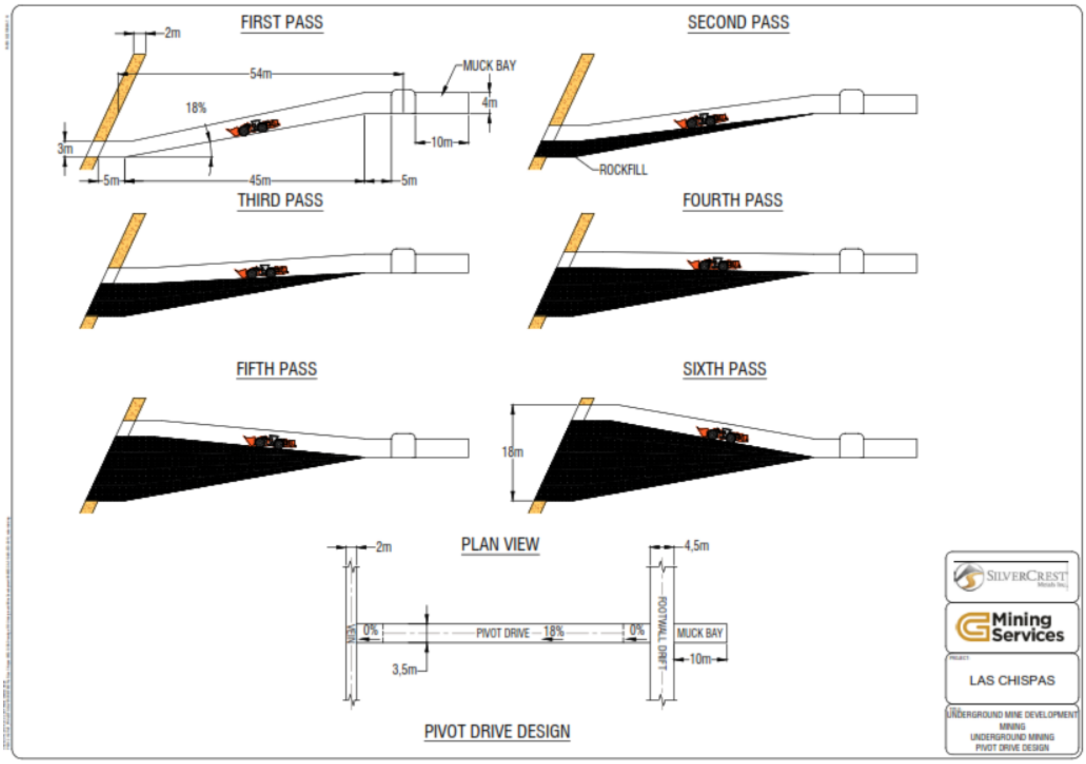


Figure 16-30: Pivot Drive Section View

16.5.3.2 Long-hole Sills Development

Long-hole sills will be excavated in the Babicanora Main mineralized zone. Dimensions are set at 3 m x 3 m. The long-hole sill will be developed following geology. Services such as fan pipes, fresh water etc. will be located directly in the long-hole sills. Long-hole sills will be used to drill and blast stopes and to mine muck. Loading and hauling will be carried out to the nearest muck bay that will be located at a maximum distance of 250 m. Dimensions will allow the use of 3 t scoops and single-boom jumbos. Figure 16-31 is a profile showing long-hole sill development.

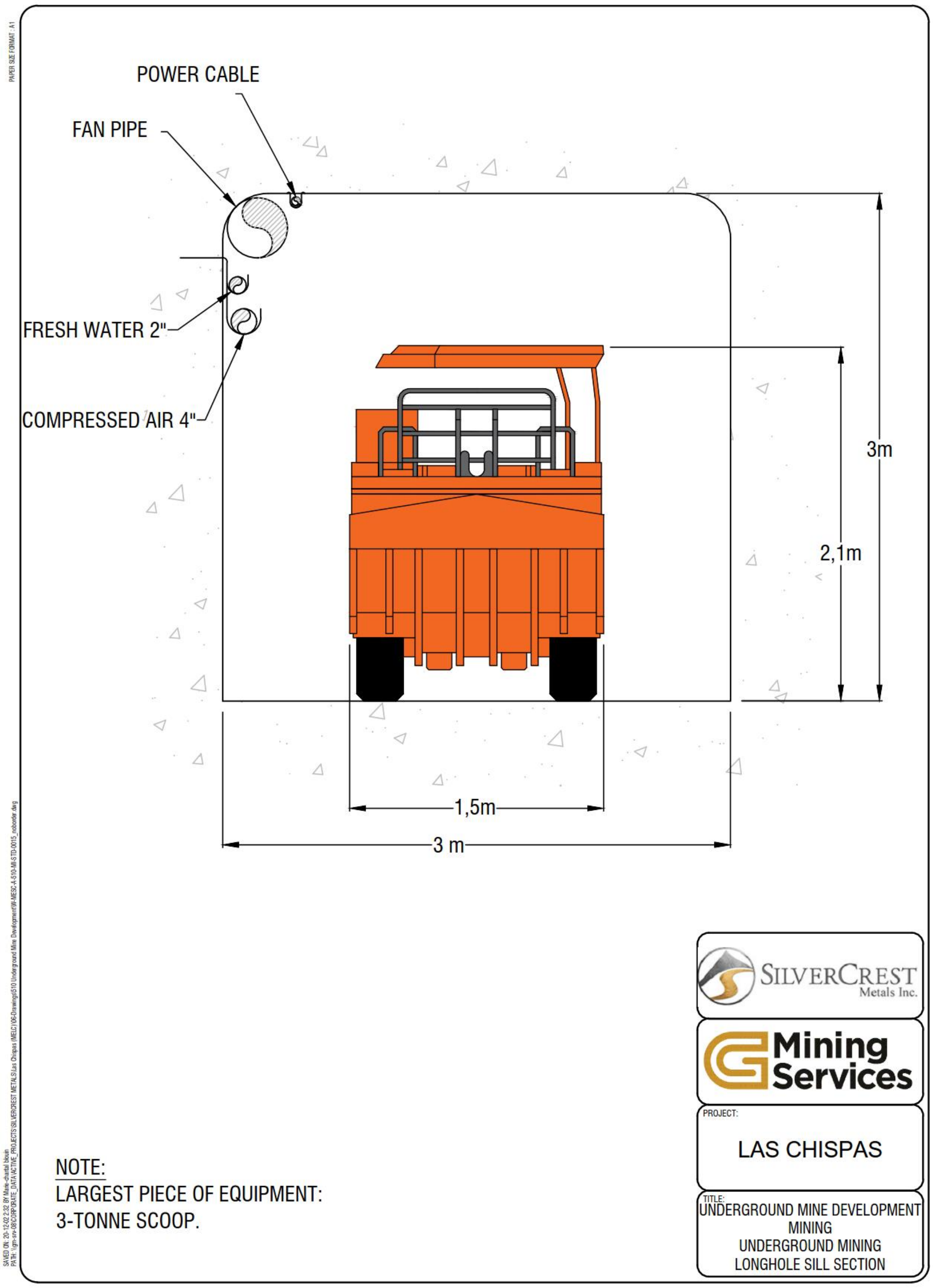


Figure 16-31: Long-hole Sills Development Profile

16.5.3.3 Cut-and-Fill Sills Development

Development of cut-and-fill sills will include sills developed by breasting, uppers and resuing. Dimensions are set at a minimum of 1.5 m L x 3 m H for upper and breasting sills and 3 m L x 3 m H for resuing sills. Services such as fan pipe, fresh water etc. will be located directly in the cut-and-fill sills. Loading and hauling will be carried out to the nearest muck bay, which will be located at a maximum distance of 250 m. Dimensions will allow for the use of 1.5 t or 3 t scoops, single-boom or two boom jumbos, or jacklegs, depending on the dimensions. A development profile is included as Figure 16-32.

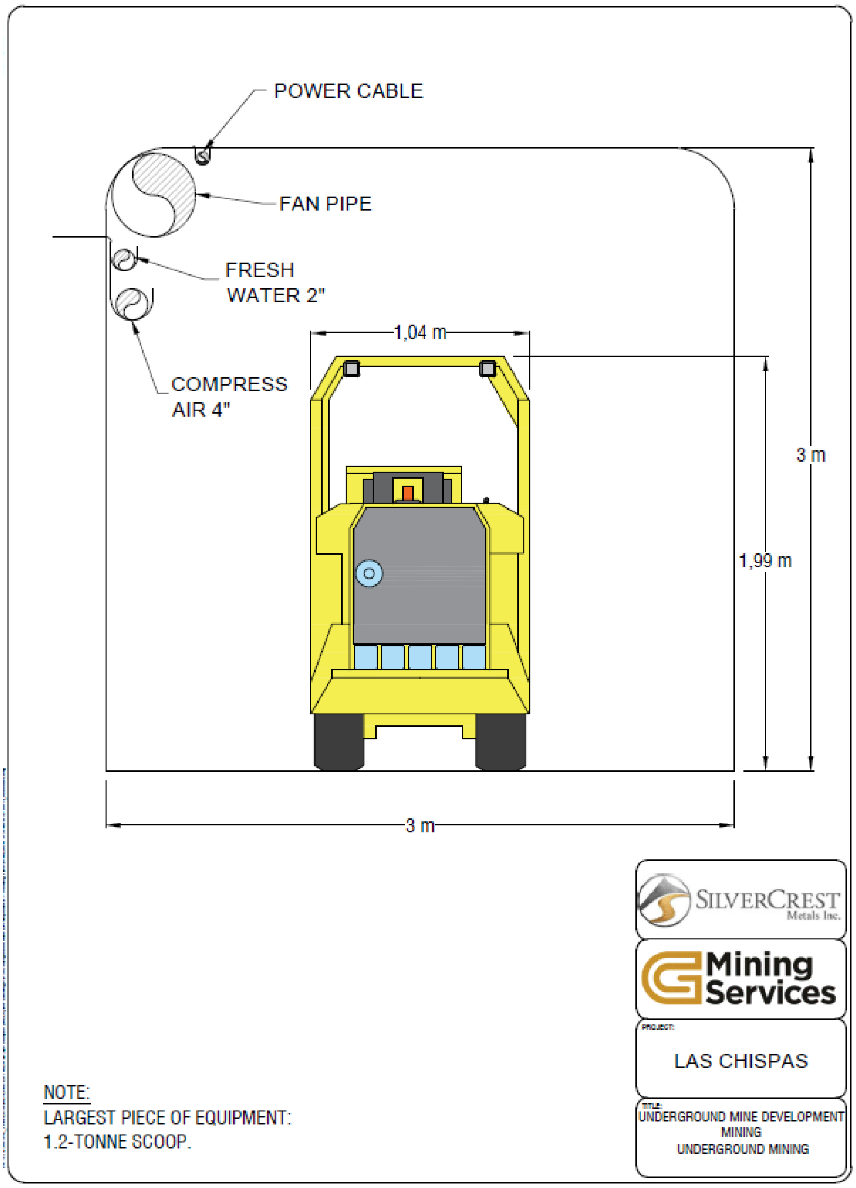


Figure 16-32: Resue Sills Development Profile

Table 16-6 summarizes the total mine lateral development metres per type of excavation and mining area.

Table 16-6: Total Development Metres

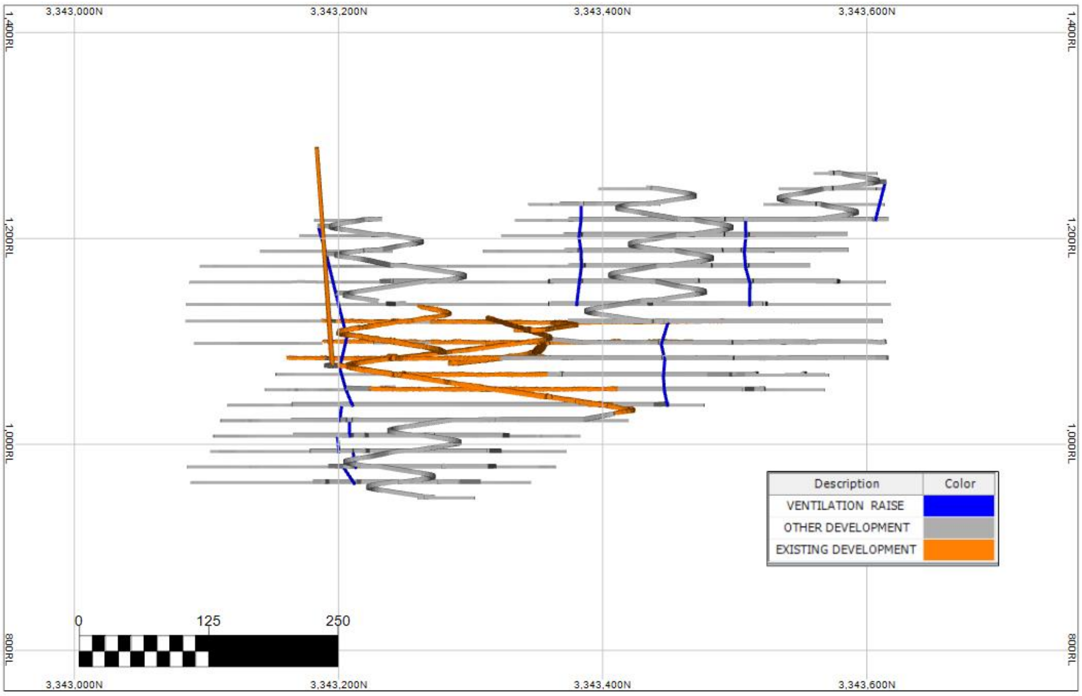
Excavation Type	Babicanora Main (m)	Babicanora Central (m)	Babicanora Sur (m)	Babi Vista (m)	Babicanora Norte (m)	Las Chispas (m)	TOTAL (m)
Sill	9,798	39	–	3,781	3,877	231	17,726
Pivot drive	–	2,307	–	2,760	4,330	5,097	14,494
Access	1,846	1,472	3,381	2,486	2,161	4,451	15,797
Haulage drift	2,111	–	–	–	–	–	2,111
Vent drift	515	386	627	370	1,167	–	3,064
Muck bay	320	180	380	120	396	732	2,128
Other	212	278	382	246	1,135	–	2,252
Decline	2,479	3,047	4,435	2,605	9,855	5,780	28,201
TOTAL	**17,282**	**7,708**	**9,204**	**12,367**	**22,922**	**16,291**	**85,773**

16.5.4 Ventilation Raises

Several raises will be required for the different mine zones. Raises for all zones will be 3 m in diameter for main raises (drilled by raise boring) and 3 m x 3 m for internal raises. Some ventilation raises will also serve as escape ways. A prefabricated modular manway system will be installed in the emergency exits. This system has been proven to save time and costs to establish the manway. Ventilation raises between the levels will be excavated by the drop raise method.

16.5.4.1 Babicanora Main

A single raise was excavated from the surface to the Santa Rosa decline and will be connected to an internal raise. Three other raises will be excavated as shown in Figure 16-33.

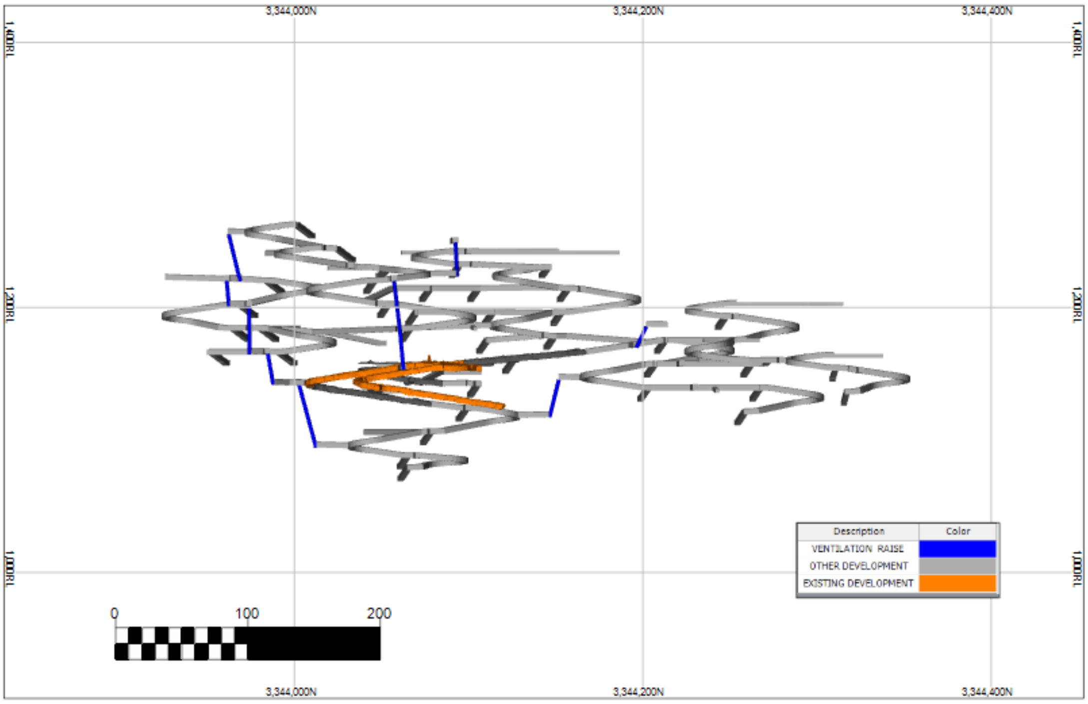


Note: Figure prepared by GMS, 2020.

Figure 16-33: Ventilation Raise – Babicanora Main (looking southwest)

16.5.4.2 Babicanora Central

A single raise will be excavated from surface and will be connected to internal drop raises (Figure 16-34).

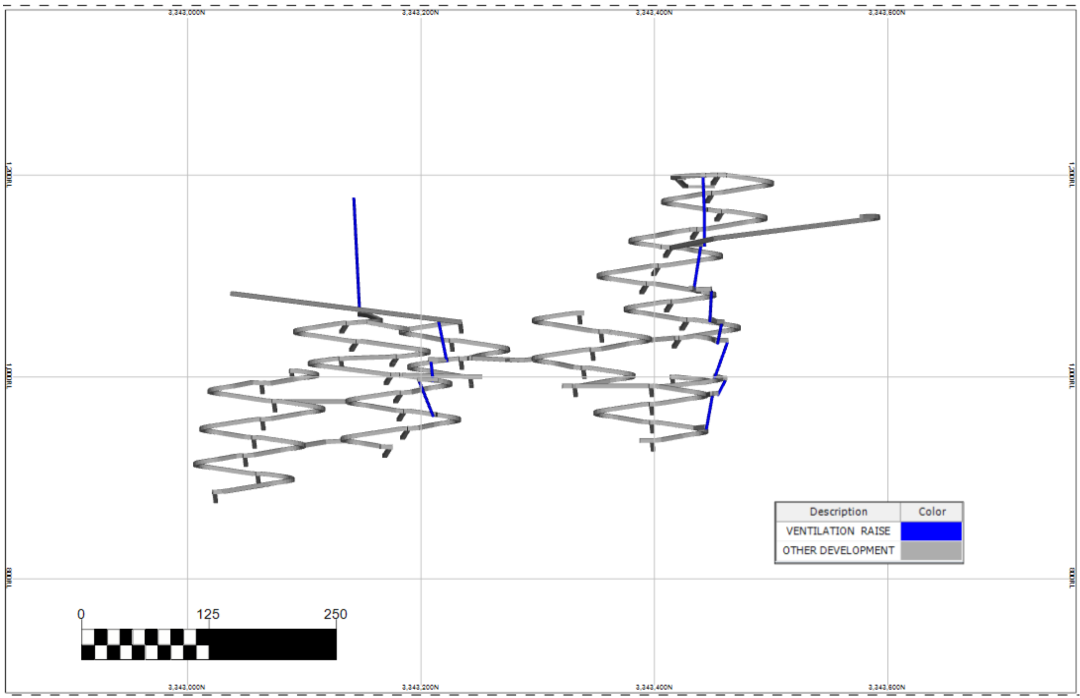


Note: Figure prepared by GMS, 2020.

Figure 16-34: Ventilation Raise – Babicanora Central (looking southwest)

16.5.4.3 Babicanora Sur

One raise will be excavated from surface and will be connected to internal drop raises, see Figure 16-35.

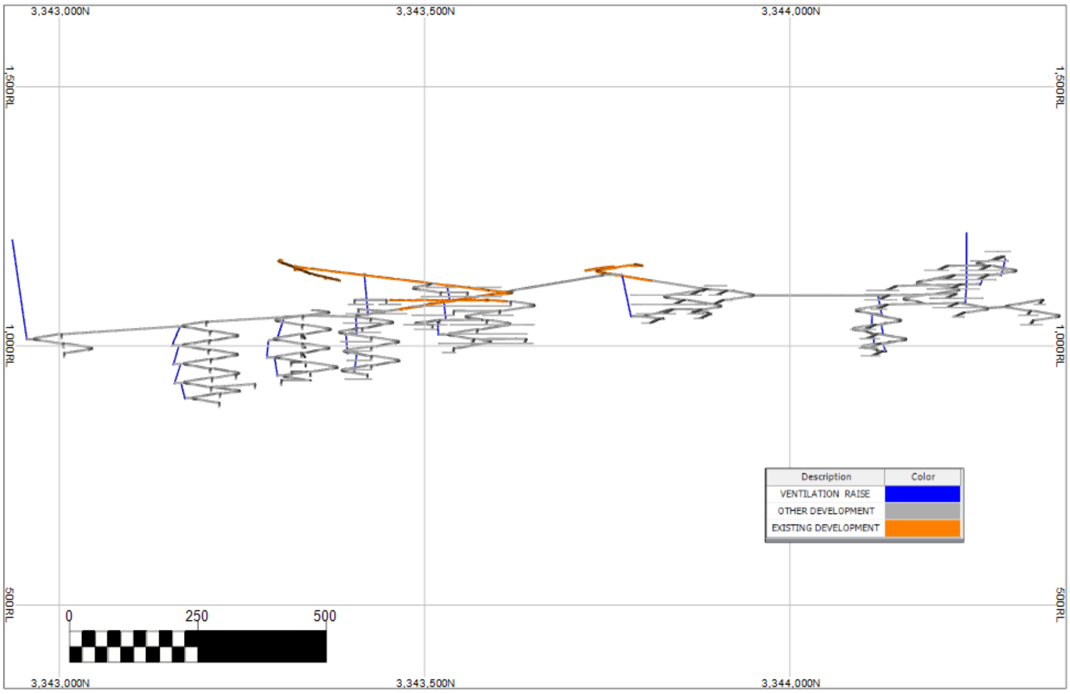


Note: Figure prepared by GMS, 2020.

Figure 16-35: Ventilation Raise – Babicanora Sur (looking southwest)

16.5.4.4 Babicanora Norte

Two raises will be excavated from surface. Two other raises will be excavated from Santa Rosa Ramp to the first accessible level. Some internal raise raises will also be excavated. Figure 16-36 is a schematic showing the proposed layout.

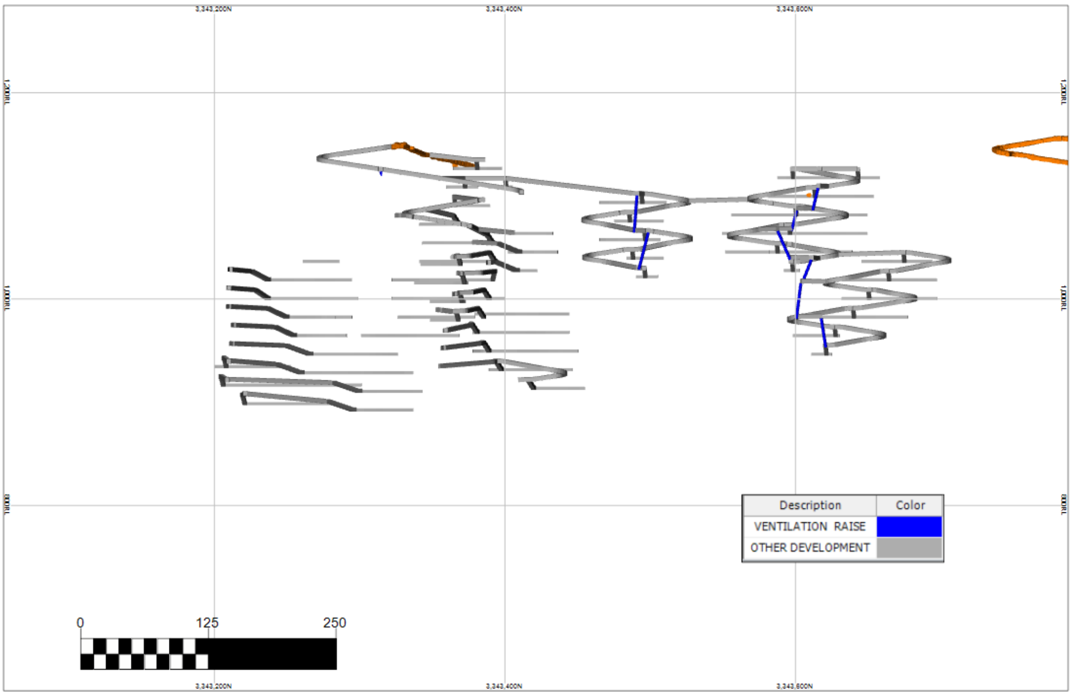


Note: Figure prepared by GMS, 2020.

Figure 16-36: Ventilation Raise – Babicanora Norte (looking southwest)

16.5.4.5 Babi Vista

A raise will be connected to the Santa Rosa ramp. Other internal raises will be excavated by drop raise methods. Figure 16-37 illustrates the planned ventilation raise configuration.

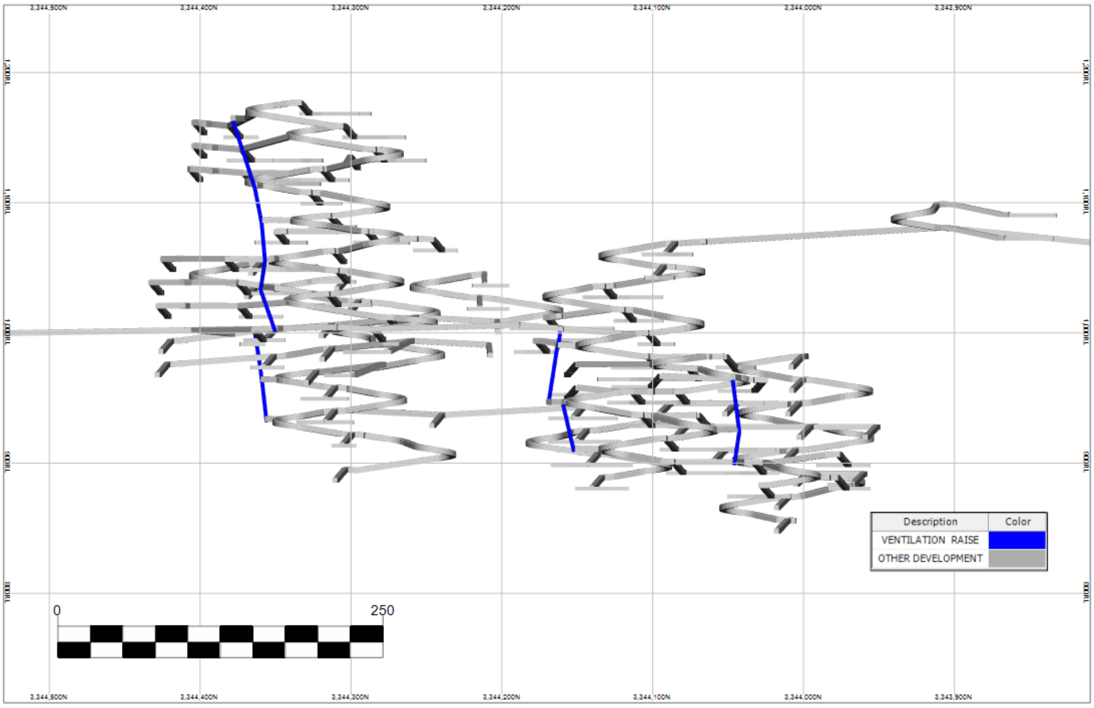


Note: Figure prepared by GMS, 2020.

Figure 16-37: Ventilation Raise – Babi Vista (looking southwest)

16.5.4.6 Las Chispas

A single raise will be excavated from surface and will be connected to internal drop raises. Figure 16-38 is a schematic showing the proposed raise configuration.



Note: Figure prepared by GMS, 2020.

Figure 16-38: Ventilation Raise – Las Chispas (looking northeast)

16.5.5 Mining Method

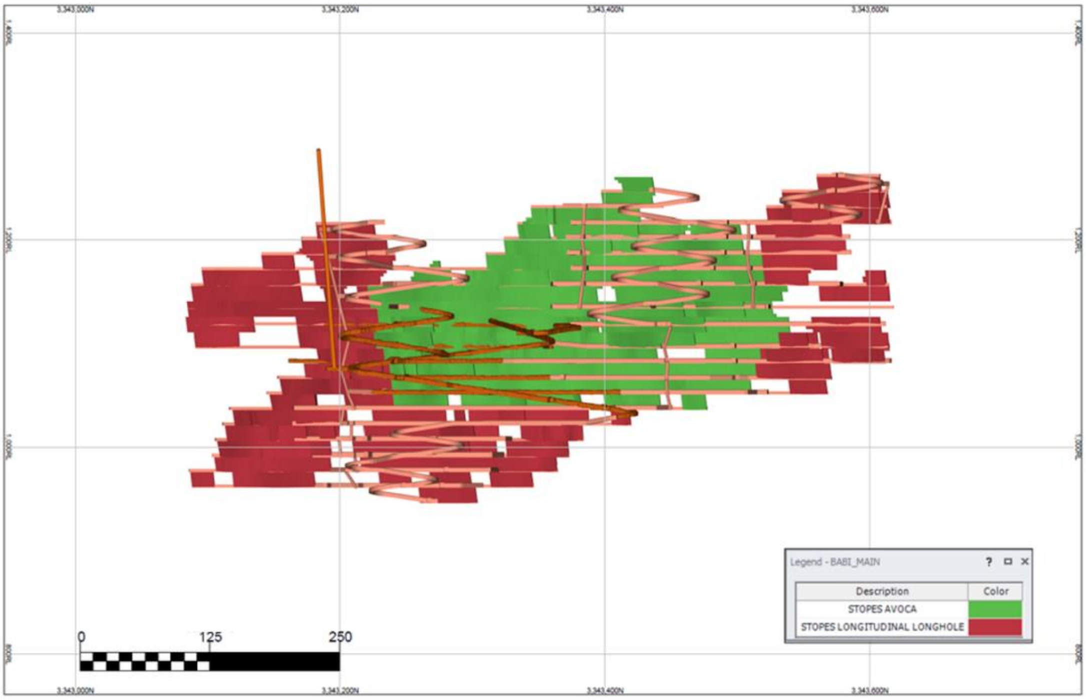
16.5.5.1 Babicanora Main

Two mining methods were selected for Babicanora Main. The preferred method, Avoca, with around 60% of the production tonnage, and longitudinal long-hole retreat stoping (LRS), which will be used wherever the footwall drift required for the Avoca method cannot be placed or if the orebody displays major discontinuities. Figure 16-39 shows where each mining method will be used.

The production levels will be advanced at 15 m intervals, and the mineralized drifts above and below the stoping area will be developed at 3 m x 3 m. The additional haulage drifts and sill accesses required for the Avoca method will be developed at 4.5 m x 4.5 m.

The typical long-hole block will have a maximum 12 m strike length and be 15 m high.

To gain flexibility and increase productivity, two mining horizons were selected; from level 946 to level 1036 and from level 1036 up. The main sill pillar will be between level 1036 level and level 1051. A high binder will be required for this pillar.



Note: Figure prepared by GMS, 2020.

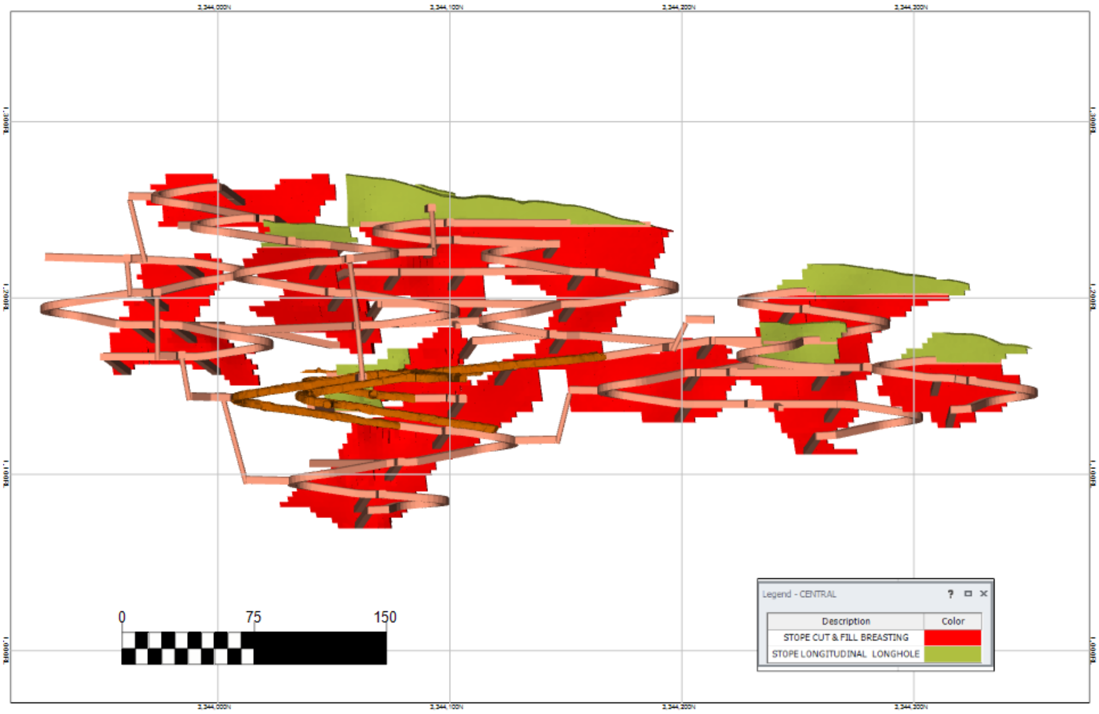
Figure 16-39: Mining Method – Babicanora Main (looking southwest)

16.5.5.2 Babicanora Central

Longitudinal long-hole and cut-and-fill breasting mining methods will be used. A long-hole mining method will be used for the stopes near the surface where the development of pivot drives is not practicable. Figure 16-40 shows where the different mining methods will be used.

For stopes that will be mined using the cut-and-fill method, a pivot drive (4 m H x 4 m W) with -18% grade will be excavated. An initial sill (3 m H x 3 m W) needs to be developed. Once the sill reaches the extremity of the mining area, rockfill will be placed, allowing a 0.3 m gap to the sill back. The next cuts will be driven by slashing the breast holes rounds into this gap to enhance productivity. Six cuts will be completed with the same pivot drive. The height of the cut-and-fill stopes, including all six cuts, will be 18 m.

Some drifts are designed to backfill the existing open stope left from historical mining before mining around and near it.



Note: Figure prepared by GMS, 2020.

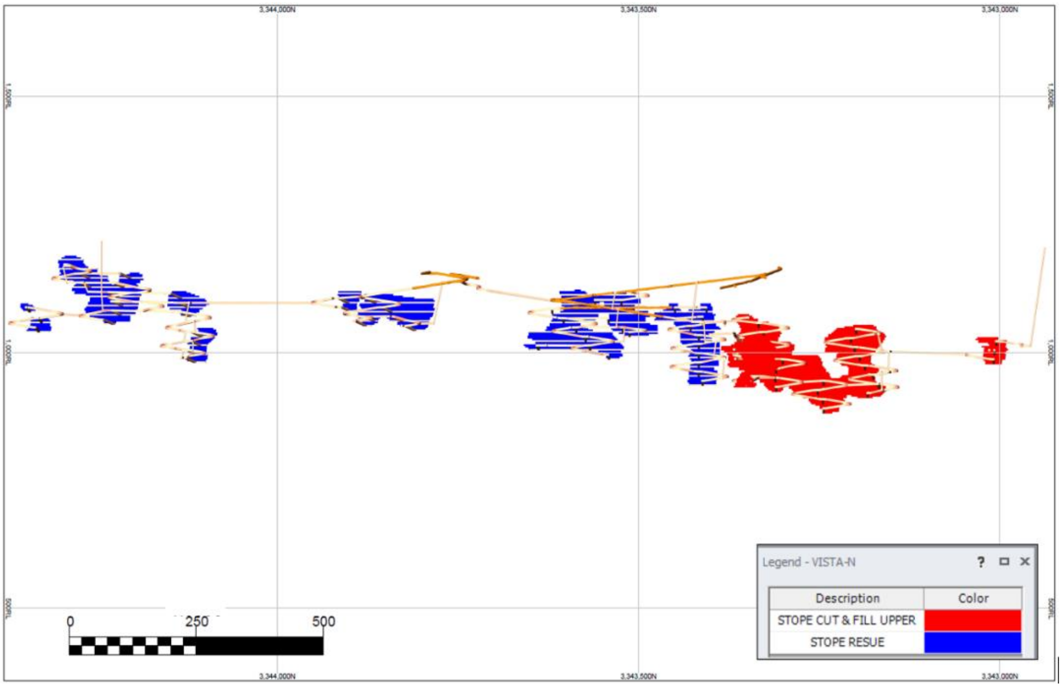
Figure 16-40: Mining Method – Babicanora Central (looking southwest)

16.5.5.3 Babi Vista and Babicanora Norte, Babicanora Sur and Las Chispas

Cut-and-fill upper and resuing mining methods will be used for these zones. Resuing is used where vein thickness is <1 m, otherwise the cut and fill mining method is used. Both methods are used for Babicanora Norte, Sur and Las Chispas, based on the vein thickness. Resuing will be the only mining method used for Babi Vista.

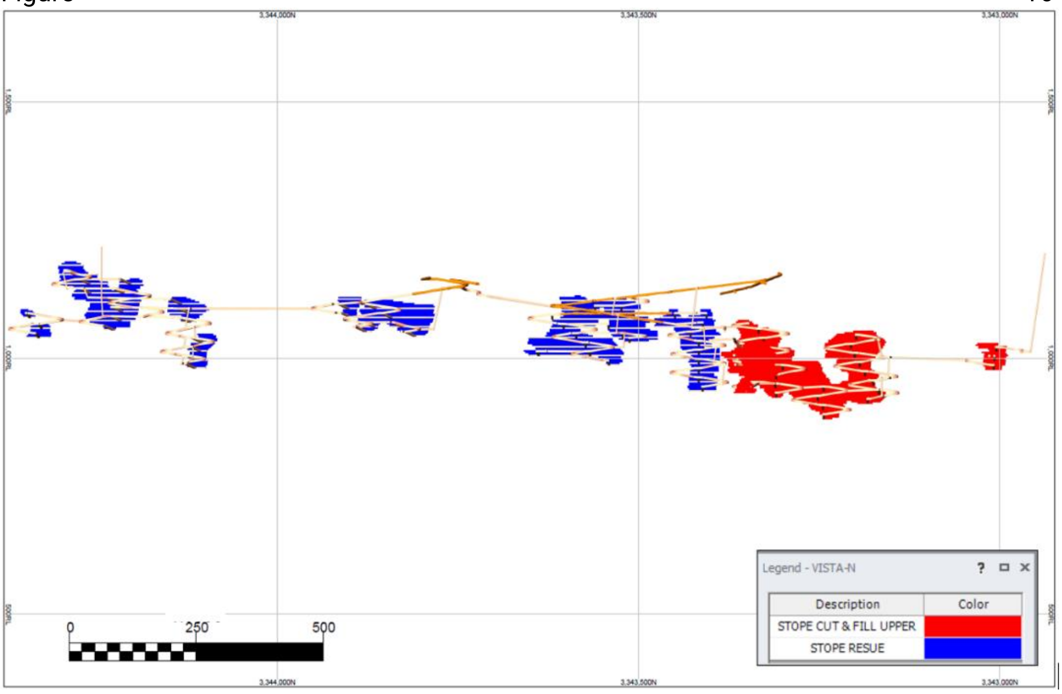
For mining stopes using the cut-and-fill method, a pivot drive (4 m H x 4 m W) with -18% grade must be excavated. An initial sill (3 m H x 3 m W) needs to be developed. Once the sill reaches the extremity of the mining area, rockfill will be placed to allow a 0.3 m gap to the sill back. The next cuts will be driven by slashing the breast holes rounds into this gap to enhance productivity. Six cuts will be completed with the same pivot drive. The height of the cut-and-fill stopes, including all six cuts, will be 18 m.

For mining stopes using resuing, a sill will be developed in the mineralized vein. Once developed, the resuing blasting technique will be used for the subsequent cuts. The mineralized material will be mucked before the waste portion of the round is drilled, loaded, and blasted. The waste will be left in place to create a new working floor for the next cut to be mined. The dimension of the final resue sill will be 2.6 m W x 3 m H



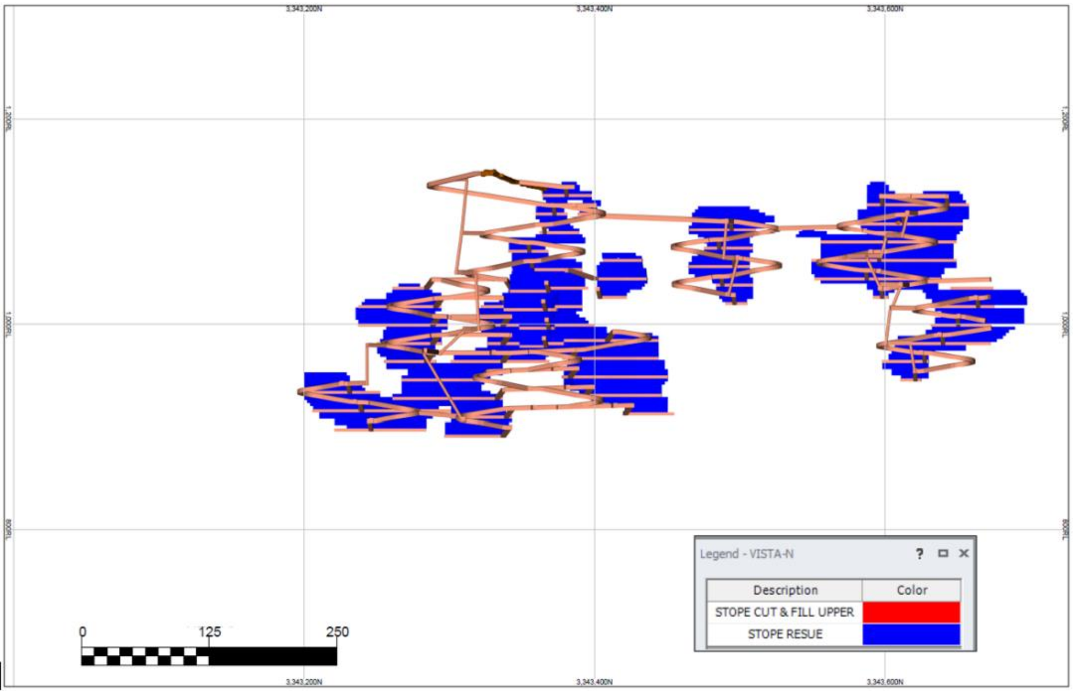
Note: Figure prepared by GMS, 2020.

Figure 16-41



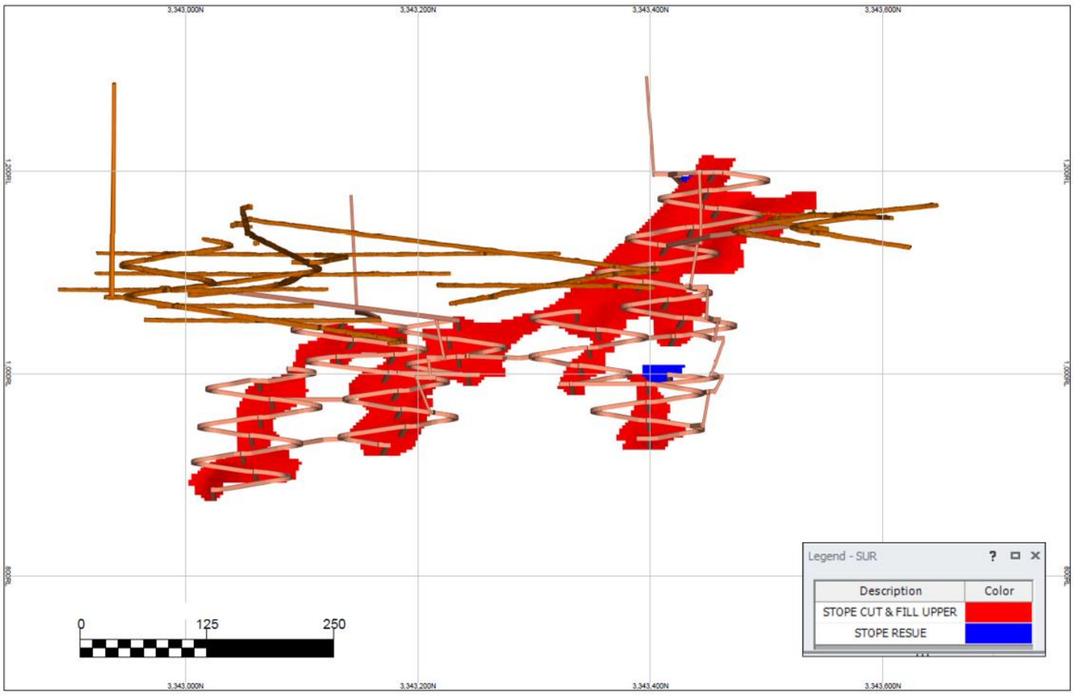
Note: Figure prepared by GMS, 2020.

Figure 16-41: Mining Methods – Babicanora Norte (looking north)



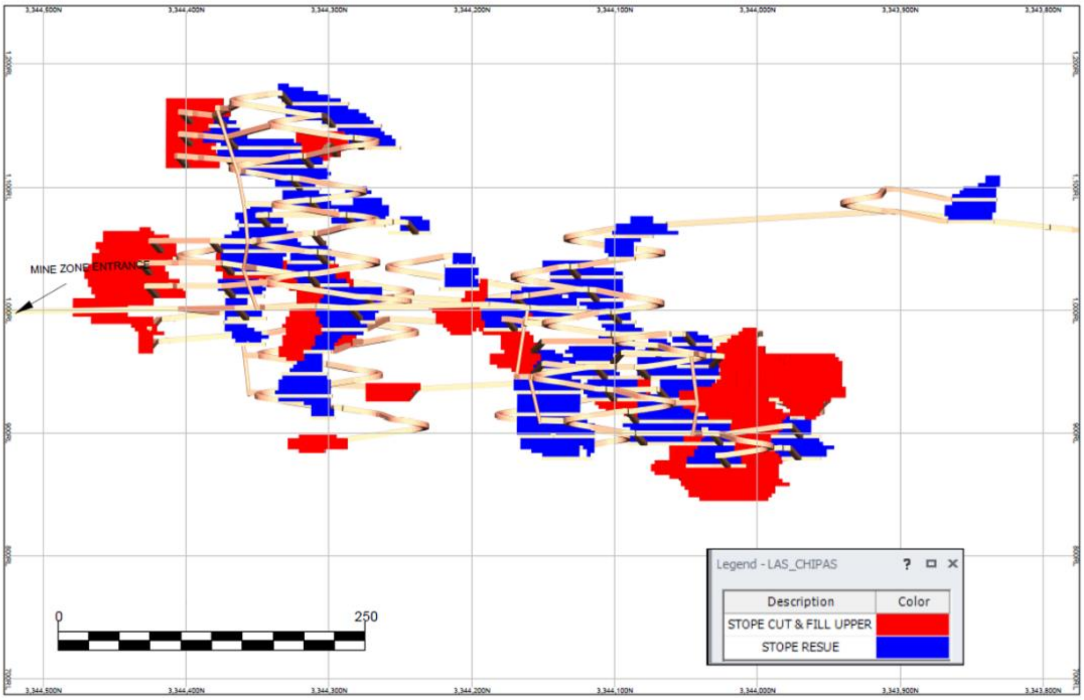
Note: Figure prepared by GMS, 2020.

Figure 16-42: Mining Method – Babi Vista (looking southwest)



Note: Figure prepared by GMS, 2020.

Figure 16-43: Mining Method – Babicanora Sur (looking southwest)



Note: Figure prepared by GMS, 2020.

Figure 16-44: Mining Method – Las Chispas (looking northeast)

16.5.6 Long-hole Production Drilling and Blasting

Three drill types will be used: two-boom electric-hydraulic jumbos for large-scale development headings, single-boom electric-hydraulic jumbos for mineralized material sill development, resue, and larger cut-and fill-mining, and finally, single-boom electric-hydraulic narrow vein jumbos will be used for the narrow cut-and-fill stopes.

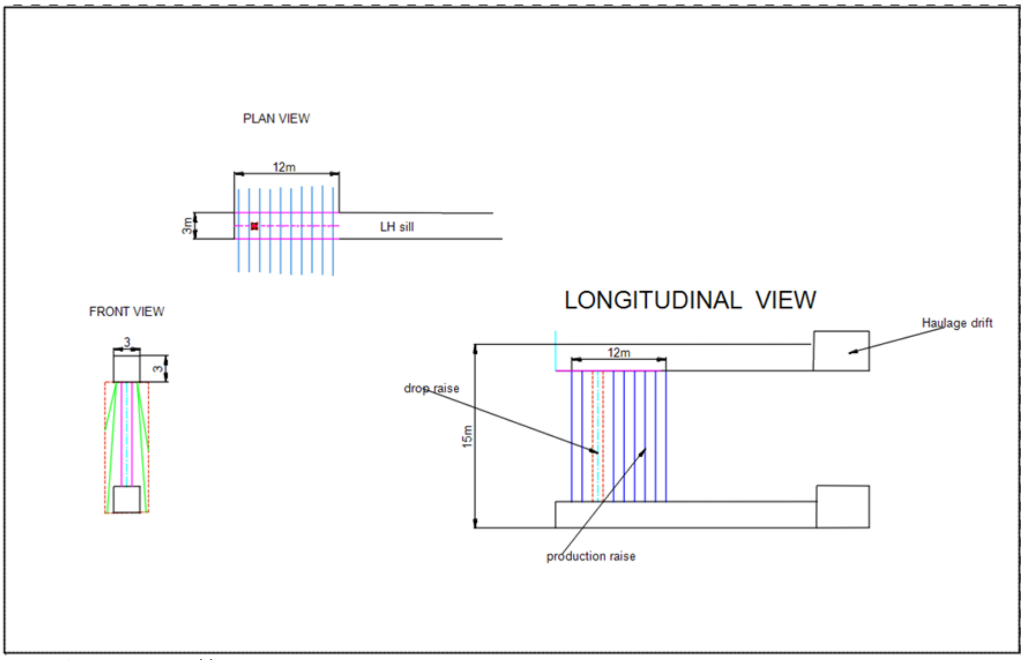
Mechanized bolters are planned for the ground support installation. Depending on the excavation size, two types of bolters will be used: standard size bolters for waste development, and narrow vein bolters in other excavations.

Jackleg and stoper drills are planned for bolting in raises, drilling safety bays, service drilling, and ground support installation.

Electric-hydraulic long-hole drills are planned for the production holes. The following guidelines were used:

- The production hole will be completed by long-holes drilled downward. For some areas near the crown pillar, drilling in an upward direction is planned;
- Typically, the drilling sections will consist of three production boreholes, and a stope will have 10 drilling sections;
- Drill hole diameter of 64 mm (2.5 in.);
- Ring burden of 1.2 m and spacing of 1.2 m. A "Dice 5" drill hole pattern will be used where the true stope width is <2 m;
- Slot raises (drop raises) incorporate a 125 mm diameter relief hole;

- The downhole hole rings will be drilled vertically. The ascending hole rings will be drilled at 80º, dumped forward;
- A 10% provision is included to account for re-drilling; and,
- The drilling factor is estimated at 4.71 t/m drilled; and,
- A typical production drilling pattern is shown in Figure 16-45.



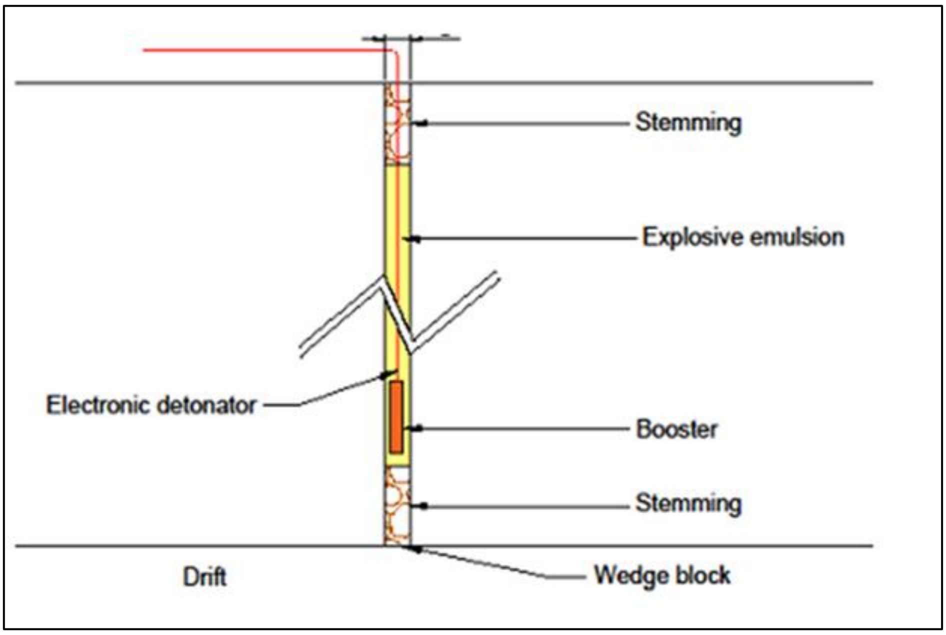
Note: Figure prepared by GMS, 2020.

Figure 16-45: Typical Production Drilling Pattern

Drilling production (drilled metre/percussion hour) was built from first principles, current operational results, and industry standards.

Blasting crews will be trained and certified for explosives use. Anfo and/or packaged emulsion will be used for production blasting and development rounds. Primers, detonators, detonation cords and other ancillary blasting supplies will also be used. There are currently no plans to use smooth blasting techniques. However, tests may need to be carried out in unfavourable ground conditions to assess the effectiveness of this technique. All blasting will be done at shift change. All personnel not cleared from the underground will be required to be in a designated safe work area during blasting. During the production period, a central blast system will be used to initiate blasts for all loaded development headings and production stopes.

Pumpable emulsion type explosives will be used in stope blastholes. Figure 16-46 shows a typical loading procedure for stope blast holes.



Note: Figure prepared by GMS, 2020.

Figure 16-46: Typical Blast Hole

16.5.7 Mucking

Based on the mining method and excavation dimensions a fleet of 1.2 t, 3 t, 3.5 t and 10 t load–haul–dump (LHD) vehicles were selected.

The 1.2 t, 3 t and 3.5 t LHD units will remove the mineralized material or waste from the stope and transport it to a muck bay, where the 10 t LHD can directly load haul trucks. In long hole stopes, an LHD equipped with remote control will be used to keep personnel away from unsupported ground. LHDs will tram the material to a nearby re-muck bay.

For the cycle time calculation, it was assumed that all the tonnes would transit via a muck bay. For the development mineralized material the assumption is 100% of the development mineralized material will transit to a muck bay and be re-handled onto the haul trucks. For waste development, most of that waste rock will be used as rockfill and the remainder will be transported to surface. Surface-stockpiled waste will be returned to the mine for rockfill when necessary.

Hauling distances were evaluated by mining sequences from the tonnage centre of gravity on each stope to the muck bays. Combined with LHD performances, time cycles were evaluated. The LHD fleet requirement was estimated based on the planned tonnage, planned hours, and the specific vehicle performances assumed.

16.5.8 Hauling

Diesel 30 t truck was selected to bring rock to surface. The trucks will be loaded by the 10 t LHD at the loading point. Each sector of the mine will have loading points on each production level. The trucks will travel to surface where broken (not crushed) mineralized material or waste will be unloaded on a surface transfer pad. From this pad, mineralized material will be transported by surface mining trucks or via loader to the process plant. The waste not used for rockfill will be transported by surface mining trucks to the appropriate WRSF.

To estimate the required truck fleet size, haulage cycle times were evaluated for each mining area and level, based on the development and production plan. Specifications of the 30 t haulage trucks were established using the following key factors: truck speed, haul cycle time, and truck capacity.

16.5.9 Mining Equipment

Mining equipment will be provided by a contractor. For areas using cut-and-fill and long hole development sill, 3 t or 3.5 t scoops will be used to haul mineralized material from the face to the re-muck bay. For resuing sills, 1.2 t scoops will be used to avoid additional dilution during mucking.

The drilling equipment requirements were determined based on optimal drilling parameters. Equipment numbers reflect the departmental needs, and the development and production profiles over time. A peak of 109 units of mobile equipment was estimated as listed in Table 16-7.

Table 16-7: Mining Fleet over LOM for All Zones

Type	LOM Quantity
Narrow Production Drill	3
Two-Boom Jumbo	4
Single-Boom Jumbo	3
Rock Bolter	4
Narrow Rock Bolter	6
Long-hole Drill	4
30-Tonne Underground Haul Truck	6
10-Tonne Scoop	6
3.5-Tonne Scoop	8
3.0-Tonne Scoop	2
1.2-Tonne Scoop	2
Shotcrete Machine	1
Backhoe	3
Loader/Forklift	1
Scissor Lift	2
Scissor Lift	1
Boom Truck	1
Fuel Lube Truck	1
Grader	1
Personnel Carrier-5 People	25
Personnel Carrier	25
Total	**109**

16.6 Mining Rate

To estimate the mining rate, two principal approaches were used. Initially, three empirical formulas were used to determine a base case mining rate (Taylor 1977; Long, 2009, McIsaac and Devon Smith (2019)).

- Taylor : t/d = 0.0143 x $(\text{tonnage})^{0.75}$;
- Long : t/d = 0.297 x $(\text{tonnage})^{0.562}$;
- McIsaac and Devon Smith : t/d = 0.02 x $(\text{tonnage})^{0.75}$.

Based on the expectation of 3.1 Mt to be mined from the Babicanora Main zone, the three formulas provide the following results:

- Taylor = 1,000 t/d;
- Long = 1,300 t/d;
- McIsaac and Devon Smith = 1,400 t/d.

Based on these empirical rules and to limit the amount of initial development, the QP determined that a production rate of 1,250 wet-t/d was optimal.

The scheduling process was carried out using the DOS software. To ensure an adequate volume of tonnes throughout the LOM, the tonnage to be produced from Babicanora Main was spread out over the course of the mine life. Forgoing this option would have resulted in tonnages coming from low-volume stopes in the last years of the LOM. It would also have impacted the mine throughput, which would have been reduced or required a significant staff increase to achieve the desired tonnage.

16.7 Mine Schedule and Sequence

Development has already begun in Babicanora Main. Material mined prior 2021 was stockpiled on surface. The basis of the Feasibility Study is that the pre-production period starts in January 2021.

16.7.1 Production schedule

The LOM plan is based on the Mineral Reserves in Section 15. Including development and production, a total of 35 Mt, including historical stockpiles, will be extracted over a period of approximately 8.5 years. The pre-production period will start in 2021 and will end during the second quarter of 2022. The average grade mined is forecast to be 4.98 gpt Au and 487 gpt Ag.

The production schedule was established for an annual mill feed production rate of 456,200 t/a. The schedule was built to maximize profitability by targeting higher-grade mining blocks in the early stage of the mine life.

Table 16-8 shows the forecasted mine production rates.

Table 16-8: Mine Task Production Rates

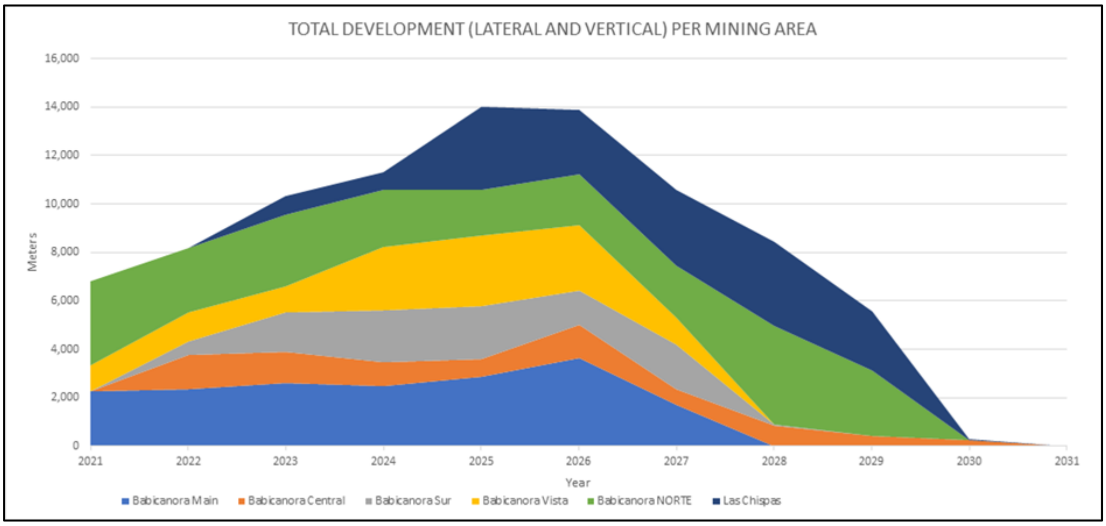
Long Hole			
Task	**Zone**	**Task Rate**	
Stope Mucking	Babicanora Main	250	t/d/stope
	Babicanora Central	150	t/d/stope
Drilling	All	250	m/d/drill
Backfill	Babicanora Main	150	t/d/stope
	Babicanora Central	50	t/d/stope

Long Hole			
Task	**Zone**	**Task Rate**	
Cut and Fill			
Stope Mucking	Las Chispas	10–110	t/d
Average*	Babicanora Central	20–150	t/d
	Babicanora Sur	35–105	t/d
	Babicanora Norte	20–90	t/d
	Babi Vista	10–50	t/d
Pivot/Sill Advance	All	2.5	m/d

As for the lateral development, the duration of the task was determined by the production rate and the number of equipment available. The single heading advance rate were set a 2.5 m/d with a scheduling maximum constraint of 300 m of lateral advance per month. This performance level was shown to be achievable at Las Chispas in 2019–2020.

Total mine underground capital and operating development will be 53,554 m and 32,220 m, respectively. Underground overall development averages 9500 m/a or 26.0 m/d from Year 1 to Year 8.

The annual development per mining area is shown in Figure 16-47.

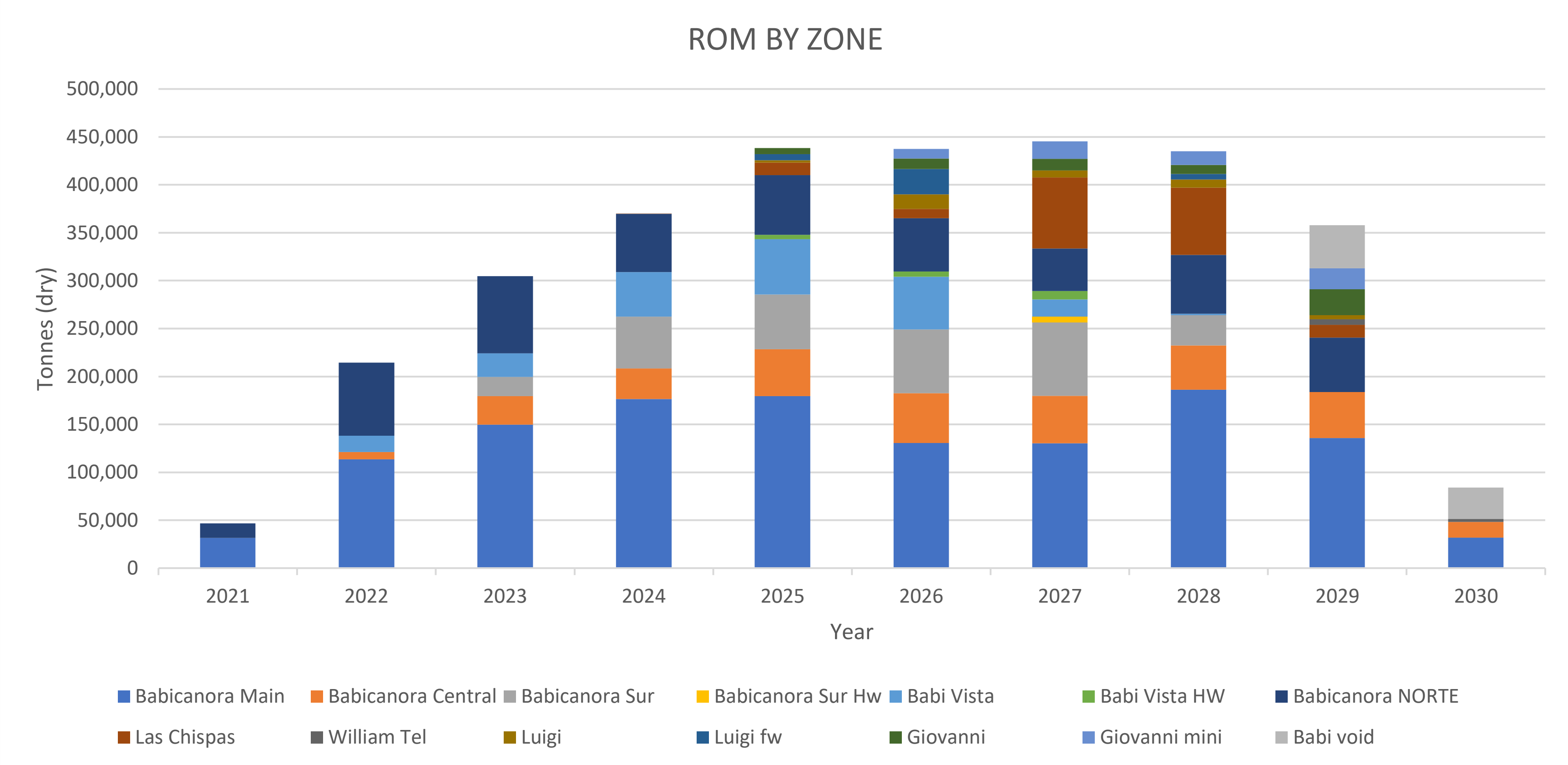


Note: Figure prepared by GMS, 2020.

Figure 16-47: Forecast Annual Total Development by Mine Area

The pre-production period covers development starting in Q1 2021 (2021-01-01) and ending in the Q2 of 2022 (2022-05-30). The development during the pre-production period will be carried out in the Babicanora Main, Babicanora Norte and Babi Vista areas. This will be followed by the production period.

The largest tonnage and contained metal is estimated for Babicanora Main, making that area the development priority. The same principles and access availability will be considered to prioritize development and mine production. The annual mineralized material production by zone is shown in Figure 16-48 and the LOM physical plan is presented in Table 16-9.



Note: Figure prepared by GMS, 2020.

Figure 16-48: Annual Run of Mine Production by Vein

Table 16-9: ROM Plan

Mining Area		2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	TOTAL
Babicanora Main	Tonnes	31,445	113,705	149,826	176,488	179,602	130,489	130,290	186,042	135,720	31,761	**1,265,368**
	gpt Au	2.26	2.73	4.51	4.44	3.86	3.91	6.02	4.56	5.87	4.50	**4.44**
	gpt Ag	386	325	358	316	377	466	500	507	617	521	**432**
	gpt AgEq	582	562	750	703	713	806	1,024	904	1,128	912	**818**
Babicanora Central	Tonnes	-	7,355	29,813	31,996	48,751	52,178	49,493	46,461	48,000	16,347	**330,395**
	gpt Au	-	1.89	2.12	2.39	2.63	2.05	2.60	7.61	3.37	2.46	**3.25**
	gpt Ag	-	198	250	220	226	234	269	601	135	165	**271**
	gpt AgEq	-	362	434	428	455	412	494	1,263	428	379	**553**
Babicanora Sur and HW	Tonnes	-	-	19,899	53,942	57,118	66,290	82,639	31,535	-	-	**311,422**
	gpt Au	-	-	5.07	3.52	4.04	5.42	7.00	2.56	-	-	**4.94**
	gpt Ag	-	-	428	346	223	305	191	146	-	-	**259**
	gpt AgEq	-	-	869	652	574	776	800	369	-	-	**688**
Babi Vista and FW	Tonnes	-	17,244	24,555	46,592	62,407	60,650	26,672	1,402	-	-	**239,523**
	gpt Au	-	16.61	15.12	13.13	11.54	9.45	6.75	2.97	-	-	**11.47**
	gpt Ag	-	1,158	1,745	823	902	845	671	285	-	-	**948**
	gpt AgEq	-	2,602	3,060	1,964	1,905	1,666	1,257	543	-	-	**1,945**
Babicanora Norte and HW	Tonnes	15,194	76,058	80,562	60,732	62,313	55,615	44,318	61,521	56,992	-	**513,304**
	gpt Au	2.87	8.28	4.46	5.09	6.09	8.62	3.01	6.67	4.79	-	**5.88**
	gpt Ag	365	963	576	541	695	885	543	795	475	-	**683**
	gpt AgEq	614	1,683	964	984	1,224	1,635	805	1,375	891	-	**1,194**
Las Chispas	Tonnes	-	-	-	396	28,114	72,312	111,824	108,205	72,272	3,167	**396,291**
	gpt Au	-	-	-	0.02	1.94	2.22	4.61	4.68	2.20	2.05	**3.54**
	gpt Ag	-	-	-	3	301	276	517	644	325	420	**456**
	gpt AgEq	-	-	-	5	470	470	918	1,051	516	598	**764**
Babicanora Central Open Stope	Tonnes	-	-	-	-	-	-	-	-	45,000	32,834	**77,834**
	gpt Au	-	-	-	-	-	-	-	-	2.51	2.51	**2.51**
	gpt Ag	-	-	-	-	-	-	-	-	301	301	**301**
	gpt AgEq	-	-	-	-	-	-	-	-	519	519	**519**
ROM	Tonnes	46,639	214,362	304,655	370,145	438,305	437,534	445,236	435,167	357,984	84,109	**3,134,137**
Au dry	gpt Au	2.46	5.79	5.16	5.33	5.03	5.00	5.21	5.06	4.20	3.23	**4.98**
Ag dry	gpt Ag	379	614	521	413	455	489	436	565	431	362	**478**
AgEq dry	gpt AgEq	593	1,117	970	876	893	924	889	1,005	796	643	**911**

16.7.2 Babicanora Area Development

Table 16-10 and Table 16-11 present, for each Babicanora zone, the annual lateral and vertical development, respectively.

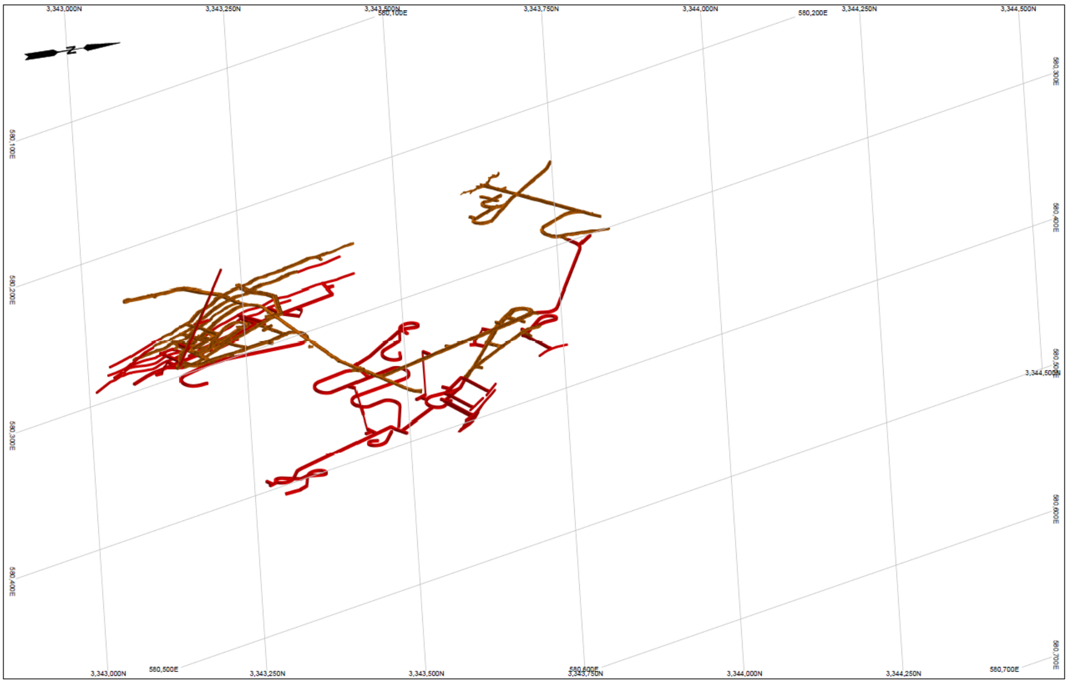
Table 16-10: Annual Lateral Development over LOM for All Zones

Zone	Units	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	TOTAL
Babicanora Main	(m)	2,197	2,274	2,523	2,412	2,734	3,514	1,629	–	–	–	17,282
Babicanora Central	(m)	–	1,330	1,290	979	705	1,349	590	815	440	212	7,708
Babicanora Sur	(m)	–	536	1,466	2,100	2,053	1,437	1,577	34	–	–	9,204
Babi Vista	(m)	1,015	1,113	1,020	2,610	2,578	2,280	685	–	–	–	11,302
Babi Vista FW	(m)	–	–	–	–	286	366	413	–	–	–	1,065
Babicanora Norte	(m)	3,355	2,339	2,765	2,213	1,840	1,925	2,154	3,750	2,579	–	22,922
TOTAL	**(m)**	**6,567**	**7,591**	**9,064**	**10,315**	**10,195**	**10,871**	**7,048**	**4,599**	**3,019**	**212**	**69,482**

Table 16-11: Annual Vertical Development over LOM for All Zones

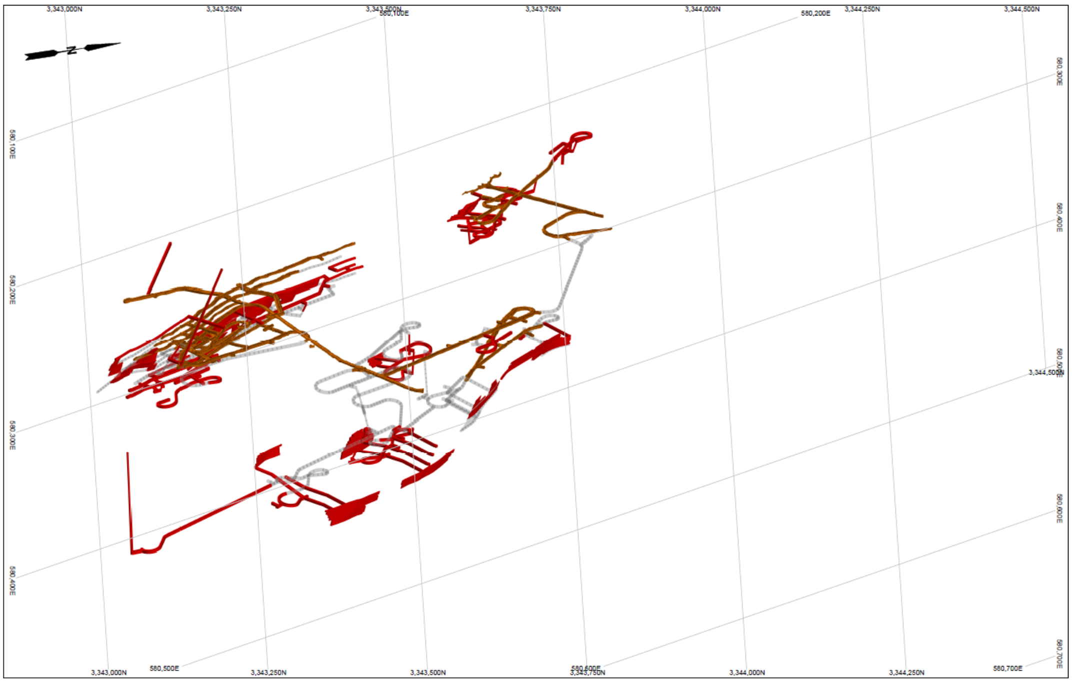
Zone	Units	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	TOTAL
Babicanora Main	(m)	65	82	99	65	128	113	79	–	–	–	633
Babicanora Central	(m)	–	96	–	–	32	24	70	39	–	57	318
Babicanora Sur	(m)	–	-	166	45	123	-	264	–	–	–	599
Babi Vista	(m)	72	73	21	22	55	63	-	–	–	–	307
Babi Vista FW	(m)	–	–	–	–	–	–	–	–	–	–	–
Babicanora Norte	(m)	109	356	184	127	55	146	–	320	111	–	1,408
TOTAL	**(m)**	**246**	**607**	**471**	**259**	**393**	**346**	**413**	**359**	**111**	**57**	**3,264**

Figure 16-49 through Figure 16-58 illustrate the LOM annual development and production sequence, coloured in red, by production year for the Babicanora areas.



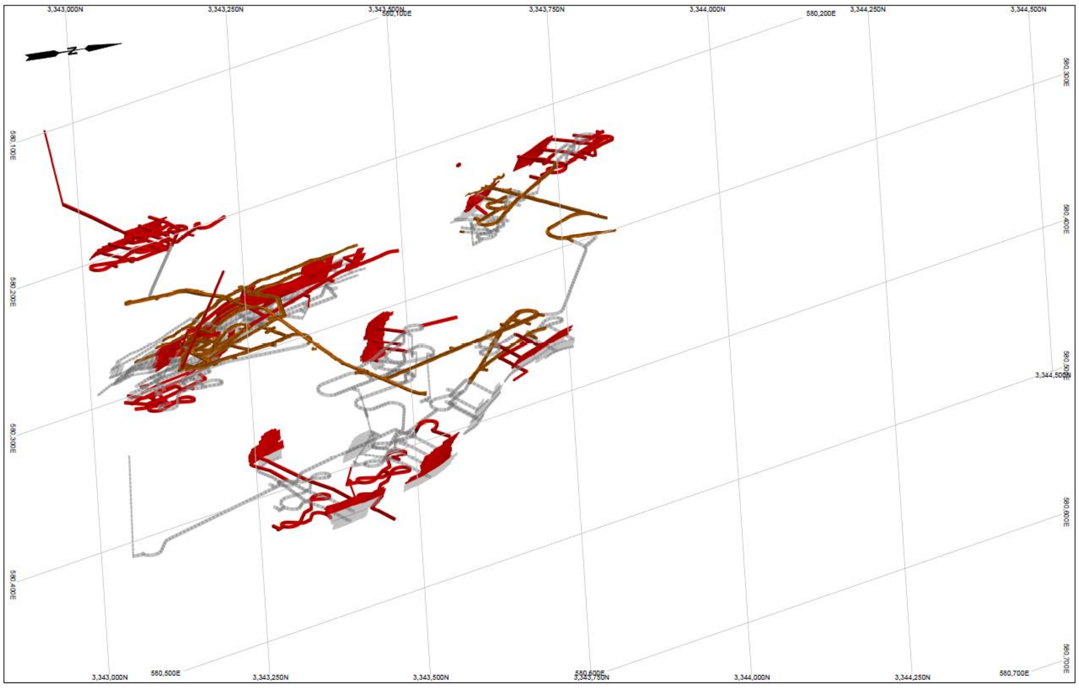
Note: Figure prepared by GMS, 2020.

Figure 16-49: Babicanora Area Development 2021, Oblique View (looking southwest)



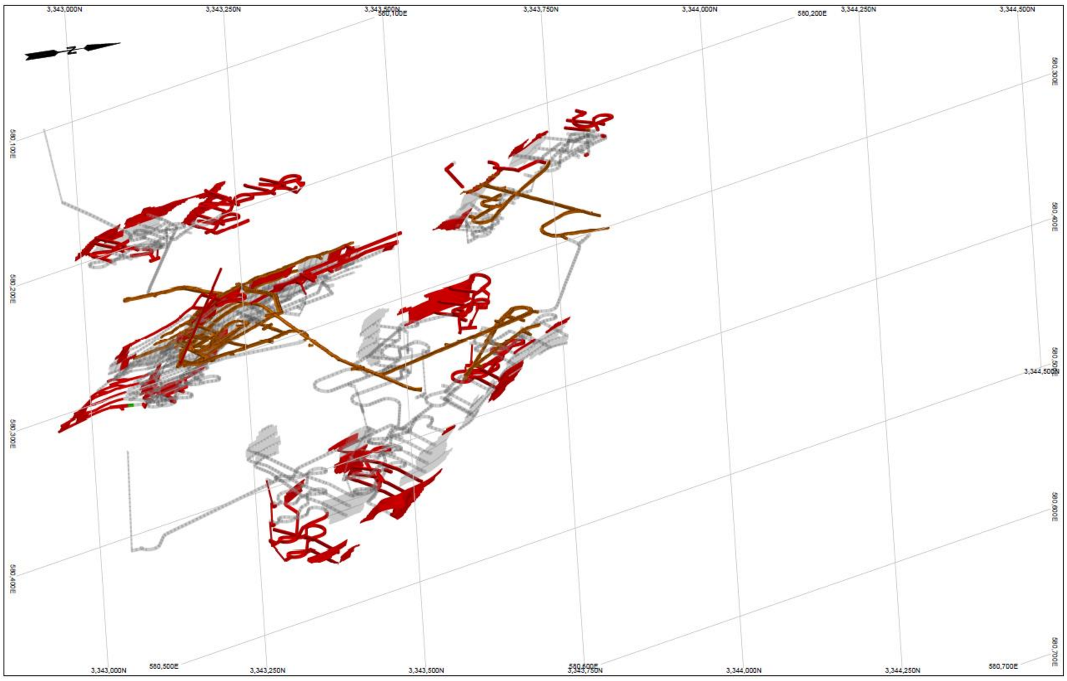
Note: Figure prepared by GMS, 2020.

Figure 16-50: Babicanora Area Development 2022, Oblique View (looking southwest)



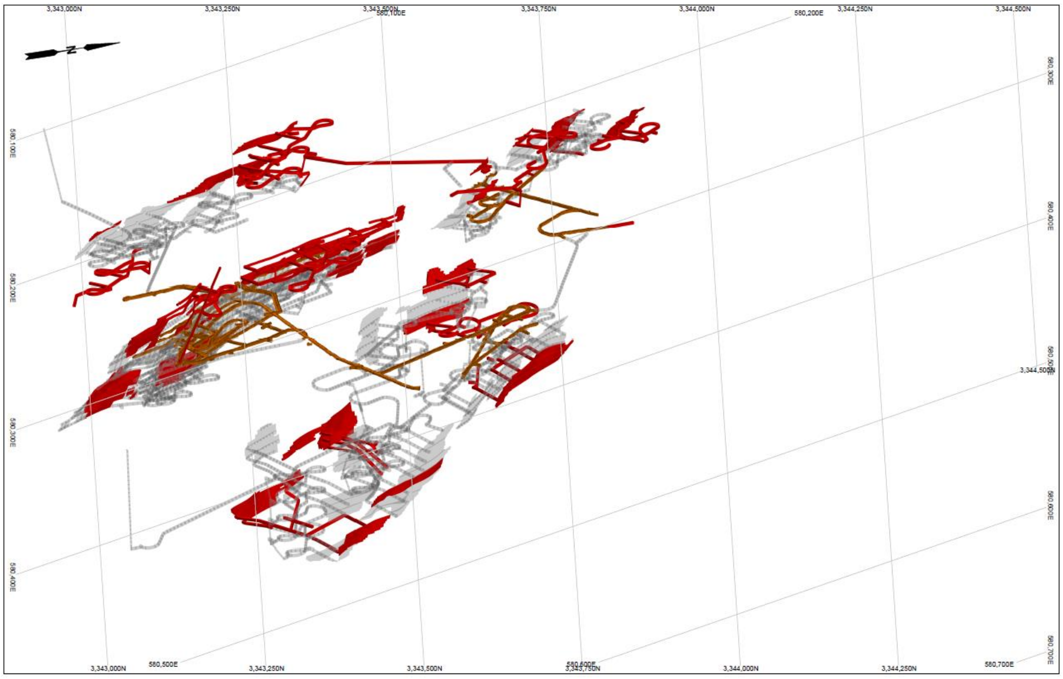
Note: Figure prepared by GMS, 2020.

Figure 16-51: Babicanora Area Development 2023, Oblique View (looking southwest)



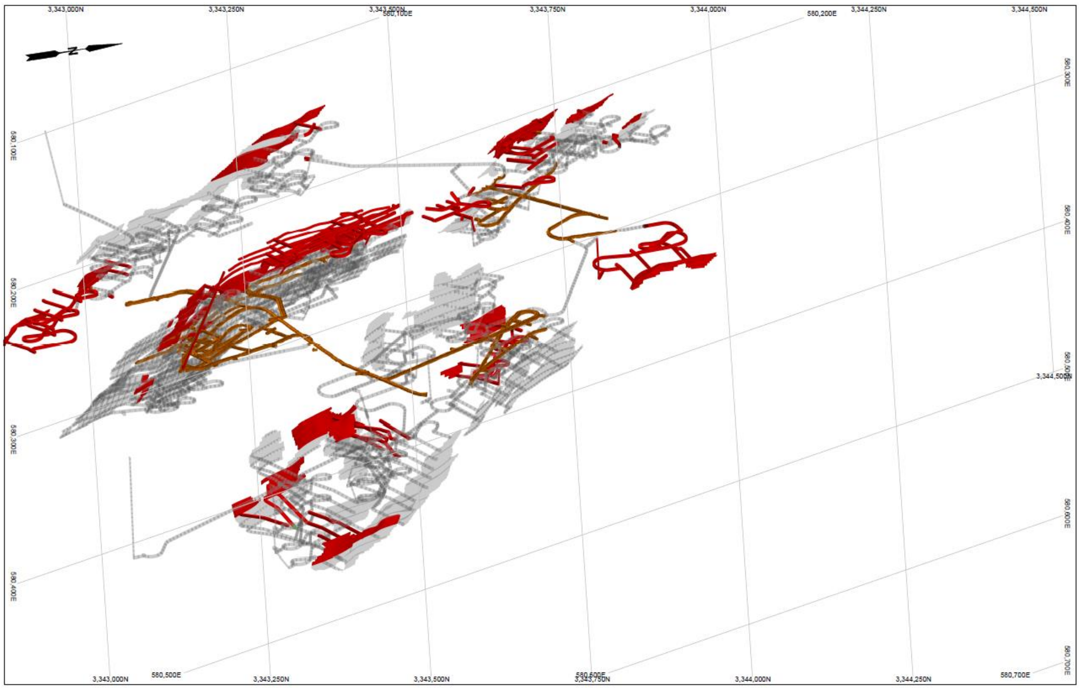
Note: Figure prepared by GMS, 2020.

Figure 16-52: Babicanora Area Development 2024, Oblique View (looking southwest)



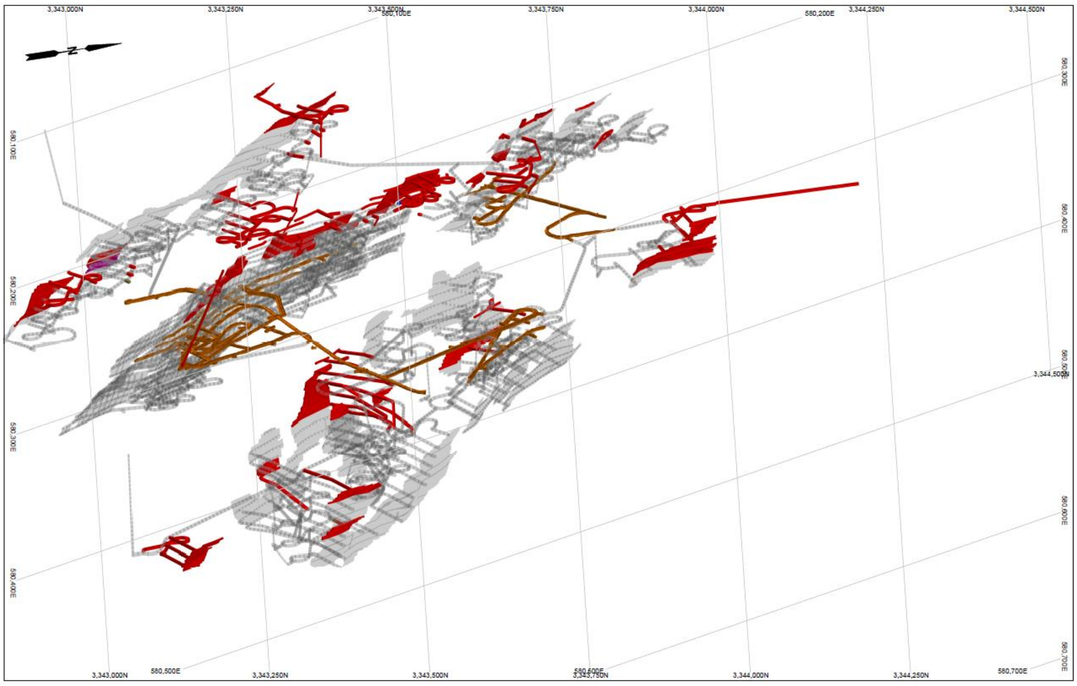
Note: Figure prepared by GMS, 2020.

Figure 16-53: Babicanora Area Development 2025, Oblique View (looking southwest)



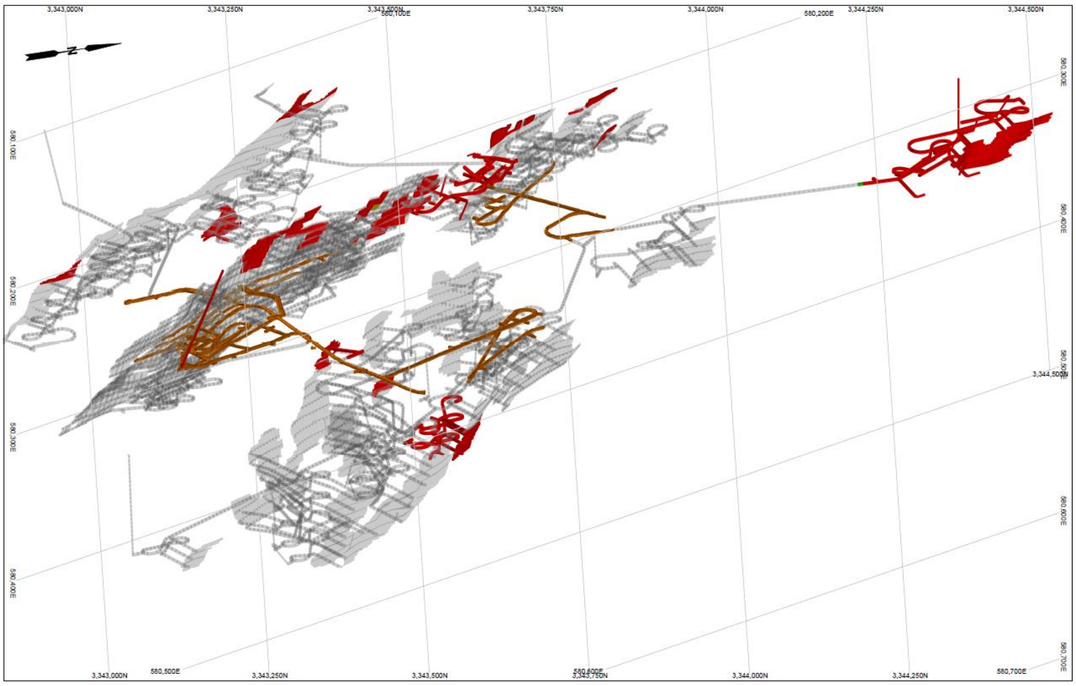
Note: Figure prepared by GMS, 2020.

Figure 16-54: Babicanora Area Development 2026, Oblique View (looking southwest)



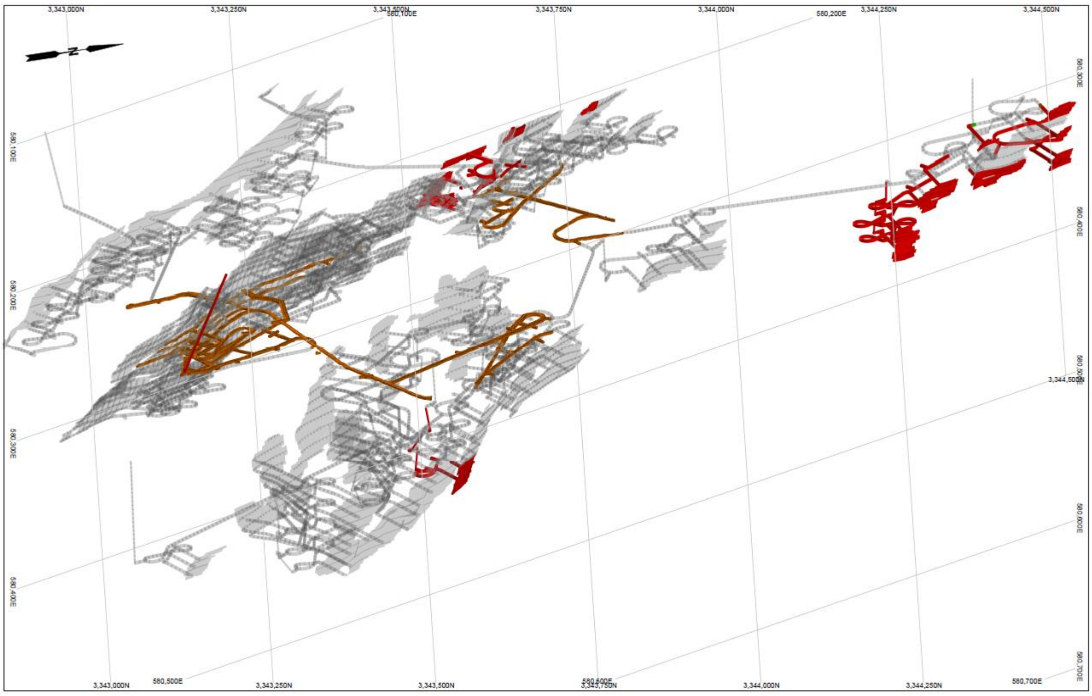
Note: Figure prepared by GMS, 2020.

Figure 16-55: Babicanora Area Development 2027, Oblique View (looking southwest)



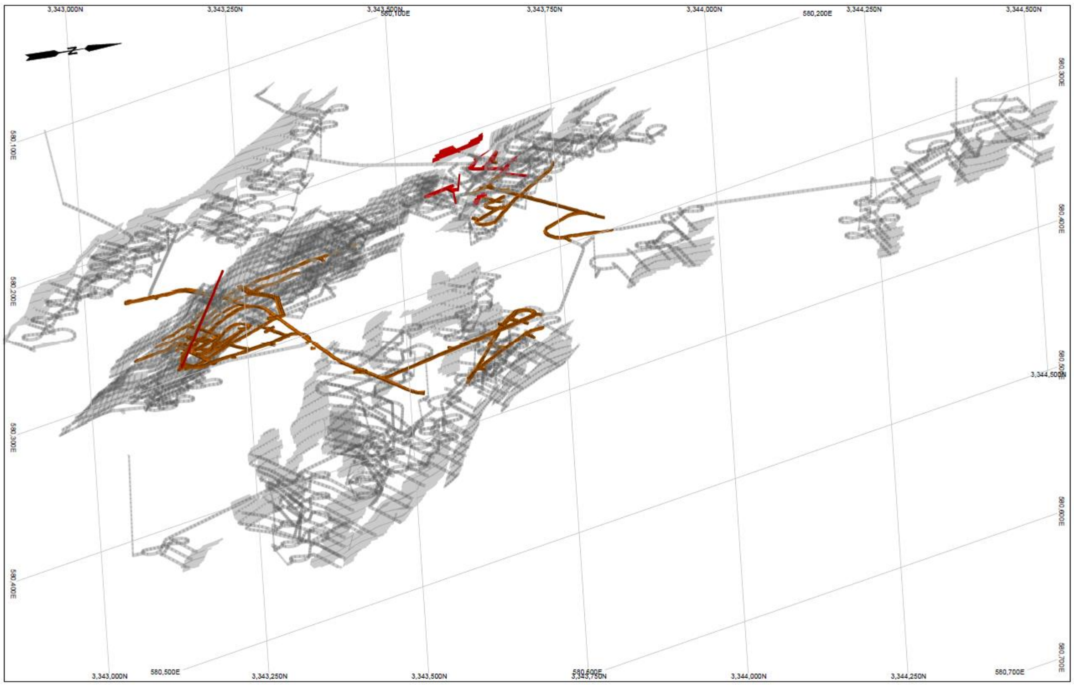
Note: Figure prepared by GMS, 2020.

Figure 16-56: Babicanora Area Development 2028, Oblique View (looking southwest)



Note: Figure prepared by GMS, 2020.

Figure 16-57: Babicanora Area Development 2029, Oblique View (looking southwest)



Note: Figure prepared by GMS, 2020.

Figure 16-58: Babicanora Area Development 2030, Oblique View (looking southwest)

16.7.3 Las Chispas Area Development

The Las Chispas zone development starts in 2023 with 750 m of decline advance. At the end of 2023, the main decline will reach the first two sub zones; Las Chispas and Giovanni, and the first stope will be mined in the Q4-2024. The production ramps up in the first year and reaches its peak in 2027 with 150,000 t of mineralized material mined.

Table 16-12 and Table 16-13 present the annual lateral and vertical development, respectively.

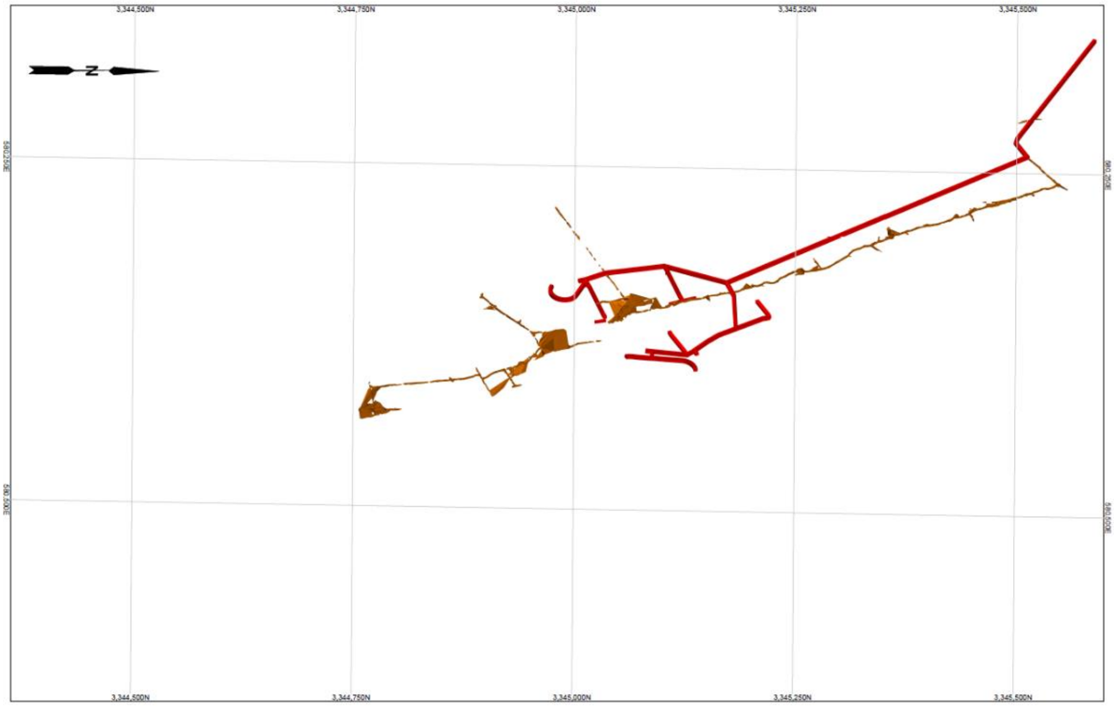
Table 16-12: Annual Lateral Development over LOM for All Las Chispas Areas

Zone	Units	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	TOTAL
Las Chispas	(m)	–	–	782	55	422	844	1,821	721	–	–	4,644
William Tel	(m)	–	–	–	–	–	–	–	–	1,078	5	1,083
Luigi	(m)	–	–	–	-	509	646	224	635	17	–	2,030
Giovanni	(m)	–	–	–	393	1,039	321	242	891	898	–	3,784
Gio Mini	(m)	–	–	–	–	223	779	780	741	379	–	2,902
Luigi FW	(m)	–	–	–	290	1,190	–	–	369	–	–	1,849
TOTAL	**(m)**	**–**	**–**	**782**	**737**	**3,383**	**2,589**	**3,066**	**3,357**	**2,372**	**5**	**16,291**

Table 16-13: Annual Vertical Development over LOM for All Las Chispas Areas

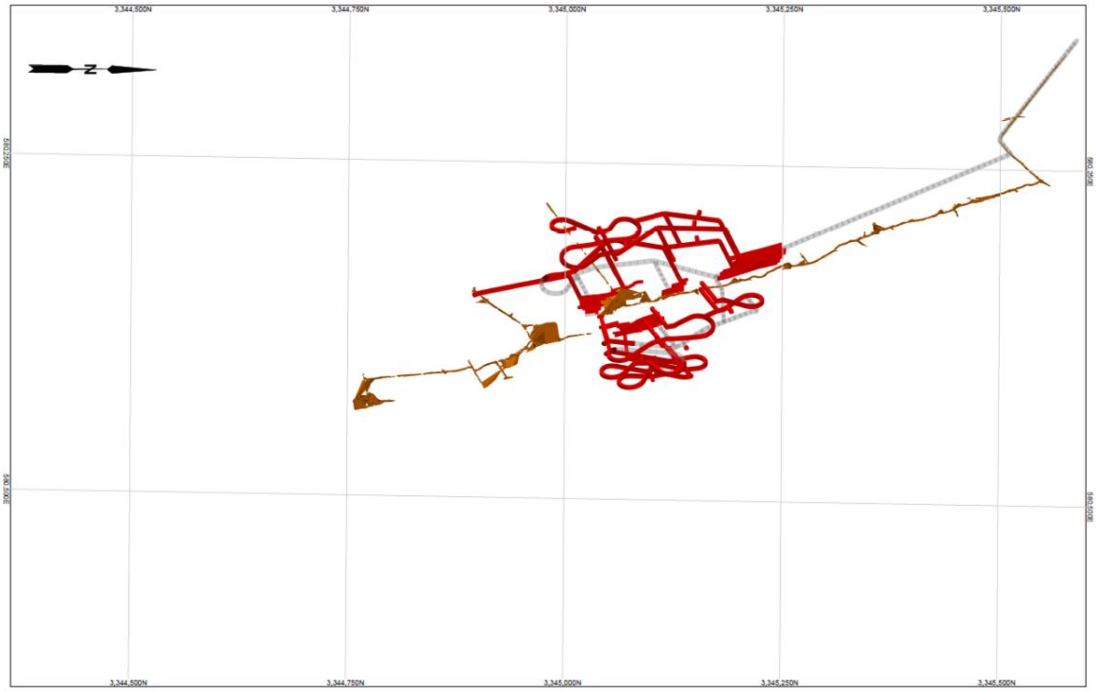
Zone	Units	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	TOTAL
Las Chispas	(m)	–	–	–	–	–	–	–	20	–	–	20
William Tell	(m)	–	–	–	–	–	–	–	–	–	–	–
Luigi	(m)	–	–	–	–	–	32	–	17	–	–	49
Giovanni	(m)	–	–	–	–	63	29	–	83	70	–	245
Gio Mini	(m)	–	–	–	–	–	27	54	13	–	–	95
Luigi FW	(m)	–	–	–	–	–	–	–	–	–	–	–
TOTAL	**(m)**	**–**	**–**	**–**	**–**	**63**	**89**	**54**	**134**	**70**	**–**	**410**

The Las Chispas LOM annual development and production sequence is shown in Figure 16-59 through Figure 16-64.



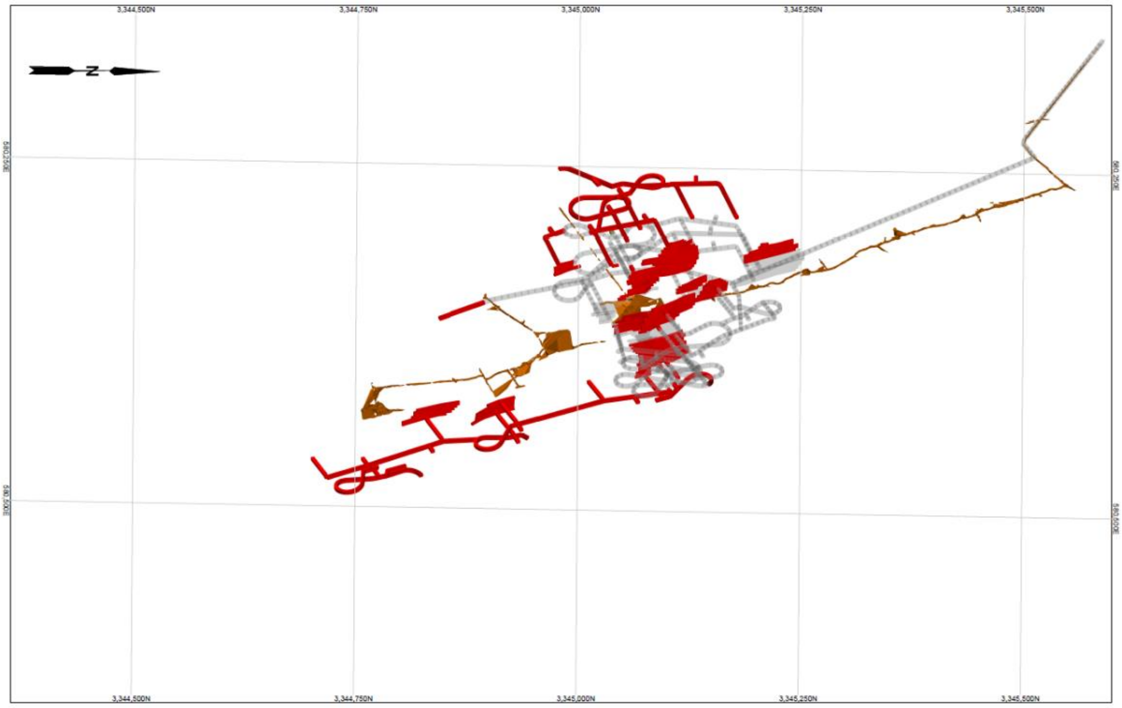
Note: Figure prepared by GMS, 2020.

Figure 16-59: Las Chispas Area Development 2023, Oblique View (looking northwest)



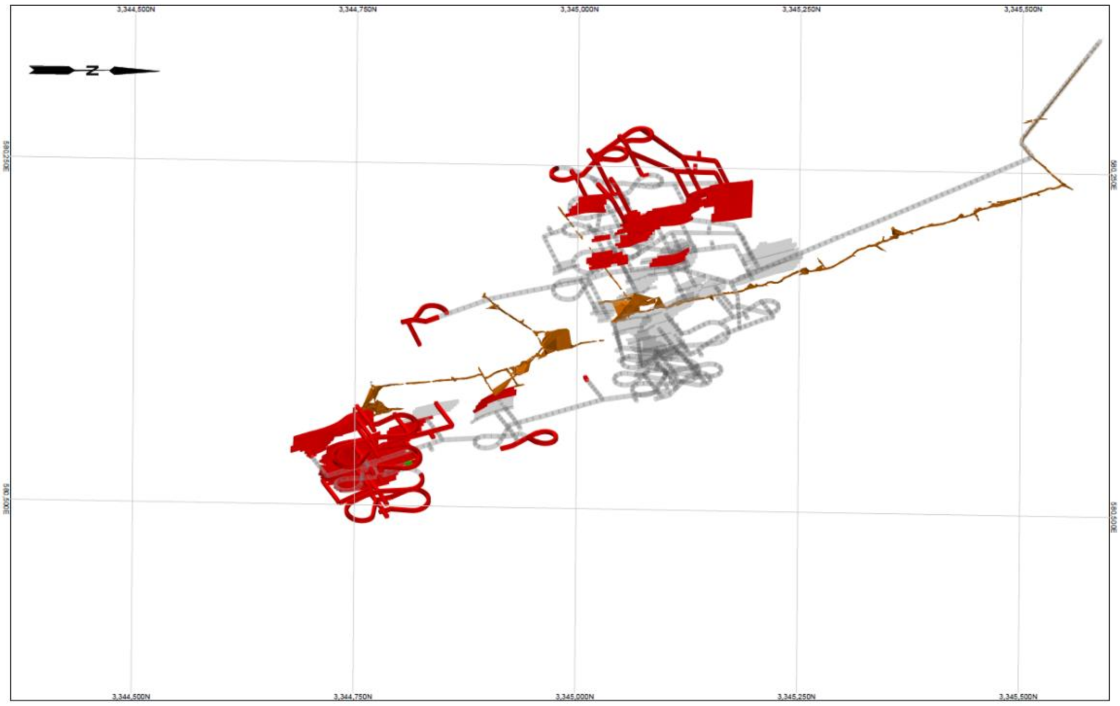
Note: Figure prepared by GMS, 2020.

Figure 16-60: Las Chispas Area Development 2024, Oblique View (looking northwest)



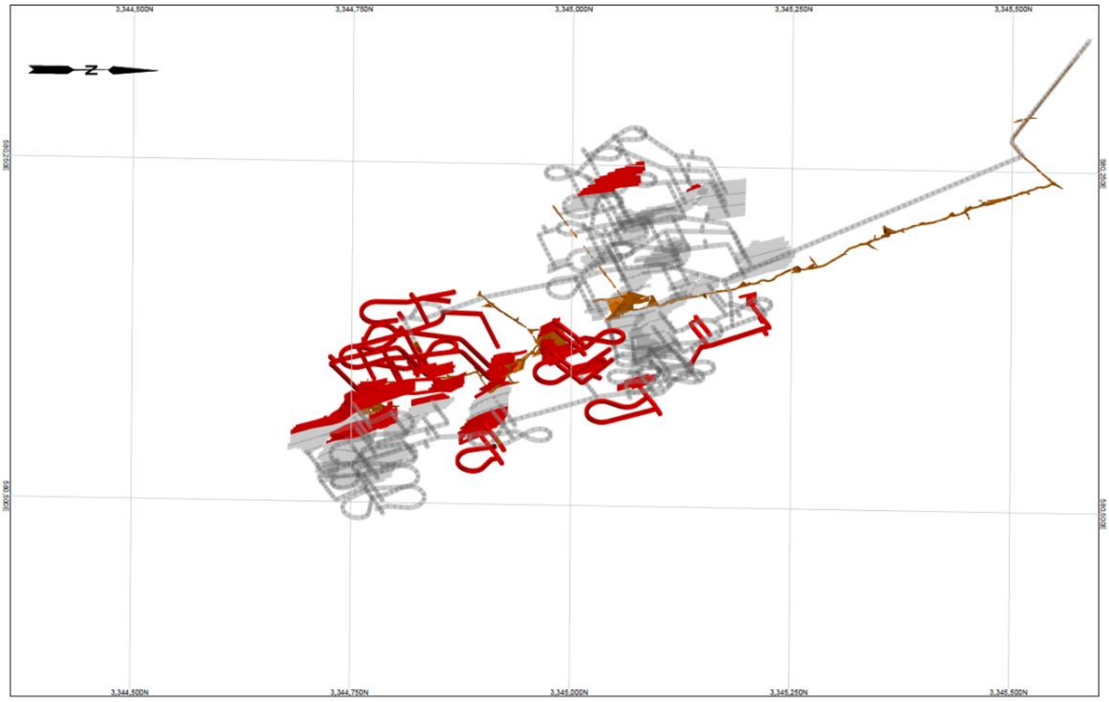
Note: Figure prepared by GMS, 2020.

Figure 16-61: Las Chispas Area Development 2025, Oblique View (looking northwest)



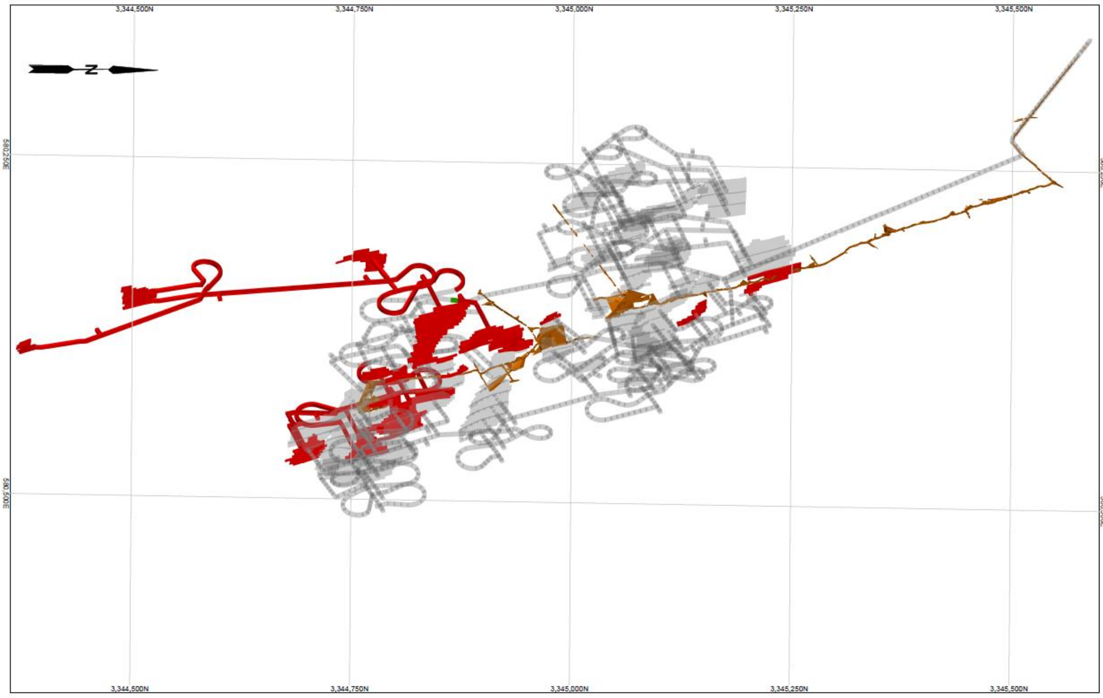
Note: Figure prepared by GMS, 2020.

Figure 16-62: Las Chispas Area Development 2026, Oblique View (looking northwest)



Note: Figure prepared by GMS, 2020.

Figure 16-63: Las Chispas Area Development 2027, Oblique View (looking northwest)



Note: Figure prepared by GMS, 2020.

Figure 16-64: Las Chispas Area Development 2028, 2029 and 2030, Oblique View (looking northwest)

16.8 Manpower and Working Schedule

All labour positions required for lateral and vertical development, as well as production stoping, will be provided by contract labour for the LOM. The Owner labour force will consist of key management positions, contract management supervision and technical services teams. Experienced professional training staff will be employed through the LOM to train local labour for safe and efficient activities within the mine.

The mine will operate seven days per week, with two shifts of 12 hours each. Development and production crews will be on a schedule of 14 days working/seven days off, for two 12-hour shifts. The maintenance crew will also be on the same schedule of 14 working days/seven days off, for 12 hr/shift, night and day, or days only. This schedule is equivalent to operating365 days per year.

Staff mine labour, including mine management and the technical department will work on five, eight-hour shifts, per week.

Table16-14 shows the total estimated manpower requirements over the LOM.

Table16-14: Annual Manpower Requirement Over LOM

		2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
SilverCrest Mine Management	Owner	4	12	12	12	12	12	12	12	12	6
Technical Services	Owner	22	31	31	31	31	31	31	31	31	25
Contractor Management	Contractor	16	16	20	20	24	24	24	24	8	4
Development Team	Contractor	42	42	63	63	84	84	63	63	42	21
Stoping/C&F mineralized material development	Contractor	21	60	72	72	92	92	101	87	45	36
Ore/Waste Handling	Contractor	6	12	15	18	24	24	27	30	21	12
Services	Contractor	2	6	6	6	6	6	6	6	6	6
CRF Mobile Plant	Contractor		12	12	12	12	12	12	12	12	
Total		**113**	**191**	**231**	**234**	**285**	**285**	**276**	**265**	**177**	**110**

16.9 Mine Services

16.9.1 Ventilation

The fresh air requirement was determined to meet the "Mexican Official REGULATION NOM-023-STPS-2012, Underground and open pit mines - Occupational safety and health conditions (*NORMA Oficial Mexicana NOM-023-STPS-2012, Minas subterráneas y minas a cielo abierto – Condiciones de seguridad y salud en el trabajo*)." The fresh air rate to dilute emissions must be 2.13 m^3/min of air for each horsepower of machinery that is powered by diesel combustion engines.

A conservative utilization rates were applied to account for the time when machines may be mechanically unavailable, or simply not in use. The utilization rates are: 80% for production equipment, and 50% for most service equipment and for machinery that operates primarily with electricity. Table 16-15 shows the ventilation rate for each piece of diesel equipment and the fresh air volumes needed to respect regulation and protect workers.

Table 16-15: Fresh Air Requirement: Ventilation Rate per Diesel Equipment

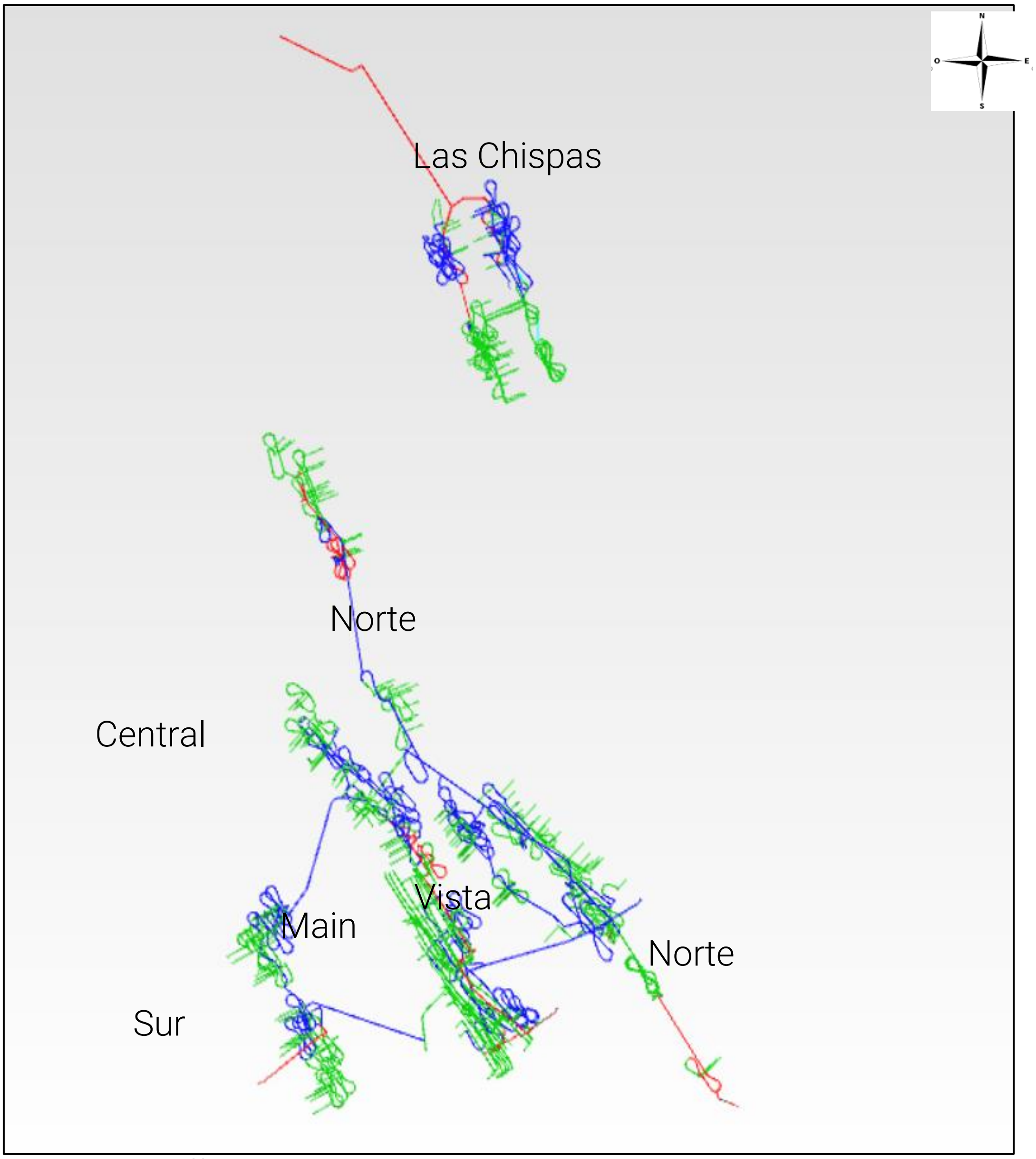
Equipment Type	Assumed Brand	Model	Power		Air Requirement / Unit		Quantity	% Utilization	Total Fleet	
			(kW)	(HP)	(CFM)	(m^3/s)			(ctm)	(m^3/s)
Narrow Production Drill	Resemin	MUKI22			8,612	4.06	4	0.5	17,224	8.1
Two-Boom Jumbo	Sandvik	DD320S	110	147	11,087	5.2	7	0.5	38,806	18.3
Rock Bolter	Sandvik	DS311D	95	127	9,576	4.5	8	0.5	38,302	18.1
30-Tonne Underground Haul Truck	Sandvik	TH430	310	415	31,247	14.8	8	0.8	199,978	94.4
10-Tonne Scoop	Sandvik	LH410	235	315	23,687	11.2	5	0.8	94,748	44.7
3.5-Tonne Scoop	Sandvik	LH203	72	96	7,207	3.4	6	0.8	34,593	16.3
3.0-Tonne Scoop	Sandvik	LH202	50	67	5,040	2.4	3	0.8	12,095	5.7
Shotcrete Machine	Multicrete	Aliva267		0	8,612	4.1	1	0.5	4,306	2.1
Backhoe	Caterpillar	430F2	80	107	8,064	3.8	3	0.5	12,095	5.7
Loader/Forklift	Kubota	R630	64	86	6,491	3.1	1	0.5	3,246	1.5
Scissor Lift	Maclean	SL2	129	173	13,003	6.1	2	0.5	13,003	6.1
Personnel Carrier	Access	Landcruiser 05-Person	94	126	9,475	4.5	25	0.5	118,434	55.9
Personnel Carrier	Kubota	RTV-1120	16	21	1,613	0.8	25	0.5	20,159	9.5
								Total	**606,989**	**287**
								Total with Contingency (10%)	**667,688**	**316**

Note: cmf = cubic feet per minute.

A contingency of 10% was applied as an allowance for any potential leaks in the system. A total of 670,000 cubic feet per minute was estimated and represents the design criterion for the underground ventilation infrastructure.

16.9.1.1 Ventilation Network

The Barbicanora Central, Babicanora Main, Babi Vista, Babicanora Norte and Babicanora Sur mining areas will be interconnected (Figure 16-65). The Las Chispas area will be independent of the other zones.

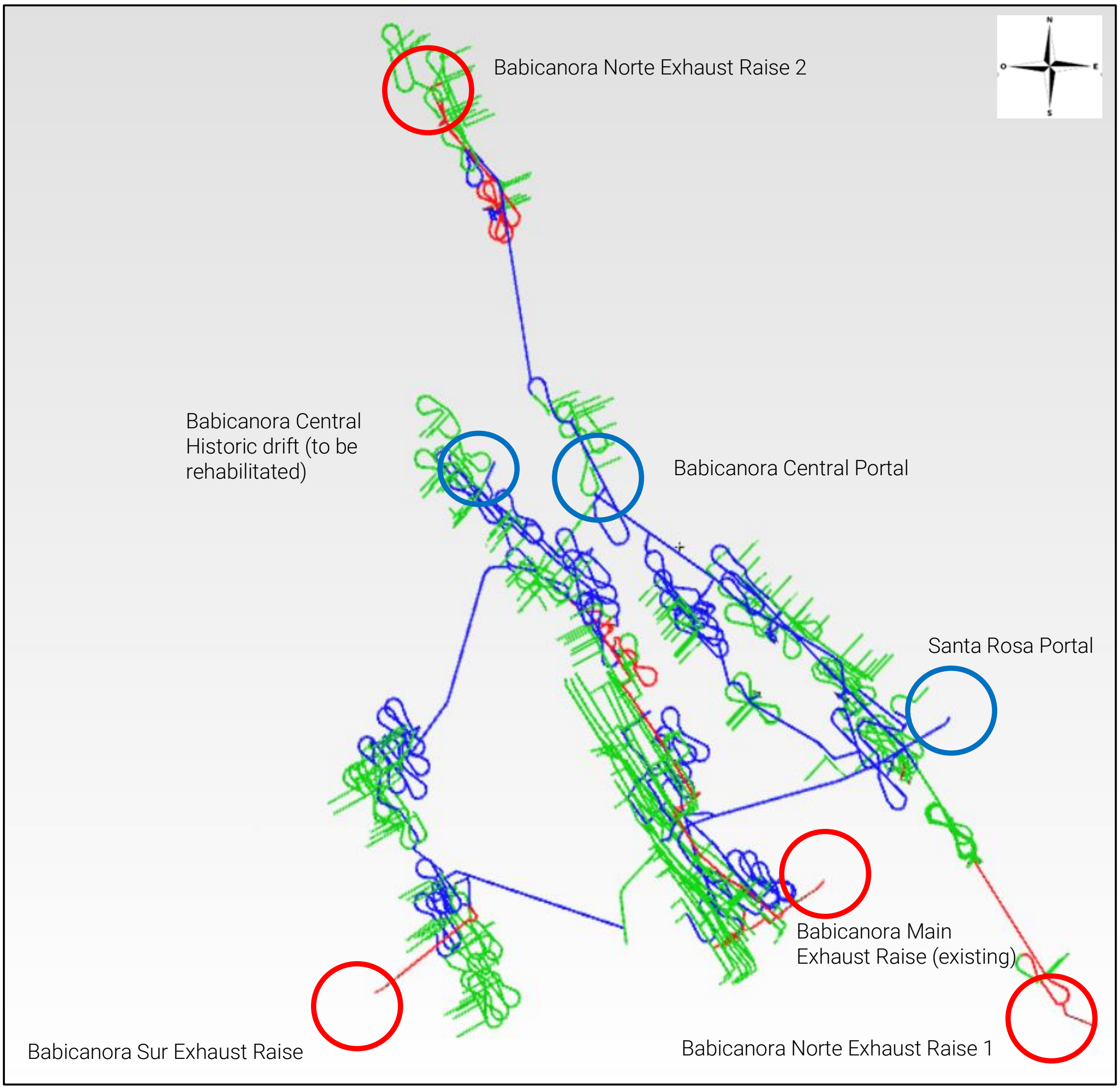


Note: Figure prepared by GMS, 2020.

Figure 16-65: Schematic Showing Mining Area Location

16.9.1.1.1 Intake and Exhaust System

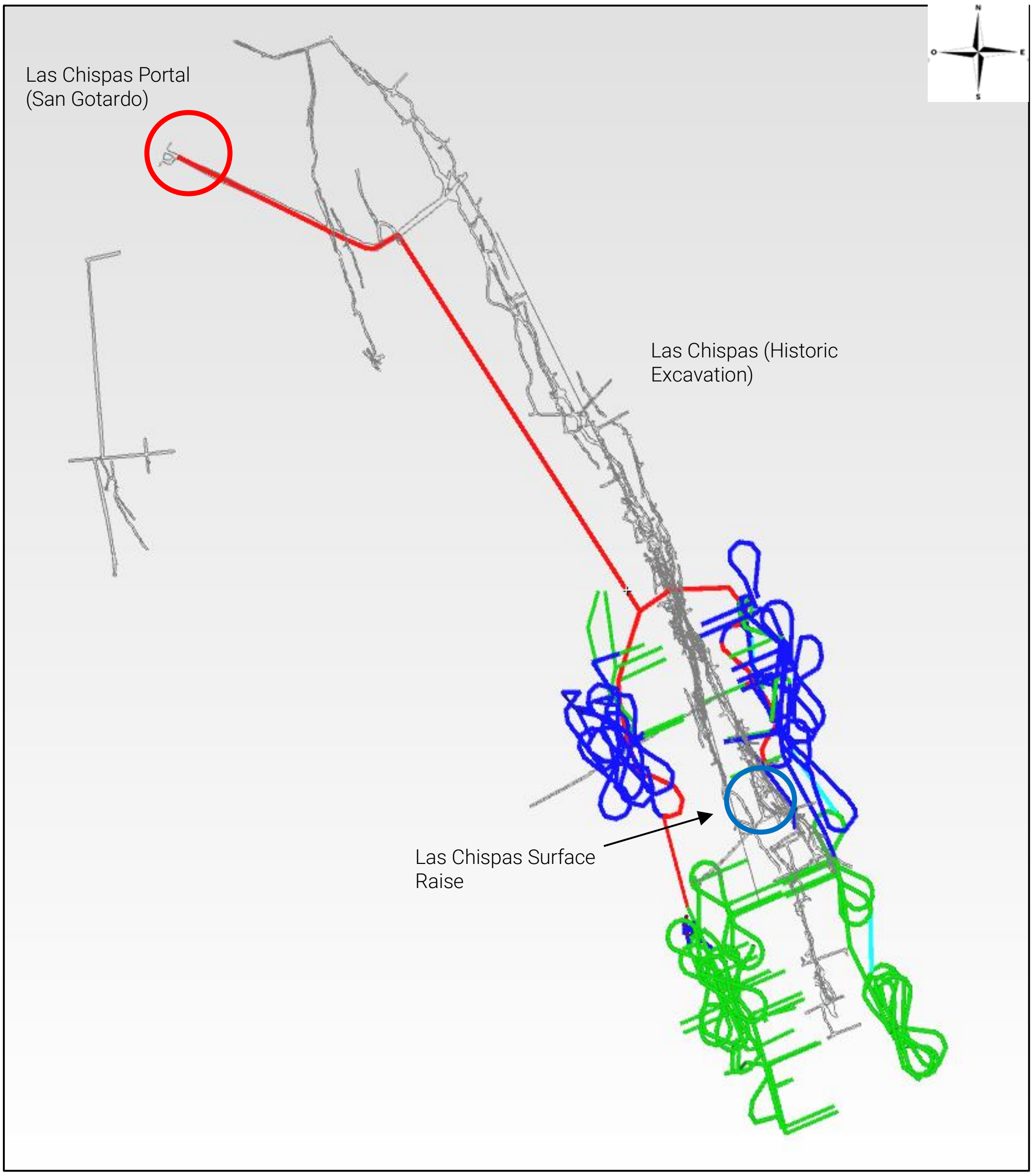
In the Central, Main, Norte-Vista and Sur Zones, the intake system will consist of three portals (blue circle on Figure 16-66). The exhaust system (red circle) will comprise four raises (3 m diameter) which will pull the air from the portal.



Note: Figure prepared by GMS, 2020.

Figure 16-66: Plan View of the Ventilation Network Schematic for the: Central, Main, Norte-Vista and Sur Zones showing Intake and Exhaust System

At Las Chispas, the ramp portal will be the exhaust and the raise will be the intake (Figure 16-67). The ventilation network will take advantage of the historical accesses by using some excavations as ventilation openings (e.g. intake raise from surface). However, the network will have to avoid the former production levels, stope openings and sublevels to avoid short-circuits and leakages.



Note: Figure prepared by GMS, 2020.

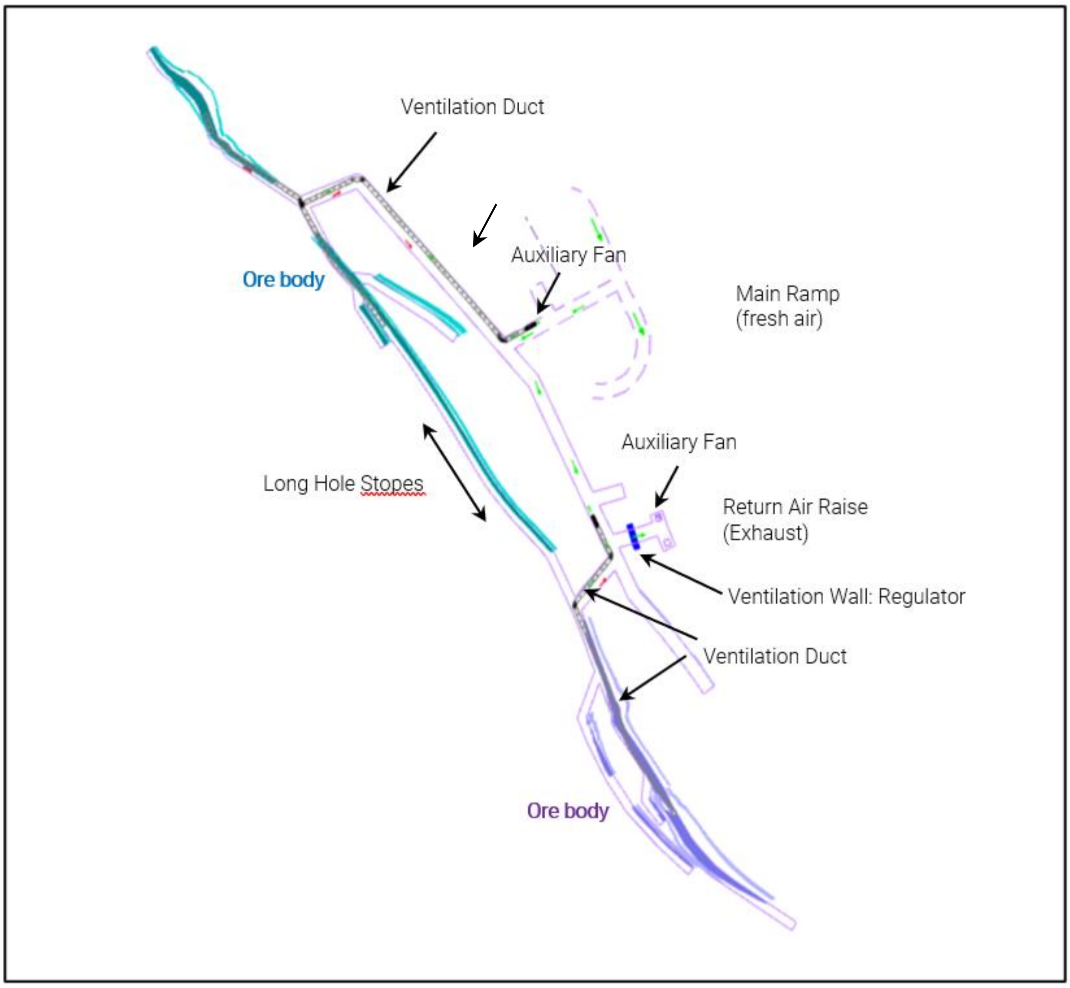
Figure 16-67: Plan View Schematic of the Ventilation Network for the Las Chispas Zones showing Intake and Exhaust Systems

The main objective of the ventilation network is to pull the entire required flow (670 kcfm) throughout the four extraction raises. Over each of these four raises, fan systems will be installed to meet the variable demands depending on the LOM. The ventilation modelling software *Venstim*™ was used to determine the total mine pressure at every phase of the LOM and to calculate the main fan duty points.

A second network objective is to generate an overall air volume for diesel equipment working in a given area. This amount of air will cumulatively dilute contaminants. This air volume must correspond to the ventilation rate required to dilute the exhaust gases described in Table 16-15.

16.9.1.1.2 Level Ventilation

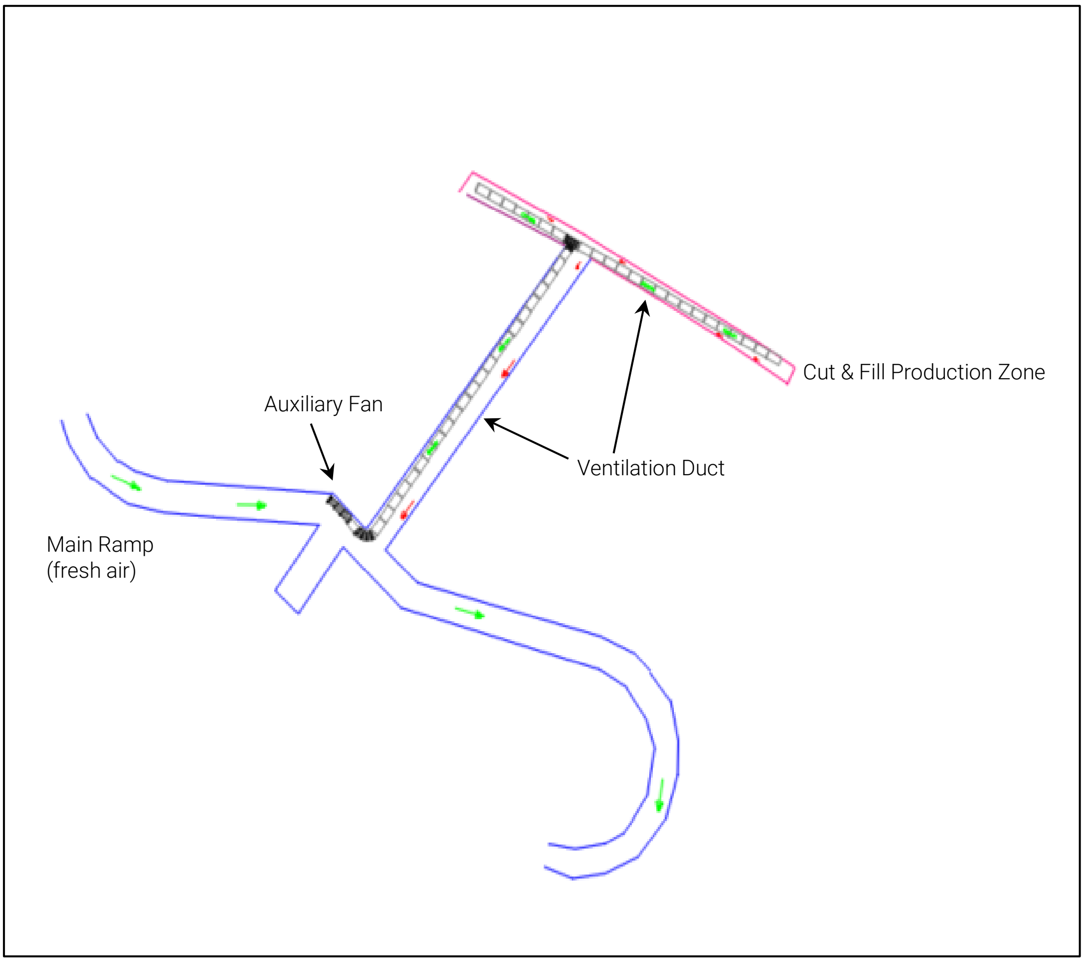
There are two ways to ventilate a level depending on whether a long-hole or cut-and-fill method is used. Figure 16-68 is a schematic showing a general production level ventilation arrangement for the Babicanora Main area. The objective for a typical long hole production level is to create a flow-through from the fresh air (ramp) to the return raise (exhaust).



Note: Figure prepared by GMS, 2020.

Figure 16-68: Ventilation Arrangement Schematic for a Typical Long Hole Production Level

Figure 16-69 is a schematic showing a general arrangement for a cut-and-fill production level. This schematic is applicable to the Babicanora Central–Vista, Babicanora Norte, Babicanora Sur, and Las Chispas Zones.

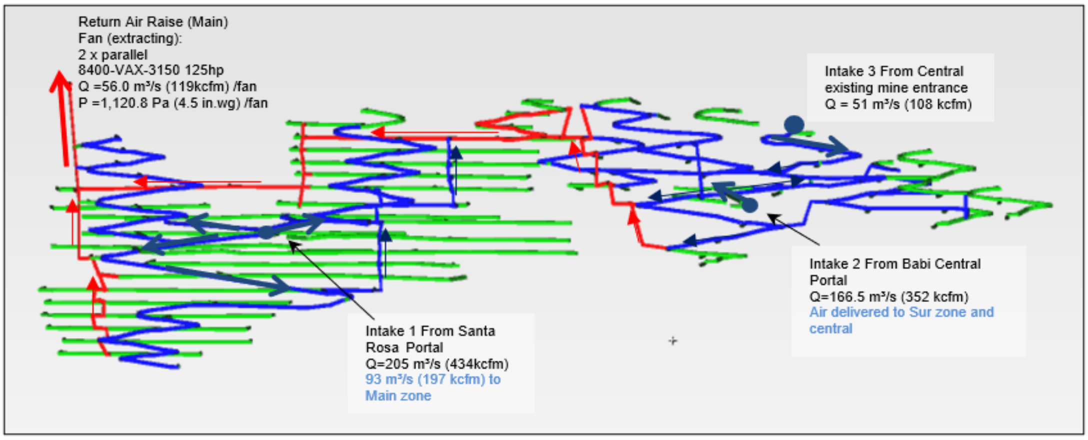


Note: Figure prepared by GMS, 2020.

Figure 16-69: Ventilation Arrangement Schematic for a Typical Cut-and-Fill Production Level

16.9.1.1.3 Babicanora Main Ventilation Network

The Babicanora Main ventilation network will use fresh air derived from the Santa Rosa portal (Figure 16-70) and fresh air coming from the existing mine entrance in the central area (intake 3). This air will be pulled out by the return air raise. This raise is currently excavated. It is planned to extend the raise as the mine deepens.



Note: Figure prepared by GMS, 2020.

Figure 16-70: Babicanora Main Vein with Babicanora Central Zone, Ventilation Network Schematic (longitudinal view)

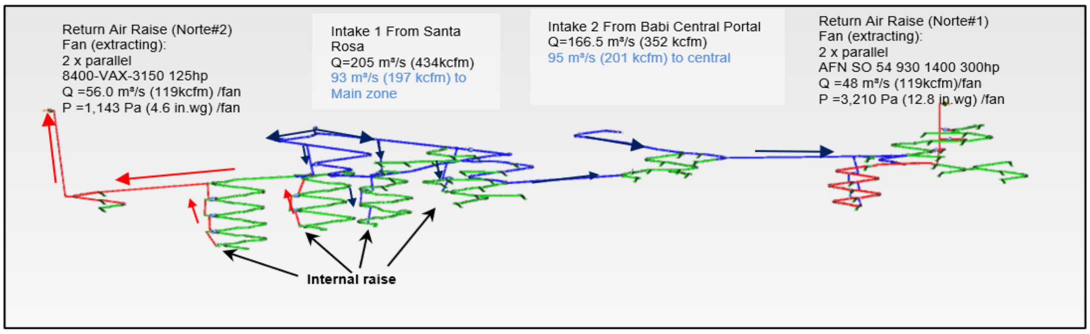
16.9.1.1.4 Babicanora Norte, Ventilation Network

The ventilation network for the Central zone will use fresh air from the Santa Rosa and Babicanora Central ramps. A return air raise located to the north will be used for the north sector of the central zone. The babicanora Central Portal and the extraction raise located to the south will be used for the south sector.

Each internal ramp will be connected to an internal raise on which a fan can be installed as operation progresses. This will ensure that the mine will have a constant flow in the ramp to be able to make any required modifications to the typical production level arrangement that was outlined in Figure 16-69.

Doors and controls for this type of network must be installed in certain areas to keep the circuits without recirculation.

A schematic illustration of the proposed ventilation network is provided in Figure 16-71.



Note: Figure prepared by GMS, 2020.

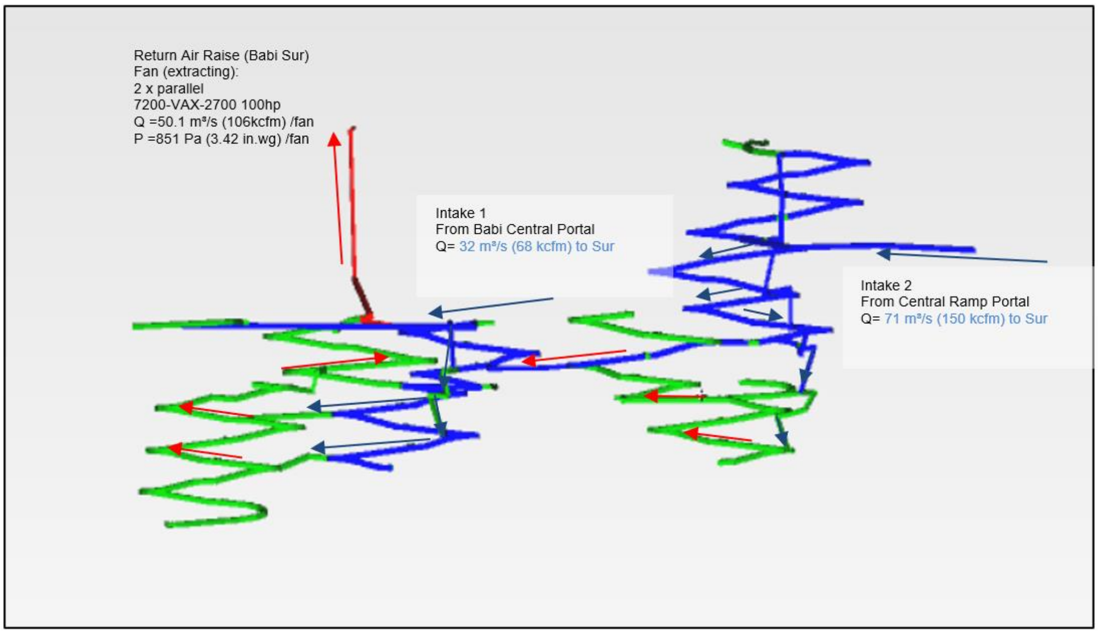
Figure 16-71: Norte Zone, Ventilation Network Schematic (longitudinal view)

16.9.1.1.5 Babicanora Sur Ventilation Network

Fresh air will originate from two transfer galleries. This zone will have its own return air raise.

Like the other zones where the cut and fill mining method will be primarily used, the network will consist of a high volume of air in the ramp.

A schematic illustration of the proposed ventilation network is provided in Figure 16-72.

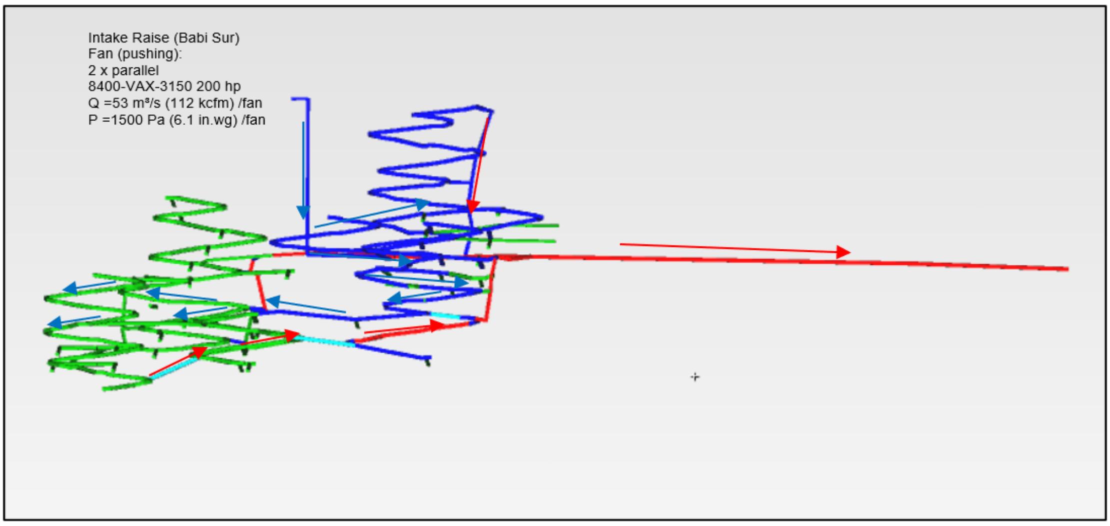


Note: Figure prepared by GMS, 2020.

Figure 16-72: Babicanora Sur Zone, Ventilation Network Schematic (longitudinal view)

16.9.1.1.6 Las Chispas Ventilation Network

The Las Chispas has area will have its own fresh air raise in positive pressure established to 210 kcfm. The air exhaust is the ramp shown in the schematic presented as Figure 16-73.



Note: Figure prepared by GMS, 2020.

Figure 16-73: Las Chispas Area, Ventilation Network Schematic (longitudinal view)

16.9.2 Water Supply

Service water will be non-potable recycled water pumped to the underground mine portals from surface. Pressure reducing valves will be installed in the main water supply line at every 150 m of decline advance to maintain the static head pressure within the rated working pressures of the pipe. The main water supply lines will run along the length of the main declines with valves at every level entrance to provide water to all working area.

16.9.3 Power

The underground power distribution follows a 4.16 kV power supply distributed through the three mine portals (Santa Rosa, Babicanora Central and San Gotardo) via 5 kV cables. Voltage drop in feeders and branch circuits will be limited to 3% as mobile substation of 1 MVA will be located at every 350 m. Mobile substations will be standardized as a step-down dry type of transformer converting 4.16 kV (distribution voltage) to 480 V (equipment utilization voltage). Located 5 kV isolation fault interrupter switches will be required for the maintenance of power feeder cables and mobile substation without interrupting downstream equipment operation or causing major underground power outages.

For lateral distribution, the required power feeder cables (mine cable) will be routed through the main ramp decline via permanently installed overhead messenger cables. Vertical cable installation (bore hole cable) could be used to reduce cable length and to maintain the overall voltage drop within 3%.

The demand load for underground activities is estimated at a peak of 2,675 kW in 2025 as showed in Table 16-16.

Table 16-16: LOM Load Demand by Mining Area

Project Development	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
BABI_SUR (kw)	0	0	127	342	439	387	396	396	515	0	0
BABI_MAIN (kw)	256	464	641	566	568	531	504	504	499	201	105
BABI_CENTRAL (kw)	167	188	313	335	379	419	381	381	381	326	299
LAS_CHISPAS (kw)	0	0	0	0	277	487	620	697	804	645	560
BABI_VISTA (kw)	0	202	219	261	285	303	18	18	0	0	0
BABI_NORTE (kw)	202	202	351	375	503	548	461	355	432	339	0
Total DRAW (kW)	**625**	**1054**	**1652**	**1880**	**2451**	**2675**	**2379**	**2351**	**2631**	**1511**	**964**
Total CONNECTED (kW)	**1450**	**2494**	**3903**	**5236**	**5827**	**6416**	**5697**	**5649**	**6267**	**3552**	**2330**

Underground power distribution single line diagrams as well as underground power load schematics were developed to explain the proposed electrical distribution system.

16.9.4 Dewatering

A hydrogeological analysis was conducted by HRI (section 16.3). The inflow in all the future mining areas is predicted to be negligible to non-existent, except for Las Chispas, which is the only area that will reach the water table (900 masl). This area currently has a dewatering pump installed at the bottom of a ventilation raise that will be used throughout the development of this zone and will support the projected peak water flow of 9.4 L/s.

The main pumping duty throughout the different mining areas will be to remove mine process water. The primary pump system requirements for each area were based on the water consumption of the proposed mining fleet as well as the water consumption required to carry out the various development activities. The secondary pump system will consist of submersible centrifugal sump pumps with a rated capacity of 15 kW.

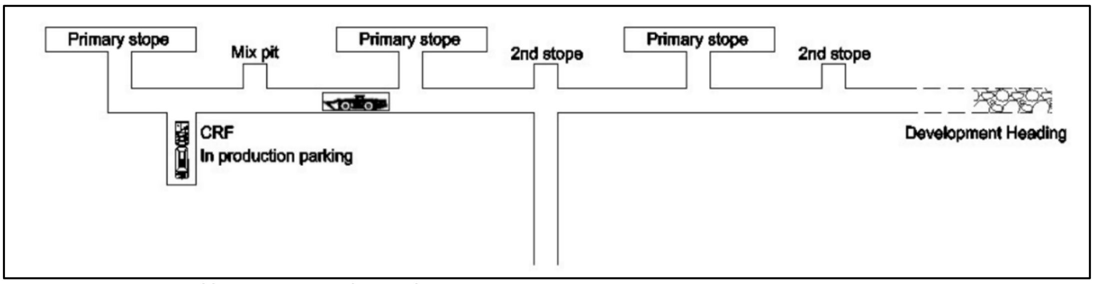
Each area will have its own dewatering system. Water will be drained via gravity through a series of borehole-connecting sumps on each zone level. The water collected in these sumps will be directed down to settling sumps located next to the permanent pump station. Each area will have at least one main pump station that will pump the water back to surface and directed to the mill. The pump system design details are provided in Table 16-17.

Table 16-17: Dewatering Pump Design Details

Mining Area	Pump Station #	Head (m)	Pump (kw)	Peak flow (L/s)	Capacity Flow @ Head
Babicanora Main	1	225		8.8	
Babicanora Central	1	75		5.5	
Babicanora Sur	1	350		5.5	
Babi Vista	1	260		4.2	
Babicanora Norte	1	130	60	5.5	13.9 L/s @ 350 m
	2	110		5.5	
Las Chispas	1	150		8.3	
Las Chispas	2	150		8.3	
Las Chispas	3	150		8.3	

16.9.5 Cemented Rockfill

When the use of CRF is required, a portable CRF plant will be used. The cement powder will be transported from the surface to a mobile plant that will be located underground. This mobile plant will produce a cement slurry that will be added to the development waste or to waste rock from the surface directly in a scoop bucket. The scoop will transport and discharge waste rock into the stope. Cement transport and operation of the mobile CRF plant will be performed by a contractor. Figure 16-74 shows an example of a level installation for an underground CRF plant. Given the significant distances between the different areas, two mobile plants are planned in the LOM. It is planned to use a binder percentage of 4% in the worksite where the CRF is required.



Note: Figure prepared by Swatcrete, date unknown.

Figure 16-74: Schematic Showing Level CRF Mobile Plant

16.9.6 Compressed Air

The compressed air supply will be provided by two series of electrical compressors. The first series will be installed at the Santa Rosa portal and the second at the San Gotardo portal. The planned capacity is 4,750 cfm per compressor.

The compressed air piping network will be installed along the ramp, in the main drifts and in the escapeways throughout the mine. Compressed air will provide power the pumps for dewatering development work, to handheld drills (for specific and limited use in planned development, and stope), and provide an emergency air supply to the refuge station.

16.9.7 Fuel Storage and Distribution

Fuel will be stored on surface. There will be no distribution system of fuel in the underground mine. A fuel truck is planned in the contractor's equipment to distribute the fuel to underground equipment that cannot travel quickly to the surface for refuelling.

16.9.8 Communications

An underground network with leaky feeder radio communication systems is already in place on site and will be expanded over the LOM.

Mobile equipment operators, light vehicles, and supervisors will be equipped with handheld radios to communicate with personnel on surface.

16.9.9 Explosives Storage and Handling

Explosives will not be stored in the underground mine. The explosive storage is already installed on surface. The blasting crews will haul the estimated quantities of explosives required for each shift using explosives transport vehicles and deliver those explosives to working faces and explosives-loading equipment underground.

16.9.10 Personnel and Underground Material Transportation

Supplies and personnel will access the underground via the main access drift. A series of personnel carriers such as access land cruisers will be used to transport workers in the underground mine from surface to the underground. Supervisors, engineers, geologists will use diesel-powered all-terrain vehicles for transportation underground. Mechanical and electrical personnel will use maintenance land cruisers. A flat bed with a service boom will be used to move supplies from the surface to the underground active heading/stope.

16.9.11 Underground Construction

Underground construction will be limited to the construction required for ventilation (wall and installation) and dewatering systems. This work is planned to be done by the contractor.

16.9.12 Equipment Maintenance

All major mechanical maintenance will be performed on surface at the workshop. Only minor maintenance and emergency work will be performed underground by mobile maintenance crews. The existing surface workshop has sufficient warehouse storage for operational requirements.

16.10 Safety Measures

16.10.1 Emergency Exits

The main ramp is planned to provide primary egress from the underground workings. The ventilation raises will be equipped with manways, and many of the drift connections between different zones will serve as second egress. All ventilation raise-safety egresses will be equipped with prefabricated modular manway systems.

16.10.2 Refuge Stations

Refuge stations will be positioned in a way that an employee will need 30 minutes or less to access the refuge from the moment they leave the workplace. At Las Chispas, engineered mobile refuge stations will be used (Figure 16-75). One refuge per mining area is planned for the LOM. Each refuge station will be equipped with the following:

- Telephone or radio to surface, independent of mine power supply;
- Compressed air, water lines and water supply;
- Emergency lightning;
- Hand tools and sealing material; and,
- Plan of underground work showing all exits and the ventilation plans.

Note: Figure from Drager.com (Drager MRC 5000 – Mine Refuge Chamber)

Figure 16-75: Typical Underground Portable Refuge Station

16.10.3 Fire Protection

Underground mobile vehicles will be equipped with automatic fire suppression systems in accordance with regulations. Fire extinguishers will be provided and maintained in accordance with regulations and best practices at the electrical installations, pump stations, service garages and wherever a fire hazard exists. Every vehicle will carry at least one fire extinguisher of adequate size and proper type.

16.10.4 Mine Rescue

Fully trained and equipped mine rescue teams will be established in accordance with regulations. Mine rescue equipment and a foam generator will be located on site. The mine rescue teams will be trained for surface and underground emergencies. An Emergency Response Plan was developed, will be kept up to date, and will be used in the event of an emergency.

16.10.5 Emergency Stench System

A mine stench gas warning system has been installed in both operating portals and will be expanded through the LOM (temporary and permanent system). Another mine stench gas warning system will be installed at the mine compressed air system as a second mean to alert underground workers in the event of an emergency.

17 Recovery Methods

17.1 Process Design

Based on the metallurgical testing results discussed in Section 13, Ausenco's design expertise, and experience from local operations treating similar types of mineralized material, the planned flowsheet, which is designed for treatment of a variety of grades, is flexible and robust. The flowsheet is based on well-proven unit operations in the industry and there are no unique or novel processing methods required for gold and silver recovery.

The key project design criteria for the plant are:

- Major equipment designed for nominal throughput of 1,250 t/d with the ability to accommodate increased throughput up to 1,750 t/d via an expansion. Future expansion of the facility can be achieved through addition of a ball mill and a pebble crusher to the grinding circuit;
- Crushing circuit availability of 70%, supported by the use of a surge bin, a dedicated feeder and an emergency stockpile to provide continuous feed to the balance of the process plant; and.
- Process flowsheet including semi-autogenous grinding (SAG), flotation, independent cyanide leaching circuits for both flotation concentrate and tailings streams, Merrill Crowe circuit, and tailings handling facilities, with an overall availability of 91.3%, given:
  - Axb of 41 and BWI of 19.4 kwh/t;
  - Design head grades of 8 gpt Au and 800 gpt Au with the ability to handle peak head grades of as much as 13 gpt Au and 1,300 gpt Ag; and,
  - Overall process recovery of 97.6% gold and 94.3% silver, given the LOM average grades.

The total operating power for the process plant will be 4.6 MW.

The process plant will be located at the mine site and will receive blended feed material from a number of different mineralized veins. Due to the anticipated variability in grade and clay content of the disparate veins within the deposit, there may be operational periods in which blending of mineralized materials from selected mining areas is the preferred means to mitigate potentially adverse effects of high clay content or exceedingly high grade feeds on plant operability or process recoveries.

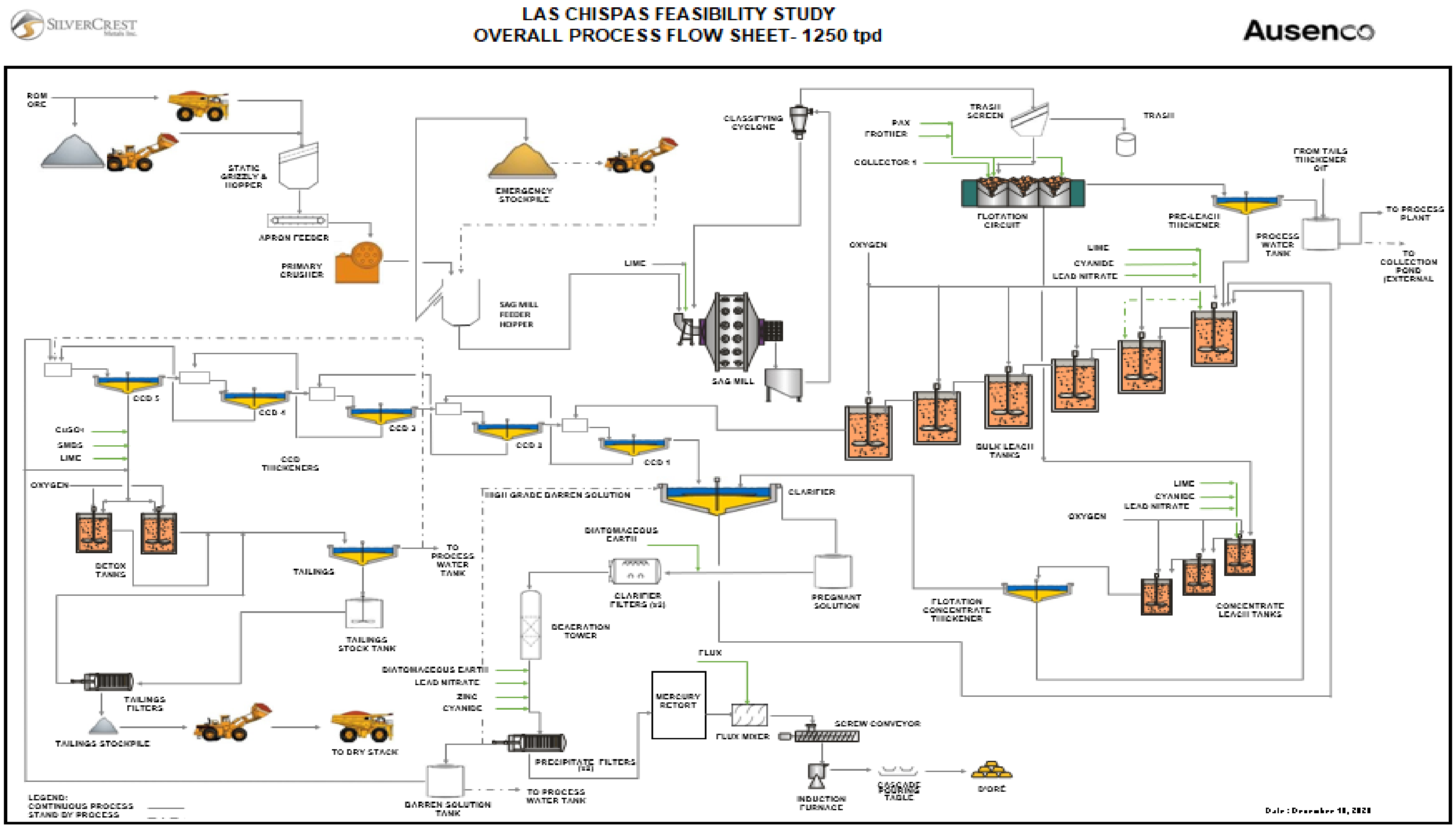
17.1.1 Selected Process flowsheet

The overall flowsheet includes the following steps:

- Primary crushing;
- Grinding (single stage SAG mill circuit closed with cyclones for classification);
- Bulk rougher flotation;
- Cyanide leach of flotation concentrate;

- Flotation concentrate thickening;
- Flotation tails pre-leach thickening;
- Cyanide leach of flotation tails;
- Countercurrent decantation (CCD) washing and pre-clarification of pregnant solution;
- Clarification, de-aeration and zinc precipitation (Merrill-Crowe);
- Mercury removal using a retort;
- Smelting to produce doré;
- Cyanide destruction of tailings;
- Tailings thickening and filtration; and,
- Transferring tails to the FTSF.

Figure 17-1 presents an overall process flow diagrams depicting the major unit operations.



Flowsheet designed by Ausenco

Figure 17-1: Overall Process Flow Diagram

The process plant will use a conventional comminution circuit, including a primary jaw crusher and a SAG mill, to reduce the feed material to the target particle size distribution for processing. The SAG mill is designed to operate in closed circuit with a cyclone cluster with the cyclone underflow returned to the SAG mill and cyclone overflow forwarded to a trash screen. Cyclone overflow will be fed to a bulk rougher flotation circuit to recover the bulk of the gold and silver in the concentrate. The flotation concentrate will be leached in a high concentration cyanide solution, in stirred tanks, to dissolve gold and silver. The leached concentrate slurry will be thickened in a thickener, where the thickener overflow is a pregnant solution for transfer to the Merrill Crowe circuit and thickener underflow is the leached residue for transfer to the bulk leach circuit for additional leaching and recycle of the unconsumed cyanide.

Rougher flotation tailings will be thickened in a pre-leach thickener prior to transferring to the bulk leach circuit. In the bulk leach circuit, gold and silver values in the flotation tailings and in the concentrate leach residue will be recovered by conventional cyanide leaching in stirred tanks.

Discharge slurry from the bulk leach will be washed in a five-stage CCD circuit and then clarified to remove fine solids. The resulting clarified solution will be deaerated and treated in a Merrill-Crowe process, which employs addition of zinc powder to precipitate gold and silver. The precious metals precipitate will be filtered and treated in a retort furnace for mercury removal and then smelted on site to produce gold–silver doré bars. The captured mercury will be collected and disposed of safely off site.

Underflow slurry from the fifth wash thickener will be treated with sulphur dioxide (added as sodium metabisulphite) and oxygen in parallel agitated tanks to detoxify residual cyanide in the leach residue. Following cyanide detoxification, the tailings slurry will be thickened and filtered, and the filter cakes will be transferred to the FTSF for impoundment.

17.1.2 Key Process Design Criteria

The key process design criteria listed in Table 17-1 form the basis of the process flowsheet design and selection of mechanical equipment.

Table 17-1: Process Design Criteria

Parameter	Unit	Value
Plant Throughput	t/d	1,250
Head Grade–nominal	gpt Au	8
	gpt Ag	800
Head Grade–design	gpt Au	11
	gpt Ag	1,100
Head Grade- peak	gpt Au	13
	gpt Ag	1,300
Overall Recovery (From mineralized material to doré)–Au	%	97.6
Overall Recovery (From mineralized material to doré)–Ag	%	94.3

Parameter	Unit	Value
Plant Availability	%	91.3
SMC- Axb-design (75th percentile)	kWh/t	41
Bond Ball Mill Work Index–design (75th percentile)	kWh/t	19.4
Flotation Concentrate Mass Pull	%	2
Flotation Concentrate Leach Residence Time	h	96
Concentrate Leach Slurry Density	% solids (w/w)	15
Concentrate Thickener Solid Loading	$t/m^2.h$	0.1
Pre-Leach Thickener Solid Loading	$t/m^2.h$	0.6
Bulk Leach Residence Time	h	96
Bulk Leach Slurry Density	% solids (w/w)	48
Total Sodium Cyanide Consumption Rate (design)	kg/t	3.5
Total Quick lime Consumption Rate (design)	kg/t	1.76
Number of CCD Wash Stages		5
CCD Wash Efficiency	%	99.7
Metal Recovery Method		Merrill Crowe
Cyanide Detoxification Method		SO_2 / Air
Tailings Thickener Solid Loading	$t/m^2.h$	0.6
Tails Filter Cake Moisture	%	18
Tailings Management	–	Dry stacking

17.1.2.1 Comminution

Design parameters for the comminution circuit were sourced from testwork completed at SGS during 2019–2020. Ausenco's review of the mineralized material breakage characterization data resulted in the following observations and conclusions:

- On average, the mineralization can be characterized as moderate mineralized material competence, with the exception of Area 51. The mineralization from this area has significant amounts of clays and testwork indicated higher Axb values (less competent);
- Significant variability in hardness is evident amongst the tested samples. Most of the samples showed a Bwi >16 kWh/t, which is considered very hard; and,
- Significant variability in mineralized material abrasiveness was observed. On average, the mineralized material can be classified as very abrasive (Ai > 0.5), and above average wear rates and media consumption are anticipated.

17.1.2.2 Flotation Circuit

A flotation circuit was included in the process design to generate a high-grade concentrate, with a 2% mass pull, which would be leached under aggressive conditions. The remaining gold and silver will be leached from flotation tails under more conventional leaching conditions in the downstream bulk leach circuit. Addition of the flotation circuit and separate cyanide leach circuits for the concentrate and tailings will provide more flexibility and robustness in processing of different grades.

For the purposes of flotation cell sizing, and downstream equipment specification, Metso: Outotec (MO Group) completed a simulation using the testwork data generated by SGS in 2019–2020. The circuit was sized based on:

- 57 t/hr at feed grades of 600 gpt Ag and 6.15 gpt Au, and with 2% concentrate mass pull.

The flotation circuit configuration for 57 t/hr, is three TankCell e5 cells with internal launders. Four flotation TankCells will be installed and the first flotation cell will be used as a conditioning tank for the nominal throughput at 57 t/hr.

The reagent addition scheme was selected based on the metallurgical testwork completed at SGS Lakefield in 2019 and 2020.

17.1.2.3 Cyanide Leach and Pregnant Leach Solution Recovery

Design parameters for the leach circuit design were sourced from testwork completed at SGS during 2019–2020

A number of trade-off studies were completed by Ausenco to determine the preferred circuit configurations for the leach circuit and recovery of precious metals. These included:

- Bulk leach residence times of 48, 72 and 96 hr;
- Number of wash stages using three, four, and five CCDs;
- CCD thickeners sizes at 12 m versus 14 m; and,
- CCD circuit versus filters.

A combination of a 96-hr leach with five CCD stages provided the highest net present value (NPV).

Reducing the size of thickeners from 14 m to 12 m resulted in minor capital savings and the 14 m option was retained to add robustness given the presence of clays.

Installation of filters instead of CCDs resulted in a smaller plant footprint, but resulted in higher capital and operating cost, increased complexity of operation and maintenance, with reduced economic benefits, therefore the CCD was retained for the flowsheet.

17.1.2.4 Thickening and Filtration

Pre-leach thickener, CCD thickeners, tails thickener and filtration parameters were sourced from testwork completed by Outotec in January 2020, using three (3) master composites of the relevant material streams provided by SGS. The SGS 2019–2020 testwork program included similar testwork on three master composite samples representing materials corresponding to Area 51, Master Comp 1, and Master Comp 3, and selected variability samples.

No thickening or filtration testwork on flotation concentrate was completed as part of the Feasibility Study. All values are based on similar projects from Ausenco's internal databases and/or vendor recommendation. Flotation concentrates from Las Chispas ore, which are largely without significant clay components, are expected to settle predictably and in line with commercial processing of similar ores at comparable facilities in Mexico.

The target moisture content of the tails filter cake was dictated by testwork performed by Wood.

17.1.2.5 Merrill Crowe Circuit

The Merrill Crowe circuit is designed for a pregnant solution feed rate up to 366 m^3/hr and peak head grades of 1,300 gpt Ag and 13 gpt Au.

17.1.2.6 Other Testwork

Cyanide detoxification circuit configuration and reagent consumptions were designed based on Ausenco's design expertise and experience from successful installation in similar operations, and the metallurgical testwork completed at SGS Lakefield in 2019–2020.

17.2 Unit Process Description

17.2.1 Crushing Area

A conventional jaw crusher was selected to reduce the feed material particle size to P80 of 63 mm, suitable for feeding a single stage SAG mill. The nominal feed throughput of the crushing circuit is approximately 74 t/hr, at 70% availability.

The crushing circuit major equipment will include:

- Static grizzly and hopper;
- Apron feeder;
- Jaw crusher (75 kW);
- Surge bin;
- Belt feeder to reclaim crushed material to feed the SAG mill;
- Emergency stockpile and reclaim; and,
- Associated material handling systems (conveyors, weightometers and tramp magnet).

Run-of-mine (ROM) mineralized material will be trucked from the underground mine either to the ROM pad stockpile or directly onto the static grizzly hopper. ROM mineralized material from the stockpile will be reclaimed using front-end loaders and dumped into the static grizzly hopper. The jaw crusher will be a Metso C80 with a closed side setting (CSS) of 80 mm and will crush the ROM mineralized material from F80 of 159 mm to P80 of 63 mm. The crushed mineralized material will be conveyed to the surge bin via the primary crusher product conveyor. A tramp metal magnet will be installed at the head end of this conveyor to remove tramp. The tramp metal will be manually removed as needed.

The surge bin will have a live capacity of 10 minutes for 9.5 t of storage.

Surge bin overflow will be transferred to an emergency stockpile via the emergency stockpile conveyor and reclaimed from the stockpile using a front-end loader when required. The emergency stockpile will provide 16 hr of storage given a plant feed rate of 57 t/hr.

Crushed mineralized material will be reclaimed via a belt feeder beneath the surge bin and conveyed to the SAG mill feed chute by the SAG mill feed conveyor.

A freshwater line is available for dust suppression in the crushing area if required.

17.2.2 Grinding Circuit

A conventional SAG mill, arranged in closed circuit with a cyclone cluster, was selected to reduce the mineralized material from a F80 of 63 mm to P80 of 100 µm. The nominal feed throughput of the grinding circuit is approximately 57 t/hr, based on 91.3% availability.

The grinding circuit will include:

- One SAG mill, 6.1 m (20 ft) in diameter by 3.66 m (12 ft) in length, powered by a 2,000-kW variable speed drive motor;
- Two 55 kW slurry pumps to pump SAG mill discharge to cyclones, with one pump in operation and one in standby;
- One cyclone cluster with ten 250 mm cyclones, eight in operation and two in standby; and,
- Associated material handling and storage systems (sump pumps, pump boxes, bins).

Crushed mineralized material will be reclaimed from the surge bin onto the SAG mill feed conveyor and discharged into the feed chute of the SAG mill. The SAG mill will be a grate discharge type mill. The grate aperture will be 15 mm and will have no pebble ports, so there will be no recycle of pebbles. Provisions were made in the plant layout to allow the installation of a ball-mill, the retrofit of conveyors and a pebble crusher in a potential expansion case.

The SAG mill product will discharge onto a trommel screen. Trommel screen undersize will report to a cyclone feed pumpbox and the oversize to a scats bunker. Process water will be added to the SAG mill feed chute and cyclone feed pumpbox to maintain a target mill discharge slurry solids density of 70%. The cyclone cluster will be fed at a nominal rate of 228 t/hr to separate the coarse and fine particles in the SAG mill trommel screen undersize. The cyclone underflow will return to the SAG mill feed. The nominal circulating load is 400%. The cyclone overflow with a particle size of P80 of 100 µm will report to the bulk rougher flotation circuit after flowing through a trash screen.

A vertical cantilevered centrifugal sump pump will service the area. Grinding media for the SAG mill will be introduced by use of a dedicated kibble and a grinding building jib crane.

17.2.3 Bulk Rougher Flotation

The bulk rougher flotation circuit was designed to generate a small amount of concentrate, about 2% mass pull, that will contain a significant portion of the gold and silver from the ore. The high-grade flotation concentrate will be leached in a concentrate leach circuit with high-intensity cyanide conditions to dissolve the gold and silver from the flotation concentrate into a small stream of high-grade pregnant solution. The produced pregnant solution will be further processed in a Merrill Crowe circuit in conjunction with the pregnant solution recovered from the flotation tails (bulk) leach circuit.

The flotation circuit will include:

- One trash screen; and,
- Four 5 m^3 forced-air tank cells, arranged in series.

Cyclone overflow will gravitate over the vibrating trash screen, to remove foreign material prior to flotation. Trash will report to the trash bunker which will be periodically removed for emptying. Screen undersize will gravitate to the first flotation cell which will be used as conditioning tank.

Pax and Aeroflot 208 will be added as preferred collectors, respectively in the first and third flotation cells using dedicated pumps. Frother will be also be dosed into the first and third flotation cells using dedicated pumps.

Fine gold and silver associated with sulphides will be floated in the bulk rougher circuit and flow by gravity to the concentrate leach feed pumbox via overflow launder. The testwork showed that the flotation concentrate has typical particle size of about 30 µm and no regrinding step is required prior to concentrate leaching. Provisions were made in the plant layout to allow the retrofit of a regrinding mill circuit if required.

The flotation tailings will flow by gravity to flotation tails pumpbox and will be thickened in a pre-leach thickener prior to leaching in the flotation tails (bulk) leach circuit.

On average recovery of 62% and 64% of gold and silver respectively has been estimated to the flotation concentrate.

17.2.4 Gold and Silver Recovery from Flotation Concentrate

The concentrate leaching circuit will leach precious metal values from the flotation concentrate in a series of stirred tanks and the corresponding leach residue will be thickened in a thickener. Thickener overflow, a pregnant solution containing high concentrations dissolved precious metals, will be pumped to a pre-clarifier for removal of fine suspended solids before being introduced to the Merrill Crowe circuit. Thickener underflow pumps will transfer the concentrate leach residue to bulk leach circuit for further extraction of the gold and silver, and for recovery of the unconsumed cyanide.

17.2.4.1 Flotation Concentrate Cyanide Leaching

The flotation concentrate will be leached in three mechanically agitated leach tanks operating in series.

The nominal feed rate of flotation concentrate to the corresponding cyanide leaching circuit will be 1.14 t/hr.

Flotation concentrate leaching will include:

- Three 6.8 m diameter x 6.8 m high leach tanks;
- One 5 m diameter high-rate thickener;

- Associated material handling and storage systems (agitators, pumps, sump pumps, pump boxes).

The concentrate leaching circuit will operate continuously; leaching reagents, including sodium cyanide and lead nitrate will be added to facilitate gold and silver extractions. The operating pH of the leach circuit will be maintained between 10.5 and 11.0 with additions of milk of lime, which will be produced in a lime slaking plant located at site, to drive leach kinetics, to limit corrosion and to prevent the loss of cyanide to gaseous hydrogen cyanide.

Gold and silver leaching will occur in a series of three tanks, providing over 96 hours of total residence time. Each leach tank will have a live volume of 229 m³, providing 32 hours retention. The target solids concentration for the leach circuit will be 15% w/w; barren solution will be added to concentrate leach feed pumpbox to achieve the desired leach density. The estimated concentrate leach circuit extraction (from concentrate) will be about 99% for gold and about 98% for silver.

Leach discharge slurry will gravitate to a thickener where it will be mixed with diluted flocculant to increase particle settling rate. The concentrate leach thickener will be a 5 m diameter high-rate thickener, which will increase the solids density of the concentrate materials to a target of 30% w/w. Thickener underflow slurry will be pumped to the feed box ahead of the bulk leach circuit using a concentrate leach thickener underflow pump. Thickener overflow, containing the bulk of the precious metal values, will be pumped to a pre-clarifier.

Oxygen will be produced at site using vacuum swing adsorption (VSA) technology. Oxygen will be supplied from the oxygen plant, as required, and delivered to the leaching circuit via the tank agitator shafts. The dissolved oxygen will be maintained at the range of 20–30 mg/L in the circuit.

To allow for maintenance of individual concentrate leach tanks, the circuit will be configured with a provision which allows for slurry to bypass any single leach tank and report directly to the subsequent leach tank, such that one tank can be removed for service without requiring the entire circuit to be stopped.

17.2.5 Gold and Silver Recovery from Flotation Tailings

Gold and silver contained in the flotation tailings solids will be extracted in the flotation tails (bulk) leaching circuit. Residue from the concentrate leach will be transferred to the bulk leaching circuit for further leaching and consumption of the free cyanide.

17.2.5.1 Flotation Tails (Bulk) Cyanide Leaching

Bulk flotation tails will be thickened and subsequently leached in the bulk leaching circuit, which will consist of six, mechanically agitated leach tanks operating in series. The nominal feed rate to the bulk cyanide leaching circuit will be about 57 t/hr, approximately equal to the plant feed as the concentrate leach residue is recycled back to this circuit.

The bulk cyanide leaching circuit will include:

- One 14 m diameter high rate thickener;
- One leach feed box;
- Six 12 m diameter x 12.7 m high leaching tanks; and,

- Associated material handling and storage systems (agitators, pumps, sump pumps, pump boxes).

The flotation tails at about 30% w/w solids density will be fed to the pre-leach thickener to increase the solids density to about 46% w/w target density prior to feed to the bulk leach circuit.

Sodium cyanide, for gold and silver dissolution, will be added to the leach circuit via cyanide ring main and dosing valves. The primary cyanide dosing point will be the leach feed distribution box, with a further addition point located in each leach tank. Lead nitrate will be added to the leach circuit to reduce the detrimental effect of metallic sulphides and decrease cyanide consumption. Milk of lime will be used to maintain the operating pH of the leach circuit between 10.5 and 11.0. The estimated bulk leach circuit extraction (from flotation tail and concentrate leached residue) will be about 97% for gold and about 90% for silver.

Oxygen will be introduced into the circuit via the leach tank agitator shafts, to maintain the desired oxygen level at 20–30 mg/L in the circuit. The bulk leach circuit will have a 96 hr retention time, equally distributed across the six tanks. Slurry exiting the leach circuit will flow by gravity to the CCD circuit to recover pregnant solution from leached slurry.

The leach circuit will be serviced by a vertical cantilevered centrifugal sump pump, which will return spillage to a nearby leach tank.

17.2.6 CCD Circuit and Pre-Clarifier

A five-stage CCD washing circuit and a pre-clarifier will be used to recover pregnant solution from the cyanide leached slurry. The nominal throughput of the circuit will be approximately 57 t/hr.

The washing circuit will include:

- Five 14 m diameter high rate thickeners;
- One 23 m diameter pre-clarifier;
- Pregnant solution storage tank, with a live volume of 460 m^3; and,
- Associated material handling and storage systems (feed boxes, pumps, sump pumps, pump boxes.

The leached slurry will gravitate to the first CCD thickener and underflow from the first thickener will be fed to the subsequent CCD thickener. The process will repeat until the solids flow reports to the last CCD thickener (CCD No. 5). The underflow of CCD No. 5 will be pumped to a cyanide detoxification circuit as washed tailings. The barren solution from the Merrill Crowe circuit will be added to CCD No. 5 as process wash water; the facility to add process water as a washing solution is included in the design. Overflow solution from the final CCD thickener will flow in a counter current mode to the preceding thickener. The overflow from the first CCD thickener will flow to a pre-clarifier feed box. The recovered pregnant solution from the concentrate leach circuit will combine with the pregnant solution recovered from CCD circuit. Both streams of pregnant solution will be clarified in the pre-clarifier prior to storage in the pregnant solution tank which will feed the Merrill Crowe circuit. The pre-clarifier underflow will be pumped to bulk cyanide leach feed box.

The washing ratio, washing solution volume to feed solution volume, will be 3.5:1 to achieve an overall CCD washing performance efficiency of higher than 99%.

Settling of solids will be aided by the addition of diluted flocculant at each stage of CCD and diluted coagulant to pre-clarifier.

Antiscalant can be added to the pregnant solution tank as required to inhibit scale formation in the Merrill Crowe circuit.

One vertical cantilevered centrifugal sump pump will be provided in the CCD area to return spillage to the circuit.

17.2.7 Merrill Crowe Circuit

Pregnant liquor from the pre-clarifier will be stored in the pregnant solution tank. Clarified pregnant solution will be treated by the Merrill Crowe process, zinc-dust cementation, to recover the contained precious metals. The barren solution will be mainly recycled to the CCD wash circuit as wash solution. The nominal solution feed rate to the Merrill Crowe precipitation circuit will be approximately 300 m^3/hr, although the circuit was designed to treat up to 366 m^3/hr of pregnant liquor.

The Merrill Crowe circuit will be provided as a vendor package, and will include:

- Two rotating disk filters as clarifier filters, each having a 139 m^2 filtration area;
- One de-aeration tower;
- One air/water separator;
- One de-aeration tower vacuum pump;
- One zinc mixing cone, including a hopper and a feeder;
- Two precipitation filter press units, each having a 185 m^2 filtration area;
- One pre-coat preparation tank;
- One body feed preparation tank; and,
- Associated material handling and storage systems (pumps, sump pumps, pump boxes, feed conveyors).

Pregnant solution from the pre-clarifier will be discharged to the pregnant solution tank which will provide about 1.5 hr of surge capacity to cater for the semi-continuous nature of the clarification and precipitation stages in the Merrill Crowe circuit.

A further stage of clarification will be required to reduce the suspended solids content to <5 mg/L for efficient zinc precipitation. The clarifying filter feed pumps will pump the pregnant solution from the pregnant solution tank to clarifying disk filters to remove any residual solids. Two filters will be provided in a duty/standby arrangement. Pre-coat will be required to enhance capture of the fine solids at the start of each cycle. At the end of the filtration cycle, the clarifying filter sludge will be pumped back to the CCD circuit via the clarifying filter sump pump, to minimize any losses of precious metals in the entrained solution.

Filtrate from the clarifying filters will feed the de-aeration tower. Dissolved oxygen will be removed under vacuum by splashing the pregnant solution over tower packings to increase the exposed surface area. De-aeration of the solution will prevent excessive zinc consumption by minimizing side reactions that oxidize zinc.

De-aerated pregnant solution will be contacted with the zinc dust slurry and pumped to the precipitate filters using precipitate filter feed pumps. Zinc dust will be slurried with barren solution in a zinc mixing cone. Cyanide can be added to the process as required, to maintain adequate free cyanide for the precipitation reaction. A small flow of lead nitrate solution will also be injected to the pregnant solution pipe prior being contacted with zinc dust to improve the precipitation efficiency. The precipitate filter feed pumps will be horizontal centrifugal pumps with mechanical seals such that air cannot enter the system.

The precipitate filters will be recessed plate filter presses furnished with filter cloths. Pre-coat will be used at the beginning of the filter cycle to prevent cloth blinding and body feed will be required to provide acceptable filtration rates. The filters will typically be operated in a duty/standby configuration and operated until the pressure reaches a predetermined value. Filtrate will report to the barren solution tank to be reused mainly as CCD wash solution. Any excess barren solution will report to the cyanide detoxification circuit to be recycled to process plant as process water or will be bled from the plant as required. The facility to recycle barren solution to the pre-clarifier feed box was included in the design in case high-grade barren solution occurs.

At the end of the filtration cycle, feed pumps will be shut down, filters drained, and compressed air may be used to further dewater the cake. The filter cake, containing approximately 50% w/w precious metals, will be dropped onto precipitate carts for transfer to the doré room for smelting. Precious metal recovery from solution to zinc precipitate will be about 99.5%.

The zinc mixing cone and precipitate filters will be located in a secured, closed room within the doré room building.

17.2.8 Doré Room

Zinc precipitates from the Merrill Crowe circuit will be loaded into a mercury retort for removal of mercury and further treated by smelting into gold–silver doré. The smelting process will be performed in batch mode. The circuit will be in a secure enclosed area with closed circuit television (CCTV) cameras and restricted access. The doré room was designed to be able to manage the volume of doré to be produced at design grades for gold and silver.

The smelting circuit will be a vendor package, and the main equipment will include:

- One 54 KW, (40 ft^3) electric retort and adsorption skid;
- One 400 KW, (10 ft^3) induction furnace;
- Flux dosing and flux mixer system;
- One gold–silver doré safe;
- Mechanized slag handling; and,
- Associated material handling and other systems (molds, dryers, dust collection system).

There will be a provision in doré room layout to install a second mercury retort and induction furnace if plant receives higher grade material and the inclusion is justified.

The wet precipitate filter cakes from the Merrill Crowe circuit will be loaded into the mercury retort for removal of mercury. The mercury retort, as part of the vendor package, will include the retort oven, condenser, mercury trap, sulphur-impregnated carbon adsorber, and a vacuum pump with seal water separator. Mercury will be collected in a mercury trap and decanted into a mercury flask.

Once the mercury free material has cooled following the retort process, it will be mixed with fluxes and loaded into the electric furnace for smelting. The fluxes will react with base metal oxides to form a slag, whilst the gold and silver will remain as molten metal. The molten metal will be poured into 40 kg moulds, to form doré ingots at nominal composition of about 0.5-1.5% Au and 90-95% Ag, and other impurities including copper and zinc. The doré bars will be cleaned, assayed, stamped, and stored in a secure vault ready for periodic transfer to market.

The slags generated from the refining treatment will be handled through a mechanized slag handling system. Once solidified, slag will be tipped from the slag pot onto spikes and broken slag will be collected in a bin underneath. The slag bin will be removed by forklift and recycled back to the SAG mill.

The doré room will operate seven days a week, for 10 hours each day.

Sufficient ventilation and off-gas handling will be provided in the doré room for a healthy work environment. Fume and dust exposure for the melting furnace and material handling will be controlled through a ventilation system installed in the doré room, including hoods, enclosures, and fans to follow local regulations/guidelines.

A sump pump, complete with precious metals trap, will be installed in the doré room to remove mercury retort condenser return water, scrubber liquid and any hose-down or spillage, and return it to pre-clarifier feed box.

17.2.9 Cyanide Detoxification

The washed leach residue slurry from the CCD circuit will be treated using a sulphur dioxide (SO_2)-O_2 process to reduce the CN_{WAD} cyanide concentration to <5 mg/L.

The nominal feed to the cyanide detoxification circuit will be 57 t/hr.

The cyanide detoxification circuit will include:

- Two cyanide detoxification reaction tanks of 4.7 m in diameter x 7 m high, operating in parallel; and,
- Associated material handling systems (pumps, pump boxes, sump pumps).

Thickened, washed tailings slurry from the final CCD thickener, with solids concentration of approximately 50%, will be pumped to the cyanide detoxification tanks. Barren solution will be used for slurry density control. In the SO_2–O_2 process, sodium metabisulphite, oxygen, copper sulphate (catalyst) and milk of lime will be added to oxidize residual free and CN_{WAD} to cyanate, thereby reducing the CN_{WAD} concentration to the target level prior to filtration and long term storage of the tailings solids. The cyanide detoxification circuit will consist of two mechanically agitated tanks, each providing a residence time of 1 hr.

Oxygen will be provided from the oxygen plant as required and will be added to the tanks via agitator shafts. CN_{WAD} levels of the cyanide detoxification discharge will be measured by analysis of regularly collected samples.

The detoxified tailings will be pumped to a thickener to thicken the slurry prior to filtration and dry stacking of the final solids.

The cyanide detoxification circuit will be serviced by a dedicated sump pump. Any spillage within this area will be returned to the cyanide detoxification feed box.

17.2.10 Final Tailings Dewatering and Disposal

Detoxified tailings will be thickened and filtered, and the filtered solids will be impounded in an on-site storage facility. The nominal throughput of the final tailings circuit will be approximately 57 t/hr.

The tailings circuit will include:

- One 14.0 m diameter high rate thickener;
- One 8.6 m diameter x 8.6 m high agitated tails filter feed tank;
- Two 2.1 m x 2.1 m, 120 chamber plate and frame pressure filters;
- One 3.5 m diameter x 3.5 m high tails filter filtrate tank;
- One 3.5 m diameter x 3.5 m high cloth wash water tank; and,
- Associated material handling systems (pumps, pump boxes, sump pumps).

Thickener underflow slurry, at approximately 53% solids (w/w), will be pumped to an agitated filter feed tank, prior to being pumped to a filtration circuit for further dewatering. This tank will provide about 6 hr of surge capacity between the thickener and filter. Two vertical plate pressure filters were selected for this purpose to increase the solid density of the tailings from approximately 53% w/w to about 82% w/w, after which the tailings will hauled to the FSTF.

Filtered solids will be impounded at the designated FTSF, to be located north east of the process plant. Filtrate will be recycled back to the tailings thickener, where it will be combined with tailings thickener overflow solution that will ultimately report to the process water tank for distribution throughout the process facilities.

Any spillage within this area will be returned to the sump pump in the cyanide detoxification area, and in turn will be pumped to a cyanide detoxification feed box.

17.2.11 Reagent Handling and Storage

The mixing and storage area for each reagent will be located proximate to various addition points throughout the plant. Some reagents will be delivered in 25 kg bags, and manually handled from a pallet to the bag breaker for mixing and further storage. Reagents delivered in bulk bags will be moved from storage to the mixing area by forklift. Electric hoists servicing in the reagent area will lift the reagents to the respective reagent bag braker that will be located above the reagent mixing area.

The reagent handling system will include unloading and storage facilities, mixing tanks, stock tanks, transfer pumps, and feeding equipment.

Quick lime will be delivered to the plant in regular 20 t bulk shipments and received in a 30 t storage silo, which, at design operating rates, will provide for about 13 days of storage. Lime will be subsequently slaked in a package slaking circuit, sized for 183 kg/hr. The resulting milk of lime slurry, at approximately 20% CaO solids %w/v, will be stored in an agitated tank, and distributed to the various addition points by way of a ring-main. At the design production rate, the process facility will consume about 800 t of lime annually.

Sodium cyanide, supplied in solid (briquette) form, will be received in regular bulk shipments from regional suppliers in 1 t bulk bags. Sodium cyanide stock solution of 20 vol% (200 g/L NaCN) will be generated on site using a solid to liquid system (SLS) to minimize potential releases and employee exposure. The stock solution storage tank will provide for 12 hrs of cyanide supply at the nominal production rates. From the stock solution storage tank, sodium cyanide solution will be provided to the leaching circuits and Merrill Crowe circuit as required via a ring-main. The SLS storage and make up, as well as the cyanide solution storage tank will be fully contained in a bunded area and separated from the plant site. The solid sodium cyanide will be stored in a fenced and locked area before being used to prepare stock solution. At the design production rate, the process facility will consume about 1,600 t of sodium cyanide annually.

Table 17-2 shows the reagents proposed for the process plant.

Table 17-2: Summary of Reagent Used in the Process Plant

Reagent	Preparation Method	Use
Lime	Received as quicklime from a 20 t pneumatic tanker truck and transferred to a silo; slaked and mixed to 20% strength; pumped to a storage tank. Dosed to concentrate leaching, bulk leaching and cyanide detoxification circuits as required	pH control added as required
Sodium Cyanide	Received in 1 t bulk bags; mixed to 20% strength; transferred to a storage tank. Dosed using the cyanide circulation pump and a ringmain system to the concentrate and bulk cyanide leaching circuits, as well as Merrill Crowe circuit if required	Leaching agent
Flocculant	Received as powder in 25 kg bags; mixed to 0.25% storing strength; transferred to a storage tank. Dosed directly to concentrate thickener, pre-leach thickener and CCD washing thickeners with dilution as required	Flocculation of concentrate leach slurry, bulk cyanide leach feed thickener and CCD washing thickeners
Tails Flocculant	Received as powder in 25 kg bags; mixed to 0.25% storing strength; transferred to a storage tank. Dosed directly to tails thickener	Flocculation of tails
Coagulant	Received as powder in 25 kg bags; mixed to 0.25% storing strength; transferred to a storage tank, and dosed directly to pre-clarifier	Clarification of the pregnant solution
Oxygen	Produced by oxygen plant, gasified, and sent to the concentrate and bulk cyanide leaching circuits and cyanide detoxification circuit	Cyanidation reagent, Cyanide detoxification reagent
Aerofloat 208	Delivered in liquid form in IBC totes. Dosed to the required locations within the flotation circuit	Flotation collector
PAX	Received as pellets in 25 kg bags; mixed to 10% storing strength; transferred to a storage tank. Multiple dosing pumps deliver the reagent to the required locations within the flotation circuit	Flotation collector
Frother	Delivered in liquid form in IBC totes. Multiple dosing pumps deliver the reagent to the required locations within the flotation circuit	Flotation frother
Diatomaceous Earth	Received as powder in 25 kg bags; mixed to about 5% solution strength. Dosed to the clarifier and precipitate filters in Merrill-Crowe circuit	Precoat and body feed in Merrill-Crowe circuit
Zinc Powder	Received as powder in 20 kg drums. Dosed to Zn mixing cone through a feeder at specific rate in Merrill-Crowe circuit	Precipitation regent

Reagent	Preparation Method	Use
Lead Nitrate	Received as powder in 1 t bulk bags, mixed to 10% strength; transferred to a storage tank. Dosed directly to the concentrate and bulk cyanide leaching circuits, as well as Merrill-Crowe circuit	Leaching aid in cyanidation and a co-precipitation regent in Merrill-Crowe
Copper Sulphate	Received as powder in 25 kg bags; mixed to 10% strength; transferred to a storage tank. Dosed to the cyanide detoxification circuit	Catalyst in the cyanide detoxification process
Sodium Metabisulfite	Received as powder in 1 t bulk bags; mixed to 20% strength; transferred to a storage tank. Dosed to the cyanide detoxification circuit.	Reactant in the cyanide detoxification process
Antiscalant	Delivered in liquid form in IBC totes. Dosed neat without dilution to pregnant solution tank and process water tank	To minimize scale build-up
Flux	Received as powder in bulk; mixed with calcined charges for smelting	Fusion agent

17.3 Plant Services

17.3.1 Fresh Water, Raw Water, Fire Water and Potable Water

Provisions will be made for the raw water to be supplied from the underground mine, the fresh water (storm) pond, the Sonora Valley, or any combination thereof pending availability and requirement. Raw water will be supplied to settling tanks for bulk removal of solids. The sediment-free water will be transferred from the raw water tanks to a fresh/fire water storage tank.

Fresh water will be used for the following duties:

- Reagent mixing and preparation;
- General process uses in crushing area and emergency stockpile; and,
- Gland water.

Wherever possible in the plant, process water or barren solution will be used to minimize fresh water consumption. The total fresh water requirement for the plant will be 3.6 L/s.

Potable water will also be sourced from the sediment-free water in the raw water tanks. The raw water will be treated in a water treatment plant prior to transferring to the potable water tank for distribution where needed.

17.3.2 Process Water, and Barren Solution

Process water will consist of reclaimed water from the pre-leach thickener overflow and tailings thickener overflow. Barren solution will be used as make-up for the process water supply as required. Process water will be stored in a process water tank and pumped to the grinding circuit, lime preparation, and cloth wash water for tailings filters. Any excess process water will be transferred to the collection pond. If impurities build-up in the process plant, a flow of process water to a collection pond will allow for the required bleeding from the system. No water is expected to be discharged to the environment.

Barren solution from the Merrill Crowe circuit will be stored in a barren solution tank, recycled to the CCD washing circuit as wash water and used for flocculant dilution. The residual barren solution will be used wherever possible, to minimize the fresh water consumption. Barren solution will be used for concentrate leach feed and cyanide detoxification feed dilution to achieve target density, and in stock cyanide solution preparation. The barren solution within the Merrill Crowe circuit will be used for pre-clarifier coagulant dilution, sluicing water for clarifier filters, pre-coat and body feed preparation, zinc dust slurry preparation, and vacuum pump gland seal water to minimize the fresh water usage. Any excess barren solution will report to the cyanide detoxification circuit.

17.3.3 Oxygen Plant

The oxygen plant will generate oxygen using vacuum swing adsorption (VSA) technology. The plant will consist of five units each with oxygen production capacity of 954 kg/day with 38 kW installed power requirement for each unit. Oxygen will be produced at 93% purity at 100 psig (6.8 barg).

Oxygen will be used for the following duties:

- Concentrate leach circuit;
- Flotation tails (bulk) leach circuits; and,
- Cyanide detoxification circuit.

The total oxygen required for the plant will be about 3.8 t/d. An oxygen utilization of 80% was used in sizing the oxygen plant.

17.3.4 Electrical Power

The total peak operating load for the project will be 7.3 MW with a normal operating load of 7.1 MW.

Power will be supplied to site from the regional grid, as described in Section 18.

17.3.5 High Pressure and Low Pressure Air

17.3.5.1 High Pressure Air for Tailings Area

High pressure air at 1,000 kPa(g) will be provided by two high pressure air compressors, operating in a lead-lag configuration for tailings filters. The portion that is required for instrument air at 700 kPa(g) for this area will be dried and filtered and distributed via the tailing area instrument air receiver.

17.3.5.2 Plant and Instrument Air for the Balance of the Process Plant

Plant and instrument air for the balance of the process plant at 700 kPa(g) will be provided by two process plant air compressors, operating in a lead-lag configuration. The entire high-pressure air produced will be dried and filtered and will be used to satisfy both plant and instrument air demand. Dried air will be distributed via the process plant air receiver, with additional receivers for precipitate filter in Merrill Crowe circuit and dust collector in the refinery area.

17.3.5.3 Low Pressure Air for Flotation Circuit

Low pressure air to flotation cells will be supplied by one dedicated blower.

17.3.6 Diesel Fuel

Diesel will be supplied to trucks and light vehicles. There will be two vessels containing at least 22,000 L of diesel each, for total of 44,000 L. The diesel fuel area will contain the following equipment:

- Two diesel storage tanks;
- Two storage tank suction pumps; and,
- Two diesel stations.

17.3.7 Instrumentation and Process Control

A distributed control system (DCS) will be designed and installed in the process plant. The process control system will consist of individual, locally mounted, control panels located near the equipment and a PC-based operator interface station (OIS) located in a centralized control room. The local control panels will act as a local point for monitoring and control of the nearby equipment and instrumentation. They will also act as the distribution point of power for instrumentation. Major process performances, including process rates, mill power draw, and motor variable speeds, will be displayed in the centralized control room. The DCS and OIS will perform process control and data management through equipment and processing interlocking, control, alarming, trending, event logging, and report generation. In this manner, the process plant will be monitored and operated automatically from operator workstations in conjunction with control systems.

17.3.8 Sampling and Quality Control

A metallurgical and assay laboratory will be provided to conduct daily assays for quality control and optimize process performance. The assay laboratory will be equipped with the necessary analytical instruments to provide all the routine assays for mine samples, geological samples, process plant samples, and samples taken for environmental monitoring. The metallurgical laboratory will undertake all basic test work to monitor metallurgical performance and to improve the process flowsheet and efficiencies.

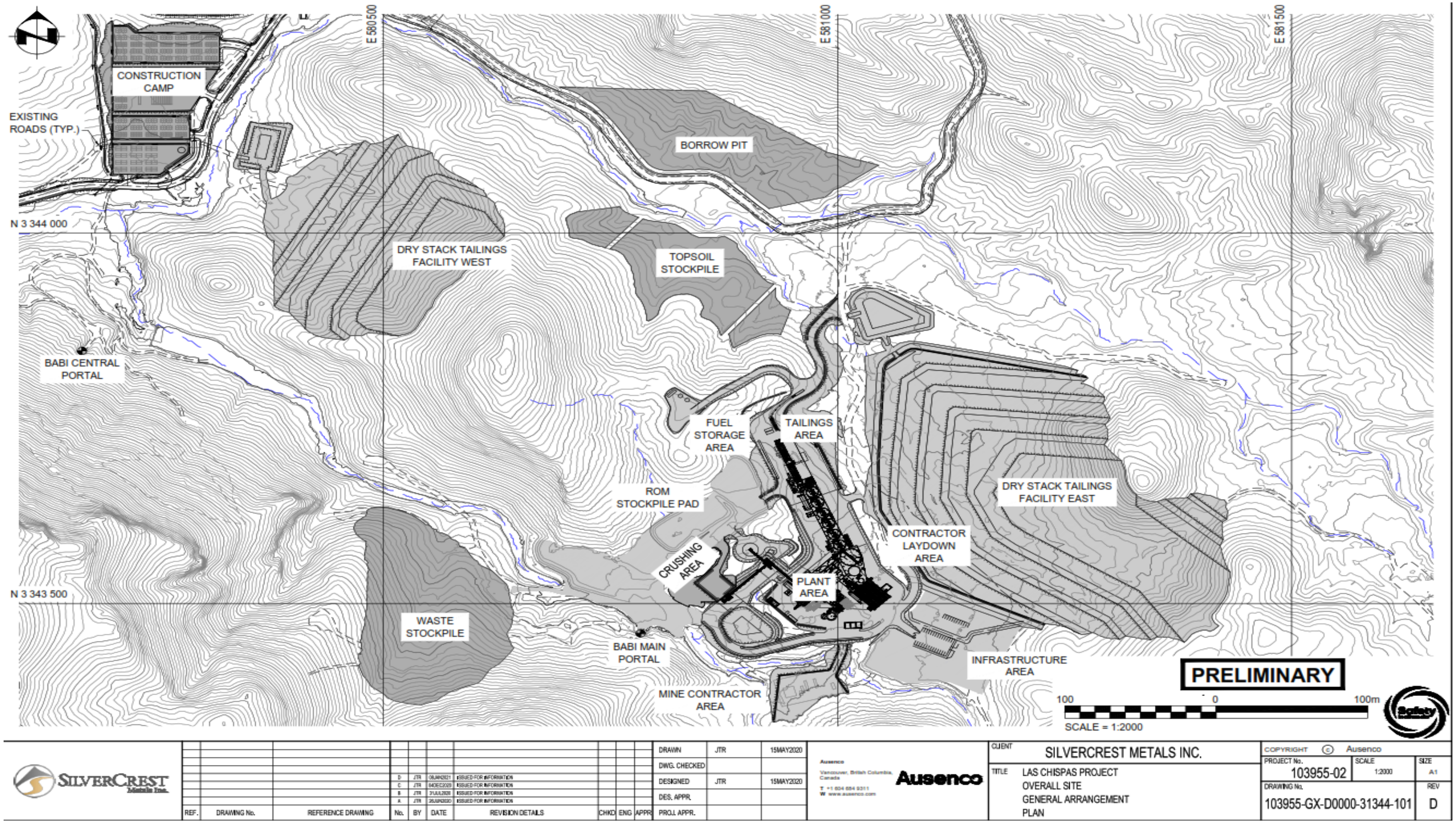
18 Project infrastructure

18.1 Introduction

The proposed mine is the site of historical mining. Infrastructure that will be required for the mining and processing operations will include:

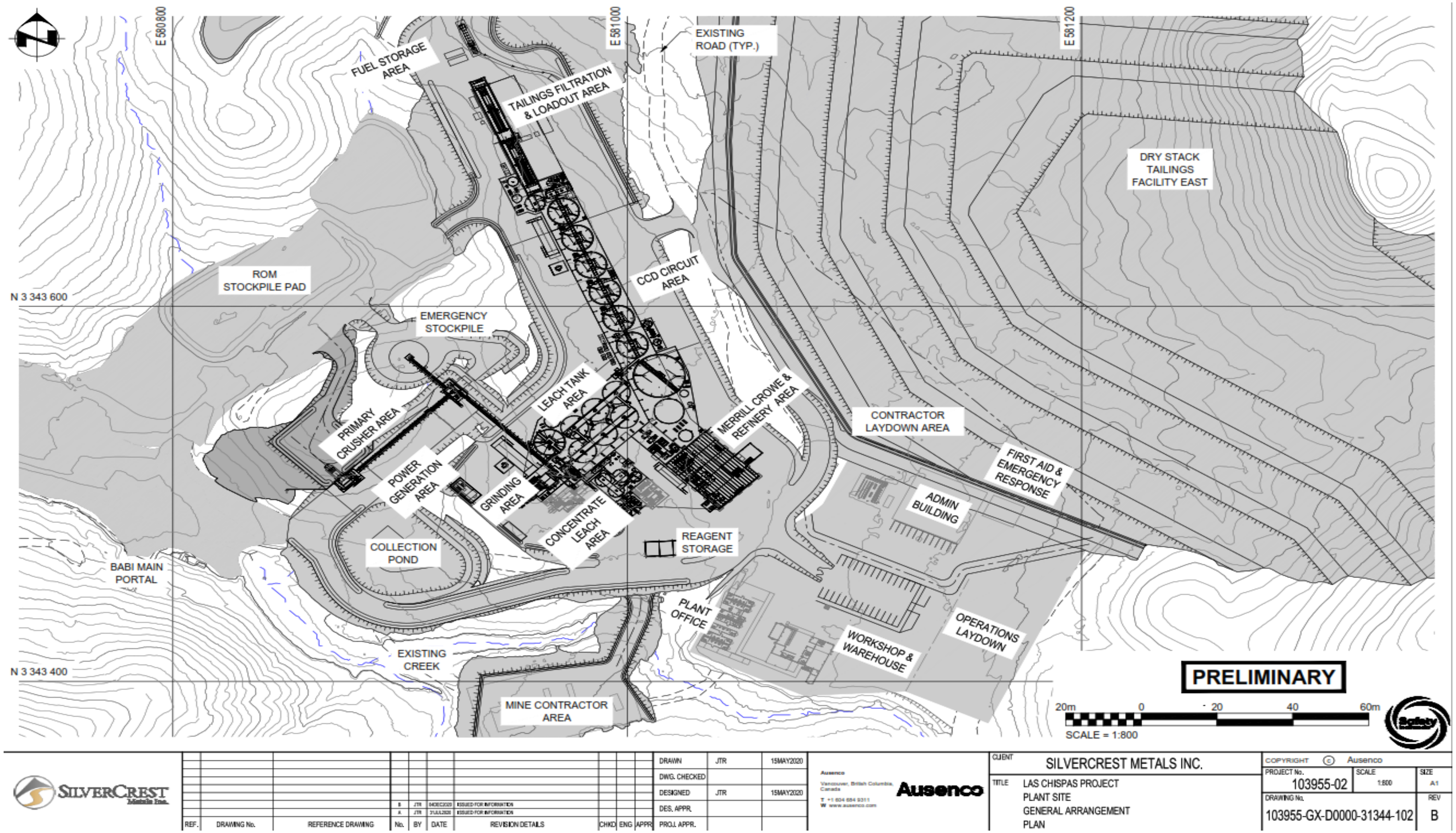
- Underground mine, including portals, ramps and vents;
- Roads: main access road, site access road, bridge crossing, borrow pit haul road, FTSF haul road, WRSF haul road, and explosives access road;
- Diversion and collection channels, culverts, and containment structures;
- Site main gate and guard house;
- Construction camp;
- Power and water distribution;
- Warehouse and truck shop, offices, process plant dry facility, medical clinic, and nursery;
- Explosives magazines,
- Processing plant;
- Control room;
- Doré room;
- Assay laboratory (off-site facility);
- Reagent storage facility;
- Water treatment plant;
- Mineralized stockpiles and WRSFs;
- FTSF;
- Hazardous waste interim storage facility; and,
- Exploration core shack.

Figure 18-1 shows the proposed site layout and Figure 18-2 shows the proposed locations of the processing plant, warehouse and shop, and administration buildings.



Note: Figure prepared by Ausenco, 2020.

Figure 18-1: Proposed Site Layout

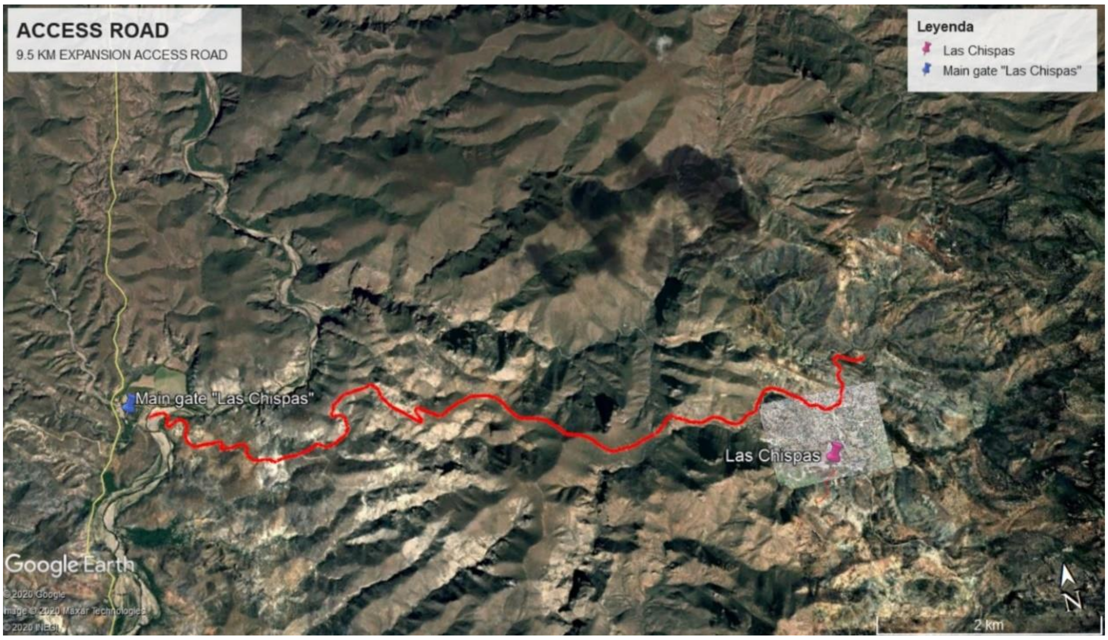


Note: Figure prepared by Ausenco, 2020.

Figure 18-2: Proposed Location of the Processing Plant and Other Buildings

18.2 Roads

The Las Chispas Property can be accessed from Highway 89 via the 10 km existing access road (Figure 18-3). The net elevation gain along the main access road is approximately 440 m (from Rio Sonora crossing to Santa Rosa Portal), towards the Property. Main access road upgrades will be required to facilitate transportation of equipment and materials during construction and operations and will include widening to 8 m over a length of 9.5 km and a bridge over the Rio Sonora. The upgrades will include the security gate on Highway 89.



Note: Figure prepared by SilverCrest, 2020, using Google Earth backdrop.

Figure 18-3: Access Road between Highway 89 and Project

A single lane bridge located approximately 250 m east of Highway 89, will be built over the Rio Sonora with a length of 171m and a capacity of 72.5 t. This capacity will be enough to support the operation. Before the bridge is completed, the existing river crossing will be used.

Additions and upgrades to existing access roads around the plant site are being planned to access mine infrastructure including mine portals, process plant, explosive magazines, potable water well, FTSF, WRSF, seepage ponds, and all other ancillary infrastructure.

18.3 Camps

18.3.1 Construction Camps

The Project will require the use of a temporary construction camp. The camp will generally be self-contained and have its own power generation and heating capabilities, potable water treatment plant, and sewage treatment plant. Garbage will be collected on site and disposed at the Arizpe municipality waste disposal facility.

In Q4 2020, SilverCrest started to build the construction camp to accommodate changes in protocols and procedures during construction which have been implemented in response to the COVID-19 pandemic. Phase 1 of the 520 man-camp (320 rooms) is scheduled to be completed in February 2021. Phase 2 (100 rooms) is scheduled to start in Q1 2021 and phase 3 (100 rooms) is scheduled for Q2 2021.

The design includes 520 single occupancy rooms with 20 rooms reserved in case of a COVID-19 outbreak at site. The single occupancy room will include a bed, toilet and shower. A total of 30 rooms have been designed slightly larger and include sufficient space for a small desk. These rooms will be later reconfigured and relocated to house the operation offices. The remaining part of this temporary facility will be sold.

18.3.2 Arizpe Camp

Given the risk associated with the operation of a camp during a pandemic, SilverCrest has implemented a second facility directly in the small town of Arizpe. The facility is already operational and for the most part supports the exploration needs of SilverCrest in a confined setting. The facility has a capacity of 150 beds to supplement the construction camp. This is intended as a temporary facility only and will be demobilized upon completion of project construction.

18.4 Fuel Storage

Diesel fuel requirements for the mining equipment, process and ancillary facilities will be supplied from two modular above-ground diesel fuel storage tanks located near the Santa Rosa portal. The diesel fuel storage will have a capacity of 44,000 L, sufficient for approximately nine days of operation. The above-ground tanks include containment, and dispensing equipment conforming to all applicable regulations.

18.5 Power Line

Electrical power will be supplied to site from the national grid, by way of an overhead power line, rated to carry 8.5MVA at 33kV. Connection to the grid will be at the Nacozari de Garcia substation, which is 74.4km from the Project. The power line will cross three municipalities, and will be divided into two sections:

The first section of the power line, from Nacozari to Los Hoyos will be 26 km long. The section is an older overhead power line, owned by Comisión Federal de Electricidad (Federal Electricity Commission – CFE), which will be upgraded (increased cable size, from 3/0 to 477 kcmil) to safely supply the power required at the Project.

The second section of the power line, from Los Hoyos to Las Chispas will be 48.4 km in length. This will be a newly constructed section. Right-of-way agreements for the construction are in place and detailed engineering for the transmission line, in accordance with local and international standards (CFE specs, NEC & IEC), is well advanced.

CFE have approved the transmission line for the Project and anticipate completion of the upgrades and construction to be completed, with power supply available at the Project in Q1 2022.

18.6 Power Distribution and Emergency Power

Power will be provided by CFE at 33 kV using overhead powerlines.

A 1.2 MW standby generator will be installed as needed to power the essential loads, in the event of the utility power failure.

Electrical power will be received at a substation on site and distributed to two major electrical rooms via above ground cable trays. Power distribution and control equipment will be installed within prefabricated electrical rooms located in the grinding and CCD areas of the process plant. The operation and control will be primarily from a centralised control room with an additional control facility provided within the refinery building.

The total normal operating load for the project will be 7.1 MW, including the process plant, ancillary facilities, camps, and mine portals. The capacity of the connection to the grid is 8.1 MW.

18.7 Site Communications

On-site communication systems will include a voice over internet protocol (VoIP) telephone system, a local area network (LAN) with wired and wireless access points, hand-held very high frequency (VHF) radios, and a leaky feeder network for the underground mine. A communications tower is in place on Babicanora Hill (Photo 18-1).

Photographs taken by SilverCrest, 2020.

Photo 18-1: Communication Tower Babicanora Hill

Off-site communications will use a satellite-based, cellular-based, or landline-based system. Evaluation of these options is currently in progress, and ultimate decision will depend on the proximity of the nearest available fibre-optic or cellular network in the region.

Point-to-point link from the general offices in Hermosillo to the operations, exploration and administration offices was installed in Q3 2020, including installation of towers and equipment at: the Hermosillo office, Cerro del Bachoco, a hill in the Aconchi Sierra, Sonora, Babicanora hill (shown in Photo 18-1) and in the administration office at site and the exploration office. The system is provided with redundant communication equipment with two services in Hermosillo: a 200 Mbps fiber dedicated internet and a 200 Mbps symmetric fiber internet, and three other satellite internet services in: administration office, exploration office and construction offices.

A second 200 Mbps point-to-point link from the general offices in Hermosillo to the operations, is planned to cover the needs of the construction camp facilities.

It is expected that there will be cell phone access for personal use in a limited area. This system will be installed and maintained by a communications service provider.

18.8 Fire Protection

A complete fire detection and alarm system, fire water supply and distribution system, water-based fire protection system, and special hazard fire protection system will be constructed and installed as per applicable regulations. Fire detectors, alarms, and extinguishers will be installed where required. The firewater distribution network will be maintained under a constant pressure with a jockey pump and will be sectionalized to minimize loss of fire protection during maintenance.

Yard hydrants will be limited to the fuel storage tank area. Wall hydrants will be used in lieu of yard hydrants, and these will be located on the outside walls of the buildings.

Fire protection within buildings will include sprinkler systems and portable fire extinguishers.

18.9 Sewage System

Sewage collected from the ancillary buildings will be pumped to the sewage treatment module for treatment prior to being discharged. The sewage treatment module will be of the rotating-biological-contactor type. Treated effluent will be pumped to the designated discharge point for release.

18.10 Hazardous Waste Interim Storage Facility

An interim storage facility for hazardous waste will be built at site to allow for storage of ancillary wastes, such as used oils, before they are transported off-site for disposal by third-party.

18.11 Plant Nursery

A nursery has been built to conserve the flora rescued from the areas where infrastructure, processing plant, roads, bridges and power lines will be built.

18.12 Mine Related Infrastructure

18.12.1 Waste Rock Storage Facility

The maximum capacity of the WRSF will be 899,500 t. This stockpile will be used to temporarily store the development waste before returning it to rockfill the mined-out stope. Although current waste rock characterization indicates no potential for acid generation, contact water channels as well as contact water ponds will be included to capture surface water runoff from this temporary structure and allow for water sedimentation/monitoring before returning to the process plant.

18.12.2 Stockpiles

The maximum capacity of the mineralized material stockpile will be 287,000 t with segregated piles based on grade (or clay content). This capacity is greater than the maximum storage requirement. Non-contact water management structures were included around the stockpile area to minimize contact water generation. Contact water channels and contact water ponds were designed to manage and capture runoff from this temporary structure.

18.13 Process Related Infrastructure

18.13.1 Plant Site Roads

The site roads will provide access to the on-site facilities. All roads will be designed for a speed of 30 km/hr and a slope limit of 12%. The majority of site roads will have two lanes.

All new facility areas and surrounding areas will be graded to ensure stormwater drains away from the facilities during rainfall. Drainage ditches will be 0.6 m bottom channels with a minimum longitudinal grade of 0.5%. At the intersection of drainage paths and access roads, water will be conveyed across the road via a culvert crossing. Culverts will be installed with rip-rap erosion protection at inlets and outlets and will require regular maintenance to keep them sediment free and free-flowing during rainfall events.

18.13.2 Truck Shop and Warehouse

The maintenance shop facility was built as part of the Early Works program. It consists of a pre-engineered steel structure with a roof and low walls and limited interior support steel structures. The building is supported on concrete spread footings and concrete grade walls along its perimeters. Sumps and trenches were constructed to collect wastewater in the maintenance bays. Floor hardener was applied to concrete surfaces in high-traffic areas.

During construction, the facility will be used to house the Construction Management Team (CMT). Upon completion, the facility will be returned to operations and will house a wash bay complete with repair bays, parts storage area, welding area, machine shop, electrical room, a training room, mechanical room, air compressor, and lube storage. The facility is designed to service and maintain the process plant and the service fleet.

18.13.3 Plant Offices

The offices for the Operation Team will be located adjacent the warehouse where the CMT will be located during construction.

Upon the start of the operation, the warehouse will be transferred back to operations and the offices will be redistributed according to operational needs.

18.13.4 Process Plant Dry Facility

A dry for the process plant will be built near the processing plant facility. The building was designed as a single-storey modular building with an area of approximately 125 m^2. The facility will provide clean and dirty areas for male and female workers, and will be complete with showers, basins, toilets, lockers and overhead laundry basket.

The mine contractor will be responsible for their own facility.

18.13.5 Primary Crushing

The primary crushing area will feature a concrete foundation, steel structures for supporting process equipment, platforms, and walkways.

18.13.6 Process Plant

Most of the process plant area will not be fully roofed and principal construction will be concrete foundations, steel structures for supporting process equipment, platforms, and walkways. Where required, there will be some areas of the process plant that will be roofed.

Process plant cranage will be provided by a mobile crane for most areas.

The process plant will have elevated steel platforms in the grinding area and over the leach tanks and other large tanks for maintenance access.

The process plant foundation will consist of concrete spread footings and containment bunds along the building perimeters and a slab-on-grade floor. The floor surfaces will have localized areas that are sloped toward sumps for clean-up operations.

18.13.7 Doré Room

The Merrill Crowe facility will be housed in a pre-engineered, fully secured building.

The doré room will be constructed with thick concrete floors and walls complete with a heavy-duty building enclosure, entry gates, CCTVs, motion sensors, and alarm. The facility will be monitored 24 hr/d by the security personnel. Access to the doré room will be restricted to authorized personnel only.

The gold and silver recovery and smelting areas will be provided with sufficient ventilation to mitigate the potential impact of off-gas produced from the melting furnace and dust generated from flux mixing.

Gold–silver doré products will be stored in the dedicated safe in the doré room. Doré product transportation will be undertaken by contractors using armour trucks.

There will be a fenced area for controlled entry and exit of the armoured transport vehicle to prevent unauthorized entries into the doré room, while the armoured vehicle is entering or exiting the facility.

18.13.8 Reagent Storage Facility

The Reagent storage shed will be an open-side structure having a footprint of 18ºm x 8 m. The shed will be of pre-engineered steel construction with a corrugated metal roof.

18.14 Water Management

18.14.1 Water Management

The key facilities for the water management plan include:

- Underground mine dewatering, predominantly from backfilling operations;
- Mill (including fresh and process water tanks);
- FTSF;
- Surface water diversion and water management structures;
- Fresh water supply system, including pumps and piping; and,
- Sediment and erosion control measures for the facilities.

The water management strategy uses water within the Project area to the maximum practical extent. The plan involves collecting and managing site runoff from disturbed areas and maximizing the recycle of process water. The water supply sources are will as follows:

- Precipitation runoff from the mine site facilities;
- Water recycled from the tailings dewatering system;
- Groundwater from the underground mine dewatering system for fresh water supply and potable water;
- Treated black and grey water, in small quantities, from the buildings; and,

- If needed, water withdrawn from the Sonora Valley for fresh water supply and potable water.

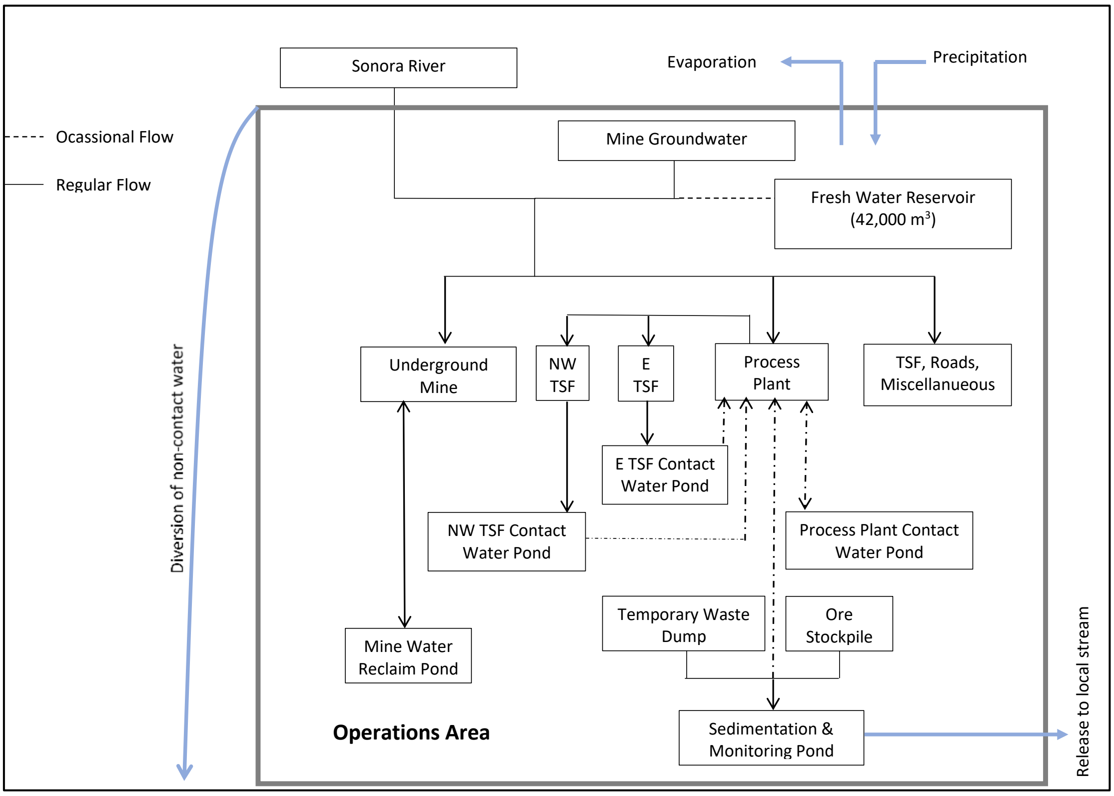
18.14.1.1 Water Balance

A deterministic annual water balance model was developed for the site using the site data, which included the characterization of climate, hydrometric, hydrogeologic and surface water conditions in the mine development area.

For the purpose of developing the water balance, the following components were considered:

1. Water entering the operations area from precipitation and surface or groundwater sources;
2. Water exiting the operations area through evaporation or infiltration; and,
3. Water entrained in the filtered tailings and exiting the system.

A conceptual water balance diagram is shown in Figure 18-4.



Note: Figure prepared by Wood, 2021.

Figure 18-4: Conceptual Water Balance Diagram of Water Sources, Pathways and Discharges for the Las Chispas Mine Operations Area

The main water sources identified for the Project consist of 1) groundwater from the underground pumping 137 system; 2) fresh water from the Sonora Valley; and 3) surface water resulting from precipitation in contact areas and retained in contact water ponds. Based on a hydrogeological characterization of the site as well as recent pumping tests conducted in the Las Chispas underground mining area (Hydro-Ressources, 2020), the groundwater source located in the historical Las Chispas workings can consistently supplement 0.3 L/s. However, based on information provided by SilverCrest and HRI, future studies will focus on fault areas within Las Chispas mine area that could produce up to 6 L/s of groundwater, potentially becoming the main source of fresh water for the Project.

SilverCrest has negotiated water rights with the local national regulators to use up to 9.5 L/s of water from the Sonora Valley. The mine groundwater and the Sonora Valley sources provide the required 8 L/s of fresh water for the LOM of the project.

Additional sources of water include an existing historical water reservoir on site, which has an effective capacity of 42,000 m^3, that can be an emergency supply of water for up to about two months (at a total mine demand of 8 L/s) depending on the season. Occasional inflows to the system will come from the FTSF contact water pond during the rainy season, which can supplement inflows from the valley and underground dewatering system to satisfy water demands.

18.14.1.1.1 System Outflows

Water demands include: the process plant, the underground mine, water entrained in the tailings and going to the FTSF, water for construction and dust suppression, and evaporative losses (Wood, 2021).

Based on the process plant water balance (with occasional bleed scenario) the water demand for the plant is approximately 3.3 L/s (Ausenco, 2020).

Current estimates of water loss as pore water in the filtered tailings is approximately 2.8 L/s. This amount would exceed optimum water for tailings compaction but evaporation loss (which exceeds average precipitation all year round) will allow for tailings placement and compaction on most days. On days when there is more precipitation and the tailings exceed the optimum moisture content, tailings that have not been graded and compacted can be covered with high-density polyethylene (HDPE) sheeting (or raincoats) until the climatic conditions allow again for moisture loss due to evaporation.

The demand from the underground mine will average 3.2 L/s with a maximum of up to 6.3 L/s during startup/commissioning. However, most of this water is anticipated to be reused, therefore a 1.3 L/s constant demand (approximately 30% loss) was estimated in balance calculations.

Water will also be required for dust suppression in the FTSF, roads and miscellaneous structures. A, 0.6 L/s demand has been assumed based on similar projects in semi-desert regions of Mexico.

18.14.1.2 Reclaim Water System

Reclaim water for use in the mill processes will be pumped from the tailings filtrate water tank to the process water storage tank. The process water storage tank will store a 24-hour supply of mill process water, which will be gravity fed to the plant site. Additional process water will be obtained from other sources described in Section 18.14.1.1.1.

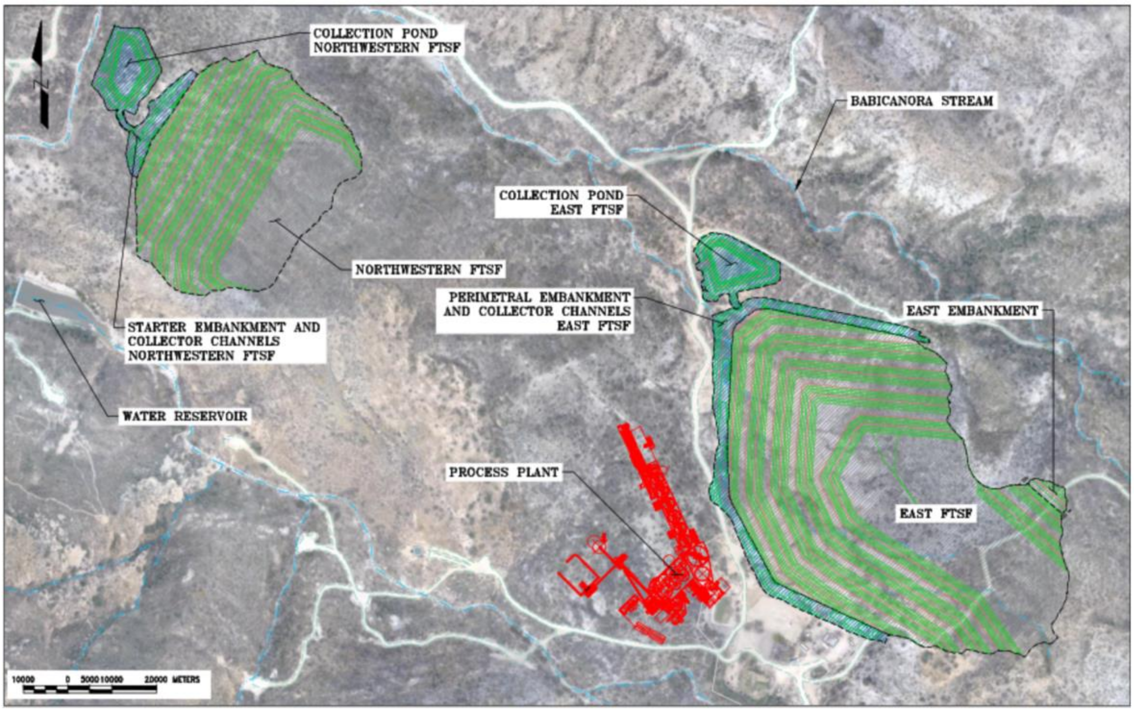
18.14.1.3 Additional Water Management Facilities

Additional facilities include contact water ponds for each FTSF, a contact water pond in the process plant area, settling and monitoring ponds for the temporary waste dump and mineralized material stockpile and a mine water reclaim pond in the proximity of the main portal.

18.15 Filtered Tailings Storage Facility

18.15.1 Overview

A FTSF concept was adopted based on the mine plan, the limited available construction materials, and to avoid risks associated with storage of conventional slurried tailings behind a dam. The tailings to be stored on surface will be thickened, filtered, and delivered by trucks to the FTSF. Two facilities have been designed to store approximately 4.5 Mt. However, based on the current LOM and estimated production, only the East FTSF will be constructed to store approximately 3.5 Mt of filtered tailings. The proposed geometry and key features of the proposed FTSFs are shown in Figure 18-5.



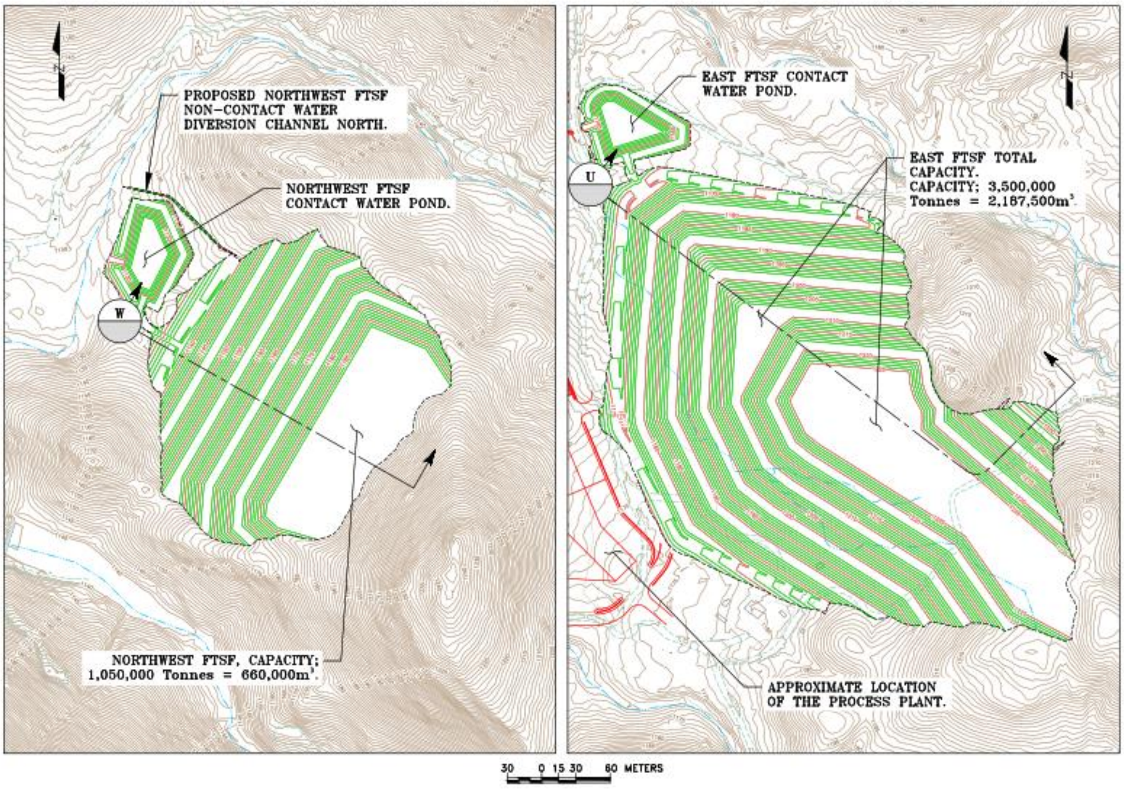
Note: Figure prepared by Wood, 2021.

Figure 18-5: Proposed FTSFs Locations and Footprints with Respect to the Process Plant

Due to their unsaturated condition and predominantly dilatant geotechnical behaviour, filter-pressed dry-stacked or filtered tailings, do not need large retention structures (dams) and allow for the mitigation of physical stability risks. Additionally, once placed and compacted, filtered tailings tend to be very low permeability materials, which in combination with adequate surface water management systems results in a structure that significantly reduces the quantity of water that can migrate into the natural environment. Another advantage of filtered tailings is that water recovered in the filtration step can be recycled into the metallurgical process. Moreover, in comparison with conventional slurry tailings facilities, filtered tailings stacks generally require a smaller footprint for tailings storage, are easier to progressively reclaim, and can have lower long-term (closure) liability in terms of potential environmental impacts.

The FTSFs locations were selected according to the criteria established by the Mexican authorities through the NOM-141-SEMARNAT-2003 standard, with a focus on the integrity and stability of the FTSFs. Accordingly, hydraulic aspects related to the catchment area and large return-period precipitation events as well as geotechnical aspects related to location and properties of the soil and rock material underlying the Project area were considered.

Based on the storage requirements (approximately 3.5 Mt of filtered tailings) as well as topographical, geotechnical and hydrological requirements, the selected locations were the most favourable (Figure 18-6).



Note: Figure prepared by Wood, 2021.

Figure 18-6: Proposed East (right) and Northwest (left) Filtered Tailings Storage Facilities

From a design perspective, the proposed FTSF sites have the following advantages:

- Proximity to the plant and mine portal;
- Closest population centre (Sinoquipe) located about 20 km downstream of the proposed FTSF structures;
- Areas with minor human activity both in the catchment basin as well as downstream of both proposed FTSFs structures;
- The proposed FTSF sites are not within the footprint of what would be considered federal surface water bodies according to the Comisión Nacional del Agua (CONAGUA);
- Absence of aquifer and groundwater production wells in the FTSF areas; and,
- Underlying bedrock with low permeability at shallow depths (5 m or less) that limits the vertical infiltration of water.

The FTSFs have a projected maximum elevation of 1,219 masl in the East FTSF and 1,187 masl in the NW FTSF. The East FTSF will be about 530 m northeast of the process plant and cover an area of 101,932 m^2. The NW FTSF would be about 300 m northwest of the process plant and cover an area of 47,758 m^2. The FTSFs will have additional infrastructure, such as: contact water collection channels, contact water collection/storage ponds, sub-drain collection systems, access roads, etc., which are part of the areas considered for rehabilitation in the Closure Plan. Non-contact water diversion channels will be constructed to reduce the amount of surface contact water generated from the FTSF exposed areas. However, these structures are assumed to remain in place as they would provide some benefit diverting water for low return period precipitation events, while they remain functional. It is also assumed that over time, these structures will be silted and slowly integrate to the surrounding landscape.

18.15.2 Geotechnical Characterization of Tailings

Two representative composite samples of tailings for geotechnical testing designated as "FS-Tailings-01" and "FS-Tailings-02" were shipped respectively to reputable soils mechanics laboratories in Lakewood, CO and Guanajuato, Mexico. Geotechnical tests were conducted on these composite tailings samples (Wood, 2021).

Based on the Unified Soil Classification System, the first sample FS-Tailings-01 classified as non-plastic sandy silt, whereas the second sample FS-Tailings-02 classified as a low plasticity silty clay. Based on information from SilverCrest, most of the tailings to be generated are anticipated to resemble the first sample tested (FS-Tailings-01), and that the second, clayey sample (FS-Tailings-02) represents a small portion of the tailings produced over the LOM.

The FS-Tailings-01 sample had approximately 70% by weight of its particles passing the No. 200 mesh (0.075 mm) and an SG of 2.69. Based on the standard Proctor compaction test, these tailings have a maximum dry density of 18.2 kN/m^3 and an optimum geotechnical moisture content (weight of water over the weight of solids) of 16.0%. The FS-Tailings-02 sample had up to 95% passing the No. 200 mesh, a maximum dry density of 17.10 kN/m^3 and an optimum geotechnical moisture content of 14.6%.

A direct shear test was performed on the FS-Tailings-01 sample re-modelled at 95% of the maximum dry density, with confining stresses of approximately 250, 500 and 1000 kPa resulted in an internal friction angle φ of 32.9º and a cohesion "c" of zero. The average saturated hydraulic conductivity was 2.3E-05 cm/s, as measured in a flexible wall permeameter, of a remoulded tailings sample to approximately 95% of the maximum Proctor density and -1.9% of optimum moisture content.

An unsaturated triaxial test on a remoulded FS-Tailings-02 sample in similar conditions of compaction and minus 2% optimum moisture, with confining stresses of 245, 440 and 650 kPa resulted in an internal friction angle φ of 35.4º and a cohesion "c" of 80 kPa.

The swell/collapse test on the FS-Tailings-01 sample at the proposed compaction conditions of 95% maximum dry density and minus 2.0% of optimum moisture content resulted in a collapse of approximately 0.5%, which translates into a low collapse potential.

Unsaturated soil mechanics capillary humidity retention tests were also conducted under similar remoulded conditions to define infiltration from precipitation and runoff in the filtered tailings once placed and compacted. These tests covered ranges of zero suction (saturation) up to a 0.6% humidity that reached a maximum suction of 1,550 bars (15,810 m of water column).

18.15.3 Geotechnical Analyses

Regardless of the plasticity and gradation differences between the tailings samples, the friction angles had a narrow range of values between 32.9–35.4º, and cohesion ranged between zero and 80 kPa. Therefore, for geotechnical design purposes, an angle of internal friction φ = 33º and cohesion C = 0 were used for limit equilibrium stability analyses.

18.15.4 Infiltration Analyses

A series of infiltration modelling analyses were performed to estimate the infiltration during construction and through the lifetime of the FTSF (Wood, 2021). These analyses were performed using the commercially available finite element 2D software VADOSE/W, which is capable to perform both steady-state and transient flow analyses in porous media. A representative cross section of the East FTSF was selected to perform the unsaturated analyses. Based on the infiltration modelling results, the native ground remained unsaturated even at the end of the 20-year modelling simulation. Considering that by this time, construction of the cover system would be completed and the FTSF vegetated and reclaimed, infiltration modelling indicated that under the anticipated filtered tailings placement and compaction conditions, and the site-specific climatological conditions, no infiltration from the FTSF into the native ground would be expected. Despite the infiltration modelling results, the FTSFs were designed with a subsurface water collection system (subdrain) to capture potential infiltration during the early construction stages and to manage contact water upstream of the starting buttress structure.

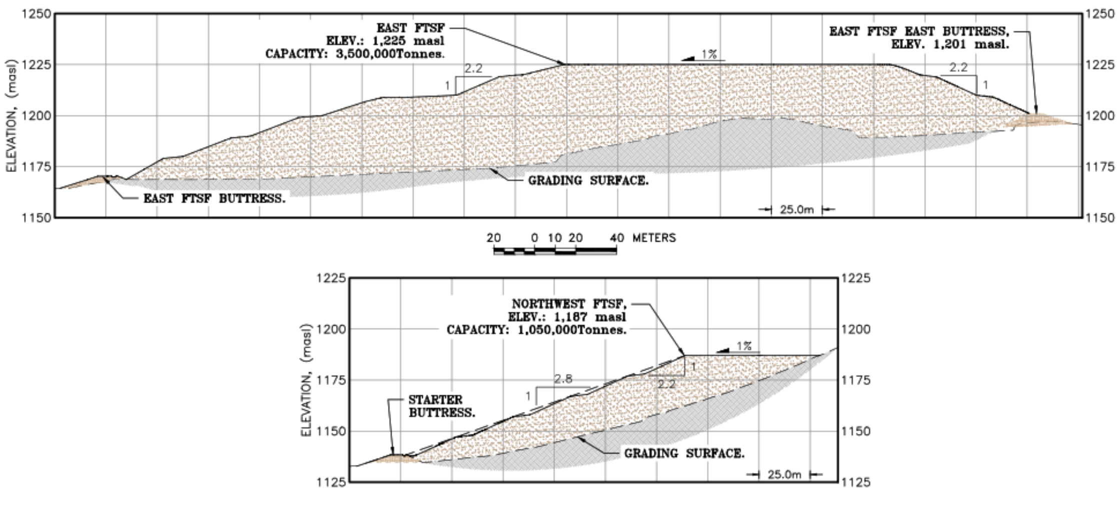
18.15.5 Geotechnical Stability Analyses

Geotechnical stability analysis was conducted using the commercially available computer program Slide v.8 (Rocscience, 2019), which enables the user to perform limit equilibrium slope stability calculations using a variety of methods and failure surface search routines. This software allows analysis of either individual slip surfaces or application of search algorithms to calculate the critical failure surface (i.e., lowest deterministic factor of safety) for a given set of soil shear strength properties, geometry, pore-water pressure and loading conditions.

The stability of the FTSF was estimated for two conditions:

1. Static loading; and,
2. Seismic loading conditions using pseudo-static analyses.

To determine the critical slip surface for static and seismic scenarios, representative cross sections were selected considering the final FTSFs geometries (Figure 18-7).



Note: Figure prepared by Wood, 2021.

Figure 18-7: Cross Sections of the East FTSF (top) and Northwest FTSF

The proposed configurations for the FTSFs met the design criteria for stability under static and seismic loading based on the site conditions and the characterization of the materials that will compose the structures.

18.15.6 Key FTSF Design Elements

The FTSF structures were designed to store together an approximate capacity of 4.5 Mt of tailings with an overall slope of 2.8:1 (H:V), slope between benches of 2.2:1 (H:V), and maximum approximate heights of 50–56 m (measured from the lowest portion of the starting buttress to the maximum elevation of the dry stacks). The required tailings storage capacity is 3.5 Mt therefore, the East FTSF alone can meet the anticipated tailings surface disposal requirements.

18.15.7 Non-Contact Surface Water Diversion Systems

Non-contact surface water diversion will consist of construction of ditches to divert the surface water uphill from the footprint of the FTSF and prevent it from encountering the tailings placed downstream. These diversion channels must be excavated prior to construction of the FTSF to minimize contact water generation and reduce the required size of the contact water management structures.

18.15.8 FTSF Foundation

A competent foundation is fundamental for the stability and adequate performance of the FTSF. Foundation preparation must start with the removal of all vegetation and roots that may remain in the native soil. This includes the removal of all brush and debris, as well as grinding and removal of stumps. These soils may be rich in organic matter and should be stored in a designated topsoil deposit for later use during revegetation and closure activities. Once clearing and grubbing is completed, most of the soil cover in the FTSF area should be removed to be used as borrow material for the starter buttress and perimeter berm structures that will confine the base of the FTSF. Therefore, most of the FTSF foundation material will consist of the shallow underlying bedrock. In those areas where soil does not need to be removed, either for grading purposes or because enough borrow is available for FTSF perimeter or toe structures, the remainder soil cover must be prepared and compacted to mitigate excessive deformation or weak foundation areas.

18.15.9 Contact Water Subdrain System Installation

This system is designed to capture water from possible infiltrations into the FTSF structure. Numerical modelling to simulate the natural processes of precipitation, infiltration and evaporation on the surface of the compacted filtered tailings indicated that infiltration into the filtered tailings would be negligible. However, as an additional environmental and geotechnical risk mitigation measure and to drain potential infiltrations in the FTSF, the installation of a subdrain system at the bottom of the facility was proposed that can direct contact water to the pond downstream for storage, monitoring and reuse in the process plant. This subdrain system will consist of 12” and 18” perforated HDPE pipe and a gravel drain cover. Subdrain details and drawings are provided in the Feasibility Design Report (Wood, 2021).

18.15.10 Contact Water Collection Ponds

Based on information from environmental testing on representative samples, the filtered tailings show little to no potential for acid generation or metal leaching. However, the metallurgical process will use cyanide leaching, which after detoxification will result in the pore water of the filtered tailings having residual CN concentrations. Therefore, contact water resulting from the filtered tailings runoff will need to be collected and stored in this pond for its subsequent reuse in the plant.

The contact water ponds have been proposed downstream of each of the FTSF starting buttresses to capture surface contact water runoff from active tailings placement areas, where there will be solids removal in a sedimentation pond prior to storage, monitoring, potential treatment or pumping back of this water to the process plant.

The contact water for the East FTSF, which will be the first structure to start operating, will have a capacity of 6400 m^3; whereas the pond for the Northwest FTSF will have a 7,000ºm^3 capacity. These ponds were designed to store a 50-year flood and its resulting contact water runoff from the active tailings placement areas of each FTSF.

18.15.11 Starter Buttress

The starter buttress will be built at the toe of the filtered tailings stacking facility to provide stability and erosion protection. This buttress has been designed with 3H:1V and 2H:1V for downstream and upstream slopes, respectively. The structure will have a 6 m wide crest, and a maximum height of approximately 6 m. The starter buttress will be built with native material excavated from the FTSF foundation or approved borrow soils or waste rock. The material that conforms the buttress must not have potential for acid generating or be ML.

18.15.12 Filtered Tailings

Once grading and foundation preparation takes place, and construction of the non-contact water diversion channels, subdrain, contact water ponds and starter buttress is completed, the facility will be ready to receive filtered tailings. The East FTSF will be commissioned first followed by the NW FTSF if required. The East FTSF may be temporarily partitioned in a northern and southern half, where the northern half is prepared first and receives tailings up to one bench height (9 m), while the southern side of the facility is temporarily used by the construction contractors.

Based on the saturated permeability tests and unsaturated testing in filtered tailings, as well as the infiltration modelling in an unsaturated porous medium (Wood, 2021), the tailings will allow only limited infiltration on the surface of the deposit and most of the precipitation will runoff the surface. None to minimal contact water is expected to be generated at the filtered tailings-native ground interface. Therefore, the design does not consider the placement of liner or geomembrane on the FTSF footprint.

The tailings will dewater in the filter plant to a gravimetric moisture content (ww/ws) of approximately 18–20%. The optimum moisture contents for the filtered tailings, based on standard proctor compaction tests, range from 14.5–16.0% and the tailings can reach 95% compaction of the standard Proctor test with approximately ±3% of the optimum moisture content. Given the semi-arid conditions of the site, evaporation will take place between loading, transport by trucks from the filter plant to the FTSF and placement in the FTSF. Therefore, the tailings could be placed and compacted with the moisture they have directly when coming from the filter plant even on light rainy days. On days when there is more precipitation and the tailings exceed the optimum humidity, tailings that have not been processed (graded and compacted) can be covered with HDPE sheeting or raincoats until the weather again allows for moisture loss due to evaporation. The geotechnical design calls for the filtered tailings to be compacted at 95% of the maximum Proctor density value ±3% of the optimum moisture content, and in lifts no greater than 0.3 m.

18.15.13 Coarse Graded Filtered Tailings Cover

After placing and compacting the filtered tailings and having built at least one bench, the surface slopes and those areas that have reached the proposed final grading will be covered with a coarse graded cover of approximately 0.5 m in thickness. The objective of this cover is to protect against erosion, mitigate tailings dust resuspension by wind action or the suspension of solids by surface water runoff and allow for revegetation of the surface as part of reclamation and progressive closure of those areas of the FTSF that have reached its target storage capacity.

18.15.14 FTSF Construction

Some of the key elements of the filtered FTSF can be constructed in parallel or the order change slightly but, in general, the proposed construction sequence of key elements of the FTSF design is as follows:

- Clearing grubbing and grading of the area that the FTSF will occupy;
- Foundation preparation of the FTSF area, subdrain system and structural elements such as starter buttress and contact water pond embankments;
- Construction of the water diversion systems to manage non-contact surface water upstream of the tailings dry stack;
- Installation of a subdrain system to funnel the surface contact water of the FTSF footprint and to collect contact water resulting from infiltration into the FTSF;
- Construction of the contact water ponds for sedimentation, temporary storage and monitoring of contact water prior to reuse in the process plant;
- Construction of the starter buttress at the toe of the FTSF;
- Filtered tailings placement, grading and compaction; and,
- Progressive placement of a coarse material cover on the filtered tailings as it reaches its proposed final grade at each bench to prevent water erosion and dust resuspension and implement its progressive closure.

18.15.15 Contact Water Collector Channels and Collection/Storage Ponds

Each FTSF will have a contact water collection system, consisting of collecting channels that will direct contact water into collection ponds. This contact water management system will be maintained in the first year of closure, with the intention of continuing to capture water runoff from still to be reclaimed areas of FTSFs. The contact water runoff may contain suspended solids and associated metals and residual cyanide. Therefore, the collection system must be maintained if there are exposed tailings areas that can generate contact water or seeps of contact water coming off the subdrain system. However, at the end of the first year if there is no runoff or seepage, the ponds could be removed, and the area regraded and rehabilitated.

18.15.16 Surface Water Monitoring

Surface water monitoring was conducted by SilverCrest environmental personnel to establish an environmental baseline of the site conditions prior to Project development. During operations as well as closure and post-closure SilverCrest must continue to collect water samples upstream and downstream of the mine site. The purpose of this sampling and testing will be to show that water quality is comparable outside and inside the Project area and that the Project is not negatively impacting water quality in the Las Chispas, Babicanora and La Culebra streams. Should the impacts on water quality be noticeable (above the baseline water quality) or significant (above the permissible maximum limits), mitigative measures through engineering controls or treatment should be implemented to comply with local regulatory guidelines and project objectives.

18.15.17 Groundwater Monitoring

Prior to the commencement of the Feasibility Study, SilverCrest had installed seven groundwater monitoring wells to monitor baseline groundwater conditions both inside and outside the mining project area. During the geotechnical investigation campaign for the FTSF design, Wood installed two groundwater monitoring wells upstream and downstream from the proposed FTSF locations. The NOM-141-SERMANAT-2003, Section 5.8.1, states that it is necessary to keep at least one upstream monitoring well and another downstream well to monitor groundwater quality before entering the FTSF area and when leaving the FTSF area. The installation of the groundwater monitoring wells is required per SERMANAT (2003) to the depth where the groundwater table is encountered or down to 50 m below existing grade (whichever is encountered first).

18.16 Off-Site Facilities

18.16.1 Assay Laboratory

The assay laboratory will be located in the town of Arizpe. At the Effective Date of the Report, SilverCrest was negotiating with a private service provider to design and operate the assay laboratory. The decision to locate the laboratory in Arizpe was based on a number of factors, including supporting SilverCrest's environmental, social and governance (ESG) efforts by establishing a business in Arizpe that could potentially outlive the Project and even grow with the addition of other clients. It is also expected that the location in Arizpe will facilitate single family parent employment, for which working remotely from town is not possible.

It is expected that the facility will be built within an existing warehouse and will have 20–30 employees. The facility will handle all the assaying needs for the Las Chispas mine and process plant. The facility will be designed to handle some environmental assays.

Discussions are also on-going to potentially certify the assay facility, so that it could meet exploration assaying needs for other SilverCrest exploration projects in Mexico. Considerations are being given to provide water assay services to the municipality, Arizpe citizens, and/or to local ranchers, since water assaying is not always readily available.

18.16.2 COVID-19 Testing Facility

SilverCrest is currently using a private service provider for all its needs concerning the COVID-19 PCR tests. Starting in 2021, SilverCrest plans to operate its own facility to better control its development schedule. As such, SilverCrest is in the process of procuring a Q-Tower PCR machine and plan to hire a private company to operate that instrument on its behalf. The facility will be located in Hermosillo.

Under the COVID-19 protocol, each employee, contractor, and consultant will require to quarantine prior to being administered the test and while awaiting test results. Upon confirmation of a negative result, the employee will be transported directly to the Project. Ongoing random testing will be conducted on the Project.

19 Market studies and contracts

19.1 Market Studies

Detailed market studies on the potential sale of gold and silver doré from the Project have not been completed. The doré bars produced at the Project can be expected to have variable gold and silver contents and a variable gold to silver ratio, depending mainly on the corresponding gold and silver grades of the feed material being processed at any given time. Over the projected LOM, the metal content is expected to be 0.5%-1.5% gold and 90%-95% silver with the balance impurities.

Gold and silver doré can be readily sold on many markets throughout the world and the market price can be ascertained on demand. A number of mining operations produce and sell gold and silver doré in Mexico and elsewhere, and there is sufficient information available in the public domain or furnished to SilverCrest directly from third party refiners or comparable doré producers to use as the basis for the economic analysis.

19.2 Refining Terms and Conditions

Gold and silver doré is typically refined by third parties before being provided to the market. As of the Effective Date of the Report, no agreements for refining of Las Chispas doré have been concluded. The refining terms used as the basis of the economic analysis in Section 22 are based on an average of payment terms and refining costs provided to SilverCrest in quotations from third-party refiners and are well aligned with other operations in the region. A summary of the terms and costs is provided in Table 19-1.

Table 19-1: Gold and Silver Doré Terms used in the Las Chispas Project Feasibility Study Financial Model

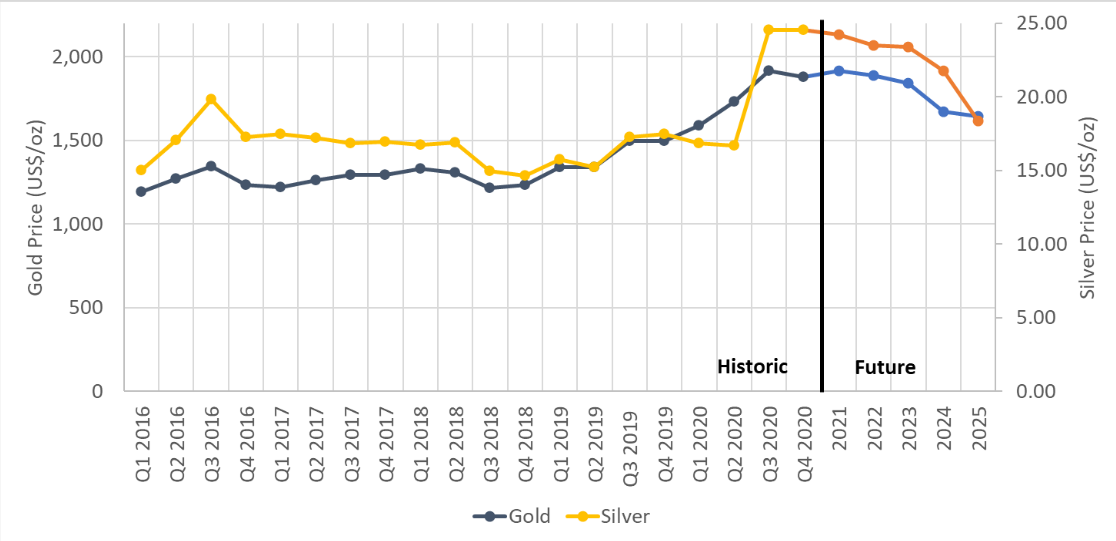
Smelter Terms (Doré)	Unit	Value Used
Gold Payable	%	99.85
Silver Payable	%	99.85
Refining, Transport, and Insurance	$/oz AgEq in doré	0.31

Prior to production, SilverCrest will engage with gold and silver buyers and refiners, and make the necessary arrangements to safely transport, refine, and sell the doré.

19.3 Metal Pricing

Metal pricing used for the Feasibility Study was agreed upon based on consideration of various metal price sources. These sources included review of consensus price forecasts from banks and financial institutions, three-year trailing average of spot prices, and current spot prices.

A summary of five-year historical gold and silver prices and the forecast prices for 2021 – 2025 is provided in Figure 19-1.



Note: Figure prepared by S&P Capital IQ, 2020.

Figure 19-1: Gold and Silver Prices – Historical and Forecast

Based on a review of forecast and current pricing, the metal pricing for the base case economic model in the Feasibility Study is:

- Gold price of $1,500/troy oz payable; and,
- Silver price of $19.00/troy oz payable.

19.4 Contracts

No contracts have been entered into at the Report Effective Date for mining, facility operations, refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements. It is envisaged that SilverCrest would sell any future production through contracts with a refiner, or on the spot market, as applicable.

It is expected that when any such contracts are negotiated, they would be within industry norms for projects in similar settings in Mexico. There are ample regional operations which provide precedent for any such contracts.

19.5 QP Comments

Doré that will be produced from any future Las Chispas operation is expected to be readily marketable, and there are no known concerns with the predicted deportment of impurity or deleterious elements which would adversely affect the refining terms and costs.

The QP is also of the opinion that the doré marketing and commodity price information is suitable to be used in the economic analysis that supports the Feasibility Study.

20 Environmental Studies, Permitting, and Social or Community Impact

20.1 Environmental Review

20.1.1 Baseline and Supporting Studies

Environmental surveys and studies for the Project were completed in support of permit applications. All studies conducted were compiled into the Environmental Baseline report prepared by LLA, that was submitted to SEMARNAT in 2020. Completed studies include climate, flora, fauna, air quality, noise, surface and groundwater quality (Table 20-1).

Table 20-1: Baseline and Supporting Studies

Study Type	Comments
Climate	Described as semi-arid and temperate.
Flora	Surveys were conducted in 2019. Identified 21 perennial terrestrial vascular plant species, including four species of arboreal, 15 shrub, and 2 herbaceous species. Based on the definitions in NOM-059-SEMARNAT-2001 none of the flora species identified in the Project area are in a special protection category that would require specific protective action.
Fauna	Surveys were conducted in 2019. Included 30 wildlife sampling points for mammals, birds, and reptiles. Based on the definitions in NOM-059-SEMARNAT-2001 none of the faunal species identified in the Project area are in a special protection category that would require specific protective action.
Air quality	Surveys were conducted in 2020. Data collected on total suspended particulates (TSP) and particulate matter <10 µm (PM10) to determine baseline compliance against NOM-035-SEMARNAT-1993. Emission values at the time of sampling were below the maximum permissible limits (PST = 2 $\mu g/m^3$; PM10 = 5 $\mu g/m^3$). Once operations commence, periodic sampling to monitor the maximum permissible levels of emissions of pollutants such as smoke (%), particulate matter (ppm), carbon monoxide (CO), sulfur dioxide (SO_2) and nitrogen oxides (NOX) will be required.
Noise	Surveys were conducted in 2020. Collected fixed-source noise data to determine baseline compliance against NOM-081-SEMARNAT-1994. Results showed 61.3 db, for a maximum limit of 68 db in the period 6:00–22:00, and 65 db for the period 22:00–06:00. Until mine development is completed, noise level measurements must be taken annually and reported to SEMARNAT.
Surface water	Surveys were conducted in 2019 and 2020. Data was collected in the dry season, and again in August, after the rainy season, to determine baseline compliance against NOM-127-SSA1-SEMARNAT-1994. Surface water quality regularly meets the current guidelines, with occasional exceedances of fecal coliforms, which appear related to grazing and ranching activities upstream of the proposed mine site.
Groundwater	Surveys were conducted in 2019 and 2020.

Study Type	Comments
	Data was collected from eight groundwater wells to determine compliance against NOM-127-SSA1-SEMARNAT-1994. Data were collected in the dry season and again in August, after the rainy season. Groundwater pH ranges from 6.5–8.6. The groundwater quality exceeds maximum allowable limits for total coliforms, iron, nitrogen and turbidity. These exceedances in baseline groundwater quality are interpreted to reflect pre-mining impacts from other activities upstream of the project area such as grazing and ranching.
Archaeology	The National Institute of Anthropology and History maintains a catalogue of monuments of cultural and historical value by municipality. The 2020 catalogue has no records for the Las Chispas mine site or any associated historical or cultural monuments in the section pertaining to the municipality of Arizpe, Sonora. No site investigation was performed.

20.1.2 Geochemistry

Samples of waste rock from exploration drill holes and test pits in the FTSF footprint area were sent to ALS in Monterrey, NL, Mexico (ALS) for determination of acid rock drainage (ARD) and metals leaching (ML) potential to determine compliance against NOM-157-SEMARNAT-2009. Results showed:

- Exploration drill holes: Potentially leachable metals included barium and lead, but in concentrations that were well below the maximum allowable limits, which are 100 and 5 mg/L, respectively. Neutralization potential as measured in $CaCO_3$ kgpt of waste was >20. The acid potential was zero. All pH measurements were >9; and,
- Test pits: Barium was the only potentially leachable metal, but in concentrations that were well below the maximum allowable limits. All pH determinations were >8. The neutralization potential as measured in $CaCO_3$ kgpt of waste ranged from 0.51–1.51. One sample returned an elevated neutralization potential/acid potential ratio of 0.025; however, this value is below the minimum ratio of three set out in NOM-157-SEMARNAT-2009.

Tailings samples generated from metallurgical testwork at SGS Lakefield were subjected to acid base accounting (ABA) and net acid generation (NAG) testwork. The majority of samples showed non-acid forming (NAF) characteristics in NAG testing.

A composite sample of tailings was sent to ALS in Monterrey to conduct environmental testing in accordance with NOM-141-SEMARNAT-2003. No potentially-leachable metals of concern were identified. The sample showed a high neutralization potential and no acid generating potential.

Humidity cell testing will be performed on material for which test-work indicated that it could be potentially acid generating or had non-acid forming characteristics.

SGS Lakefield completed cyanide destruction testwork on tailings samples. The aim was to produce samples from the optimized circuit conditions that would have treated pulp containing <1 mg/L of residual CN_{WAD} using the SO_2/air detoxification process. At a bench scale, based on 48% solids w/w, the amount of SO_2 and copper addition as well as the pH conditions to reach the target <1 mg/L residual CN_{WAD} were determined.

20.1.3 Environmental Liabilities

No known environmental liabilities exist in the Project from historical mining and processing operations. Soil and tailings testing were conducted as part of the overall sampling that has been ongoing at site. To date, there are no known contaminants in the soils. Water quality testing is currently ongoing through baseline environmental studies.

20.2 Permitting

20.2.1 Overview

SEMARNAT requires that a number of studies be completed to support award of environmental permits to conduct exploration, or construct and operate a mine:

- Mining Exploration Permit I: Defined in a 2011 SEMARNAT standard, NOM-120-SEMARNAT-2011, which sets out what environmental protections actions must be taken for exploration activities in active or inactive agricultural areas, or in areas that have a dry or temperate climate and are home to deciduous, coniferous or oak forests, or dry-adapted scrub;
- Environmental Impact Assessment (MIA in the Spanish acronym): sets out the environmental impacts of any work/activity and outlines the mitigation strategies proposed. It is supported by baseline studies and surveys; and,
- Change in Land Use (CUS in the Spanish acronym): applies when partial or total vegetation is planned due to non-forestry activities, and the area to be cleared is larger than that allowed under NOM-120-SEMARNAT-2011. A technical justification study (ETJ in the Spanish acronym) must be prepared and presented to SEMARNAT in support of any change in land use proposed.

In addition to the SEMARNAT requirements, permits must also be obtained in certain instances from the (CONAGUA), Comisión de Ecología y Desarrollo Sustentable del Estado de Sonora (CEDES), Secretaría de la Defensa Nacional (SEDENA) and local municipal authorities.

The final licence requirement is the environmental operating licence (LAU in the Spanish acronym). The LAU sets out operating conditions, including specifications around equipment and processes, production, air emissions, hazardous waste and water impact obligations.

20.2.2 Permits to Support Construction and Operations

LLA worked with its permitting team in Mexico to identify the key environmental permits and other Mexican regulatory permits required to construct and operate a mine in Sonora state, Mexico, and to identify which regulatory authorities grant such permits. These are summarized in Table 20-2. The table also shows the current permitting status, indicating which permits have been granted and the status of others.

Table 20-2: Key Permit List

Permit	Current Status	Agency
Mining Exploration Permit on first exploration phase	Completed	SEMARNAT
Mining Exploration Permit on second exploration phase	Completed	SEMARNAT

Permit	Current Status	Agency
Mining Exploration Permit on third exploration phase	Completed	SEMARNAT
Mining Exploration Permit on fourth exploration phase	Completed	SEMARNAT
Mining Exploration Permit on fifth exploration phase	Completed	SEMARNAT
MIA titled, “Las Chispas project”	Completed	SEMARNAT
MIA titled, “Ampliación de camino de acceso a Las Chispas”	Completed	SEMARNAT
MIA titled, “Ampliación de Mina Las Chispas”	Completed	SEMARNAT
Land use change document titled, “Ampliación de Mina Las Chispas”	Completed	SEMARNAT
MIA titled, “Ampliación 2 Las Chispas”	Completed	SEMARNAT
MIA titled, “Acceso a mina Las Chispas Project”	Completed	SEMARNAT
Land use change document titled, “Ampliación 2 Las Chispas project” (also cover “Acceso a mina Las Chispas project”	Completed	SEMARNAT
MIA and land use change document titled, “LT Los Hoyos – Mina las Chispas Project”	Pending	SEMARNAT
Permit to allow bridge construction	Pending	CONAGUA
Title on water rights grant (300,000 m3)	Completed	CONAGUA
Water rights transfers (modification to title)	Pending	CONAGUA
Permit allowing sewage discharge	Pending	CONAGUA
Federal registration as a hazardous waste generator	Completed	SEMARNAT
State registration as a hazardous waste generator	Completed	CEDES
Municipal permit for garbage disposal	Completed	Local municipality
Hazardous and mining waste management plan	Not started	SEMARNAT
Waste requiring special handling plan	Completed	CEDES
Renewal of annual permit for storage and use of explosives for 2020	Completed	SEDENA
Renewal of annual permit for storage and use of explosives for 2021	Completed	SEDENA
Quarterly request for permit to allow purchase of explosives for 2020	Completed	SEDENA
General permit for the purchase, storage and use of explosives (and modifications)	Completed	SEDENA
Environmental operating license	Not started	SEMARNAT

Table 20-3 summarizes the duration and purposes of the granted permits.

Table 20-3: Current Permits and Validity

Permit Name	Permit Number	Grant Date	Term	Comment
MIA for the Las Chispas Project	DS-SG-UGA-IA-0669-16	September 2016	10 years	Exploration and mineral extraction activities associated with underground mining. Originally allowed for usage of 3.1 ha for stockpiles and waste rock storage. Subject to four modifications approved by SEMARNAT. Disturbed area allowance expanded to 22.95 ha, to allow for portal and decline expansion, laydown area and explosives magazine.
MIA for access road and bridge	DS-SG-UGA-IA-0268-19	June 2019	14 years	Construction of a road from Km 86 of the Mazocahui–Cananea highway to the mine site. Approval also covers any required turn-out and rest areas. Allowed disturbance area of 14.3 ha. Agreements reached with six surface rights owners affected by the easement.
Updated MIA for planned expansion of area that will be affected by development of Las Chispas Project	DS-SG-UGA-IA-0341-19	July 2019	14 years	Covers the area required for process plant, FTSF, support and administrative facilities, internal roads, environmental monitoring equipment, and fencing/berms. Allowed disturbance area of 96.7 ha.
Updated MIA for planned expansion of area that will be affected by development of Las Chispas Project	DS-SG-UGA-IA- 0204/20	August 2020	14 years	Covers additional area required for internal roads, water pipelines, ventilation raises and fans, laydown areas, WRSF areas, and topsoil storage. Allowed disturbance area of 1,414.7 ha.
Updated MIA for access road and bridge	DS.SG-UGA-IA-0244/20	August 2020	14 years	Covers final road design, and vehicular and pedestrian bridge to be constructed over the Sonora River. Allowed disturbance area of 26.9 ha.

Permit Name	Permit Number	Grant Date	Term	Comment
Land use change permit in support of Las Chispas Project development	DFS/SGPA/UARRN/59/2020	June 2020	3 years	Authorizes removal and clearing of vegetation. Requires appropriate environmental offset activities, including rescue and relocation of affected species, mitigation of effects on surface and groundwater, and soil conservation. Work is being supervised by a forestry expert from SEMARNAT.
Water usage permit	No. 826243	October 2020	10 years	Allows for use of water from underground, at the rate of 300,000 m^3/year
Disposal of hazardous waste	No. MLA2600600003	May 2019	Unlimited	Regulates the generation, handling, storage and disposal of hazardous waste
Disposal of waste requiring special handling	No. CEDES-RGRME-19-121 (2020)	October 2020	1 year	Revalidated every year. It refers to those wastes that can be recovered through recycling.
Disposal of solid waste	Not applicable (consent letter from municipality)	August 2019	Unlimited	Approval from the municipality of Arizpe, Sonora, to use the municipal landfill facility.
General permit for the purchase, storage and use of explosive materials for 2020	No. 5131-SON	January 2020	1 year	Current permit restricted to explosives usage in six mineral concessions. Application made to extend permit to a further 18 mineral concessions. Permit allows for explosive storage in two mineral concessions. Application lodged for renewal of permit for 2021.

20.2.3 Power Line Los Hoyos – Mina Las Chispas Project

Documentation was lodged with SEMARNAT to allow construction of a 33 kV medium-voltage overhead transmission line that will run from the settlement of Ejido Los Hoyos, in the municipality of Cumpas, Sonora to the proposed mine site. The proposed powerline easement crosses surface rights on 13 properties. LLA has signed contracts with the affected parties in relation to the easement and has started process to register 2 of these contracts with the Registro Agrario Nacional (RAN).

20.2.4 Permit to Allow Bridge Construction

The proposed bridge design for the Sonora River crossing is currently being evaluated by CONAGUA, which is responsible for issuing the bridge construction permit. This bridge will be needed slightly ahead of the start of operation.

20.2.5 Water Rights Transfers

LLA has a concession title for water use und registration number 826243 that was granted by CONAGUA (National Water Commission) in October 2020. This authorization allows the use of 300,000 m^3/year subsurface water for mining purposes.

This license was obtained through an initial agreement for a partial assignment of rights contract signed in 2019, the procedure for which was evaluated and authorized by CONAGUA. CONAGUA has not yet closed the administrative procedure on the license transfer as the initial water rights license included other assignees i.e., the initial water rights license area was larger than the LLA 300,000 m^3/year.

20.2.5.1 Sewage Discharge Permit

This discharge permit is pending water rights transfer as per above. Once CONAGUA allows the transfer, LLA will apply for the permit.

20.2.5.2 Hazardous and Mining Waste Management Plan

This permitting process will be initiated in May 2021 when LLA current permit will be due for a renewal. The current registration is valid until that date.

20.2.5.3 Renewal of Annual Permit for Storage and Use of Explosives for 2021

This permit is required annually through SEDENA. It is usually filed in August of each year. SEDENA typically grant all these permits in January of each year.

20.2.5.4 General Permit for the Purchase, Storage and Use of Explosives

LLA has held a 5131-SON permit granted by SEDENA since 2019. A modification of this permit was requested in September 2020 to include 18 additional LLA mining concessions to those covered by the permit that allow the use of explosives for mining development and services.

20.2.5.5 Environmental Operating License

The LAU has not yet been applied for, as this requires the process plant to have been built and tested. It will be applied for once all construction is complete and the plant has been certified as operational.

20.3 Social and Community Requirements

Arizpe has a population of 3,037, of which 1,571 are male and 1,466 are female. According to the 2010 INEGI Census, this represents 1% of the population of the state of Sonora. The average household size in the municipality is 3.3 members, while in the state the average size is 3.7. The population of Arizpe is divided into 971 minors and 1,988 adults, of which 523 are over 60 years old.

The Sonora Valley includes several isolated municipalities set in a region of rugged topography. The areas planned for mining activity are not visible from the local communities or from adjacent roads.

As of November 2020, SilverCrest employed 85 people from the Sonora Valley.

There are four main ejido groups that SilverCrest have been engaging with, three of which will be impacted by mining operations (Ejido Bamori, Ejido Arizpe, and Ejido Sinoquipe) and the fourth (Ejido Los Hoyos) will be impacted by the powerline:

- Ejido Bamori had 84 members as of November 2020. LLA maintains constant and direct dialogue with the Ejido members. LLA attends the monthly Board of Directors meetings, the annual general meeting (AGM) and, as required, may attend periodic meetings. LLA has a 20-year lease agreement with Ejido Bamori that applies to 400 ha of land within the Project area. The Ejido Bamori controls 9,184 ha;

- Ejido Arizpe had 348 members and Ejido Sinoquipe had 116 members as of November 2020. LLA maintains constant and direct dialogue with both group of Ejido members. LLA attends the monthly Board of Directors meetings, the AGMs, and, as required, may attend periodic meetings; and,

- Ejido Los Hoyos has 156 members. In 2020, LLA contractually agreed upon a right of way relationship with the Ejido to have land access to build the Project power line. Cattle ranching is a significant part of the economy in Sonora state and in the Sonora Valley. SilverCrest established a ranching business, Babicanora Agricola del Noroeste S.A de C.V (BAN)., and as of November 2020, the ranch has 75 cows and seven bulls. BADN formally applied to become a member of the local cattle ranching association, the Asociación Ganadera Local de Arizpe in February 2020. Due to COVID-19, this application has not been formally processed and no formal association meetings have taken place in 2020. This association typically holds its AGM in the first quarter, annually. Active participation in this organization is an important community communication channel.

LLA started a Social Baseline Study in March of 2019 and the report was finalized in January of 2020. The main findings were as follows:

- The community lacked details about and knowledge of the Project;

- The community had concerns relating to a 2014 environmental incident in Cananea (some 100 km north of Las Chispas), and outside the control of SilverCrest;

- The community wished to see improvement to their local infrastructure;

- The communities requested that adequate environmental safety and appropriate mine closure protocols were in place in order to protect the region at the end of the mine life;

- The communities desired more jobs with a focus on providing women with opportunities; and,

- The communities mentioned water safety and scarcity as regional concerns.

In early 2020, SilverCrest engaged two third-party ESG consultants, the Business for Social Responsibility (BSR) and the Stonewave Group Ltd. (Stonewave), to complete a Materiality Assessment designed to identify the key risks facing SilverCrest including potential risks relating to SilverCrest's relationship with and impact on local communities. A detailed stakeholder analysis was completed that included interviews with the mayor of Arizpe, the owner of a key contracting partner and a workshop with members from the local community. Key findings predominantly echoed those found within the social baseline study and were centered around climate and water risks, community health issues (mining, food, water), environmental safety of the local river and agriculture, employment opportunities, a desire for improved infrastructure (sports, recreation, health) and a concern regarding a potential influx of people from outside the community taxing local infrastructure.

As part of SilverCrest's ESG framework, and in response to the issues raised within the social baseline study and the Materiality Assessment, SilverCrest formalized a community communication strategy that includes direct outreach, use of social media, presentation of company-generated videos, flyers, posters and workshops. SilverCrest set up a whistle blower policy and hotline and is in the process of finalizing a grievance mechanism. This mechanism is planned to be in place in the first quarter of 2021. SilverCrest is planning to have Stonewave create an action plan that converts the Materiality Assessment findings of the into a company-wide Environmental and Social Management System (ESMS).

SilverCrest joined the Sonoran Mining Cluster, an organization consisting of mining companies based in Sonora, that aims to share best practices on social license concerns, innovation, sustainability, community relations and responsible mining.

SilverCrest is one of the major sponsors in a non-profit organization (Impulso Koria A.C.) located in Arizpe. Impulso Koria's objectives include supporting local infrastructure, education and health care needs. SilverCrest communicates with Impulso Koria representatives on a regular basis as part of local CSR efforts.

Impacts to Indigenous populations were examined. There are no indigenous populations located within 10 km of the Project.

20.4 Closure Considerations

20.4.1 Conceptual Closure Plan

A Conceptual Closure Plan was prepared in general accordance with applicable Mexican standards and Wood's experience with similar projects. Under Mexican law, mining may be initiated under a Conceptual Closure Plan with a Detailed Closure Plan being developed later in the Project life.

The Conceptual Closure Plan incorporates the most recent engineering information from the Feasibility Study, as well as environmental information provided by SilverCrest, which includes on-going environmental baseline studies, MIAs, environmental laboratory testing results and data that supplements the granted environmental permits.

The Conceptual Closure Plan focuses on ensuring the post-mining landscape is safe and physically, geochemically, and ecologically stable. The plan ensures that the quality of water resources (possible effluents) in the area is protected and that the restitution plan is welcomed by communities and regulators. The optimum performance of reclamation activities heavily depends on stakeholder participation and adequate monitoring of the reclaimed site conditions.

The objectives of the Closure Plan include minimizing long-term environmental liabilities, complying with current legislation, and observing international standards and best practices for long-term environmental protection. The reclamation process should lead to a stable terrain configuration that can be used for other purposes, such as conservation, recreation or other services.

The main objectives of the Conceptual Closure Plan are:

- Cessation of activities that cause disturbances or impacts (noise, lights, dust, vehicle traffic, etc.);
- Physical, chemical, and biological stabilization of impacted land;
- Ensuring appropriate environmental compliance;
- Minimizing risks to safety and public health; and,
- Reclamation of the mine site to similar site conditions that were present prior to mining.

20.4.2 Closure and Reclamation Areas

The total anticipated disturbed area from mining operations is expected to be approximately 95 ha, including:

- FTSFs, associated water management structures (ponds, channels, etc.): 17 ha;
- Access roads: 13 ha;
- Temporary WRSF and stockpile: 5 ha; and,
- Buildings, yards, process plant, power plant, and miscellaneous infrastructure: 12 ha.

The FTSFs represent the biggest surface-impacted areas to be reclaimed and the most challenging in terms of ensuring the long-term physical and chemical stability of the waste to remain on site in the post-closure stage. The greatest closure efforts, starting with progressive reclamation, should be focused on these facilities.

Closure will include:

- FTSFs: Scarifying and grading of temporary access roads, benches and slopes; use of an inert cover material; covering the facility with a layer of topsoil to promote vegetative growth; closure of water management infrastructure; and revegetation;
- Facilities: Buildings will be dismantled, donated, retired, and/or kept;
- Portal, shafts and adits: Will be sealed to prevent access from surface;
- WRSF and stockpile: Planned to be depleted prior to cessation of mining. Disturbed footprint areas will be graded and reclaimed;
- Waste and water storage ponds: Will be demolished, and/or filled, graded and reclaimed;

- Water and miscellaneous tanks: May be donated, sold, dismantled or demolished;
- Water reservoir: Will be left in place to supply local pasture or farming water needs;
- Pipelines: Will be dismantled and recycled; and,
- Access roads: The main access roads will be maintained during the monitoring phase. Secondary roads that are no longer needed will be regraded, closed, and revegetated.

20.4.3 Conceptual FTSF Closure

The proposed East and NW FTSFs will be located close to the Babicanora stream, and therefore, water management will be a prime consideration during the closure and post-closure periods. The potential impact of any FTSF contact water runoff on existing surface water streams will need to be properly mitigated.

A progressive reclamation approach is recommended where, as each slope or bench of the FTSF is completed, it is immediately covered with a coarse protective layer to minimize surface erosion. This process will result in progressive reclamation of a great portion of the FTSFs areas prior to closure. Therefore, most of the costs for the cover materials for the FTSFs will be incurred during operations rather than closure. The final operational bench of each FTSF will require, at the time of closure, grading of the disposed tailings to provide positive drainage towards the slopes and perimeter channels of the facilities. The cover layer will be composed of inert, non-acid-generating material with a minimum thickness of 0.5 m. A layer of organic soil that promotes the growth of native vegetation will be placed on top of each FTSF. The organic soil layer is proposed to have a uniform thickness of 0.2 m to facilitate root and vegetation development.

The FTSF design includes contact water collection structures during operation, to keep contact water from impacting the native ground, surface and subsurface water in the Project. These structures will need to be well maintained and remain operational during the lifetime of the FTSFs. The contact water ponds and channel systems will remain active for at least one year after operations cease in case there is a need to capture contact water that could drain from the FTSFs. However, based on infiltration tests and numerical modeling, and assuming proper construction and operation of the facility, infiltration into the tailings is expected to be negligible. Once the contact areas are reclaimed and if the monitoring results indicate that water is of acceptable quality for discharge, the ponds will be filled, the surface will be graded and rehabilitated. Water runoff will be directed to the natural downstream creek bed, and finally the closed and revegetated FTSF would integrate into the surrounding environment.

20.4.4 Conceptual Closure Cost Estimate

Wood prepared a conceptual closure cost estimate for the planned operation, using a combination of information derived from the Feasibility Study, drone imagery of existing facilities and landforms, a database of itemized costs from local contractors working on similar projects in the area, and assumptions derived from Wood's experience in mine closure. The estimated cost is approximately $3.4 M. Closure costs are assumed to be disbursed over a period of approximately three years, following the cessation of production.

21 Capital and Operating Costs

21.1 Introduction

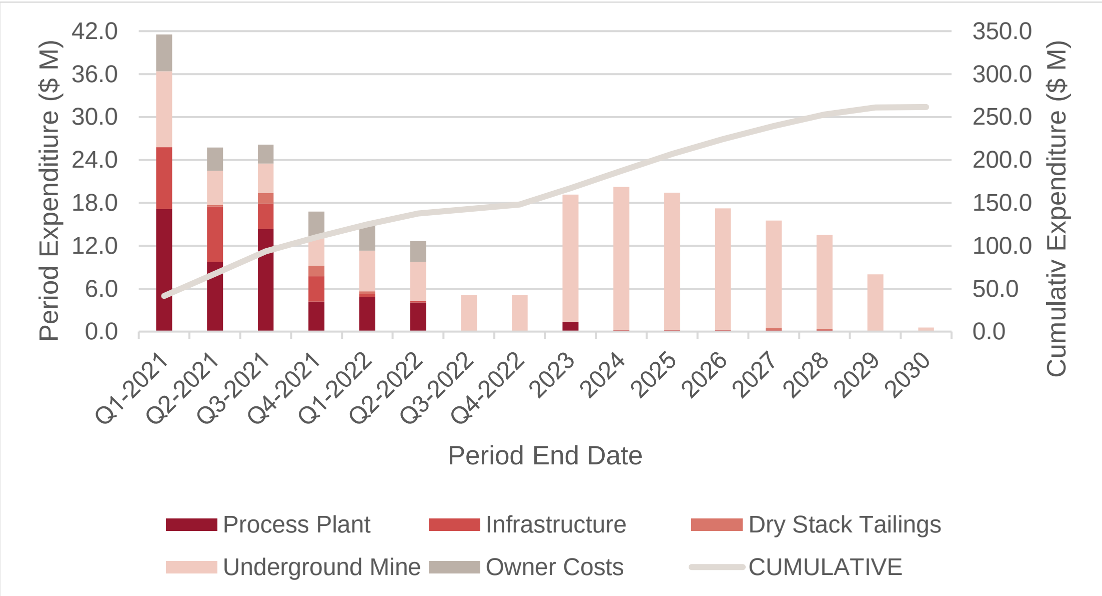
LOM Project capital costs total $265.0 M, which can be broken down to the following phases:

- Initial capital cost: includes the costs required to construct all of the surface facilities, and underground development to commence a 1,250 t/d operation. The remaining initial capital cost is estimated to be $137.7 M after the subtraction of $25.8 M of sunk capital expensed in 2020 (from a total of $163.5 M). The sunk capital was expensed for initial earthworks, some surface infrastructure, detailed engineering, the initial procurement and contract management (EPCM) milestone payment (to finance the purchase of long-lead equipment items);

- Sustaining capital costs: include all the costs required to sustain operations, with the most significant component being underground mine development. Sustaining capital costs total $123.9 M over the LOM; and,

- Closure costs: include all the costs required to close, reclaim, and complete ongoing monitoring of the mine once operations conclude. Closure costs total $3.4 M.

Figure 21-1 presents the forecast annual capital cost expenditures over the LOM (excludes closure costs).



Note: Figure prepared by Ausenco, 2021.

Figure 21-1: LOM Capital Cost Profile (excluding closure costs)

21.2 Initial Capital Cost Estimate

The capital cost estimate was developed by Ausenco, with input from SilverCrest, Wood and GMS.

The capital cost estimate was developed using a hierarchical work breakdown structure (WBS). The base currency of the estimate is US dollars.

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is $137.7 M. Table 21-1 shows a summary breakdown of the initial capital cost estimate.

Table 21-1: Initial Capital Cost Summary

Project Scope	Total Cost ($ M)
Mine	27.7
Process plant	68.0
Tailings management	3.1
Infrastructure	23.3
Owners costs	18.2
Subtotal	**140.3**
Contingency	23.3
Total initial capital cost	**163.5**
Sunk Capital	25.8
Total initial capital cost (remaining)	**137.7**

Note: Total may not add due to rounding.

21.2.1 Basis of Capital Cost Estimate

21.2.1.1 Estimate Base Date and Validity Period

The capital cost estimate was prepared with a base date of Q3-2020 except for owner's costs which are based on Q4-2020.

21.2.1.2 Class of Estimate and Accuracy

This is a Class 3 estimate prepared in accordance with AACE International's Cost Estimate Classification System. The accuracy range of the capital cost estimate is ±15%.

21.2.1.3 Foreign Exchange

Table 21-2 shows the assumed foreign exchange rates for the US dollar to the Canadian dollar (CAD), and for the US dollar to the Mexican peso (MXN), which were applied as required.

Table 21-2: Foreign Exchange Rates

Base Currency ($)	Currency
1.00	CAD 1.325
1.00	MXN 20.00

21.2.1.4 Exclusions

The following items were excluded from the capital cost estimate:

- Financing costs;
- Refundable taxes and duties;
- Land acquisition;
- Currency fluctuations;
- Lost time due to force majeure;
- Additional costs for accelerated or decelerated deliveries of equipment, materials, or services resultant from a change in project schedule;
- Warehouse inventories, other than those supplied in initial fills;
- Any Project sunk costs prior to January 2021 (exploration and preliminary mine development) and $25.8 M for initial earthworks, some surface infrastructure, detailed engineering and the initial EPCM milestone payment; and,
- Escalation costs.

21.2.2 Mining Capital Cost Estimate

SilverCrest elected to have the mining capital cost estimate based on contractor mining for both underground development and underground mining of stopes. Capital cost estimates were based on a combination of budgetary quotes from the mining contractor, GMS cost databases, and review of costs from similar mines in Mexico and South America.

It is anticipated, with a few exceptions, that the mining contractor would own the various mobile mining equipment. Fixed under-ground infrastructure was assumed to be procured by SilverCrest and installed by a mining contractor.

Pre-production costs (operating costs) were not included in the initial capital cost.

A total of $27.7 M was estimated for initial underground mining capital for the period of Q1-2021 to Q2-2022. Table 21-3 shows a summary of mining capital costs, as included in the financial model in Section 22.

Table 21-3: Initial Mining Capital Cost Summary

Area	Initial Capital ($ M)
Mine development	17.4
Mine infrastructure	10.2
Total initial mining capital cost	**27.7**

Note: Total may not add due to rounding.

Starting in January 2021, the total initial capital required for development was estimated at $17.4M for 6,773 m of development, for an average all-inclusive cost of $2,574/m (contractor overhead, drill, blast, haul, ground control, ventilation, explosive and services). The cost per metre was calculated from first principles by GMS and validated by the actual cost experienced from >6,000 m of development completed in 2020 by SilverCrest at Las Chispas.

This cost estimate excludes the pre-production development of an additional 3,266 m required for in-vein drifting and pivot drives at a cost of $7.0 M or $2,441/m. This cost is treated as operating costs in the financial model as SilverCrest uses the International Accounting Standards (IAS) 16, Property, Plant, Equipment – Proceeds before Intended Use. An amendment to IAS 16 prohibits SilverCrest from deducting the cost of an item of property, plant and equipment or any proceeds from selling items produced, while bringing that asset for its intended use. With the adoption of the amended standard, revenue from sales of precious metals recovered and related costs while bringing a mine in a condition necessary for it to be capable of operating in the manner intended by management were recognized as part of the profit or loss. SilverCrest will use IAS 2, inventories to measure the cost of those items.

The initial costs for underground infrastructure were estimated from first principles by GMS at $10.2 M.

21.2.3 Processing and On-Site Infrastructure Capital Cost Estimate

21.2.3.1 Processing Plant

Major mechanical costs were prepared based on firm quotations from qualified vendors. All equipment and material costs were based on free carrier (FCA) or free board marine (FOB) Incoterms and were exclusive of spare parts, taxes, duties, and freight. These costs, where applicable, were considered to be indirect costs.

Where appropriate, material quantities were developed from process flow diagrams, mechanical and electrical equipment lists, and the 3D model of the process plant. Instrumentation and piping costs were based on historical information from similar projects.

Discipline labour rates, rates of placement and productivity estimates were estimated based on unit rate quotations received from experienced contractors. Estimates of labour hours included an allowance for the impacts of COVID-19 restrictions at site and were based on a three-week-in, one-week-out rotation.

Project indirect costs, including construction indirect costs, spare parts, and freight and logistics, were calculated on a percentage basis, based on Ausenco experience. EPCM, commissioning and start-up, and vendor assistance allowances were estimated from first principles.

SilverCrest has entered into a fixed price Engineering, Procurement and Construction (EPC) contract with Ausenco Engineering Canada Inc. and one of its affiliates, for construction of the process plant. The EPC contract provides SilverCrest with price certainty for the process plant, consistent with the capital cost estimate.

21.2.3.2 On-Site Infrastructure

Estimates for on-site infrastructure were compiled from quotations received from experienced contractors for design, supply and installation of fixed work scopes. Selected components of these works were commenced as part of an early works program at site.

21.2.3.3 Process Plant and On-Site Infrastructure

Table 21-4 shows the estimated total initial capital cost for process plant and on-site infrastructure.

Table 21-4: Initial Process Plant and On-Site Infrastructure Capital Cost Summary

Description	Initial Capital ($ M)
Process plant	44.0
Crushing, stockpile and reclaim	2.1
Grinding and flotation	8.3
Leaching and counter current decantation	10.7
Merrill Crowe and refinery	6.1
Detoxification and tailings filter	9.0
Reagents	6.7
Plant utilities	1.1
Related infrastructure	9.0
Power supply and distribution	7.4
Utilities	1.6
Subtotal direct costs	**52.9**
Field indirect costs	2.5
EPCM services	12.6
Subtotal indirect costs	**15.1**
Total initial capital cost	**68.0**
Sunk Capital	**23.2**
Total initial capital cost (remaining)	**44.9**

Note: Total may not add due to rounding.

21.2.4 Off-Site Infrastructure

Table 21-5 shows the estimated initial capital cost for the off-site infrastructure.

Table 21-5: Initial Off-Site Infrastructure Capital Cost Summary

Description	Initial Capital ($ M)
Site development	2.7
Off-site access (road and bridge)	3.1
High and medium voltage power supply	10.0
Assay laboratory	2.3
Buildings	1.4
Mobile equipment	3.0
Water supply	0.7
Total initial infrastructure capital cost	**23.3**
Sunk capital	**2.7**
Total initial infrastructure capital cost (remaining)	**20.6**

Note: Total may not add due to rounding.

The cost estimate for the 9.5-km long access road was estimated from first principles and validated with contractor quotes based on material take-offs. The estimate for the 171 m single-lane vehicular bridge across the Rio Sonora was based on first principles calculation and validation with contractors. As of the Effective Date of the Report, the contracts for both the road and the bridge were being finalized.

The power line initial cost estimate was provided by an independent Mexican contractor with the support from CFE and SiverCrest, and totals $10.0 M (including on-re medium voltage power reticulation). The initial cost estimate includes up-grade of the existing CFE infrastructure over a distance of 26 km, and a new 33 kV double-wooden pole line over a 49 km distance. The cost estimate covers the right-of-way that was negotiated in Q4 2020 within nine different agreements. It also incorporates the expected forestry compensation, including fauna and flora re-location, and other indirect costs such as legal and permitting.

The initial cost estimate for the assay laboratory was provided by a specialized firm involved in the development of mining-related assaying laboratory and provision of some support from SilverCrest. The initial capital estimate of $2.3 M includes land acquisition costs, the building, equipment and equipment installation, and commissioning.

21.2.5 Initial Filtered Tailings Storage Facility Capital Cost Estimate

Initial capital costs for the FTSF were based on earthworks unit rates and labour cost for installation of a drainage system as provided by Mexican contractors. The material take-off for earthworks and mechanical equipment was developed by Wood.

The capital cost for the FTSF is $3.1 M as shown in Table 21-6.

Table 21-6: Dry Stack Tailings Facility Capital Cost Summary

Area	Initial Capital ($ M)
Tailings management	2.4
Water management	0.3
Subtotal direct FTSF capital cost	**2.7**
Indirect Costs - EPCM services	0.4
Total FTSF capital cost	**3.1**

Note: Total may not add due to rounding.

21.2.6 Owner Costs

Owner's costs were prepared by SilverCrest. The estimate was calculated from first principles and based on the manpower required to operate the Project. The manpower ramp-up from operation was assumed to start in Q1-2021 adding to the current 76 employees.

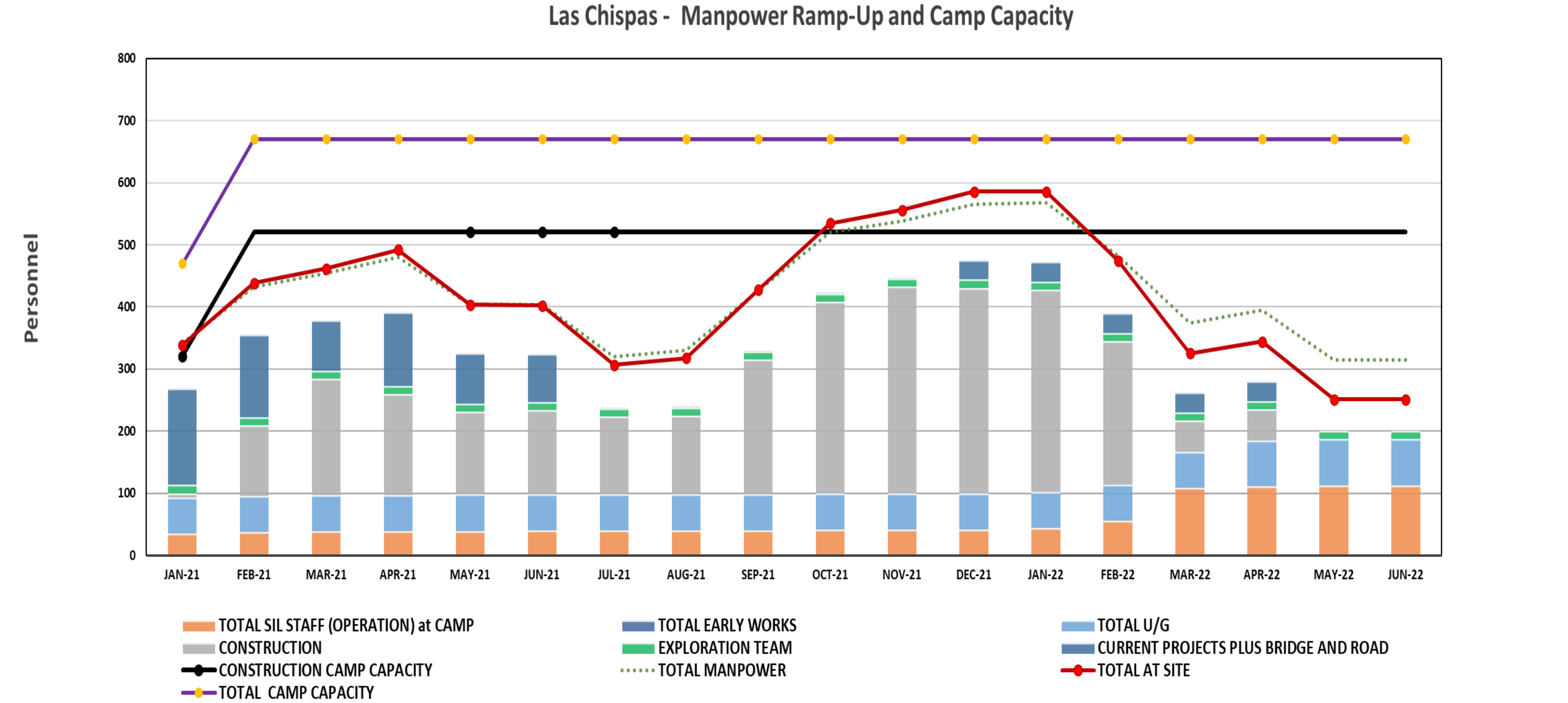
Given the effects of the pandemic at the Report Effective Date, SilverCrest decided to initiate Project construction using a confined camp (Section 18.2). The manpower loading was estimated inclusive of the consultants and contractors. Figure 21-2 provides the manpower ramp-up estimated for all employees expected at site during the 17 months of construction.

The Owner's costs, as presented in Table 21-7, include costs for all of the operating and service departments during the construction period, starting in January 2021 and finishing in May 2022. Personnel employed will increase from the current 76 to a peak of 212. The manpower hiring plan includes provision for the required training to ensure a successful operation ramp-up. The construction team is expected to peak in November 2021 with 330 workers and staff. As the construction manpower requirement declines in Q1 and Q2 of 2022, the LOM assumes a gradual ramp-up of the underground contractor manpower. The overall peak manpower is expected to reach 585 (including contingency) in the period from December 2021 to January 2022.

The Owner's cost estimate covers direct costs associated with construction camp operation, and operation of a second isolated camp located in Arizpe that has been designed for up to 150 workers. It is expected that the average manpower during construction will be slightly below 450 workers. The two camps will be capable of housing a maximum of 670 workers.

The Owner cost estimate includes all costs associated with COVID-19 protocols, including quarantining in hotel rooms in Hermosillo, COVID-19 polymerase chain reaction (PCR) testing and controlled direct transportation to site. The protocols also cover medical staff at site, including additional random testing.

Stacked Bar Graph Shows Total Manpower At Camp



Note: Figure prepared by SilverCrest, 2020.

Figure 21-2: Site Staffing and Camp Capacity

Table 21-7: Owners Costs (January 21 to May 22)

Item	Manpower ($ M)	Contract ($ M)	Supplies ($ M)	Initial Capital ($ M)
Accounting/procurement/administration	0.9	0.4	0.5	1.7
Management and community relations	0.4	1.4	0.0	1.8
Human resources and information technology	0.4	0.2	0.2	0.8
Health/environment/safety/security	0.9	0.2	0.1	1.2
Process plant/tailings/assaying	0.7	0.2	0.4	1.3
Mine supervision/geology/engineering	1.4	0.5	0.3	2.2
Projects and site services	0.4	0.3	1.1	1.9
Covid-19 operations/protocols	0.1	7.1	0.0	7.2
Total Owner Costs	**5.3**	**10.2**	**2.7**	**18.2**

21.2.7 Contingency

The estimated contingencies were allowances for undefined items of work which could be incurred within the defined scope of work covered by the estimate. Each discipline was allocated different contingency factors due to the varied risk level. The average contingency was 16.6%, equivalent to $23.3 M, which was included in the total initial capital. The contingency provision included $8.4 M that was payable as part of the EPC agreement.

21.2.8 Reclamation and Closure Capital Cost Estimate

Reclamation and closure costs were estimated at $3.4 M. Closure costs were assumed to be disbursed over a period of approximately three years, following the cessation of production. The major costs for reclamation were related to the FTSF.

21.3 Sustaining Capital Cost Estimate

Sustaining capital costs consisted of the direct costs of mine development, process plant, site infrastructure, FTSF development, and mobile equipment. Sustaining capital costs incurred by SilverCrest that were related to the cost of operating and maintaining the mine and plant were excluded, as was the closure cost.

The sustaining capital cost estimate from the start of operations to the end of the LOM is provided in Table 21-8.

Table 21-8: Sustaining Capital Cost Summary ($ M)

Calendar Year	2022	2023	2024	2025	2026	2027	2028	2029	2030	LOM
Production Year	1	2	3	4	5	6	7	8	9	
Process plant		1.4								1.4
Mobile equipment			0.3	0.3	0.3	0.3	0.3			1.3
Dry stack tailings						0.2	0.1			0.4
Mine	10.3	17.7	20.0	19.2	17.0	15.0	13.1	8.0	0.6	120.9
Total sustaining capital costs	**10.3**	**19.1**	**20.2**	**19.4**	**17.2**	**15.5**	**13.5**	**8.0**	**0.6**	**123.9**

Note: Totals may not add due to rounding.

Costs included in mining sustaining costs include underground development that is not included in operating costs. Items that were classified as sustaining capital included ramp development, waste lateral development, raise boring, and drop raising to create ventilation circuit linkages and escape ways.

The sustaining capital estimate includes mining infrastructure that will be installed after Q3 2022. The mining sustaining capital was calculated from first principles and validated based on rates provided by SilverCrest's current mining contractor. The contractor cost included the equipment, blasting, ground support, supplies, backfill operation, labour costs, and support equipment costs over the LOM.

The remaining underground development includes in-vein lateral development, driving of crosscuts and pivots drifts., and muck bays. This cost is included in the mining operating costs in the financial model.

21.4 Operating Cost Estimate

The average LOM operating cost, at a design mill feed rate of 1,250 t/d, was estimated at $118.49/t of material milled. The operating cost is defined as the total direct operating costs including mining, processing, and G&A costs. Table 21-9 shows a summary breakdown of the operating costs.

Table 21-9: Operating Cost Summary

Area	LOM Average Operating Cost ($/t milled)
Mining	71.40*
Process and tailings management	31.69
G&A	15.40
Total LOM operating cost	**118.49**

Notes: *Includes stope development but excludes capitalized underground development.

It is assumed that once construction is complete, operations personnel will reside, or be available, in nearby towns or villages. There will be no accommodation provided at site; Personnel will be transported to site by the Owner. It is expected that the mining contractor will hire personnel throughout Mexico and be responsible for lodging and catering of those personnel.

The operating costs exclude doré shipping and refining charges. Costs associated with doré transport and refining were included in the financial analysis in the applied payabilities for gold and silver values recovered.

21.4.1 Basis of Operating Cost Estimate

21.4.1.1 Estimate Base Date and Validity Period

The operating cost estimates were based on SilverCrest salary matrix as of Q4-2020.

21.4.1.2 Foreign Exchange

All the costs provided in the operating cost estimate were estimated in US dollars, unless specified, and where required were converted to US dollars using the exchange rates shown in Table 21-2.

21.4.2 Mining Operating Cost Estimate

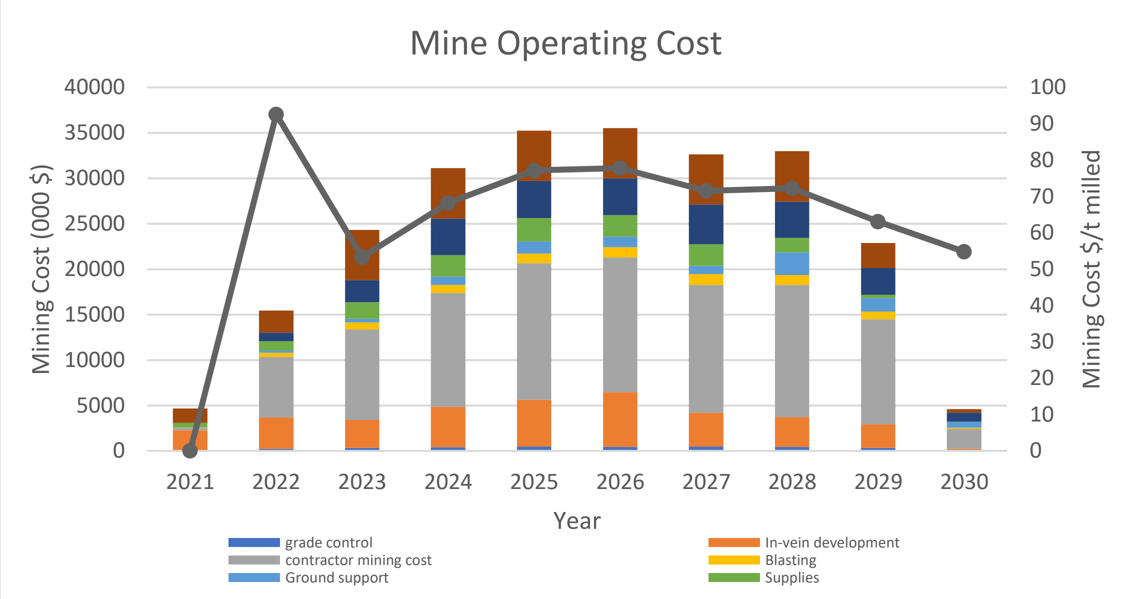
Table 21-10 summarizes the mining costs over the LOM. An average cost of $71.40/t milled was estimated. Costs vary for each year based on the annual scheduled throughput and on haul distances from the stopes to the mill.

Table 21-10: Mining Operating Cost Summary

Area	Total LOM Cost ($000)	Unit Cost ($/t milled)	Percentage (%)
Grade control	3,257	0.97	1
In-vein development	34,040	10.16	14
Contractor mining cost	101,796	30.38	43
Blasting	7,546	2.25	3
Ground support	9,671	2.89	4
Supplies	14,940	4.46	6
Backfill	27,780	8.29	12
Mining G&A and contractor overhead	40,237	12.01	17
Total mining operating cost	**239,265**	**71.40**	**100**

Note: Per tonne total LOM. Existing stockpile reclaim was included in the G&A costs.
Total may not add due to rounding.

Average annual mining unit costs, which ranged between $53/t and $92/t of mill feed, were largely dependent on the amount of development completed and the mining method. Figure 21-3 and Table 21-10 present the annual mine operating costs.



Note: Figure prepared by GMS, 2021

Figure 21-3: Annual LOM Operating Cost Profile

21.4.2.1 Mining Model

The operating cost estimate considered the use of a contractor for the development and mining. The contractor cost includes the equipment, blasting, ground support, supplies, backfill operation, labour costs, and support equipment costs over the LOM.

The mining cost estimates were based on a combination of budgetary quotes from SilverCrest's current mining contractor, GMS' cost databases and consideration of similar mines in South America and Mexico. The following costs were considered to be free issued by SilverCrest, and were not included in the contractor costs:

- CRF binder;
- Electrical power for the underground operation;
- Diesel consumption;
- Explosive;
- Consumable (piping, ground support etc.); and,
- Other general mine construction including ventilation and mine infrastructure.

A portion of fixed contractor and mining costs were capitalized based on the representational amount of underground development completed over the same period.

The key assumptions used in the estimate of the mining costs include:

- Deswick-Sched mine schedule;
- Contractor rates obtained through a bidding process, were applied to underground development and mining;
- Mining owner oversight costs (mining G&A costs and technical services) were included as a fixed cost for each month starting in June 2022. Prior to this date, these costs were included in the Owner's costs;

- Power costs of $0.10/kWh;
- Fuel cost of $1.00/L;
- Explosive cost of $0.79/kg for ANFO and $1.97 for emulsion or cartridge explosives;
- Cement cost of $100/t for backfill;
- No contingency has been applied to operating cost;
- Equipment hours were estimated over the LOM and were used to estimate the mine power and ventilation requirement. Support equipment was added based on production requirements;
- Ventsims software simulation was completed to estimate the ventilation requirement over the LOM;
- Pumping hours were estimated over the life of mine considering the mine schedule and dewatering requirement; and,
- Ventilation ducts, dewatering pipe, power cable and compressed air hose quantities were adjusted to account for re-use within the operation.

Costs were applied to the quantities to generate a cost per tonne, as shown in Table 21-10.

21.4.2.2 Stoping Costs

The mining cost estimate was completed for each stope by considering the mining tasks required. Table 21-11 shows the average mining costs per mining method.

Table 21-11: Mining Operating Cost per mining method

Mining Method	Unit Cost ($/t)
Long hole	58.3
Cut-and-fill and resue	89.0

21.4.2.3 Cut-and-Fill and Resue Mining Costs

For each cut-and-fill and resue stope, the following elements were considered:

- Production cycle to drill, blast, and muck a sill drive at the base of each stope, mining a 3 m high sill drive at the base of each stope;
- Production cycle to drill, blast, and muck 3 m high lifts to complete the stope, requiring six lifts to complete the stope;
- Slashing the pivot drive to gain access to each lift as the stope is mined from the base upwards;
- Backfilling of each lift in 3 m high lifts, filling the stope and providing a base on which the subsequent lift is mined;
- Mineralized material handling to surface;
- Grade control; and,
- Contractor supervision, SilverCrest supervision and technical services.

For each long hole stope, the following elements were considered:

- Production cycle to drill, blast, and muck a sill drive at the base and on the top of each stope;
- Production cycle to drill, blast, and muck opening raise for each stope;
- Production cycle to drill, blast, and muck long hole stope;
- Backfilling of each stope;
- Ore handling to surface;
- Grade control; and,
- Contractor supervision, SilverCrest supervision and technical services.

Table 21-12 provides the mining operating costs estimated per year of operations. The peak costs were anticipated in 2022 and were largely attributable to a the relatively low processing rate expected during the plant ramp-up period.

Table 21-12: Mining Operating Costs Estimated per Year of Operations ($ 000)

Area	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	TOTAL
Grade control	74	240	328	411	470	443	464	451	324	52	**3,257**
In-vein development	2,212	3,430	3,075	4,420	5,161	6,031	3,728	3,253	2,569	195	**34,040**
Contractor mining	265	6,673	9,979	12,556	15,031	14,826	14,063	14,577	11,610	2,214	**101,796**
Blasting	32	446	757	858	1,059	1,121	1,207	1,090	823	125	**7,546**
Ground support	18	261	391	922	1,346	1,175	917	2,479	1,495	667	**9,671**
Supplies	484	1,024	1,834	2,354	2,577	2,365	2,367	1,576	355	5	**14,940**
Backfill	-	951	2,436	3,994	4,059	4,042	4,379	3,987	2,945	986	**27,780**
Mining G&A and contractor overhead	1,597	2,414	5,363	5,431	5,297	5,322	5,297	5,431	3,383	702	**40,237**
Total mining operating cost	**4,682**	**15,439**	**24,164**	**30,975**	**35,001**	**35,325**	**32,421**	**32,845**	**23,504**	**4,911**	**239,265**
Total mining cost per ROM tonnes ($/t)	**N/A**	**92.54**	**52.96**	**67.89**	**76.71**	**77.43**	**71.06**	**71.99**	**64.79**	**58.38**	**71.40**

21.4.3 Process Operating Cost Estimate

The process operating cost estimate was based on a 1,250 t/d mill comprising crushing, grinding, flotation, cyanide leaching, countercurrent decantation washing, Merrill-Crowe precious metals recovery, and smelting unit operations to produce gold–silver doré bars. Effluent treatment costs were also included.

Costs related to the doré shipping and refining were included in the financial analysis in Section 22.

21.4.3.1 Processing Operating Cost Summary

The unit process operating cost was estimated at $31.69/t milled, based on a 1,250 t/d concentrator with crushing availability of 70%, mill availability of 91.3%, and 365 operating days per year. Table 21-13 summarizes the operating costs expected for the process area.

Table 21-13: LOM Process Operating Cost Summary

Area	Total LOM ($ M)	Unit LOM [1] ($/t milled)	Percentage (%)
Power	22.4	6.68	21
Reagents & consumables	47.4	14.14	45
Labour	14.1	4.20	13
Maintenance supplies	12.0	3.59	11
Laboratory	3.4	1.00	3
Dry stack tailings	7.0	2.09	7
Total process operating cost	**106.2**	**31.69**	**100**

Note: Total LOM milled mineralized material tonnage of 3.35 Mt derived from mine plan.

Table 21-14 provides the milling operating costs estimated per year of operations. The peak costs were anticipated in 2022 and were largely attributable to a the relatively low processing rate expected during the plant ramp-up period.

Table 21-14: Milling Operating Costs Estimated per Year of Operations ($ M)

Area	2022	2023	2024	2025	2026	2027	2028	2029	2030	TOTAL
Power	1.17	3.04	3.04	3.04	3.04	3.04	3.04	2.42	0.56	**22.39**
Reagents & Consumables	2.04	6.78	6.38	6.43	6.52	6.31	6.79	5.02	1.12	**47.39**
Labour (Maintenance & Process Plant)	1.83	1.53	1.53	1.53	1.53	1.53	1.53	1.53	1.53	**14.06**
Maintenance Supplies	0.60	1.64	1.64	1.64	1.64	1.64	1.64	1.30	0.30	**12.02**
Laboratory	0.17	0.46	0.46	0.46	0.46	0.46	0.46	0.36	0.08	**3.35**
Dry Stack Tailings	0.35	0.96	0.96	0.96	0.96	0.96	0.92	0.73	0.17	**7.00**
Total milling operating cost ($ M)	**6.15**	**14.41**	**14.01**	**14.06**	**14.15**	**13.94**	**14.37**	**11.36**	**3.76**	**106.21**
Total milling cost per ROM tonnes ($/t)	**36.86**	**31.58**	**30.70**	**30.82**	**31.01**	**30.56**	**31.50**	**31.33**	**44.72**	**31.69**

21.4.3.2 Power

The electrical energy consumption is estimated to be 30,406 MWh per year or approximately 66.6 kWh/t milled and includes the operation in the process plant and ancillary buildings, the maintenance shop, plant offices, administration building and warehouse.

Electricity will be provided to site by CFE at a unit cost of $0.10/kWh. Shortly after it has been put in service, maintenance of the power line will be the responsible of CFE.

The unit power cost is estimated at $6.68/t milled and comprises 21% of the overall process operating cost.

21.4.3.3 Reagents and Consumables

Individual reagent and consumable consumption rates were estimated from metallurgical testwork results, Ausenco's in-house database and Ausenco's experience. Each unit cost was obtained from supplier quotations. The LOM average cost of reagents and consumables was estimated at $14.14/t milled, which accounts for 45% of the overall process operating cost and is the largest Project operating cost factor. LOM mill reagents cost about $8.60/t milled and the operation consumables were estimated at $5.54/t milled.

The consumables cost at $5.54/t milled, was mainly for liners and grinding media in the comminution circuit, as well as for operational consumables associated with screening, filtration, refinery, and dust collection processes.

21.4.3.4 Labour

The estimated average labour cost was $4.20/t milled. It was based on fully loaded Q4 2020 labour rates from SilverCrest, and a consumer price index (CPI) adjustment of 5% for 2021 and the first half of 2022. A total of 89 persons were required for the mill, FTSF, and the process maintenance shop. Personnel required for the assay laboratory were included in the laboratory cost estimate.

The staff employees will work on site for eight hours per day from Monday to Friday. A management staff member will work on site through the weekend. The operators will work on a shift that is two-weeks on-site and one-week off-site.

- 53 persons were projected for the mill and FTSF areas, comprising eight management staff and 45 operators; and,
- 36 persons were projected to work in the process maintenance shop, consisting of five management staff. and 31 maintenance crew.

21.4.3.5 Maintenance Supplies

The cost for maintenance supplies was estimated at $3.59/t milled, based on 8.5% of the equipment direct capital cost.

21.4.3.6 Laboratory

The average operating cost of the assay laboratory was estimated at $1.00/t milled, as provided by a specialized firm. This unit rate was exclusive of some services provided by the Owner that were included in the G&A costs. This covered the required process analyses on routine samples.

21.4.4 Filtered Tailings Storage Facility Operating Cost Estimate (Wood)

Wood completed the operating cost estimate for operation of the FTSF. The cost was estimated for an initial facility in the first six years and an expansion in year 7 for the remainder of the LOM.

The operating cost of $2.11/t milled was applied to the first six years followed by $2.02/t milled for the remaining LOM. Table 21-15 summarizes the operating cost for the filtered tailings stacking operation.

Table 21-15: FTSF Operating Cost Summary

Area	Annual ($ M/a)	Unit Cost ($/t milled)
East FTSF (2.6 Mt)[1] for years 1 to 6	5.5	2.11
East FTSF expansion (0.9 Mt)[1,2] for years 7 to 8	1.8	2.02

Notes:
1. Assumed the use of 14 m^3 haul trucks with a distance within 1 km for tailings transportation.
2. Includes the collection pond of the NW FTSF to store bleeding solution.

21.4.5 General and Administrative Operating Cost Estimate

G&A costs covered the expenses of the service departments staring in June 2022. Operating departments (mine, geology, mine engineering, plant operation/maintenance) were covered in the plant and mine operating costs. SilverCrest personnel numbers will peak at 212 (exclusive of the mine contractor personnel employees), of which 64 employees were covered under the G&A costs.

Overall, the G&A costs included:

- Manpower: Included general manager and staffing in community relations, government relations, accounting, purchasing, environmental, health and safety, security, human resources, information technology and site services inclusive of mobile equipment on surface. Personnel working at both the Project site and at SilverCrest's Hermosillo office are included;
- Manpower salaries and wages: Based on SilverCrest salary matrix in Mexico as of Q4-2020. This includes base salary or wage and related burdens, covering retirement savings plans, various life and accident insurances, extended medical benefits, unemployment insurance, and other benefits;
- Contract expenses: Included general administration, contractor services, insurance, security, medical services, legal services, human resources, travel, communication services/supports, permits obligations, enviro assay/testing, surface rights payments, overall site maintenance, surface electricity requirement, and engineering consulting; and,
- Operating and supplies expenses: Included costs associated with employment such as telephones, information technology equipment (computers and software). This section also included the surface mobile fleet operating costs.

The total annual G&A cost was estimated at an average of $5.9 M during production which equated to an average LOM G&A cost of $15.40/t milled. G&A costs are summarized in Table 21-16.

Table 21-16: General and Administration for LOM

Item	Total LOM ($ M)	Unit Cost ($/t milled)
Accounting/procurement/administration	7.5	1.62
Management and community relation	12.3	3.96
Human resources and information technology	7.9	0.62
Health/environment/safety/security	7.0	2.34
Process plant/tailings/assaying	0.9	2.10
Projects and site services	15.9	4.75
Total	**51.6**	**15.40**

22 Economic Analysis

22.1 Forward-Looking Information Cautionary Statements

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

- Proven and Probable Mineral Reserves that have been modified from Measured and Indicated Mineral Resource estimates;
- Cash flow forecasts;
- Assumed commodity prices and exchange rates;
- Proposed mine and process production plan;
- Projected mining and process recovery rates;
- Ability to have doré refined on favourable terms;
- Proposed capital and operating costs;
- Assumptions as to closure costs and closure requirements; and,
- Assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what is assumed;
- Unrecognised environmental risks;
- Unanticipated reclamation expenses;
- Unexpected variations in quantity of mineralization, grade or recovery rates;
- Geotechnical or hydrogeological considerations during operations being different from what was assumed;
- Failure of mining methods to operate as anticipated;
- Failure of plant, equipment or processes to operate as anticipated;
- Changes to assumptions as to the availability and or generation of electrical power, and the power rates used in the operating cost estimates and financial analysis;
- Ability to maintain the social licence to operate;
- Accidents, labour disputes and other risks of the mining industry;
- Changes to interest rates, tax rates or applicable laws;
- Receipt of any required permits, beyond those already held by SilverCrest; and,
- Impacts to manpower availability and delays to the construction schedule due to the COIVID-19 global pandemic.

22.2 Methodology

A pre- and post-tax economic analysis was completed on the basis of a discounted cash flow model featuring a 5% discount rate. The analysis used constant (real) 2020 US$ and the project cash flows were modelled in annual periods.

The model assumed a 17-month physical construction period, and production period of 8.5 years, including the first year and final year which will see production for only a portion of those two years.

Although SilverCrest is assuming a targeted timeline for initial operation and ramp up of production from the Project, calendar years used in the economic analysis are provided for conceptual purposes only.

22.3 Financial Model Parameters and Assumptions

22.3.1 Mineral Resources, Mineral Reserves and Production Schedule

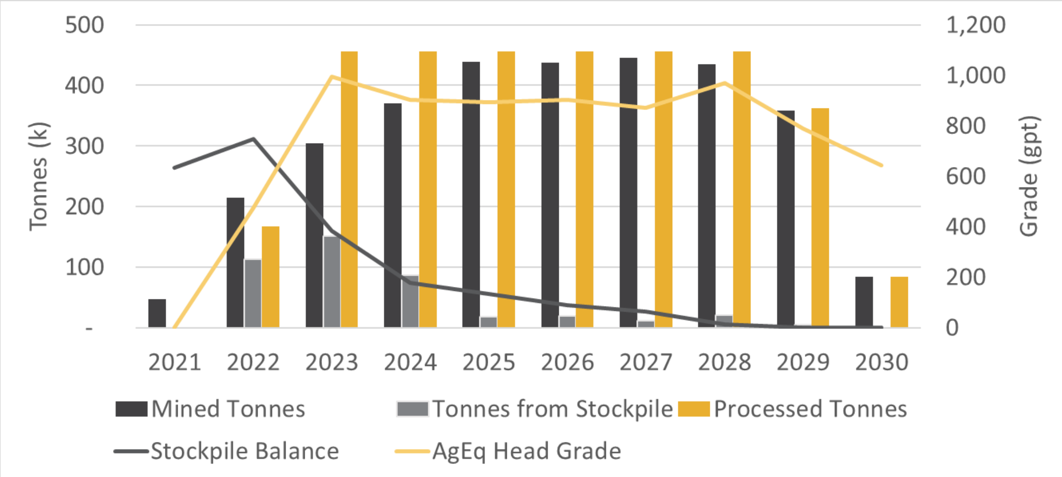
The mine plan is based on the estimated Mineral Reserves for the Project. No Inferred Mineral Resources were included in the material scheduled for processing.

Table 22-1 provides the LOM doré production forecast. Figure 22-1 and Figure 21-2 summarize the material movement and anticipated production schedules.

Table 22-1: LOM Doré Production Forecast

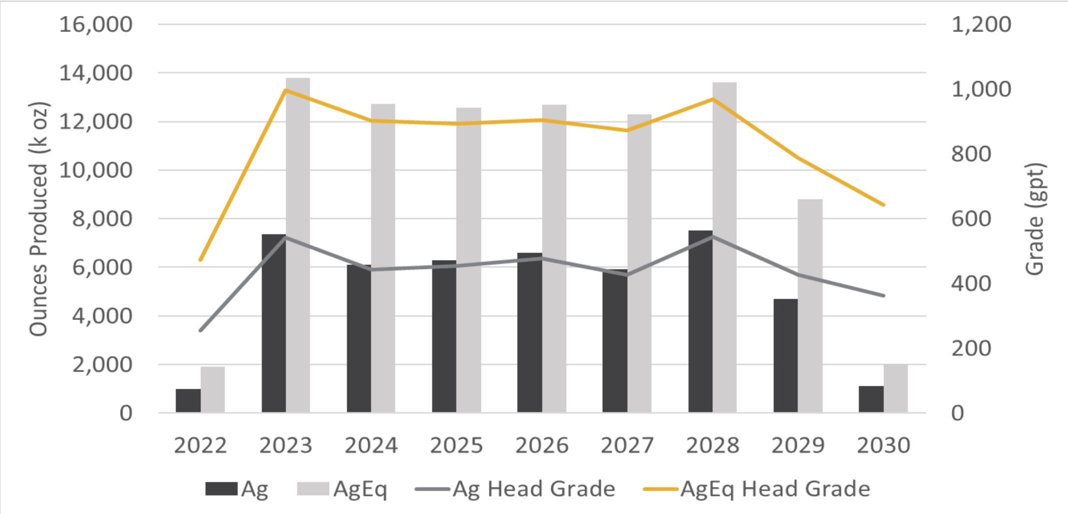
	Unit	Total	2022	2023	2024	2025	2026	2027	2028	2029	2030
Mill Feed	kt	3,351	167	456	456	456	456	456	456	363	84
Mill Feed Grade	gpt Au	4.81	2.53	5.23	5.3	5.05	4.9	5.12	4.89	4.16	3.23
	gpt Ag	461	254	541	442	455	478	428	544	427	362
	gpt AgEq	879	474	996	903	893	904	873	968	789	643
Process Recovery	% Au	97.0	90.1	96.2	97.6	97.6	97.6	97.6	97.6	97.6	97.6
	% Ag	93.7	87.0	92.8	94.3	94.3	94.3	94.3	94.3	94.3	94.3
Production in Doré	**koz Au**	**504**	**10**	**74**	**76**	**72**	**70**	**73**	**70**	**47**	**10**
	koz Ag	**46,629**	**1,004**	**7,367**	**6,116**	**6,291**	**6,608**	**5,920**	**7,524**	**4,695**	**1,104**
	koz AgEq	**90,392**	**1,912**	**13,786**	**12,715**	**12,571**	**12,708**	**12,284**	**13,602**	**8,812**	**2,002**

Notes:
- The AgEq is based on Au:Ag ratio of 86.9:1, calculated using metal prices of $1,410/oz Au and $16.60/oz Ag, and metal recovery values of 96% Au and 94% Ag.
- All numbers are rounded.



Note: Figure prepared by SilverCrest, 2021.

Figure 22-1: Material Movement Schedule



Note: Figure prepared by SilverCrest, 2021.

Figure 22-2: Annual Production Schedule

22.3.2 Metallurgical Recoveries

Metallurgical recoveries were applied to the economic model in accordance with the metallurgical testwork and the projected metal recoveries described in Section 13. The annual average ROM grades delivered to the process plant, as predicted by the mine plan, were used as the basis for the recovery calculations. The recovery algorithms employed are detailed in Section 13, and the overall achieved LOM recoveries are shown in Table 22-2. Note that these recoveries are assumed to be realized outside of the ramp-up period.

Table 22-2: LOM Processing Recoveries

Metal	Unit	Recovery
Gold	%	97.6%
Silver	%	94.3%

22.3.3 Freight, Smelting and Refining

Assumed terms for smelting and refining of the gold and silver product are set out in Section 19. Rates are based on term sheets obtained by SilverCrest and are in line with industry rates.

Table 22-3 shows the payment, smelting, and refining terms that were applied in the economic analysis.

Table 22-3: Payment, Smelting and Refining Terms

Term	Unit	Amount
Gold	%	99.85
Silver	%	99.85
Refining, Transport, and Insurance	$/oz AgEq	0.31

22.3.4 Metal Prices

The economic model is based on the following assumptions:

- Gold price of $1,500/oz; and,
- Silver price of $19.00/oz.

Base case metal prices selected for the Feasibility Study were based on consensus average long-term forecast gold and silver prices from Canadian financial institutions. These metal prices were selected in Q4 2020 with consideration of three-year trailing prices and spot prices at the time.

The economic analysis is based on flat prices for the projected LOM.

22.3.5 Operating Costs

The operating costs are detailed in Section 21. Table 22-4 summarizes the overall unit costs. The projected mine operating costs are $71.40 per tonne of material milled and total operating costs are estimated to average $118.49/t of material milled.

22.3.6 Capital Costs

Capital costs used for the economic evaluation are detailed in Section 21 and summarized in Table 22-4. Initial capital costs were scheduled based on projected milestone payments under the EPC (Engineering, Procurement, and Construction) contract with Ausenco and forecast underground development based on the GMS mine schedule. Additional capital costs related to the FTSF, powerline, additional surface infrastructure and Owner's costs, were scheduled based on a proposed execution plan developed by SilverCrest.

Sustaining capital costs for mining were estimated by GMS, scheduled to match the expected spend profile developed as part of the mining cost estimation process, and were based on the production and waste movement profile. Sustaining capital costs for the processing plant, tailings facility and other surface infrastructure were scheduled based on projected capital expenditures.

Table 22-4: Capital Cost Estimates

Area	Units	Initial Capital	Sustaining Capital
Mine	$M	27.7	120.9
Process Plant	$M	44.9	1.4
Tailings Management	$M	3.1	0.4
Infrastructure	$M	20.6	1.3
Owners Costs	$M	18.2	
Contingency	$M	23.3	
Project Total	**$M**	**137.7**	**123.9**
Closure Costs	$M		3.4

22.3.7 Royalties

The royalties and fees applied to the Feasibility Study cash flow include the following:

- Government earnings before income, taxes, and depreciation and amortization (EBITDA) royalty of 7.5% of income less authorized deductions, applicable to mining companies;
- Extraordinary government royalty of 0.5% of net revenue (NSR), applicable to gold and silver operations; and,
- Concession fees (included in G&A operating costs).

The government EBITDA royalty is predicted to total $80.5 M over the LOM, with the extraordinary government royalty expected to total $8.0 M over the LOM.

The Feasibility Study cash flow anticipates a total of $88.6 M will be payable to the Mexican government for royalties. This excludes income taxes, payroll taxes, fees, and sales taxes.

Although there is a 2% royalty on the Nuevo Lupena and Panuco II concessions for material that exceeds specified grade thresholds, none of the estimated Mineral Reserves are within these concessions; therefore, this royalty is not applicable and was excluded from the economic analysis.

22.3.8 Working Capital

Working capital for the Project is estimated to be $25.6 M. This estimate was reached based on consideration of required inventory and value added taxes (IVA) and duties required at the outset of the Project. The model assumes that value added taxes will be collectible two years from the date the related costs are incurred. Other working capital items are recoverable at the end of the LOM. In addition to working capital adjustment over the LOM, there is an adjustment of $11.5 M for value added taxes that is recoverable as of December 31, 2020 which is assumed to be recovered in 2022.

22.3.9 Taxes

Orbe Advisors completed an estimate of Mexican corporate income tax.

Allowable deductions were applied to cash flows based on estimated capital costs and expenses that SilverCrest has incurred to date, which include:

- Capital costs depreciated at 12%;
- Non-fixed development capital depreciated at 10%;
- Sustaining capital expenses, depreciated in the year expensed;
- Pre-development exploration costs of $91.5 M depreciated at a rate of 10%; and
- Historical net operating losses (NOLs) applied in 2023 of $55.9 M.

The resulting taxable income is estimated at $715.1 M. SilverCrest applied a tax rate of 30% to this amount over the LOM for an estimated tax amount of $214.5 M over the LOM. A review was completed with the NOLs excluded and the Project economics were still positive.

22.3.10 Closure Costs and Salvage Values

An allowance of $3.4 M was made for closure, based on an estimate developed by Wood. The spending was scheduled to occur across the three years following the cessation of production. No provision or accrual for closure was made (cash or otherwise) for the purposes of the economic evaluation. Any change in regulations that would require SilverCrest to undertake progressive closure, or to post a cash bond, would affect the timing of these cash flows.

No salvage value was assumed for any items.

22.3.11 Financing and Inflation

No consideration of financing was made. The model considers the cash flow only at an asset level and assumes 100% equity ownership.

The modelling was undertaken in real 2020 US$ with no inflation applied to either commodity prices or costs. An assumption of US$ accounting was made.

22.4 Financial Results

The economic analysis demonstrates that the mine plan has positive economics under the assumptions used. The project post-tax (NPV) at a 5% discount rate is estimated to be $486.3 M, with an internal rate of return (IRR) of 52%. The Project will achieve payback in 1.0 year. A project financial summary is shown in Table 22-5.

Table 22-5: Economic Analysis Summary

Project Metric	Unit	Value
Gold Price	$	1,500
Silver Price	$	19.00
Mine Life	Year	8.5
Nominal Process Capacity	t/d	1,250

Project Metric	Unit	Value
Average Annual Gold Production (LOM)	koz Au	55.96
Average Annual Silver Production (LOM)	koz Ag	5,181
Average Annual Silver Equivalent Production (LOM)	koz AgEq	10,044
Average Annual Gold Production (2023-2029)	koz Au	68.97
Average Annual Silver Production (2023-2029)	koz Ag	6,360
Average Annual Silver Equivalent Production (2023-2029)	koz AgEq	12,354
Initial Capital Expenditure	$M	137.7
LOM Sustaining Capital Expenditure	$M	123.9
LOM C1 Cash Costs (LOM)	$/oz AgEq	4.40
LOM C1 Cash Costs (2023-2029)	$/oz AgEq	4.13
Pre-Tax NPV (5%)	$M	655.9
Pre-Tax IRR	%	63
Post-Tax NPV (5%)	$M	486.3
Post-Tax IRR	%	52
Undiscounted Post-Tax Cash Flow (LOM)	$M	656.4
Payback Period (undiscounted, post-tax cash flow)	Year	1.0

Note: C1 cash costs represent costs incurred at each processing stage, from mining through to recoverable metal delivered to market, less net by-product credits.

The production schedule was incorporated into a pre-tax financial model to develop the annual recovered metal production. The annual at-mine revenue contribution of each metal was determined by deducting the applicable treatment, refining, and transportation charges (from mine site to market) from gross revenue.

The cash flow is based on payable ounces of 502.8 koz gold and 46,559 koz silver.

Initial capital expenditures were based on the required construction and development beginning in 2021 and continuing until commercial production is achieved in 2022; these initial expenditures also included underground development expenses. However, underground development and construction expenses incurred prior to January 2021 were not included in the financial model as these were considered sunk costs at the point of a production decision, which is expected in early 2021. Total construction and development expenses considered sunk capital prior to January 2021 were estimated to total $25.8 M.

Sustaining capital costs were incorporated on a year-by-year basis over the LOM, and operating costs were deducted from gross revenue to estimate annual mine operating earnings.

The financial model includes a mine closure and reclamation cost forecast of $3.4 M, which is incurred after the completion of production, over a three-year period. The model includes $25.6 M in working capital during the construction period that is assumed to be recovered throughout the LOM.

The operating costs are expected to average $118.49 per tonne milled over the LOM. All-in sustaining costs, which is an industry-accepted metric used to better reflect the recurring costs to achieve production, average $7.07/oz AgEq payable when averaged over the LOM and $6.68/oz over the seven full years of production. The standard definition for all-in sustaining costs includes all on-site costs, royalties and production taxes, derivative gains or losses, permitting costs, refining costs, corporate G&A, stockpile inventory adjustments, corporate G&A and share based compensation, and all sustaining capital costs. Given that the cashflow analysis is presented at the project level only, all corporate related costs have been excluded from this analysis.

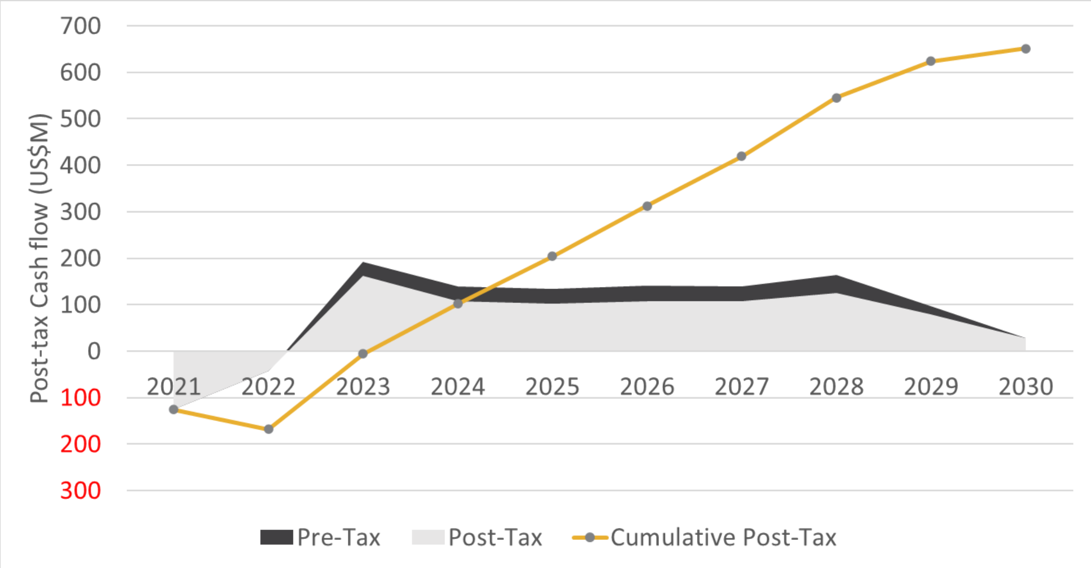
Table 22-6 is an overall cost summary for the Project.

Table 22-6: Cost Summary

Cost	LOM		2023-2029	
	($M)	($/oz AgEq)	($M)	($/oz AgEq)
Mine Operating Cost	239.3	2.65	214.2	2.48
Process Operating Cost	106.2	1.18	96.3	1.12
General and Administrative	51.6	0.57	45.8	0.53
Total Operating Costs	**397.1**	**4.40**	**356.4**	**4.13**
Refining Costs	28.8	0.32	27.5	0.32
Government Royalties	88.6	0.98	79.7	0.92
Sustaining Capital	123.9	1.37	113.1	1.31
All-in Sustaining Costs	**638.3**	**7.07**	**576.6**	**6.68**

The post-tax financial analysis was completed using the pre-tax cash flow by applying the tax model provided by Orbe Advisors(refer to Section 22.3.9).

Figure 22-3 shows the annual after-tax net cash flows (NCFs) and cumulative net cash flows (CNCFs).



Note: Figure prepared by SilverCrest, 2021.

Figure 22-3: After-Tax Cash Flow

The Project cash flow on an annualized basis is provided in Table 22-7.

Table 22-7: Las Chispas Feasibility Study Project Cash Flow

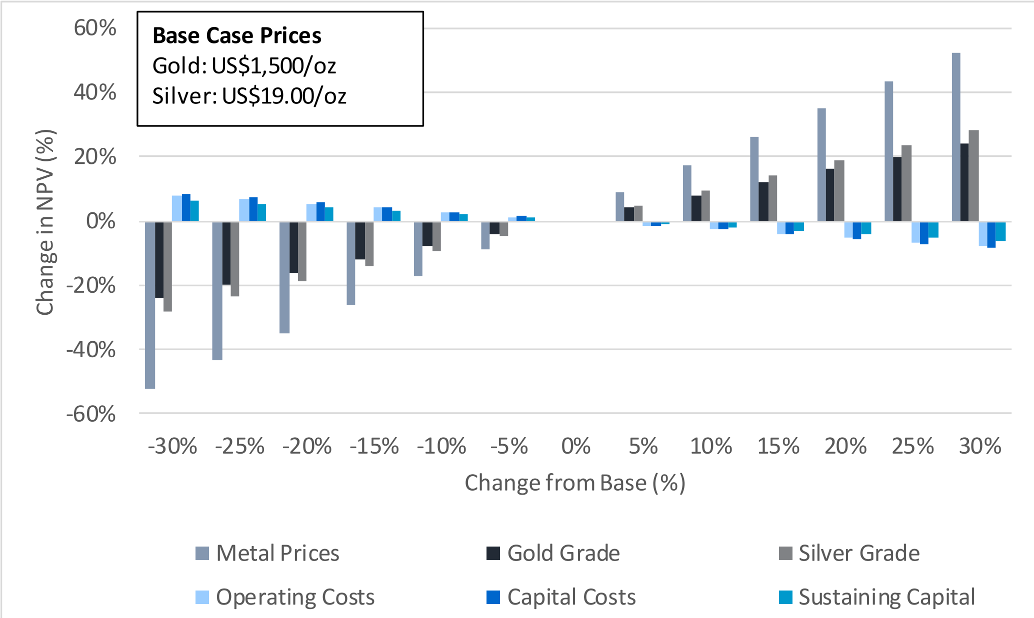
Calendar year	Year	LOM Total	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033
Metal prices															
Au	US$/oz	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500
Ag	US$/oz	19.00	19.00	19.00	19.00	19.00	19.00	19.00	19.00	19.00	19.00	19.00	19.00	19.00	19.00
Mine production															
Mined tonnes	kt	3,134	46.6	214.4	304.7	370.1	438.3	437.5	445.2	435.2	358.0	84.1	-	-	-
Milled tonnes	kt	3,351	-	166.8	456.2	456.2	456.3	456.3	456.3	456.3	362.8	84.1	-	-	-
Mine Grades															
Au	gpt	5.0	2.46	5.8	5.2	5.3	5.0	5.0	5.2	5.1	4.2	3.2	-	-	-
Ag	gpt	478	379.31	614	521	413	455	489	436	565	431	362	-	-	-
AgEq	gpt	911	593	1,117	970	875	893	923	889	1,005	796	643	-	-	-
Metal production															
Head grades															
Au	gpt	4.8	-	2.5	5.2	5.3	5.0	4.9	5.1	4.9	4.2	3.2	-	-	-
Ag	gpt	461	-	254	541	442	455	478	428	544	427	362	-	-	-
AgEq	gpt	879	-	474	996	903	893	904	873	968	789	643	-	-	-
Au ounces contained in mill feed	koz	518.1	-	13.6	76.8	77.8	74.0	71.9	75.0	71.7	48.5	8.7	-	-	-
Ag ounces contained in mill feed	koz	49,678.9	-	1,360.7	7,938.9	6,485.8	6,670.9	7,007.6	6,277.4	7,978.3	4,979.1	980.2	-	-	-
AgEq ounces contained in mill feed	koz	94,704.5	-	2,542.1	14,610.9	13,247.4	13,105.5	13,257.2	12,798.5	14,206.3	9,196.9	1,739.6	-	-	-
Dore production															
Au	koz	503.6	-	10.4	73.9	75.9	72.3	70.2	73.2	69.9	47.4	10.3	-	-	-
Ag	koz	46,628.8	-	1,003.8	7,367.3	6,116.1	6,290.7	6,608.2	5,919.6	7,523.5	4,695.3	1,104.3	-	-	-
AgEq	koz	90,391.7	-	1,911.8	13,785.8	12,715.4	12,570.8	12,707.8	12,284.2	13,602.1	8,811.9	2,001.9	-	-	-
Payable metal															
Au	koz	502.8	-	10.4	73.7	75.8	72.2	70.1	73.1	69.8	47.3	10.3	-	-	-
Ag	koz	46,558.8	-	1,002.3	7,356.2	6,106.9	6,281.3	6,598.3	5,910.7	7,512.2	4,688.2	1,102.7	-	-	-
AgEq	koz	90,256.1	-	1,909.0	13,765.1	12,696.3	12,552.0	12,688.7	12,265.8	13,581.7	8,798.7	1,998.9	-	-	-

Revenue from Payable Dore	000$	$1,638,886	-	$34,694	$250,393	$229,773	$227,584	$230,496	$222,000	$247,499	$160,027	$36,421	-	-	-
Dore selling costs	000$	($28,812)	-	($752)	($4,503)	($3,796)	($3,890)	($4,067)	($3,681)	($4,585)	($2,947)	($591)	-	-	-
Total project revenue	**000$**	**$1,610,074**	-	**$33,941**	**$245,891**	**$225,977**	**$223,694**	**$226,429**	**$218,319**	**$242,913**	**$157,081**	**$35,829**	-	-	-
Operating costs															
Mining operating costs	000$	($239,265)	($4,682)	($15,439)	($24,164)	($30,975)	($35,001)	($35,325)	($32,421)	($32,845)	($23,504)	($4,911)	-	-	-
Process operating costs	000$	($106,210)	$0	($6,149)	($14,406)	($14,005)	($14,061)	($14,149)	($13,942)	($14,373)	($11,363)	($3,761)	-	-	-
G&A	000$	($51,604)	$0	($4,093)	($6,510)	($6,472)	($6,723)	($6,574)	($6,481)	($6,496)	($6,574)	($1,681)	-	-	-
Total site operating costs	000$	($397,079)	($4,682)	($25,681)	($45,080)	($51,452)	($55,786)	($56,048)	($52,844)	($53,714)	($41,441)	($10,352)	-	-	-
Operating costs per AgEq	**US$/oz**	($4.40)	-	**($15.91)**	**($3.27)**	**($4.05)**	**($4.44)**	**($4.42)**	**($4.31)**	**($3.95)**	**($4.71)**	**($5.18)**	-	-	-
Earnings before interest, tax, depreciation and amortization															
EBITDA	**000$**	**$1,212,996**	-	**$3,579**	**$200,811**	**$174,525**	**$167,908**	**$170,382**	**$165,475**	**$189,199**	**$115,640**	**$25,477**	-	-	-
Royalties	EBITDA	7.5% EBITDA	-	-	-	($13,730)	($11,590)	($11,154)	($11,507)	($11,283)	($13,205)	($8,073)	-	-	-
	Environmental	0.5% NSR	-	-	($173)	($1,252)	($1,149)	($1,138)	($1,152)	($1,110)	($1,237)	($800)	-	-	-
Royalty	**000$**	**($88,554)**	-	-	**($173)**	**($14,982)**	**($12,738)**	**($12,292)**	**($12,659)**	**($12,393)**	**($14,442)**	**($8,873)**	-	-	-
Capital costs															
Direct															
U/G development	000$	(27,655)	($18,783)	($8,872)	-	-	-	-	-	-	-	-	-	-	-
Operations Team	000$	(18,200)	($12,554)	($5,646)	-	-	-	-	-	-	-	-	-	-	-
Process Plant	000$	(44,891)	($38,391)	($6,500)	-	-	-	-	-	-	-	-	-	-	-
Tailings	000$	(3,060)	($2,697)	($362)	-	-	-	-	-	-	-	-	-	-	-
Infrastructure	000$	(20,648)	($20,398)	($250)	-	-	-	-	-	-	-	-	-	-	-
Contingency		(23,253)	($7,038)	($16,215)	-	-	-	-	-	-	-	-	-	-	-
Sustaining			-	-	-	-	-	-	-	-	-	-	-	-	-
U/G development		(120,883)	-	($10,264)	($17,741)	($19,997)	($19,189)	($16,957)	($15,035)	($13,135)	($7,999)	($566)	-	-	-
Infrastructure		(1,250)	-	-	-	($250)	($250)	($250)	($250)	($250)	-	-	-	-	-
Processing	000$	(1,395)	-	-	($1,395)	-	-	-	-	-	-	-	-	-	-
Tailings	000$	(361)	-	-	-	-	-	-	($241)	($120)	-	-	-	-	-
Total capital costs	**000$**	(261,595)	($99,862)	($48,108)	($19,136)	($20,247)	($19,439)	($17,207)	($15,526)	($13,505)	($7,999)	($566)	-	-	-
AISC	**000$**	**(638,333)**	-	**(41,378)**	**(68,892)**	**(90,477)**	**(91,853)**	**(89,614)**	**(84,710)**	**(84,198)**	**(66,829)**	**(20,383)**	-	-	-
AISC/oz AgEq	**US$/oz**	(7.07)	-	(21.68)	(5.00)	(7.13)	(7.32)	(7.06)	(6.91)	(6.20)	(7.60)	(10.20)	-	-	-
Other expenses															
Working capital adjustments	000$	11,500	(25,579)	1,635	10,665	310	(1,727)	(233)	1,058	1,333	3,775	11,928	6,859	1,478	-
Reclamation expenses	000$	(3,400)	-	-	-	-	-	-	-	-	-	-	($1,133)	($1,133)	($1,133)
Total other expenses	**000$**	8,100	($25,579)	$1,635	$10,665	$310	($1,727)	($233)	$1,058	$1,333	$3,775	$11,928	$5,725	$345	($1,133)
Pre-tax net cash flow	**000$**	**$870,947**	**($125,441)**	**($42,894)**	**$192,166**	**$139,606**	**$134,004**	**$140,650**	**$138,348**	**$164,634**	**$96,974**	**$27,965**	**$5,725**	**$345**	**($1,133)**
Cumulative pre-tax net cash flow	**000$**	**$870,947**	**($125,441)**	**($168,335)**	**$23,831**	**$163,436**	**$297,441**	**$438,090**	**$576,438**	**$741,072**	**$838,046**	**$866,011**	**$871,736**	**$872,081**	**$870,947**
Taxable income	**000$**	**$715,098**	-	-	**$97,813**	**$109,207**	**$104,329**	**$108,362**	**$104,164**	**$129,980**	**$61,244**	-	-	-	-
Total taxes	000$	**($214,529)**	-	-	**($29,344)**	**($32,762)**	**($31,299)**	**($32,509)**	**($31,249)**	**($38,994)**	**($18,373)**	-	-	-	-
Net post-tax cash flow	000$	**$656,418**	**($125,441)**	**($42,894)**	**$162,822**	**$106,844**	**$102,705**	**$108,141**	**$107,098**	**$125,640**	**$78,601**	**$27,965**	**$5,725**	**$345**	**($1,133)**
Cumulative post-tax net cash flow	**000$**	**$656,418**	**($125,441)**	**($168,335)**	**($5,513)**	**$101,330**	**$204,036**	**$312,177**	**$419,276**	**$544,916**	**$623,516**	**$651,481**	**$657,207**	**$657,551**	**$656,418**

Note: Table prepared by SilverCrest, 2021.

22.5 Sensitivity

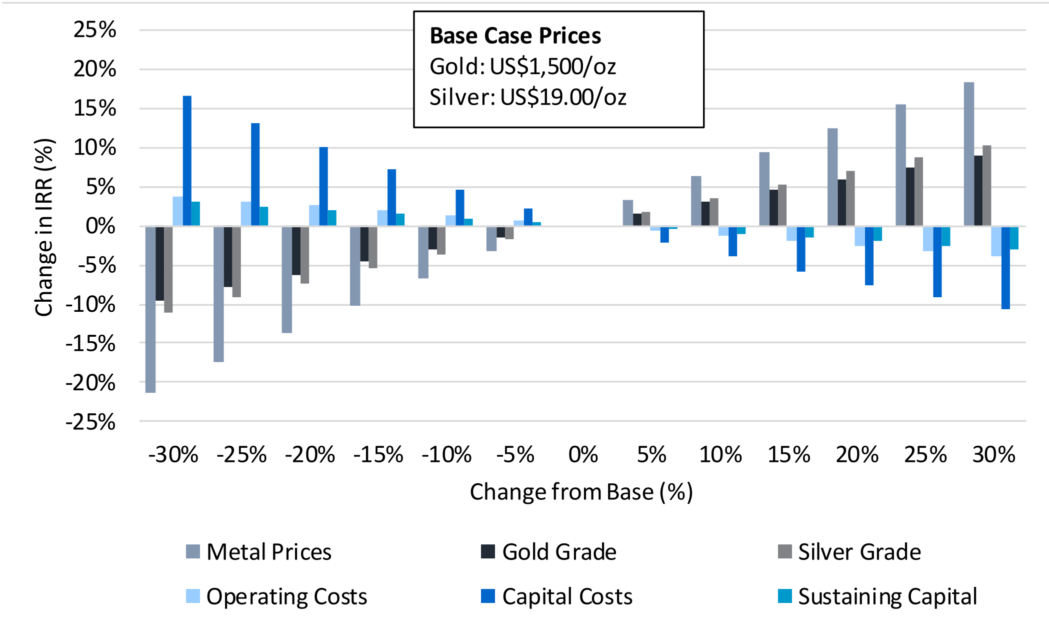
Sensitivity analysis was completed to evaluate the response of the project NPV and IRR to changes in assumptions on key inputs of metals prices, grades, capital costs and operating costs. The after-tax results across a range of ±30% from the base case assumption value are shown in Figure 22-4 and Figure 22-5. The Project maintains a positive NPV across the range tested. Operating costs, capital costs, and sustaining capital costs have a similar sensitivity profile.



Percent Change from Base (%)	NPV ($ M)						
		Metal Prices	Gold Grade	Silver Grade	Operating Costs	Capital Costs	Sustaining Capital Costs
	-30%	232.2	369.3	349.3	524.7	527.7	517.1
	-25%	274.5	388.8	372.1	518.4	520.8	512.0
	-20%	316.9	408.3	395.0	512.0	513.9	506.9
	-15%	359.3	427.8	417.8	505.6	507.0	501.7
	-10%	401.6	447.3	440.7	499.2	500.1	496.6
	-5%	444.0	466.8	463.5	492.8	493.2	491.5
	0%	**486.3**	**486.3**	**486.3**	**486.3**	**486.3**	**486.3**
	5%	528.7	505.9	509.2	479.9	479.4	481.2
	10%	571.0	525.4	532.0	473.5	472.6	476.1
	15%	613.3	544.8	554.8	467.1	465.7	471.0
	20%	655.6	564.3	577.6	460.7	458.8	465.8
	25%	697.8	583.8	600.4	454.3	451.9	460.7
	30%	740.1	603.2	623.2	447.8	445.0	455.6

Note: Figure prepared by SilverCrest, 2021.

Figure 22-4: Post-tax NPV Sensitivities (base-case is bolded)



		IRR (%)					
		Metal Prices	Gold Grade	Silver Grade	Operating Costs	Capital Costs	Sustaining Capital Costs
Percent Change from Base (%)	-30%	30%	42%	40%	55%	68%	55%
	-25%	34%	44%	42%	55%	65%	54%
	-20%	38%	45%	44%	54%	62%	54%
	-15%	41%	47%	46%	53%	59%	53%
	-10%	45%	48%	48%	53%	56%	53%
	-5%	48%	50%	50%	52%	54%	52%
	0%	**52%**	**52%**	**52%**	**52%**	**52%**	**52%**
	5%	55%	53%	53%	51%	49%	51%
	10%	58%	55%	55%	50%	48%	51%
	15%	61%	56%	57%	50%	46%	50%
	20%	64%	58%	59%	49%	44%	50%
	25%	67%	59%	60%	48%	42%	49%
	30%	70%	60%	62%	48%	41%	49%

Note: Figure prepared by SilverCrest, 2021.

Figure 22-5: Post-tax IRR Sensitivities (the base-case is bolded)

The Project is most sensitive to changes in metal prices, less sensitive to changes in capital and sustaining costs, and least sensitive to changes in operating capital costs. Grade sensitivity mirrors the sensitivity to metal prices.

22.6 Gold and Silver Price Scenarios

A sensitivity analysis was performed to assess the impact of changing gold and silver prices on the Project, as outlined in Table 22-8. The base case is bolded in the table.

Table 22-8: Economic Results for Different Metal Price Scenarios

Price Case	Gold Price ($/oz)	Silver Price ($/oz)	Post-Tax NPV 5% ($M)	Post-tax IRR (%)
Base Case	**1,500**	**19.00**	**486.3**	**52**
Three-year Trailing Average	1,788	17.73	530.7	55
Upside (Spot Case)	1,946	27.36	802.5	74
Downside (PEA Base Case Prices)	1,269	16.68	370.4	42

Note: Five-year Trailing prices and Spot Prices are based on data as of January 4, 2021. PEA Case is based on the pricing from the PEA report with effective date of May 15, 2019 and as amended July 8, 2019.

23 Adjacent properties

No advanced exploration or operating properties are known to exist immediately adjacent, or contiguous to, the Las Chispas Property that have relevance to the Report.

23.1 Nearby Properties and Operating Mines

Numerous operating mines exist along the Rio Sonora valley in proximity to the Las Chispas Property. These include the nearby Santa Elena Mine, operated by First Majestic Silver Corp., and the Mercedes Mine, operated by Premier Gold Mines Ltd. The Santa Elena Mine is a gold-silver underground mine, processing approximately 3,000 t/d and is located approximately 22 km south–southwest of Las Chispas (First Majestic, 2019). The Mercedes Mine is also a gold–silver underground mine, processing approximately 2,000 t/d and is located approximately 33 km to the northwest of Las Chispas (Premier, 2019).

The mineral deposits being exploited at these mines are low to intermediate sulphidation epithermal veins with associated breccia and stockwork over varying widths of <1 m to >10 m. The deposits are hosted in volcaniclastic host rocks with similar age of precious metal emplacement (late Cretaceous to Paleogene Neogene) to Las Chispas. The gold–silver mineralization found on these properties are similar to Las Chispas in lithology, structural controls, alteration, and geochemistry with some variations. Current mining operations may differ from a potential future operation at Las Chispas.

24 Other relevant data and information

24.1 Project Execution Plan

24.1.1 Overview

The Project Execution Plan (PEP) sets out the execution, monitoring, and controls for the construction of the Project, to bring it into operation while meeting the following objectives:

- Zero harm to personnel involved with construction, operation, and maintenance of the facilities;
- Zero unintended environmental impact or incidents;
- Preserve or improve the Project value through effective control of Project costs and completion of construction and commissioning on or ahead of schedule;
- Satisfy quality and performance targets;
- Comply with company policies and legislative requirements; and,
- Maintain positive community relations.

24.1.2 Strategy

An experienced construction management team will be appointed to supervise contractors for the construction of the process plant, the FTSF and all on-site and off-site infrastructure, including the 33 kV HV power supply to site.

Mine development will to be directly managed by SilverCrest under the supervision of a Mine General Manager. A comprehensive Operation Readiness Plan (ORP) is being developed under the supervision of the Mine General Manager and will be implemented to support an efficient ramp-up of both the mine and the plant.

To limit the impact of SARS CoV-2 (COVID-19) on the construction workforce and the surrounding communities of Arizpe and Banamichi, SilverCrest will build a 520-room accommodation camp at the Project site. The camp will be built in three stages to align with the forecasted construction ramp-up schedule. Entry to the site will be strictly controlled. The following will be in place;

- All people seeking to enter the project site will be mustered in Hermosillo, Sonora, and tested for the COVID-19 with a PCR test. Before being tested, individuals will start a quarantine that will be completed upon reception of negative results. Those personnel who test negative will be allowed entry and will be transported to the site in SilverCrest's vehicles;

- The construction workforce will be accommodated in single-occupancy rooms with self-contained ablution facilities. Personnel will not be permitted to leave site, unless authorized;

- Control measures such as the use of masks, indoor occupancy limits, and promoting social distancing and hygiene practices are already in place at the site; and,

- Protocols have been established for on-site case management of people who develop COVID-19 symptoms.

The 520-room camp will be supplemented with a contingency camp (up to 150 rooms) located in Arizpe, and which, when used, will operate in a similar manner.

24.1.3 Schedule

The Project will be constructed in two overlapping phases:

- An Early Works (July 2020 to Q1-2021) program during which key infrastructure items such as the administration building, warehouse, and construction camp and related facilities have been and/or will be erected, and bulk earthworks for the process plant will be completed; and,

- Construction of the process plant, and the FTSF will be completed over 17 months, starting in January 2021.

The schedule was adjusted to reflect productivity losses associated with the conditions required to operate under the COVID-19 pandemic.

The target date for the plant start-up is June 2022. To achieve this target, project execution is running in parallel with completion of the Feasibility Study. Detailed engineering and design of the process plant commenced in August 2020 and procurement of long-lead equipment items was authorised to start in September 2020.

As of January 2021, >8,000 metres of underground development was complete, and the ROM stockpile counted 48,218 tonnes (plus an additional 162,561 tonnes of historical stockpile).

24.1.4 Project Management

The Project team is organized based on an integrated team approach, minimising the duplication of roles and activities between the Owner’s Team and their major delivery partners.

24.1.5 Engineering

Consultants engaged by SilverCrest to complete engineering and design are to:

- Develop designs for facilities that are safe to construct, commission, operate and maintain;
- Deliver a cost-effective design that considers capital and operating costs to optimize Project lifecycle costs;
- Meet production throughput, recovery, and availability targets; and,
- Achieve the required quality standards.

The construction cost estimate and delivery schedule were developed through early engagement with preferred sub-contractors identified during the Feasibility Study. This has allowed detailed engineering to reach 60% in Q4-2020, As a result, both the cost estimate and the proposed schedule benefited from more advanced engineering than a typical Feasibility Study.

24.1.6 Procurement and Contracting

Procurement tasks will be prioritized by equipment delivery time, and to support engineering progress. Purchase orders for non-critical equipment and materials supplied from Canada, USA, or Mexico will include transport to site. Transport of goods supplied from outside the United States-Mexica-Canada Agreement (USMCA) zone or critical items will be managed by a freight forwarder.

The contracting strategy was developed to minimize the number of contracting companies required on site. This strategy will lower Project risk by reducing the number of interfaces and simplify logistics to efficiently move workers through COVID-19 testing and transport them to site at the start of each rotation.

24.1.7 Logistics and Materials Management

A route survey was commissioned to assess road conditions from main supply points to site, identify upgrade or relocation works that may need to be completed, and set the transport envelopes for delivery of goods and materials to site.

Responsibility for movement of goods and materials during execution will be assigned based on criticality and port of origin. Criticality will be judged based on value, load size or weight, potential for damage during transport, and construction schedule float.

The transport of critical items will be entrusted to a major logistics provider to manage ex-works loading and transport, customs clearance (where required), and coordination and tracking of deliveries to site. Non-critical items will be transported by original equipment manufacturer (OEM) suppliers.

24.1.8 Construction Execution

SilverCrest will manage the Las Chispas site through all phases of the Project. Contractors responsible for mine development, and delivery of the process plant will report to the Mine General Manager. Regular communication meetings will be held between all contractors working on site, facilitated by SilverCrest.

The health and safety of all workers on the construction site is a primary concern for SilverCrest. A Health and Safety Management Plan has been developed that outlines the obligations for all contractors on site. A safe work site will be established by:

- Implementing of Health and Safety Plan systems and procedures;
- Ensuring that contractors provide documents that verify personnel as qualified or competent, and plant, machinery, equipment and tools as fit for purpose;
- Auditing the execution of the contractor’s hazard and risk management processes, including risk profiles, safe work method statements and work permits;
- Implementing Rules of Safety and Fit for Work Programs;
- Maintaining Health and Safety records and providing weekly and monthly reports; and,
- Conducting emergency response drills and participating in the project emergency response program.

24.1.9 Commissioning

The EPC contractor will complete all pre-commissioning and wet-commissioning tests, prior to turnover to SilverCrest for commissioning and ramp-up. The same contractor will provide support for commissioning and ramp-up as required.

24.2 Risks

24.2.1 Introduction

A structured risk management process was established to promote early identification of risks, determine the likelihood and consequence of risk actualisation, and propose risk mitigation plans to reduce the likelihood and/or impacts. The same process was used to identify and promote opportunities.

The process included input from all sub-contractors. The risk and opportunity register is updated quarterly, with critical risks reviewed more frequently as required.

24.2.2 Construction Schedule

The major risk to construction of the process plant and infrastructure is disruption due to a COVID-19 outbreak on site or in the local community. To reduce the likelihood of this risk materializing, the construction workforce will be accommodated at the Project site, isolated from the local community. Access to and from site will be strictly controlled, including quarantine, and testing prior to authorising access to site. Control measures such as the use of masks, indoor occupancy limits, and promoting social distancing and hygiene practices has already been implemented on the Project site. Monitoring measures such as random testing will also be implemented.

Worker safety on site is protected through implementation and enforcement of a comprehensive safety management program. The protocols and procedures in the management plan are reinforced through site induction, training for higher risk tasks, daily toolbox talks and behavioural observations and interventions by the CMT.

24.2.3 Mineral Resource Estimates

Confidence in the grade, vein thickness and corresponding volume of material above a COG is influenced by several factors, including the distance between drill core samples, the direction between samples, and proximity to high-grade shoots within the vein. Lower confidence is thus associated with widely spaced drilling and higher confidence is associated with closer-spaced drilling. The actual drill sample spacing varies by vein and the classification of Mineral Resource Estimates was assigned based on the level of confidence based on drill core sample spacing. Risk is associated with all classifications of Mineral Resource Estimates; however, the greatest risk is associated with Inferred Mineral Resource Estimates.

24.2.3.1 Wireframe

There is a risk that the Mineral Resource Estimate wireframes (>150 gpt AgEq) may be moderately high biased with respect to the representative volume, and subsequent estimated tonnage and metal content. This potential bias could be where the wireframes extend somewhat too far into lower-grade (<150 gpt AgEq) assay areas of influence. When the first stope mining operation commences in each vein, a follow-up rolling reconciliation is recommended to allow for any mine call factor adjustments to be made.

24.2.3.2 Accuracy of Localized Extreme High-Grade Samples

Localized extremely high-grade samples were encountered in drill core sampling as part of the mineralization system. From limited duplicate sampling, the grade behaved as nuggety gold and silver mineralization, which was described from the duplicate core samples as having comparable order of magnitude grades within a margin of error of approximately ±10%, for grades >10 gpt Au and >1,000 gpt Ag. Locally, this represents a risk in the accuracy of grade estimation for Mineral Resource and subsequent Mineral Reserve estimation, and to operational grade control.

24.2.3.3 Grade Capping

Where only widely spaced sampling is available, the spatial extent of the high-grade mineralization may be uncertain. The selection of grade estimation methods to extrapolate high-grade samples can strongly influence regionalized vein grades and volumes. Controls such as grade capping and search distance constraints were implemented to manage the local influence of individual high-grade samples. However, there is a risk that the grade capping methodology used was too liberal during its attempt to preserve local grade variability. In areas with widely spaced sampling and containing high-grade assays, somewhat subjective grade capping strategies and restrictive search ellipse strategies were used that may introduce a risk for grade overestimation. This risk can be reduced through future close-range sampling to better delineate high-grade shoots within the vein systems, thereby allowing the highest-grade material to be domained to constrain spatial influence of these samples within delineated shoots. Closely spaced pre-production definition drilling in combination with duplicate sampling protocols for high-grade samples should be implemented to mitigate excessive extrapolation of high-grade values and to better inform the local, short-range, grade variability.

24.2.4 Mineral Reserve Estimates

24.2.4.1 Mining Recovery and Mining Dilution

The main risks that can affect the Mineral Reserves are the decrease in mining recovery and the increase in mining dilution due to the narrow veins that make up the deposit. To mitigate this risk, the mine ramp-up will be gradually increased to design level and stabilized in 2025. This will provide additional time to implement appropriate grade control measures and proper mining execution. These measures will help maintain minimal unplanned dilution and optimize mining recovery, which would minimize potential impacts on grade, throughput, and operating costs.

24.2.4.2 Open Stope in Babicanora Central and Potential Unknown Historical Excavations

There is a known open stope area in the Babicanora Central zone. This area could cause recovery problems because although the general area is known, the exact size and geometry of the open stope is unknown. To mitigate the possible impact of this risk, mining recovery was tested in Q3 and Q4 2020 with successful results. This area required additional backfilling, and grade monitoring during operations will be critical.

Although SilverCrest has some details as it relates to historical production, old unknown excavations may be encountered during mining. To mitigate this risk, a test hole program will be needed during development and stoping.

24.2.5 Underground Mine Development

A portion of the mine will require stringent measures to maintain/control good ground conditions. In addition, the mine plan requires a significant amount of development, and a portion of production will rely on lower-productivity mining methods. This combination of factors represents a risk to ensure the plant operates at its designed capacity.

SilverCrest developed the following plan to mitigate that risk:

- Development started early, and more than 8 km of development have been completed as of the end of Q4-2020. This development has been helpful to define productivity, costs, and mining methods, but also to understand ground conditions;
- The site management team has already been hired. This team is partnered with a mining contractor that has the capacity and desire to expand its contribution;
- The historical stockpile was estimated at approximately 162,561 t, which on its own represents more than four months of plant production at the forecast throughput rate. In-vein drifting completed as of the end of Q4-2020 added about 48,218 t of high-grade mineralization. The LOM plan presented in this Feasibility Study estimates that by the end of 2022, the high-grade ROM stockpile will contain approximately >261,000 t of high-grade material; and,
- The LOM will include a gradual ramp-up to take advantage of the stockpiled material. The mining rate is expected to ramp-up through 2025 and then stabilize. This gradual ramp-up is expected to help with all aspects of mining, including safety, planning, reconciliation, dilution, mining recovery and cost control. There is potential to accelerate the ramp-up to improve mine flexibility.

24.2.6 Tailings Filter Capacity

Sizing of the tailings filters was based on a limited number of samples with different filtration rates. The location of high clay/mica zones within the mineralization is not thoroughly understood. There is a risk that spikes of high clay/mica content material, with poor filtration properties, may occur in the process plant and cause reduced capacity through the filters.

To mitigate this, the plant design team used the sample that had the lowest results for sizing and assumed a duty/stand-by configuration for the filter system. In extreme cases, it is more likely that SilverCrest would operate the filters on a duty/duty basis when higher clay/mica contents are being experienced.

During detailed design, SilverCrest can further characterize the mineralization types and identify areas that have high clay/mica contents and thus will be able to better plan for the treatment of these materials in the plant.

24.2.7 Water Balance and Buildup of Impurities

When treating very high-grade gold–silver–copper that requires high cyanide and zinc reagent additions, there is potential for impurities to build up in the recirculating process water. A high circulating load of copper and cyanide can result in increased zinc demand, further exacerbating the problem.

To mitigate this, the cyanide detoxification circuit was designed to treat an additional barren bleed stream to purge impurities from the process water. Water that is bled from the circuit can be stored in contact water ponds where it will evaporate during the dry months.

24.2.8 Filtered Tailings System Facility (FTSF)

The optimum moisture content for the filtered tailings, based on standard Proctor compaction tests, ranges from 14.5–16.0% and the tailings can reach 95% compaction of the standard Proctor test with approximately ±3% of the optimum moisture content. Based on tailings filtering assays (Outotec, 2020) a tailings sample representative of Area 51 dewatered to a gravimetric moisture content of approximately 19%. However, a tailings composite sample from this filtering assays showed high clay/mica content and could only be dewatered to a 22% gravimetric moisture content, which is above the range of optimum compaction contents determined from geotechnical laboratory testing. While this sample represents a fraction of the mineralization, there is a risk that a larger portion of the tailings could exhibit higher clay contents than anticipated. This could translate into greater moisture than the target at the filter plant, and longer times and greater effort to process and compact the filtered tailings at the FTSF.

This risk will be mitigated by providing sufficient area for the FTSF, in the early stages of stacking, where tailings that do not meet the design specifications or higher clay content tailings can be temporarily placed in the interior portion of the FTSF. Filtered tailings could then be extended and compacted when conditions allow, without the need to stop tailings disposal.

Additional tailings storage capacity was designed to a feasibility level of confidence to address the scenario of higher clay content tailings being produced in the second half of the LOM. This design assumed that an additional facility, the NW FTSF, could be used as an alternative for temporary, non-specification tailings storage. The capital cost estimate to construct the NW FTSF and mitigate this risk, if it materializes, is approximately $0.64 M.

24.3 Opportunities

24.3.1 Introduction

The following sub-sections summarize opportunities that could potentially improve upon the economics of the Feasibility Study. Alone or combined, these opportunities could change the approach to development, timelines, capital requirements and operating costs described within the Feasibility Study with potential to change the scale, economics and/or the Project value.

24.3.2 Exploration & Mineral Resource Estimation

The most significant upside is the potential for; 1) conversion of Inferred Resources to Indicated Resources and possibly to Reserves in the future, 2) conversion of excluded Indicated Resources to Reserves in the future, and 3) discovery of additional mineralization that may support Mineral Resource estimation.

The 2021 drill program should focus on increasing confidence and growing resources for possible conversion to reserves. Work will consist of infill drilling, in-vein development and completing other studies (geotechnical, metallurgical, mine design) on the Indicated Mineral Resources that were excluded from Mineral Reserves. Also, work will be completed on Inferred Mineral Resources to improve confidence for possible conversion to Indicated Resources and consideration for being converted to reserves in 2021. Also, testing new targets along vein strike and to depth is suggested. The 2021 priorities for exploration are as follows (listed in order of priority):

(1) Inferred Mineral Resources are estimated at 29.7 Moz AgEq (refer to Table 14-13) with a majority of these resources located as follows;

(a) Babi Vista Vein Splay has an estimated 13.85 Moz AgEq in Inferred Resources. Encouraging high grade mineralization has been intercepted in 22 holes with two drill holes averaging 256.7 gpt Au and 11,141 gpt Ag, or 33, 450 gpt AgEq (uncut, undiluted, 86.9:1 Au:Ag) with an average est. true width of 2.9 m. These holes are spaced approximately 60 m apart along an interpreted high-grade shoot (est. at least 90 m long by 200 m high) within the Babi Vista Vein Splay. This mineralization is located approximately 400 m southeast of the current ongoing underground development and within 200 m of the proposed mine design for the Babi Vista Vein. Additionally, there is an estimated 1.5 km of vein strike length that remains untested for these veins towards the southeast.

(b) El Muerto Zone (NW part of the Babicanora Main Vein) has an estimated Inferred Resource of 4.75 Moz AgEq. Drilling of the El Muerto Zone at depth below the Babicanora Central Zone has potential to intercept additional high-grade mineralization below the limit of the current Mineral Resource Estimate.

(c) Babicanora Sur Vein has an estimated 1.73 Moz AgEq in Inferred Resources. Deeper drilling is required to expand and infill for conversion of resources. Underground drilling from existing development may be considered.

(d) Granaditas 1 and 2 veins have an estimated Inferred Resources of 1.1 Moz AgEq combined. Drilling suggests that these veins may be the southeastern extension of the Babi Vista Vein and Babi Vista Vein Splay. Expansion drilling along strike between the Granaditas and the Babi Vista Veins veins haves the potential to intercept mineralization over a strike length of 300 m. Additional drilling may support potential conversion of Inferred Mineral Resources to Indicated Resources.

(e) Babicanora Norte Vein has been recently exposed by underground in-vein development for approximately 20 m along strike. This development shows a mineralized clay-rich structure (0.1 m to 1.5 m wide) as part of the vein which was not previously identified by drilling. This occurrence is similar to the previous discovery of a clay-rich structure parallel and part of the nearby Babicanora Main Vein. Further development of this vein may show this mineralized structure to have continuity. Ongoing in-vein development shows strong mineralization outside the current resource model suggesting potential to increase resources adjacent to the proposed mine plan. The vein remains open for further exploration along strike and to depth.

(2) A majority of the Indicated Mineral Resources that were not converted to Mineral Reserves nor included in the Feasibility Study mine plan are estimated at 14.8 Moz AgEq contained in 2.4 Mt at 1.04 gpt Au and 102.9 gpt Ag are adjacent or in close proximity to the proposed mine plan. Further infill drilling and studies may convert these resources to reserves in the future based on the following;

(a) The Babicanora Main Vein has an estimated Indicated Resource of 7.0 Moz AgEq contained in 1.0 Mt at 1.18 gpt Au and 108.6 gpt Ag that was not incorporated into the Mineral Reserves. Work in 2021 will include underground in-vein development, further infill drilling and mine plan optimization which may help to convert resources to reserves.

(b) The Babicanora Norte Vein has an estimated Indicated Resource of 1.8 Moz AgEq contained in 0.3 Mt at 0.86 gpt Au and 96.0 gpt Ag that was not incorporated into the Mineral Reserves. In-vein development and further infill drilling may help to convert resources to reserves.

(c) The Babi Vista Vein has an estimated Indicated Resource of 1.3 Moz AgEq contained in 0.2 Mt at 1.54 gpt Au and 129.3 gpt Ag that was not incorporated into the Mineral Reserves. In-vein development and further infill drilling may help to convert resources to reserves.

(d) The Las Chispas Vein has an estimated Indicated Resource of 1.6 Moz AgEq contained in 0.2 Mt at 1.25 gpt Au and 184.7 gpt Ag that was not incorporated into the Mineral Reserves. A majority of these ounces are located proximal to old historic workings with large pillars. Further geotechnical and mine design work may help to convert these resources to reserves.

Exploration and study information collected in 2021 will be considered for a possible update of resources and reserves in 2022. Further exploration and drilling do not guarantee success in the identification of additional mineralization, nor collection of information required for the confidence necessary to delineate mineral resources and mineral reserves.

24.3.3 Bulk Density

Results from independent site visit due diligence and metallurgical testing indicated that the bulk densities vary according to host rock and style of mineralization. In some areas of the deposit, bulk densities are higher than the 2.55 t/m^3 value used in Mineral Resource and Mineral Reserve estimation.

If higher bulk densities are confirmed, it is estimated that the Mineral Reserve tonnage could increase by 5% to 7% and the Mineral Resource tonnage by 4% to 6%.

24.3.4 Pillar Recovery

The mine design assumes that there are numerous pillars which will remain throughout the mine life. With the mine expected to showcase good ground conditions and cemented rock fill being employed in mined-out stopes, there is potential that some pillars could be recovered.

24.3.5 Mine Design and Schedule Optimization

Positive exploration drilling results may present an opportunity for further optimization of the mine design and schedule ahead of commercial production. Several of the high priority exploration opportunities are within or close to the proposed footprint of underground development. This could allow for low capital cost modifications to the mine design to incorporate any such opportunities.

24.3.6 Plant Expandability

Although the processing plant will be built at a capacity of 1,250 t/d, it is designed to accommodate a future expansion of up to 1,750 t/d. While additional studies and engineering would be required to execute a future plant expansion; conceptually, such an expansion could be achieved through the addition of a ball mill, pebble crusher and additional flotation capacity. Given the limited capital involved between thickener sizing (14 m vs 12 m), the CCD circuit was already designed to accommodate future additional capacity.

24.3.7 Sale of Flotation Concentrate

During testing of high-grade Au/Ag mineralized material through the flotation circuit a saleable concentrate was produced. Preliminary engineering work was conducted to include a flotation concentrate thickener and filter for direct shipping the concentrate to market. Further work is planned in this area.

24.3.8 Electrowinning

There is an opportunity to directly electrowin pregnant leach solution produced by cyanidation of flotation concentrate. When testing very high-grade gold-silver material, increasing cyanide, zinc and subsequently cyanide destruction reagents were required to accommodate the high metal load. Electrowinning of the high-grade solution provides an opportunity, during periods of very high metal grade feeds, to reduce operating cost. Further work is planned to evaluate electrowinning to produce doré versus sale of flotation concentrate.

25 Interpretation and Conclusions

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for the Report.

25.2 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

Information from legal and SilverCrest experts support that the tenure held is valid and sufficient to support a declaration of Mineral Resources and Mineral Reserves.

The Project consists of 28 mineral concessions, totalling 1,400.96 ha. Concessions have expiry dates that run from 2022–2067. One concession is in the grant process, and one concession is the subject of legal proceedings following cancellation. The mineral concessions that host the Mineral Resources and Mineral Reserves are in good standing. At the Effective Date of the Report, all required mining duties were paid.

There are three option agreements. Option 1 is for the La Fortuna mining concession applications No. 082/39410 and 082/38731, which cover the Panuco II and Carmen Dos Fracción II mineral lots; title transfer to LLA of the new concession are pending until the applications are issued as mining concessions. Under Option 2, LLA has a 67% ownership interest in the Lopez concession. Option 3 provides LLA with 100% rights to Panuco II, pending title for ownership. All three options are not material to the Feasibility Study.

SilverCrest holds sufficient surface rights to support planned operations. The surface rights overlying the Las Chispas mineral concessions and road access from local highway are either owned by LLA or held by LLA under negotiated lease agreements.

A 2% royalty is payable to the Gutierrez-Perez-Ramirez optionees, on the Nuevo Lupena and Panuco II concessions (when granted) for material that has processed grades of ≥0.5 oz/tonne gold and ≥ 40 oz/tonne silver, combined. No Mineral Reserves exist on these concessions.

To the extent known to the QP, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the Project that are not discussed in the Report.

25.3 Geology and Mineralization

Mineral deposits in the Las Chispas district are classified as gold and silver, low to intermediate sulphidation epithermal systems.

The understanding of the vein settings, lithologies, mineralization, and the geological, structural, and alteration controls on mineralization is sufficient to support estimation of Mineral Resources and Mineral Reserves.

There is remaining exploration potential in the Project area. A number of the known veins remain open along strike (e.g. Babi Vista Splay) and at depth (e.g. Amethyst and Babicanora Norte Veins). Surface geological mapping identified several mineralization structures to the north of the Las Chispas Area that require drill testing (e.g. Los Chiltepins, El Cumaro and Ranch Veins).

25.4 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

The exploration programs completed to date are appropriate for epithermal-style mineralization.

Sampling methods are acceptable to support Mineral Resource estimation.

Sample preparation, analysis and security were generally performed in accordance with exploration best practices and industry standards at the time the information was collected.

The quantity and quality of the logged geological data, collar, and downhole survey data collected in the exploration and infill drill programs are sufficient to support Mineral Resource estimation.

No material factors were identified with the data collection from the drill programs that could significantly affect Mineral Resource estimation.

Sample preparation and analyses were performed by independent accredited laboratories. The sample preparation, analysis, and security practices and are acceptable, meet industry-standard practices at the time they were undertaken, and are sufficient to support Mineral Resource estimation.

QA/QC submission rates meet industry-accepted standards at the time of each of the drill campaigns. The QA/QC programs did not detect any material sample biases in the data reviewed that supports Mineral Resource estimation.

The data verification programs concluded that the data collected from the Project adequately support the geological interpretations and constitute a database of sufficient quality to support the use of the data in Mineral Resource estimation.

25.5 Metallurgical Testwork

Metallurgical testwork completed is conventional for gold–silver epithermal deposits and suitable to support process design.

The materials tested are considered to be representative of the Las Chispas deposit, both with respect to the global average materials characteristics and with respect to high-grade, low-grade (waste) and high-clay-containing zones within the deposit.

Testwork indicated that the preferred flowsheet for the Las Chispas deposit should include flotation, cyanide leaching of both the concentrate and tails fractions, CCD washing, and precious metal recovery by the Merrill Crowe process.

Based on the selected flowsheet for plant operation, over the range of grades (variability samples) tested, the recoveries for gold and silver varied from 97.2–98.3% and 90.5–96.6%, respectively. Gold recoveries were shown to be relatively independent of grade over the range tested and use of an average recovery value of 97.6% would be appropriate. Conversely, overall silver recovery was shown to vary almost linearly with grade over the range tests. The silver recovery is relatively predictable therefore and can be estimated at 94.3% on the basis of the LOM average grade determined in the mine plan.

Based on the testwork performed to date, two deleterious elements, antimony and mercury, were identified. Antimony showed no adverse effect on gold and silver recoveries. Mercury is anticipated to be extracted to a small extent and that dissolved is expected to report directly to the zinc precipitate. A mercury retort facility is recommended for the commercial facility to recover the mercury attendant with the zinc precipitate and minimize its deportment to the doré product. There are no other deleterious elements in the mineralization which would require special processing techniques or compromise doré marketability.

Area 51 Zone mineralization with high clay content did not present any problematic issues for gold and silver recoveries or material handling.

25.6 Mineral Resource Estimates

Mineral Resource estimation has been prepared in accordance with the 2019 CIM Best Practice Guidelines and has been reported using the 2014 CIM Definition Standards.

The Mineral Resource Estimate includes in-situ narrow vein gold and silver mineralization at the Babicanora and Las Chispas areas that is potentially amenable to underground mining methods, and gold and silver mineralization contained within surface stockpiles that resulted from historical operations.

The Mineral Resources have been validated and classified as Inferred, Indicated and Measured resources. The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to the Indicated and Measured Mineral Resource and were not eligible for consideration as Mineral Reserves. It is reasonably expected that the majority of the Inferred Mineral Resource could potentially be upgraded to an Indicated Mineral Resource with continued exploration.

25.7 Mineral Reserve Estimates

Mineral Reserve estimation uses industry-accepted practices, and the estimate is reported using the 2014 CIM Definition Standards, and the 2019 CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines.

Mineral Reserves were converted from Measured and Indicated Mineral Resources and do not include any Inferred Mineral Resources. Inferred Mineral Resources contained within the Mineral Resource block models were treated as waste at zero grade.

Factors that may affect the Mineral Reserve Estimates include: geological complexity, geological interpretation, and Mineral Resource block modelling; COG estimations; commodity prices, market conditions and foreign exchange rate assumptions; operating cost assumptions; sustaining capital costs to develop; rock quality and geotechnical constraints, dilution and mining recovery factors; hydrogeological assumptions; and metallurgical process recoveries. There are no other environmental, legal, title, taxation, socioeconomic, marketing, political or other relevant factors known to the QP that would materially affect the estimation of Mineral Reserves that are not discussed in the Report. It is reasonably expected that all necessary government approvals will be issued for the project to proceed.

25.8 Mine Plan

25.8.1 Geotechnical Considerations

The rock quality of the various Las Chispas mining areas can be broadly divided into two main domains, "Poor–Fair/Good" and "Fair–Good/Very Good". The lower part of the Babicanora Main vein has a better rock quality than the upper portion. The Babicanora Norte, Babicanora Norte Northwest, Babi Vista, Babicanora Sur, Babicanora FW, Las Chispas, La Blanquita zone, and Giovanni Veins have a range of "Fair-Good/V. Good" rock quality.

Ground support selection considered industry-standard empirical design guidelines and Rockland's experience with variable ground conditions. Ground support was recommended based on rock quality, the period in use (long-term or short-term), and the size of headings (excavation). The ground support consisting of Inflatable bolts (e.g., Swellex) or resin rebars with Split Sets and mesh was specified.

25.8.2 Hydrological Considerations

The water table is at approximately 900 m elevation, and the perched phreatic surface is at about 1,032 m elevation. The perched phreatic surface does not impact the historical workings, and for the purposes of the mine plan, will not require dewatering. As the majority of the workings will be above the water table elevation of 900 masl, groundwater inflows are not expected to be a concern to mining operations.

The mine plan in the Las Chispas mining area were designed with a dewatering system in the lower levels, with an estimated pumping capacity of 9.4 L/s; however, this pumping system will not be required until late in the mine life.

25.8.3 Mining Method

The mine design was based on a production rate of 1,250 t/d. The proposed underground mining approach will use variations of long-hole stoping and cut-and-fill mining methods via several access drifts and ramps. These methods are appropriate to the sub-vertical vein geometry, where the veins have thicknesses ranging from 0.5–10 m.

Mining operations will extract from 15 principal veins divided into six mining areas, which will be accessed via three portals. Each mining area will be serviced by supporting infrastructure including power distribution, compressed air distribution, water supply, ventilation, dewatering and communications.

The planned equipment fleet is conventional for underground mining operations.

All mining activities will be completed via a contractor. It is expected that the contractor will supply adequate underground mining equipment for the different mining activities.

25.9 Process Plant

The process plant will be located at the mine site and will receive blended feed material from a number of different mineralized veins.

The planned flowsheet is flexible and robust, and is based on well-proven unit operations in the industry, and there are no unique or novel processing methods required for gold and silver recovery. The flowsheet includes semi-autogenous grinding (SAG), flotation, independent cyanide leaching circuits for both flotation concentrate and tailings streams, Merrill Crowe circuit, and tailings handling facilities, with an overall availability of 91.3%.

The major equipment was designed for a nominal throughput of 1,250 t/d with the ability to accommodate increased throughput up to 1,750 t/d via an expansion.

25.10 Infrastructure

The Project, as envisaged, will include the following infrastructure: underground mine, including portals and ramps; various roads such as the main access road to site, borrow pit haul road, FTSF haul road, WRSF haul road, and explosives access road; diversion and collection channels, culverts, and containment structures; site main gate and guard house; construction camp; warehouse and truck shop, offices, process plant dry facility, medical clinic, and nursery; explosives magazines, exploration core shack; processing plant; control room; doré room; assay laboratory (off-site facility); reagent storage facility; water treatment plant; stockpiles and WRSFs; FTSF; hazardous waste containment facility; and COVID-19 testing facility (off-site facility).

Electrical power will be supplied to site from the national grid, by way of an overhead power line.

25.11 Markets and Contracts

Detailed market studies on the potential sale of gold and silver doré were not completed; however, gold and silver doré can be readily sold on many markets throughout the world and the market price can be ascertained on demand.

No contracts were entered into at the Report Effective Date for mining, facility operations, refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements. It is envisaged that SilverCrest would sell any future production through contracts with a refiner, or on the spot market, as applicable. It is expected that when any such contracts are negotiated, they would be within industry norms for projects in similar settings in Mexico.

Metal pricing used in the economic analysis is based on consideration of various metal price sources. This included review of consensus price forecasts from banks and financial institutions, three-year trailing average of spot prices, and current spot prices.

25.12 Environmental, Permitting and Social Considerations

25.12.1 Environmental Considerations

Environmental surveys and studies for the Project were completed in support of permit applications. Completed studies include climate, flora, fauna, air quality, noise, surface and groundwater quality.

ML testing showed that potentially leachable metals included barium and lead, but in concentrations that were well below the maximum allowable limits. ARD testing indicated that the majority of the rocks showed no ARD potential. The majority of tailings samples showed non-acid forming (NAF) characteristics in NAG testing.

No known environmental liabilities exist in the Project area from historical mining and processing operations. Soil and tailings testing were conducted as part of the overall sampling that has been ongoing at site. To date, there are no known contaminants in the soils. Water quality testing is currently ongoing through baseline environmental studies.

25.12.2 Permitting Considerations

LLA worked with its permitting team in Mexico to identify the key environmental permits and other Mexican regulatory permits required to construct and operate a mine in Sonora state, Mexico, and to identify which regulatory authorities grant such permits. A total of 27 key permits were identified, of which 20 have been granted, five are pending, and the remaining two permits have not had the application process initiated. Granted permits have varying terms, ranging from one year to unlimited terms. Permits will be renewed as required.

25.12.3 Closure and Reclamation

A Conceptual Closure Plan was prepared in general accordance with applicable Mexican standards. Under Mexican law, mining may be initiated under a Conceptual Closure Plan with a Detailed Closure Plan being developed later in the Project life.

The closure cost forecast is $3.4 M. Closure costs were assumed to be disbursed over a period of approximately three years, following cessation of production.

25.12.4 Social Considerations

A social baseline study, completed in 2019–2020, found key areas of community concern were: water usage, and water safety; a lack of information on the Project; concerns around an environmental incident in 2014 that was caused by a different mining company (100 km from Project site); a wish to see improvements in the local infrastructure; that environmental safety and appropriate mine closure protocols should be in place to protect the region at the end of the LOM; and job creation with a focus on opportunities being made available for women.

A 2019–2020 materiality assessment will form the basis of a company-wide Environmental and Social Management System. Key findings from the materiality assessment were centered around climate and water risks, community health issues (mining, food, water), environmental safety of the local river and agriculture, employment opportunities, a desire for improved infrastructure (sports, recreation, health) and a concern regarding a potential influx of people from outside the community taxing local infrastructure.

SilverCrest has formalized a communication strategy that employs direct outreach, social media, company-generated videos, flyers, posters and workshops. SilverCrest has set up a whistle blower policy and hotline and, at the Report Effective Date, was in the process of finalizing a grievance mechanism process.

25.13 Capital and Operating Cost Estimates

Capital and operating cost estimates have an accuracy range of ±15%.

The Project LOM capital costs total $265 M, consisting of an initial capital cost estimate of $137.7 M and sustaining capital cost estimate of $123.9 M and closure costs of $3.4 M. Approximately $25.8 M had been invested in the Project as of the Effective Date and this sum is excluded (sunk cost) from the LOM capital costs noted above.

The average LOM operating cost, at a design mill feed rate of 1,250 t/d, was estimated at $118.49/t of material milled. The operating cost is defined as the total direct operating costs including mining, processing, and G&A costs

25.14 Economic Analysis

A pre- and post-tax economic analysis was completed on the basis of a discounted cash flow model featuring a 5% discount rate. The model assumed a 17-month physical construction period, and production period of 8.5 years, including the first year and final year that will see production for only a portion of those years.

The economic model was based on a gold price of $1,500/oz and a silver price of $19.00/oz. The refining terms used as the basis of the economic analysis are based on an average of payment terms and refining costs provided to SilverCrest in quotations from third-party refiners and are well aligned with other operations in the region. The freight terms were based on local rates.

The taxable income was estimated at $715.1 M. SilverCrest applied a tax rate of 30% to this amount over the LOM for an estimated tax amount of $214.5 M over the LOM. A review was completed with the net operating losses excluded, and the Project economics remained positive.

The Project post-tax NPV at a 5% discount rate was estimated to be $486.3 M, with an IRR of 52%. The Project would achieve payback in 1.0 year.

The Project is most sensitive to changes in metal prices, less sensitive to changes in operating costs, and least sensitive to changes in capital costs. Grade sensitivity mirrors the sensitivity to metal prices.

25.15 Risks

25.15.1 COVID-19

The major risk to construction of the process plant and infrastructure is disruption due to a COVID-19 outbreak on site or in the local community. To reduce the likelihood of this risk materialising, the construction workforce will be accommodated at the Project site, isolated from the local community. Access to and from site will be strictly controlled, including quarantine, and testing prior to authorising access to site.

25.15.2 Mineral Resource Estimates

The drill sample spacing varies by vein and the classification of Mineral Resource estimates was assigned based on the level of confidence based on drill core sample spacing. Risk is associated with all classifications of Mineral Resource estimates; however, the greatest risk is associated with the Inferred Mineral Resource estimate.

There is a risk that the Mineral Resource Estimate wireframes (>150 gpt AgEq) may be moderately high biased with respect to the representative volume, and subsequent estimated tonnage and metal content. This potential bias could be where the wireframes extend somewhat too far into lower-grade (<150 gpt AgEq) assay areas of influence. When the first stope mining operation commences in each vein, a follow-up rolling reconciliation is recommended to allow for any mine call factor adjustments to be made.

Localized extremely high-grade samples were encountered in drill core sampling as part of the mineralization system. Locally, this represents a risk in the accuracy of grade estimation for Mineral Resource and subsequent Mineral Reserve estimation, and to operational grade control.

Where only widely spaced sampling is available, the spatial extent of the high-grade mineralization may be uncertain. This risk can be reduced through future close-range sampling to better delineate high-grade shoots within the vein systems, thereby allowing the highest-grade material to be domained to constrain spatial influence of these samples within delineated shoots. Closely spaced pre-production definition drilling in combination with duplicate sampling protocols for high-grade samples should be implemented to mitigate excessive extrapolation of high-grade values and to better inform the local, short-range, grade variability.

25.15.3 Mineral Reserve Estimates and Mine Plan

The main risks that can affect the Mineral Reserves are the decrease in mining recovery and the increase in mining dilution due to the narrow veins that make up the deposit. To mitigate this risk, the mine ramp-up will be gradually increased to design level and stabilized in 2025.

There is a known open stope area in the Babicanora Central zone that could cause recovery problems, because, although the general area is known, the exact size and geometry of the open stope is unknown. To mitigate the possible impact of this risk, mining recovery was tested in Q3 and Q4 2020 with successful results. This area required additional backfilling, and grade monitoring during operations will be critical.

Historical, excavations may be encountered during mining. To mitigate this risk, a test hole program will be needed during development and stoping.

A portion of the mine will require stringent measures to maintain/control good ground conditions. In addition, the mine plan requires a significant amount of development, and a portion of production will rely on lower-productivity mining methods. This combination of factors represents a risk to ensure the plant operates at its designed capacity. Planned mitigation measures include early commencement of development to provide information ground conditions, productivity, costs, and mining methods; early hire of site management personnel; stockpiling material so that the plant has an early supply of material for treatment, and gradual ramp-up.

25.15.4 Metallurgical Testwork and Recovery Plan

There is a risk that spikes of high clay/mica content material, with poor filtration properties, may occur in the process plant and cause reduced capacity through the tailings filters. During detailed design, SilverCrest can further characterize mineralization types and identify areas that have high clay/mica contents and thus will be able to better plan for the treatment of these materials in the plant. Additionally, a provision has been made to blend feed materials, to provide a consistent clay content, or grade to the plant which will mitigate this risk.

When treating very high-grade gold-silver-copper mineralized material that requires high cyanide and zinc reagent additions, there is potential for impurities to build up in the recirculating process water. To mitigate this, the cyanide detoxification circuit was designed to treat an additional barren bleed stream to purge impurities from the process water. Water that is bled from the circuit can be stored in contact water ponds where it will evaporate during the dry months.

There is a risk that a larger portion of the tailings could exhibit higher clay contents than anticipated. This could translate into greater moisture than the target at the filter plant, and longer times and greater effort to move and compact the filtered tailings at the FTSF. This risk will be mitigated by providing sufficient area for the FTSF, in the early stages of stacking, where tailings that do not meet the design specifications or have higher clay contents can be temporarily placed in the interior portion of the FTSF. Filtered tailings could then be moved and compacted when conditions allow, without the need to interrupt operations. Additional tailings storage capacity was designed to a feasibility level of confidence to address the scenario of higher clay content tailings being produced in the second half of the LOM. This design assumed that an additional facility, the NW FTSF, could be used as an alternative for temporary, non-specification tailings storage.

25.16 Opportunities

25.16.1 Exploration

The most significant upside is the potential for conversion of Inferred Mineral Resources to Mineral Indicated Resources and possibly to Mineral Reserves, conversion of excluded Indicated Mineral Resources to Mineral Reserves, and discovery of additional mineralization that may support Mineral Resource estimation.

Inferred Mineral Resources are estimated at 1.24 Mt grading 4.35 gpt Au, and 367 gpt Ag, or 745 gpt AgEq, for 29.7 Moz AgEq. The majority of these resources are located in the Babi Vista Vein Splay, Granaditas 1 and 2 veins, El Muerto Zone and the Babicanora Norte Vein. The most significant potential for adding resources and reserves is the Babi Vista Vein Splay with an estimated 211.4 kt grading 13.00 gpt Au and 909 gpt Ag, or 2,039 gpt AgEq for 13.85 Moz AgEq in

Indicated Mineral Resources that were not used for vein reserve calculation, not converted to Mineral Reserves, nor included in the Feasibility Study mine plan are estimated at 14.8 Moz AgEq (14.8 Moz 1.04 X 86.9 + 102.9 = 193 gpt AgEq) contained in 2.4 Mt at 1.04 gpt Au and 102.9 gpt Ag, and are either adjacent or proximate to the proposed mine plan. The most significant of these is the Babicanora Main Vein which has an estimated excluded Indicated Resource of 7.0 Moz Ag Eq contained in 1.0 Mt at 1.18 gpt Au and 108.6 gpt Ag or 8.41 Moz AgEq.

Up to October 16, 2020, 45 veins have been identified, but only 21 of those veins have had sufficient drilling to support at least an Inferred Mineral Resource Estimate. Surface exploration and drill-testing has identified over 30 km of potential vein strike length that remains to be tested. Future drilling should focus on step-out drilling within the known mineralization zones and testing deeper host lithologies, parallel veins and newly identified areas that had limited historical workings.

In some areas of the deposit, bulk densities are higher than the 2.55 t/m^3 value used in Mineral Resource and Mineral Reserve estimation. If higher bulk densities are confirmed, there is potential to slightly increase the tonnages in the estimates.

25.16.2 Construction & Mine Plan

The contingency cost for construction is $23.3 M. By achieving construction on-time and on budget, this cost may be unnecessary.

With the mine expected to showcase good ground conditions and cemented rock fill being employed in mined-out stopes, there is potential that some pillars could be recovered.

Positive exploration drilling results may present an opportunity for further optimization of the mine design and schedule ahead of commercial production. Several of the high priority exploration opportunities are within or close to the proposed footprint of underground development.

The Report is based on contractor development and mining. A cost benefit analysis may suggest reduced cost using owner-mining in the future.

25.16.3 Recovery Plan

While additional studies and engineering would be required to execute a future plant expansion; conceptually, such an expansion could be achieved through the addition of a ball mill, pebble crusher and additional flotation capacity.

25.17 Conclusions

Based on the assumptions and parameters presented in the Report, the Feasibility Study shows positive economics.

On the basis of the Project's positive economics, SilverCrest has elected to proceed with construction and has entered into an EPC agreement for delivery of the process plant and associated infrastructure.

26 Recommendations

26.1 Introduction

A two-phase program is recommended. The work recommended in the first phase relates to additional drilling, comprising infill and step-out drilling in the area where Mineral Resources have been estimated, includes exploration drilling based on targets from an extensive surface exploration program, and includes additional exploration activities such as geological mapping, sampling, mineralogy and geophysical surveys. The second phase focuses on studies including additional metallurgical testwork to identify areas of high mica/clay content in the veins in the mine plan, and updating the Mineral Resource Estimate using results of drilling, bulk density and geometallurgical testwork.

The majority of the second work phase can be completed in conjunction with the first work phase. A portion of the density determination and SWIR work suggested in Phase 2 will require channel sampling from in-vein drifting. Resource estimation would be completed once results of the Phase 1 drilling are available and would be updated to incorporate information from the proposed density and SWIR programs as those data became available.

The Phase 1 work program is estimated at $39 M. The Phase 2 program is estimated at $235,000.

26.2 Phase 1 – Drilling and Exploration

Contingent of the success of the drilling program, between 300,000–350,000 m of drilling is recommended, of which about 175,000 m (50%) would be allocated to infill drilling, 105,000 m (30%) to step-out or expansion drilling, and 70,000 m (20%) would be allocated to exploration drilling. Specific collar locations have yet to be determined. Successive drill collar locations will be dependent on the results of each drilled hole for the step-out and exploration programs. The program budget assumes about 70,000 samples will be taken and submitted for assay.

Infill drilling should be planned for the Babi Vista Vein Splay, and areas where Inferred Mineral Resources are currently estimated.

Step-out drilling should be planned for the Babicanora Area including the Babi Vista, Bab Vista Splay, El Muerto Zone, the area between the Granaditas and Babi Vista veins, Amethyst Vein, Babicanora Norte Vein, and in the Las Chispas Area including Area 118, La Blanquita, William Tell, and Giovanni Vein.

Exploration drilling should be planned for the Los Chiltepins veins, El Cumaro Vein, La Martina Vein and Ranch Vein areas.

Surface exploration surveys should include additional geological mapping and sampling, mineralogical studies and geophysical surveys.

The recommended budget for the Las Chispas Geology and Exploration program is approximately $39 M, comprising:

- Drilling, including assays, supervision and QAQC: $35.8 M; and,
- Surface exploration, surveys and studies, and administration: $3.2 M.

26.3 Phase 2 – Studies

26.3.1 QA/QC

The following QA/QC measures should be contemplated for operational purposes:

- When the first stope mining operation commences in each vein, a follow-up ongoing reconciliation is recommended to allow for any mine call factor adjustments to be made. Perform early-stage stope production tonnage and grade reconciliation; and,
- Create internally generated standard(s) for QA/QC purposes. Ensure that at least one standard is high-grade, to match the higher-grade mineralization noted in the veins.

This work program is estimated at $15,000.

26.3.2 Bulk Density

A bulk density measurement program should be completed to provide additional density data for use in resource estimation. Density measurements should be conducted on the channel samples collected in the proposed in-vein development program and on the core in the in-fill drilling program. A minimum of 1,000 density determinations, in waste and mineralization is recommended to provide adequate coverage across the different vein sets. If appropriate, the data should also be used to refine the tonnage estimate.

Assuming the work is mainly done by SilverCrest staff, the cost of this program is estimated at $20,000 for laboratory and administration costs.

26.3.3 Geometallurgical and Process Studies

SilverCrest should undertake a geometallurgical characterization program to identify areas of the vein systems within the current mine plan that may have high clay/mica contents that could potentially affect process recoveries. It is recommended that available core and the core to be collected in the Phase 1 drill programs be scanned using a portable shortwave infrared (SWIR) instrument to help characterize the clay and mica types, and their relative proportions in the drilled core. The results should be incorporated into a geometallurgical model to identify where the major clay and mica concentrations are in each of the vein sets.

This work program is estimated at $100,000.

26.3.4 Resource Estimation

Prior to the next resource update, the following should be completed:

- Investigate the use of density-length-weighted composites for grade interpolation;
- Use results of reconciliation, geometallurgical study and underground development to optimize the geological model and grade estimation procedure; and,
- Complete an updated Mineral Resource Estimate to incorporate results of new surface and underground exploration.

This work program is estimated at $100,000.

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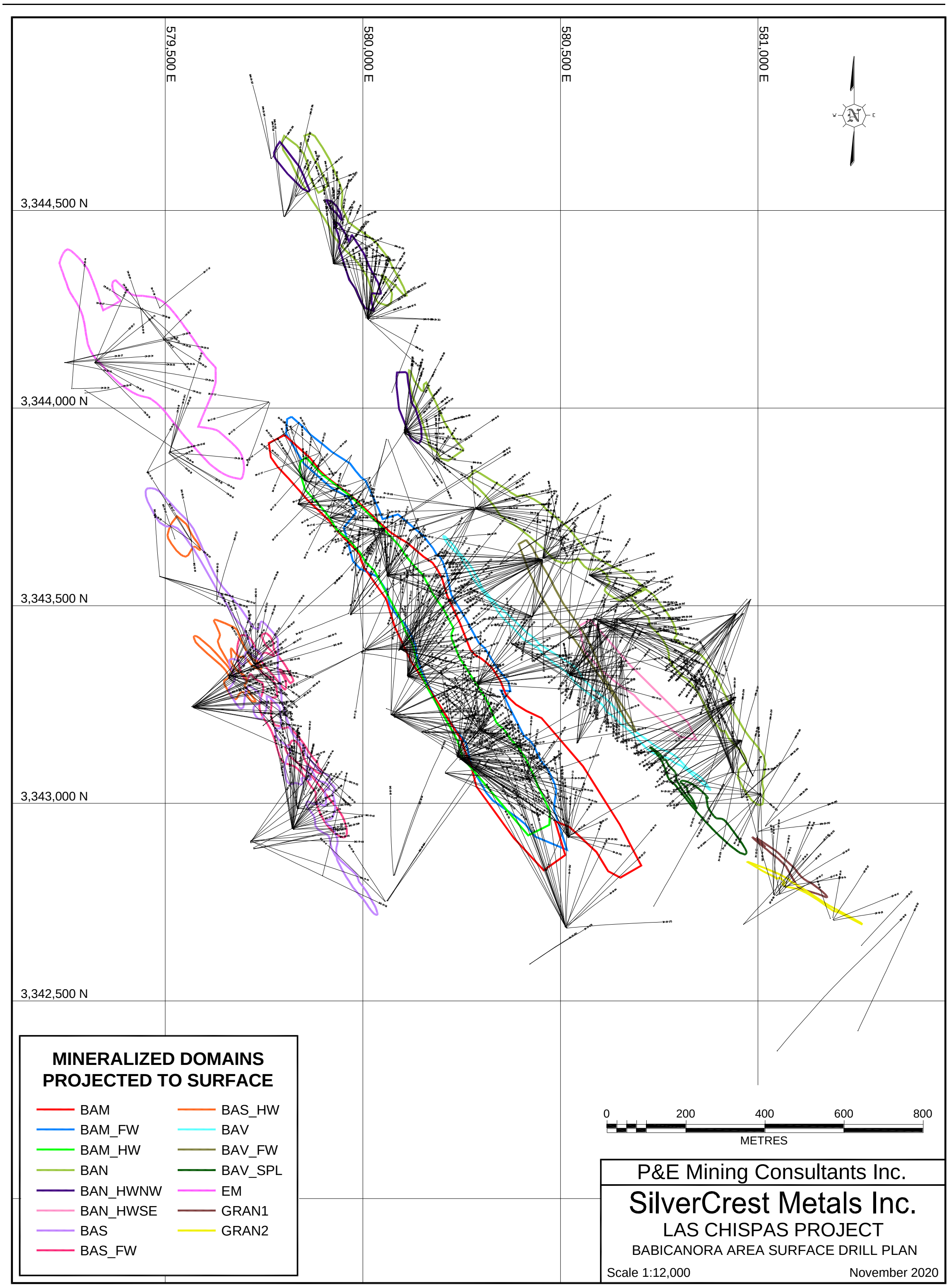
APPENDIX A SURFACE DRILL HOLE PLAN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MINERALIZED DOMAINS PROJECTED TO SURFACE
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BAN
BAN_HWNW
BAN_HWSE
BAS
BAS_FW
BAS_HW
BAV
BAV_FW
BAV_SPL
EM
GRAN1
GRAN2
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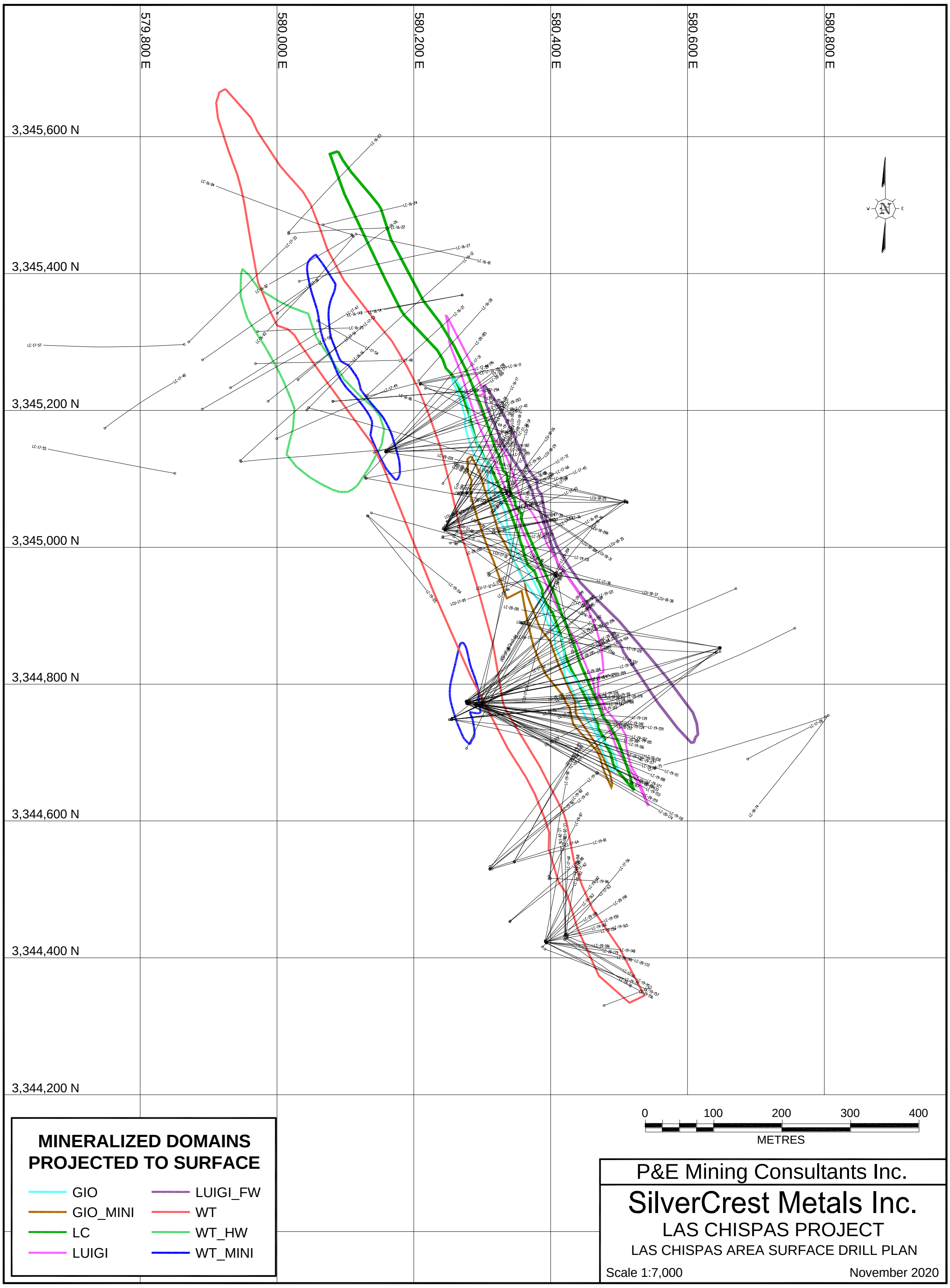
P&E Mining Consultants Inc.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
BABICANORA AREA SURFACE DRILL PLAN
Scale 1:12,000
November 2020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METRES
MINERALIZED DOMAINS PROJECTED TO SURFACE
GIO
GIO_MINI
LC
LUIGI
LUIGI_FW
WT
WT_HW
WT_MINI
P&E Mining Consultants Inc.
SilverCrest Metals Inc.
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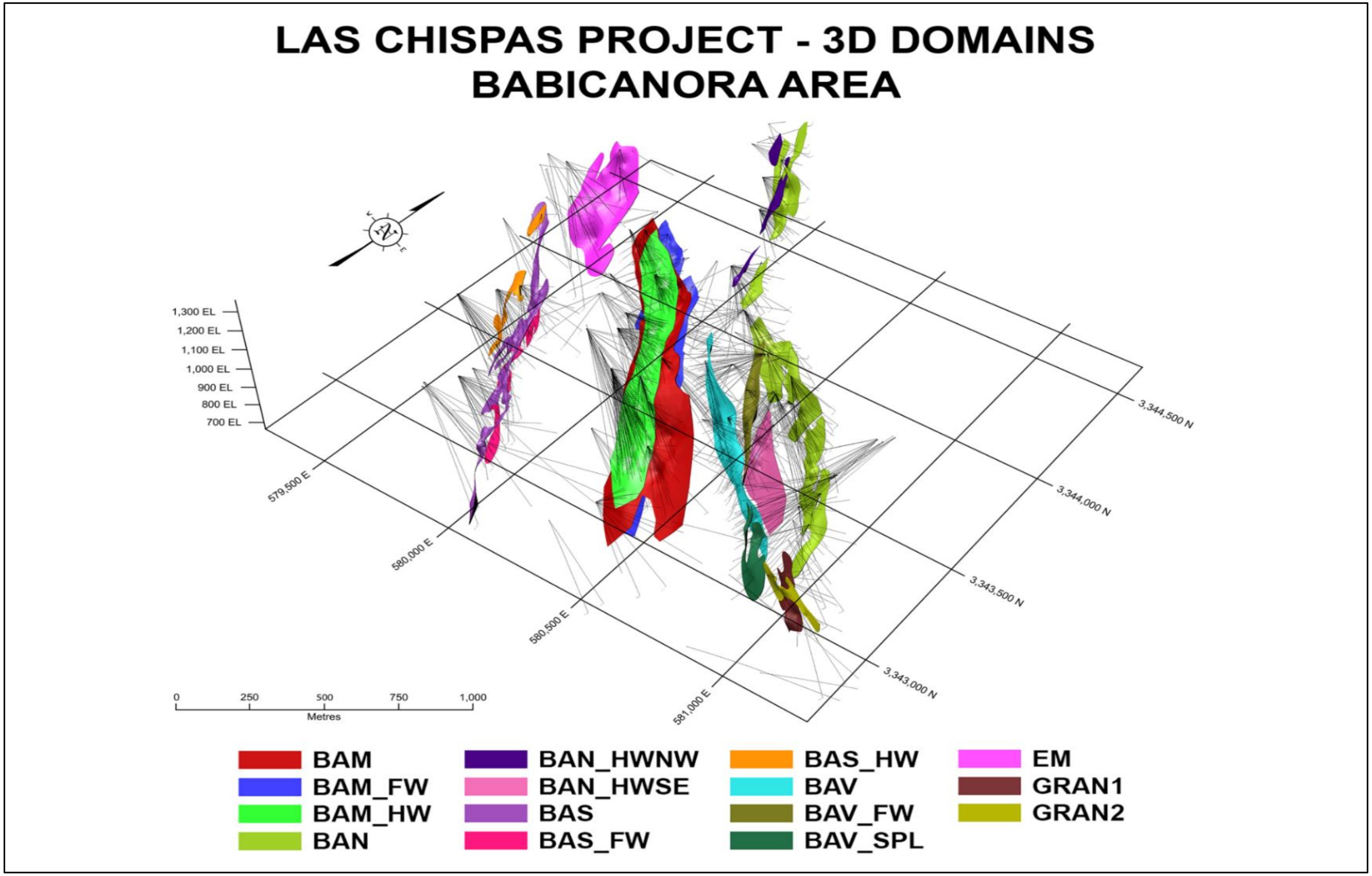
November 2020

APPENDIX B 3-D DOMAINS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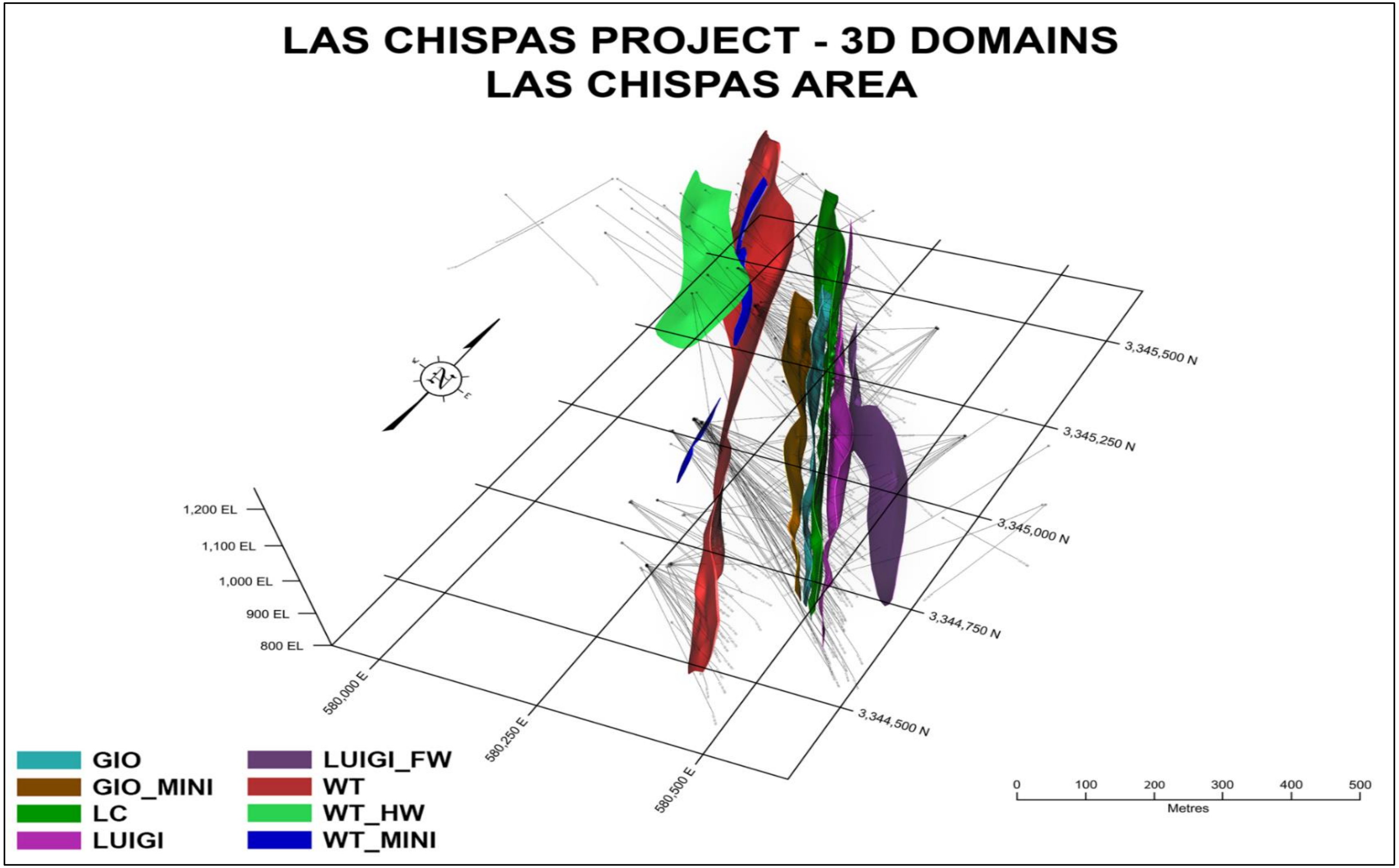
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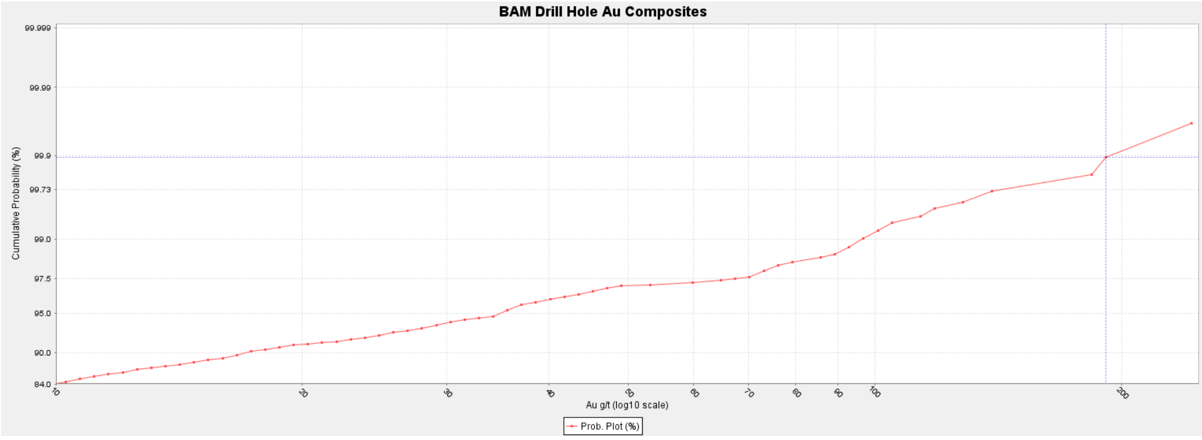
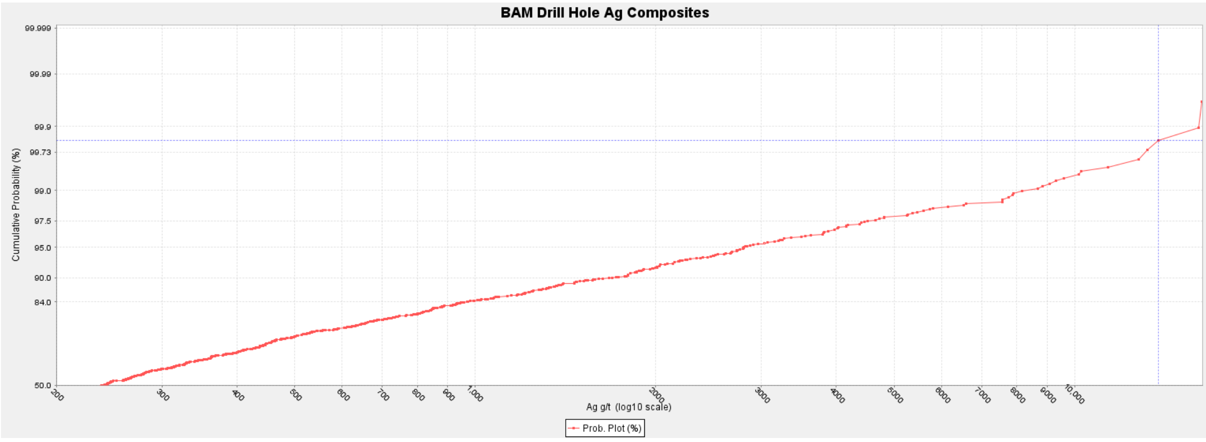
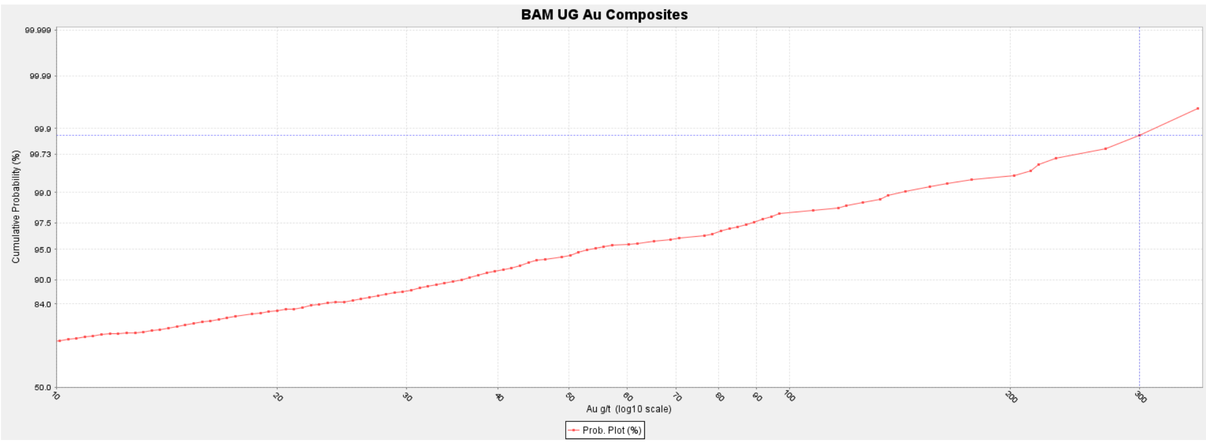
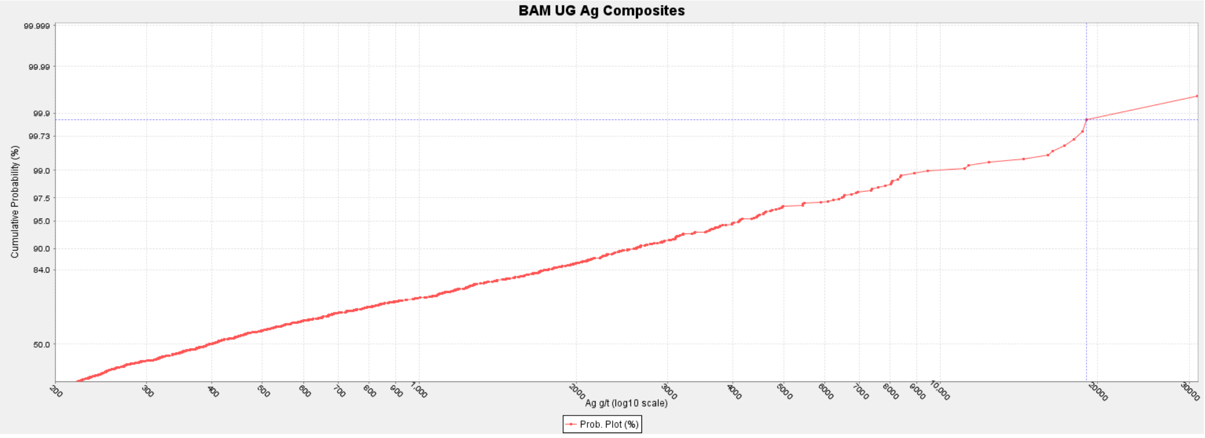
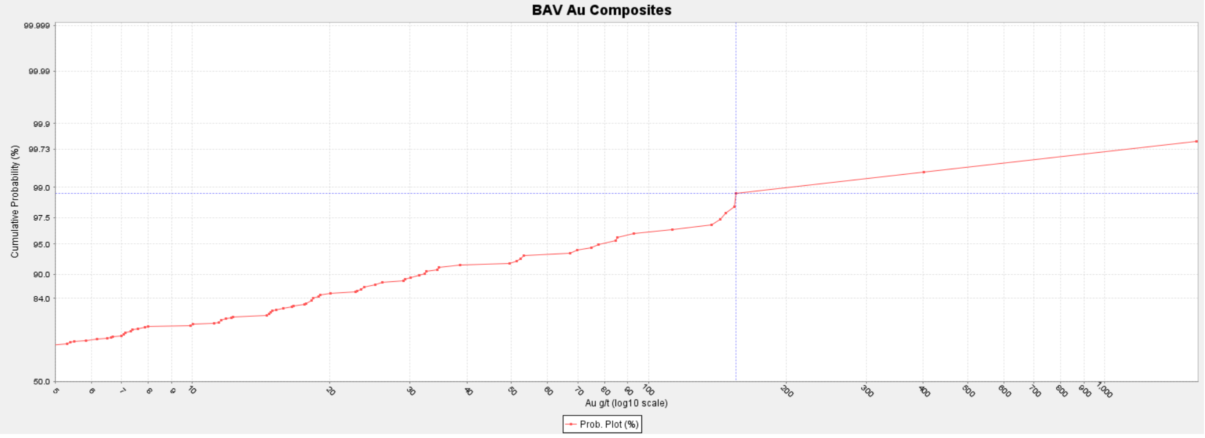
APPENDIX C LOG PROBABILITY PLOTS

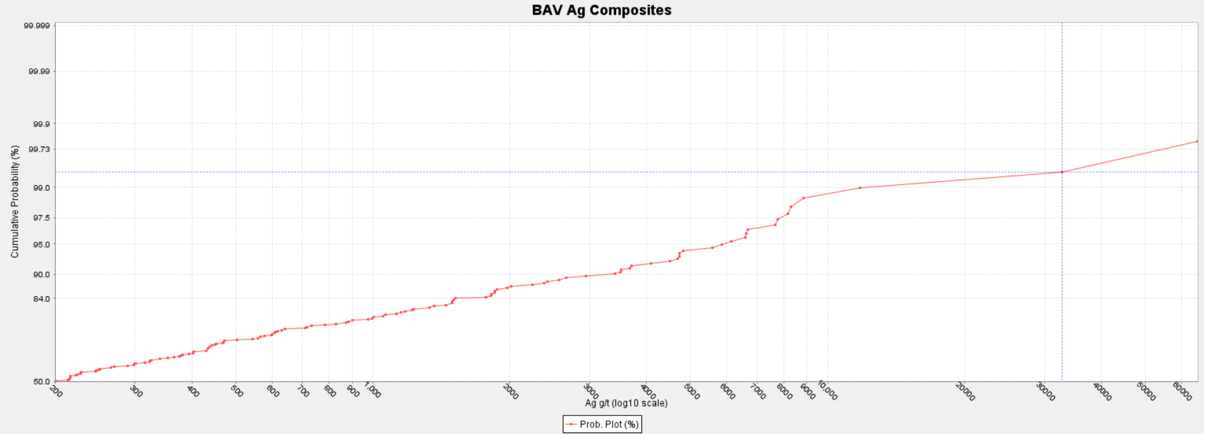


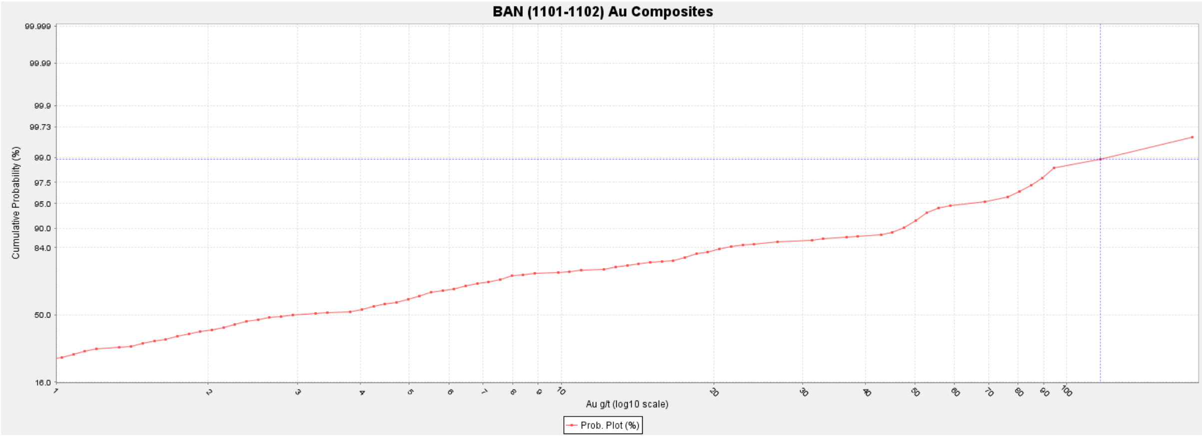


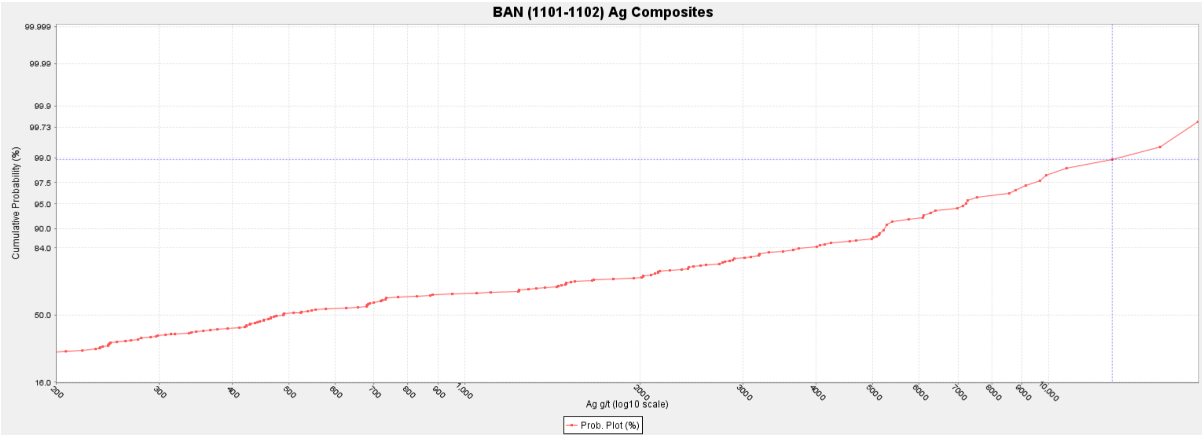


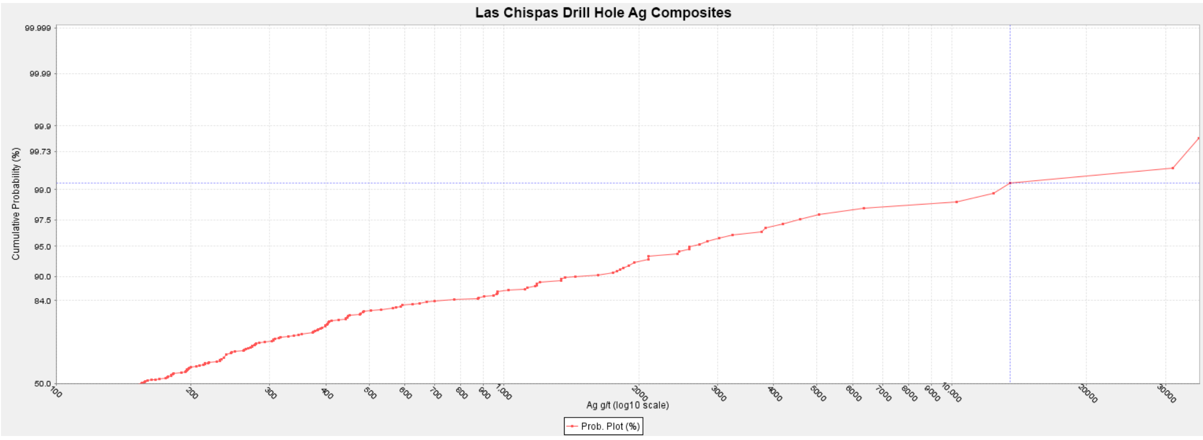

BAM UG Ag Composites
Cumulative Probability (%)
Ag g/t (log10 scale)
Prob. Plot (%)

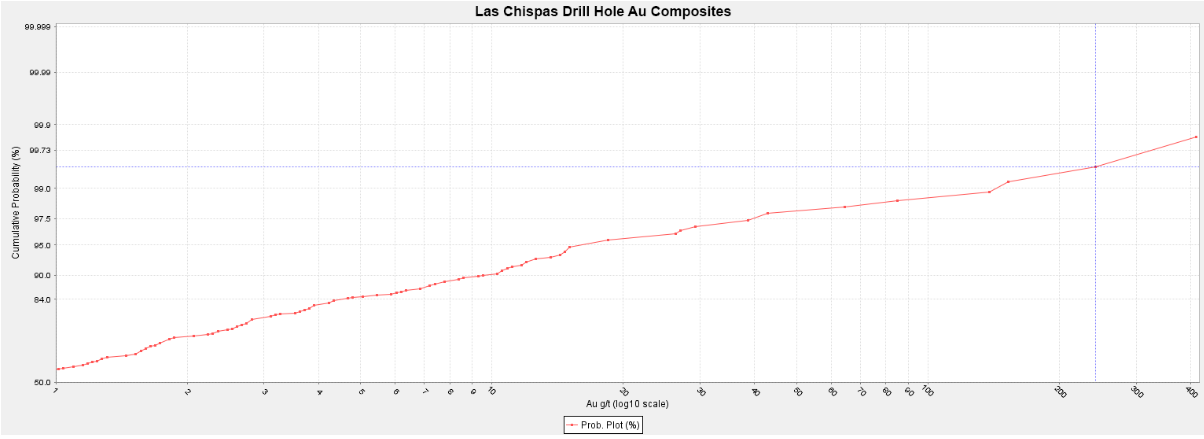

BAV Au Composites
Cumulative Probability (%)
Au g/t (log10 scale)
Prob. Plot (%)

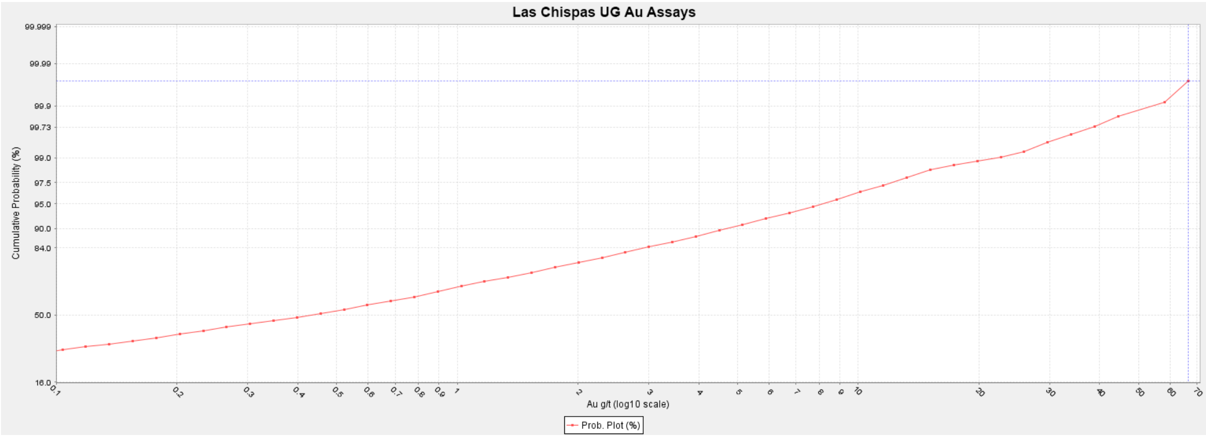

BAV Ag Composites
Cumulative Probability (%)
Ag g/t (log10 scale)
Prob. Plot (%)

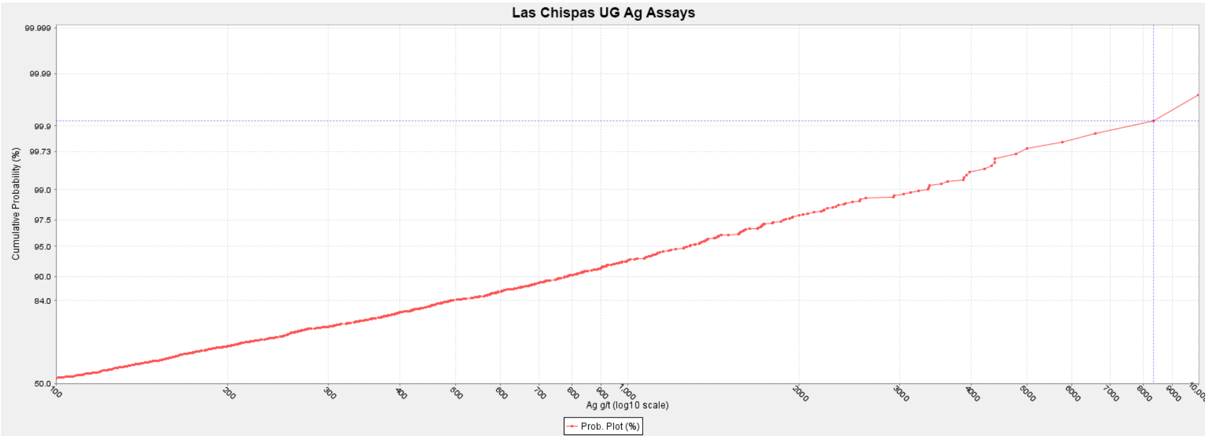

BAN (1101-1102) Au Composites
Cumulative Probability (%)
Au g/t (log10 scale)
Prob. Plot (%)

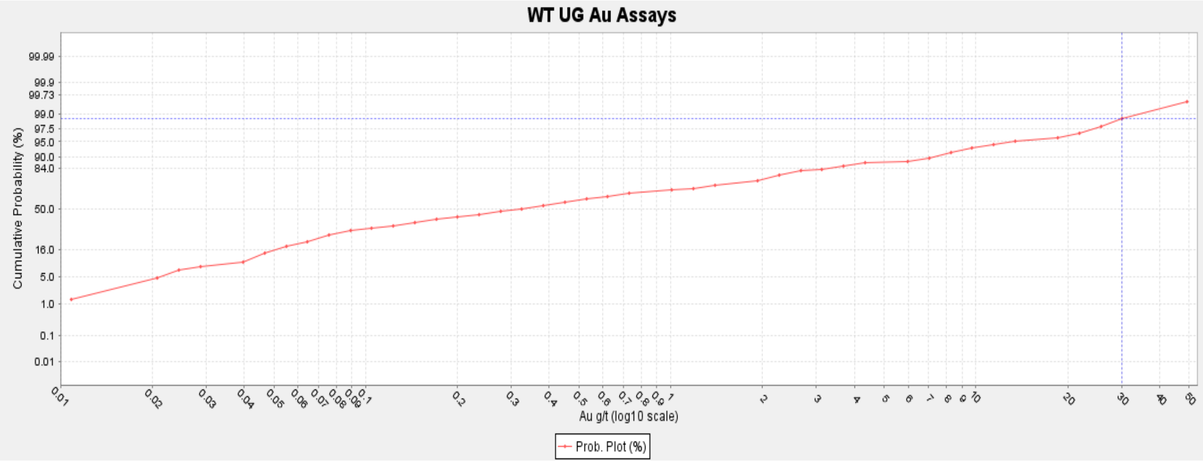

BAN (1101-1102) Ag Composites
Cumulative Probability (%)
Ag g/t (log10 scale)
Prob. Plot (%)

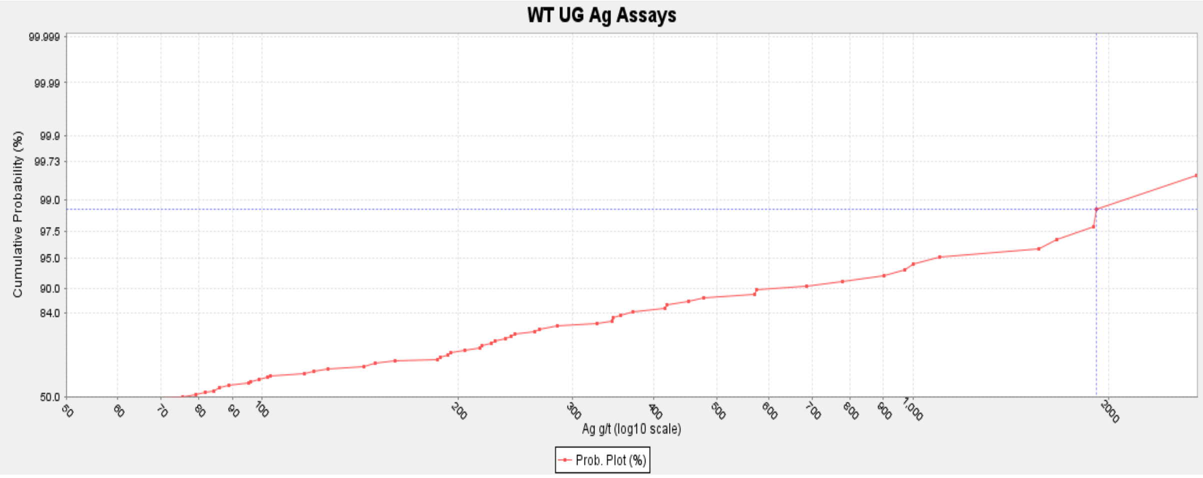

Las Chispas Drill Hole Ag Composites
Cumulative Probability (%)
Ag g/t (log10 scale)
Prob. Plot (%)

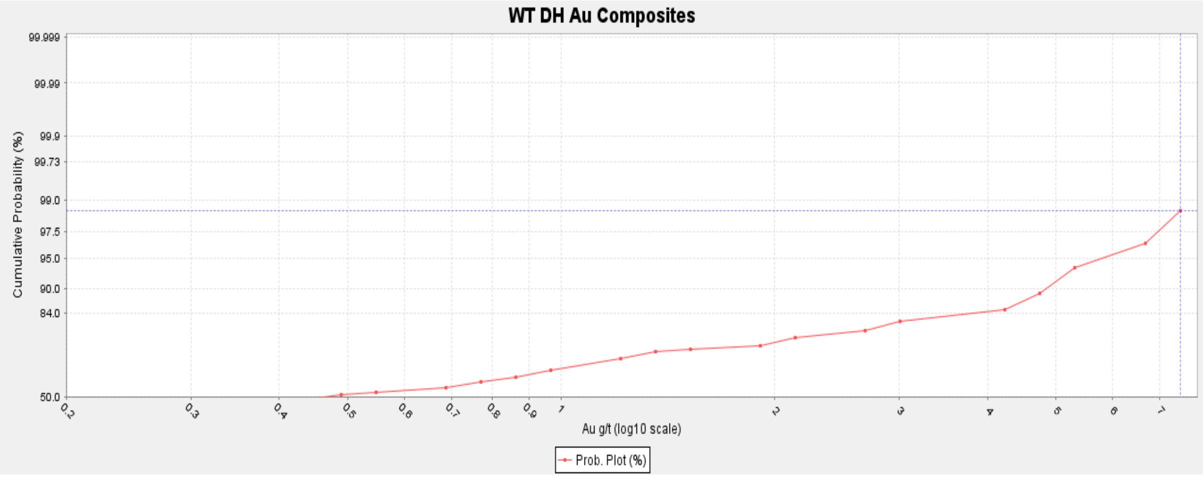

Las Chispas Drill Hole Au Composites
Cumulative Probability (%)
Au g/t (log10 scale)
Prob. Plot (%)

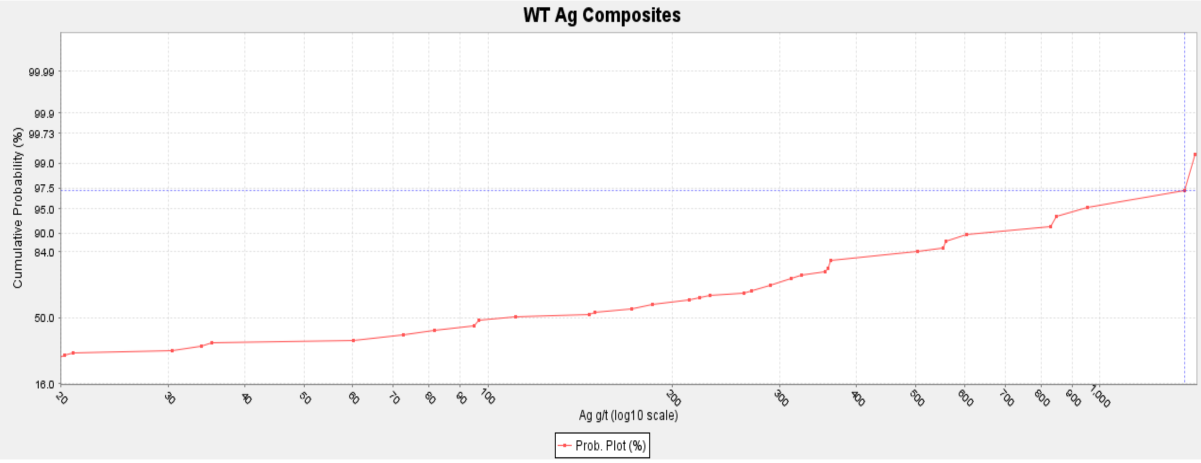

Las Chispas UG Au Assays
Cumulative Probability (%)
Au g/t (log10 scale)
Prob. Plot (%)


Las Chispas UG Ag Assays
Cumulative Probability (%)
Ag g/t (log10 scale)
Prob. Plot (%)

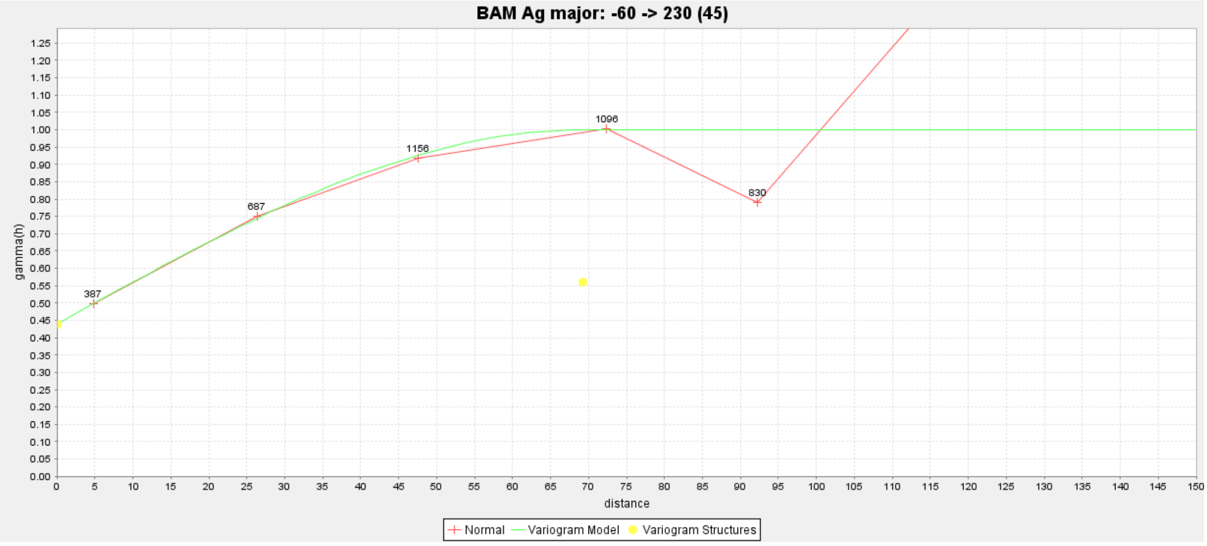

WT UG Au Assays
Cumulative Probability (%)
Au g/t (log10 scale)
Prob. Plot (%)

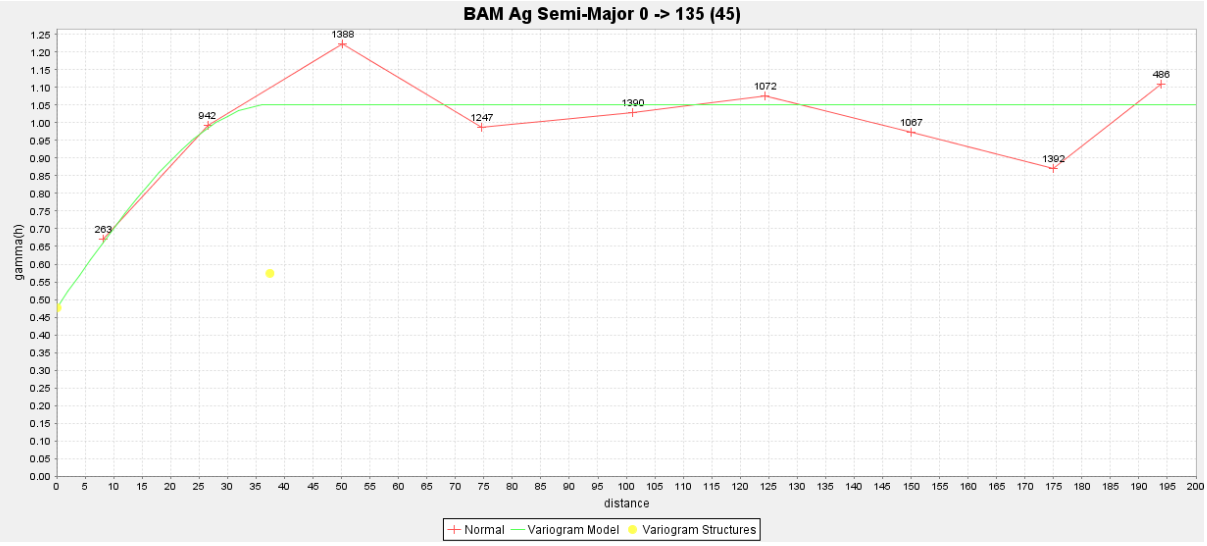

WT UG Ag Assays
Cumulative Probability (%)
Ag g/t (log10 scale)
Prob. Plot (%)

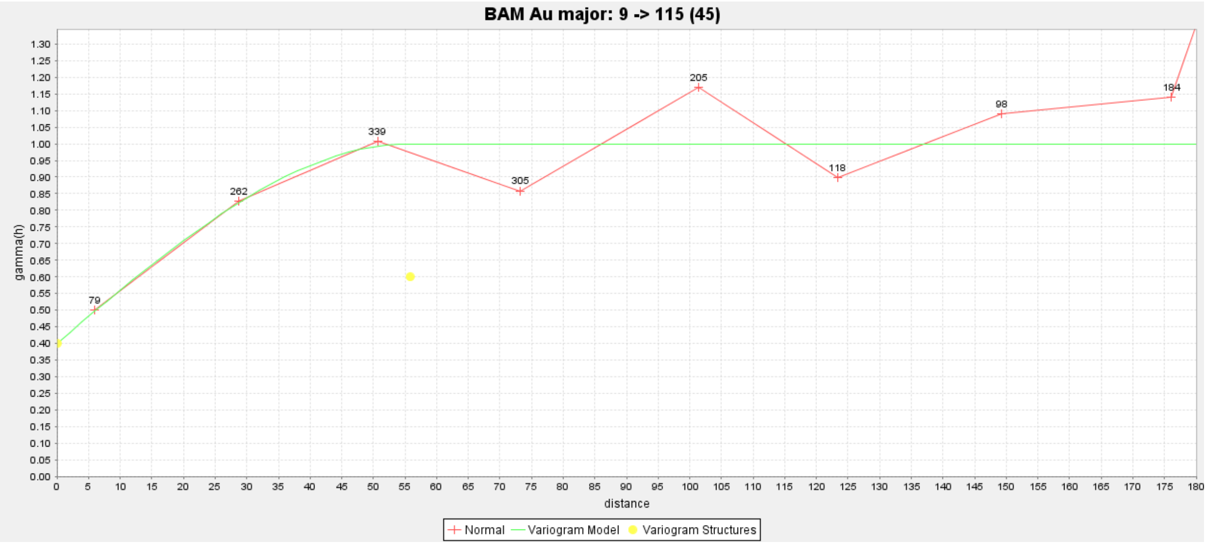

WT DH Au Composites
Cumulative Probability (%)
Au g/t (log10 scale)
Prob. Plot (%)

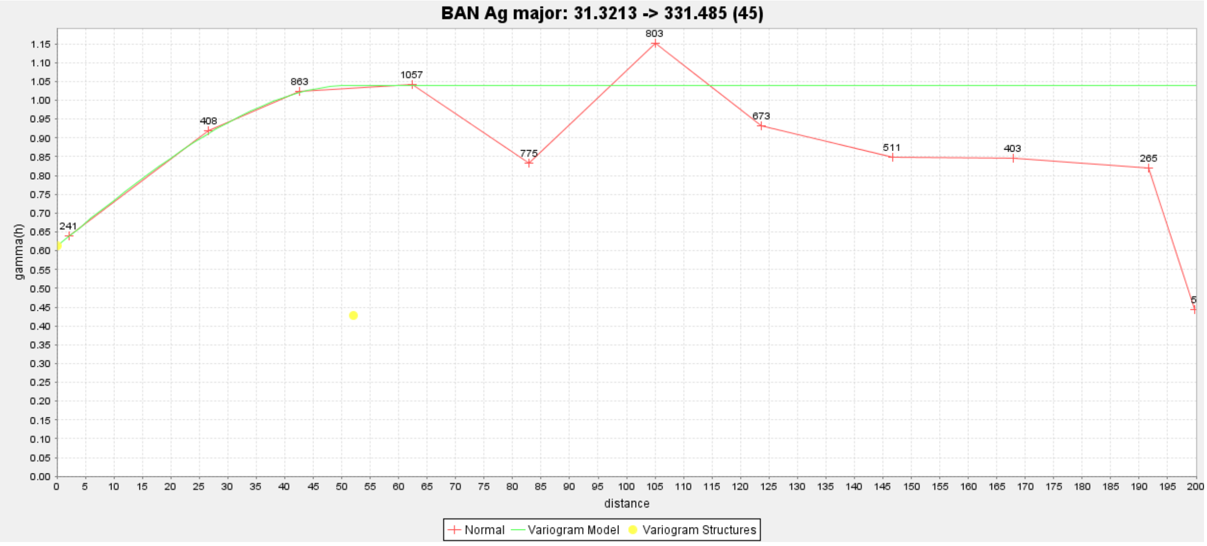

WT Ag Composites
Cumulative Probability (%)
99.99
99.9
99.73
99.0
97.5
95.0
90.0
84.0
50.0
16.0
20
30
40
50
60
70
80
90
100
200
300
400
500
600
700
800
900
1000
Ag g/t (log10 scale)
Prob. Plot (%)

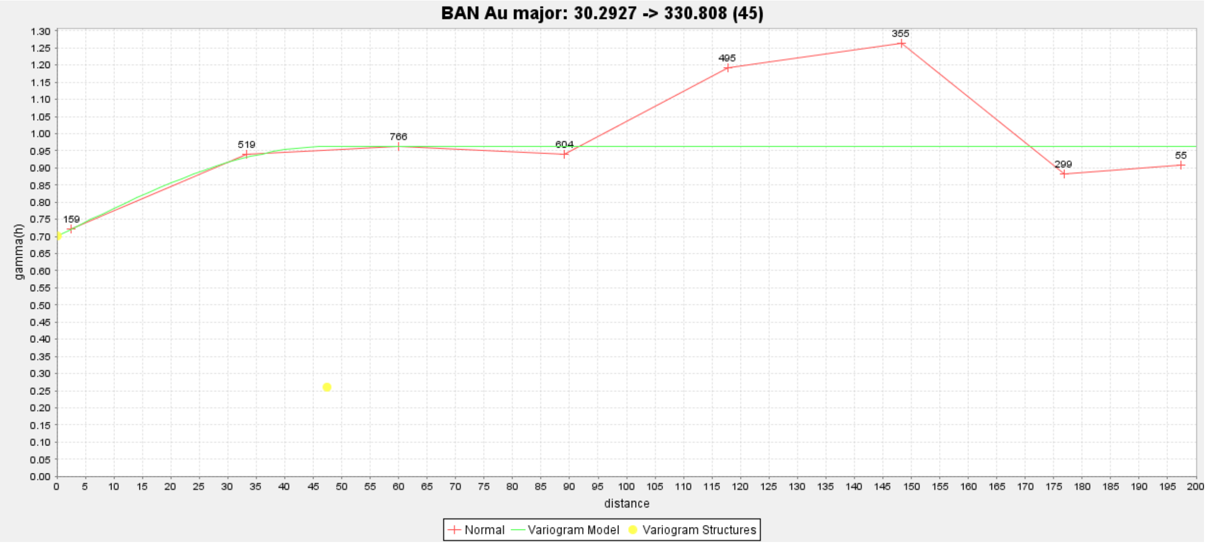
APPENDIX D VARIOGRAMS

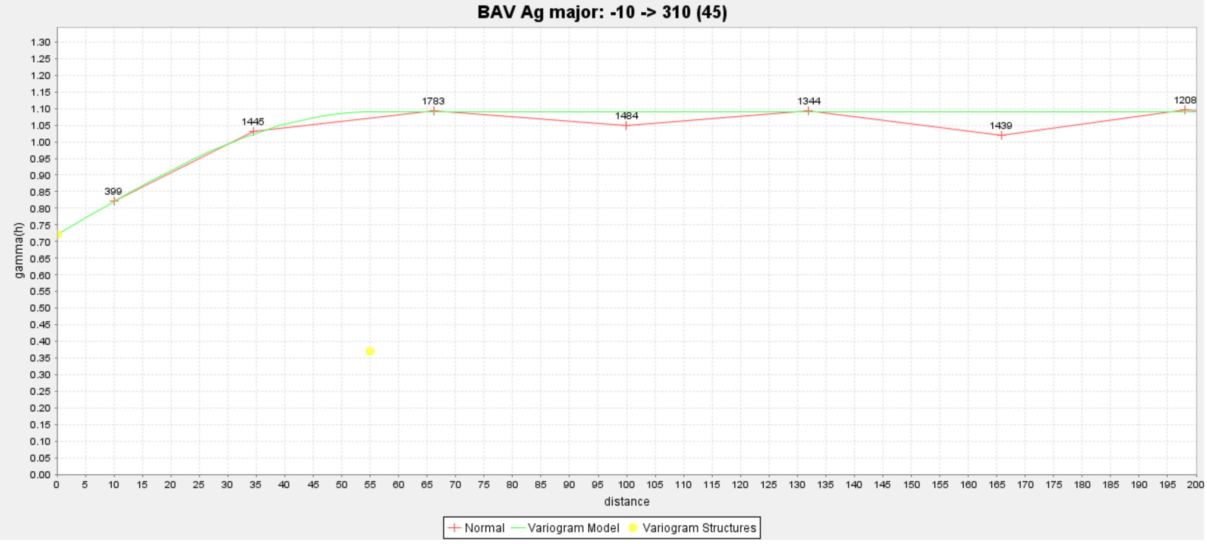

BAM Ag major: -60 -> 230 (45)
gamma(h)
distance
387
687
1156
1096
830
Normal
Variogram Model
Variogram Structures

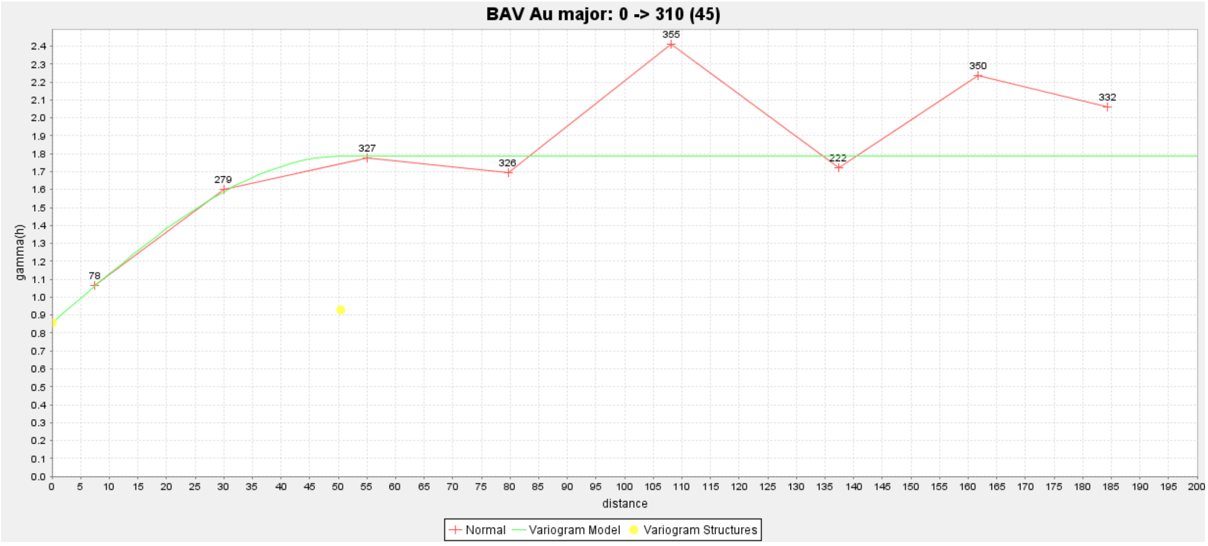

BAM Ag Semi-Major 0 -> 135 (45)
gamma(h)
distance
263
942
1388
1247
1390
1072
1067
1392
486
Normal
Variogram Model
Variogram Structures

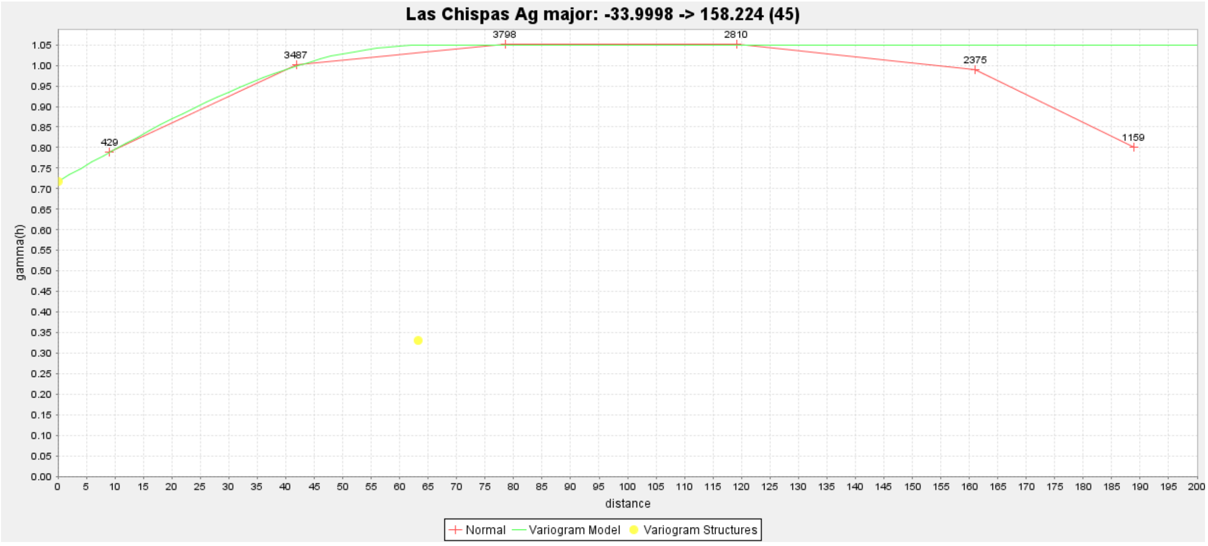

BAM Au major: 9 -> 115 (45)
gamma(h)
distance
79
262
339
305
205
118
98
184
Normal
Variogram Model
Variogram Structures

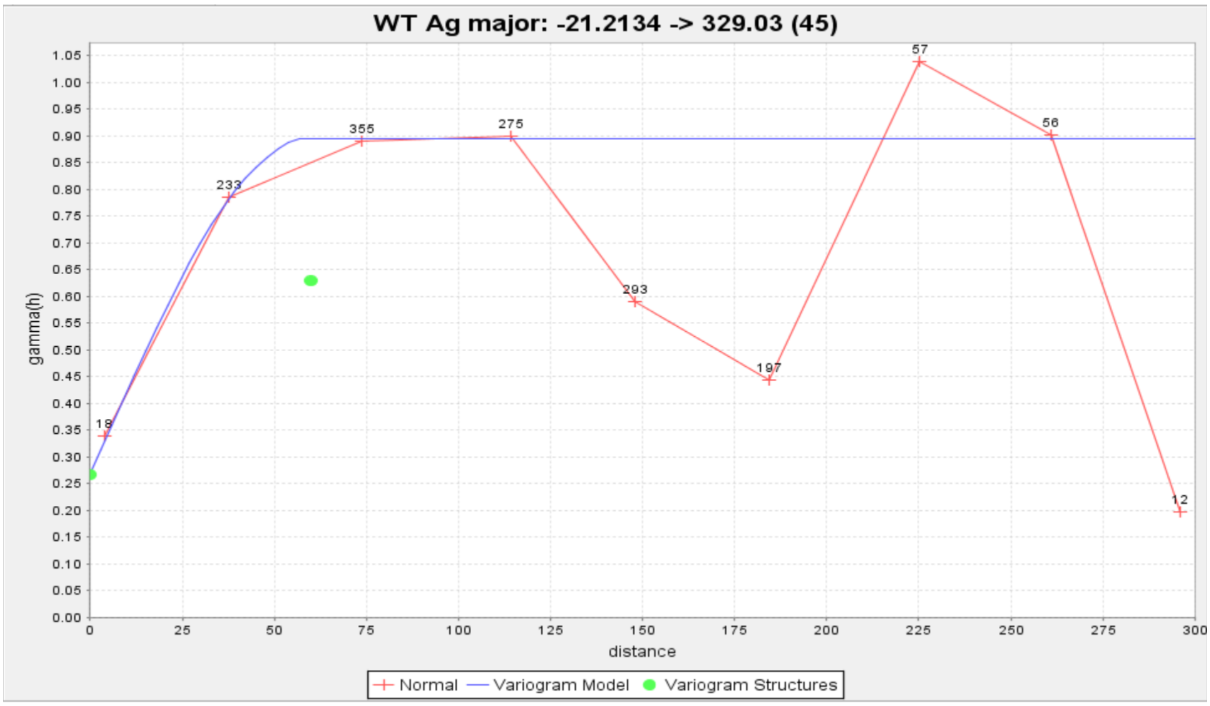

BAN Ag major: 31.3213 -> 331.485 (45)
gamma(h)
distance
241
408
863
1057
775
803
673
511
403
265
Normal
Variogram Model
Variogram Structures

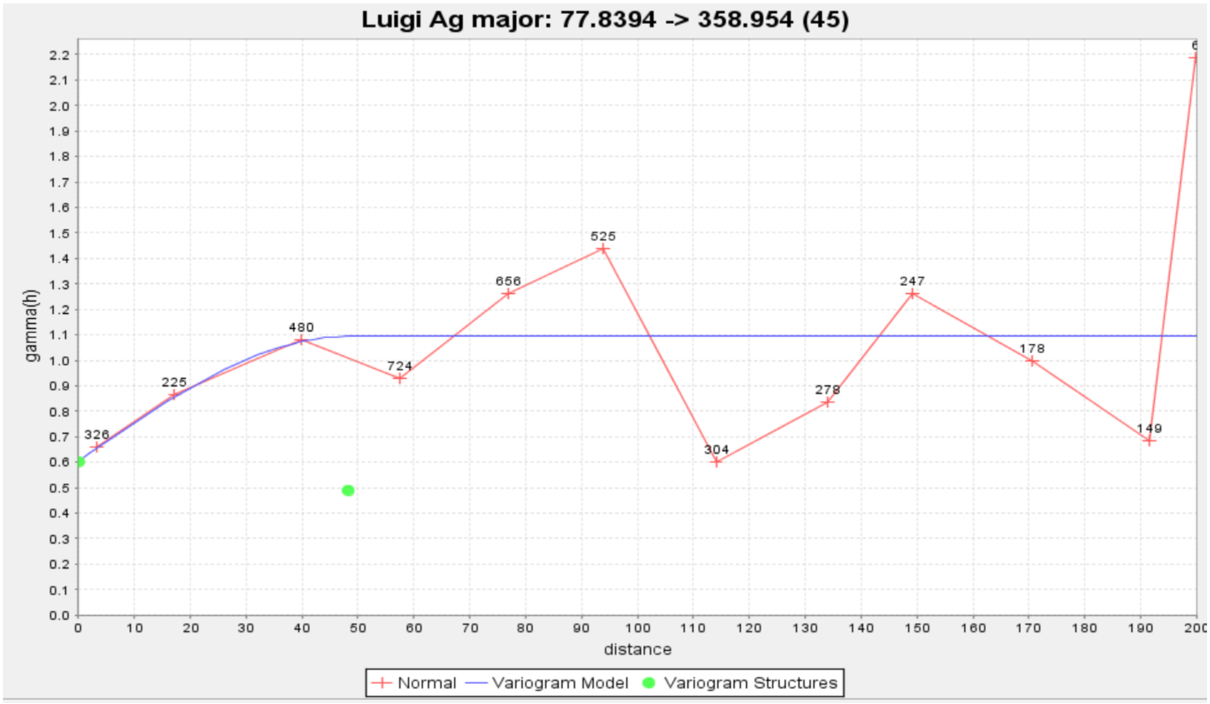

BAN Au major: 30.2927 -> 330.808 (45)
gamma(h)
distance
159
519
766
604
495
355
299
55
Normal
Variogram Model
Variogram Structures


BAV Ag major: -10 -> 310 (45)
gamma(h)
distance
399
1445
1783
1484
1344
1439
1208
Normal
Variogram Model
Variogram Structures

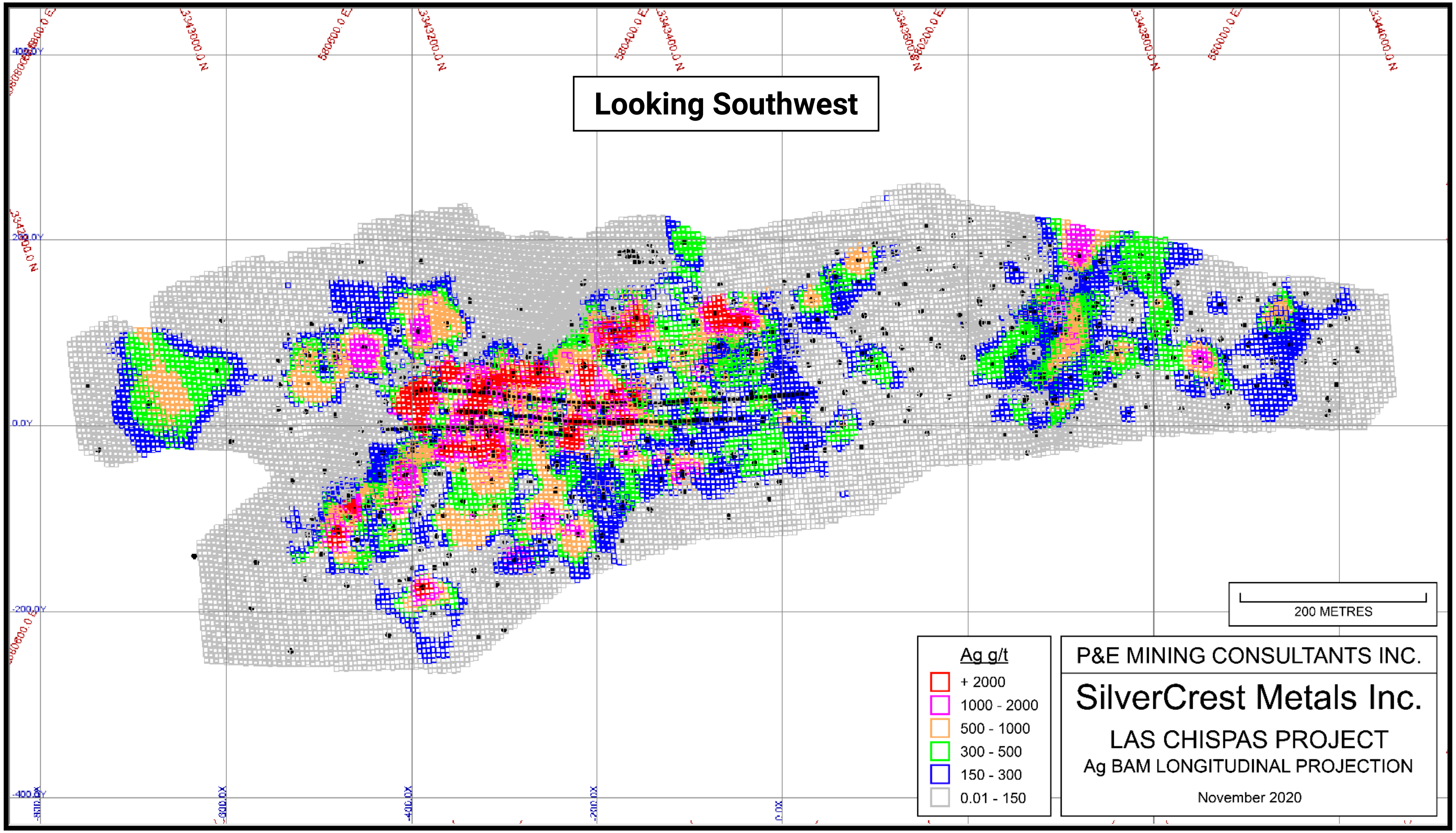

BAV Au major: 0 -> 310 (45)
gamma(h)
distance
355
350
332
327
326
222
279
78
Normal
Variogram Model
Variogram Structures


Las Chispas Ag major: -33.9998 -> 158.224 (45)
gamma(h)
distance
3798
2810
3487
2375
1159
429
Normal
Variogram Model
Variogram Structures

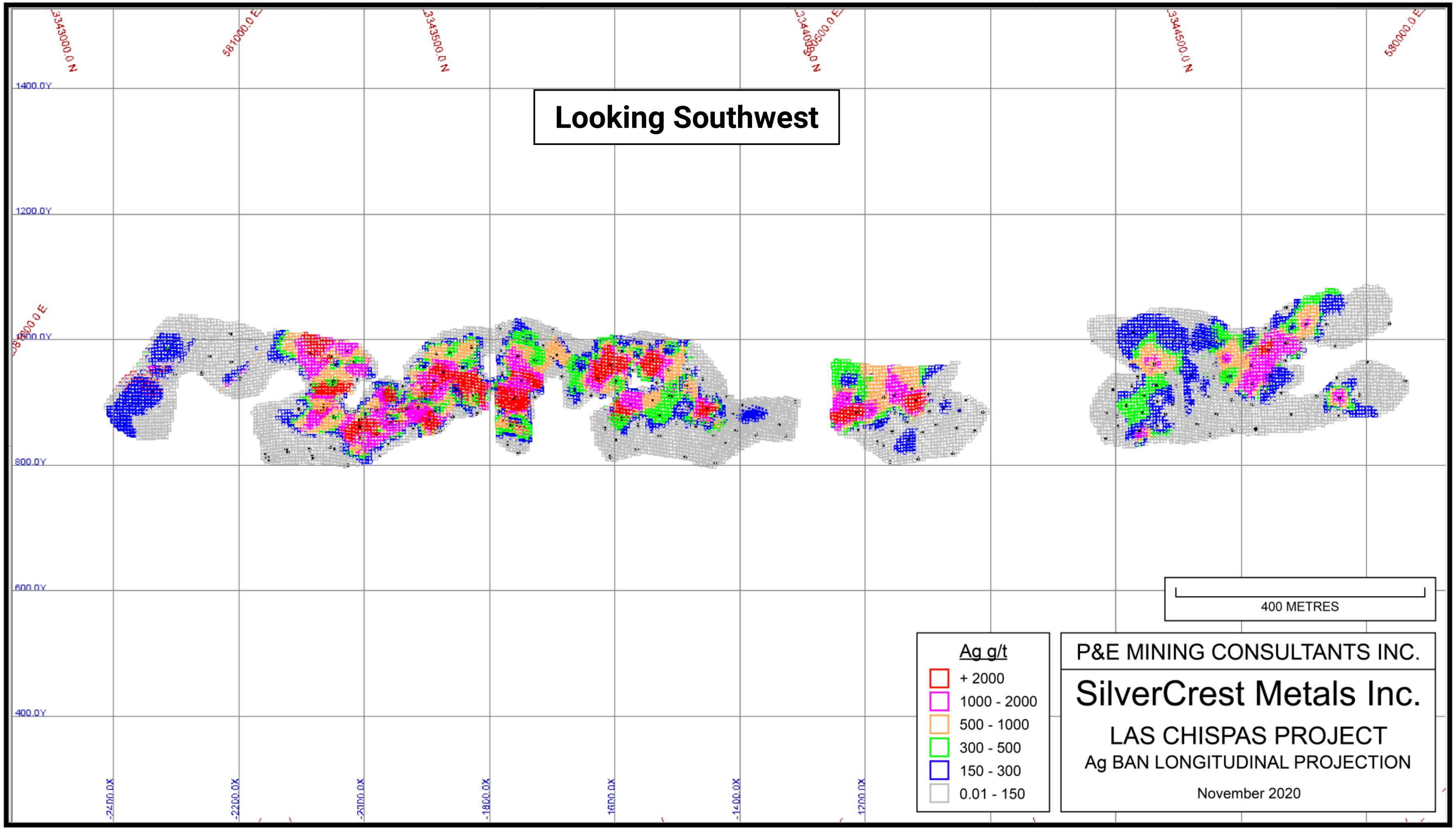

WT Ag major: -21.2134 -> 329.03 (45)
gamma(h)
distance
Normal
Variogram Model
Variogram Structures

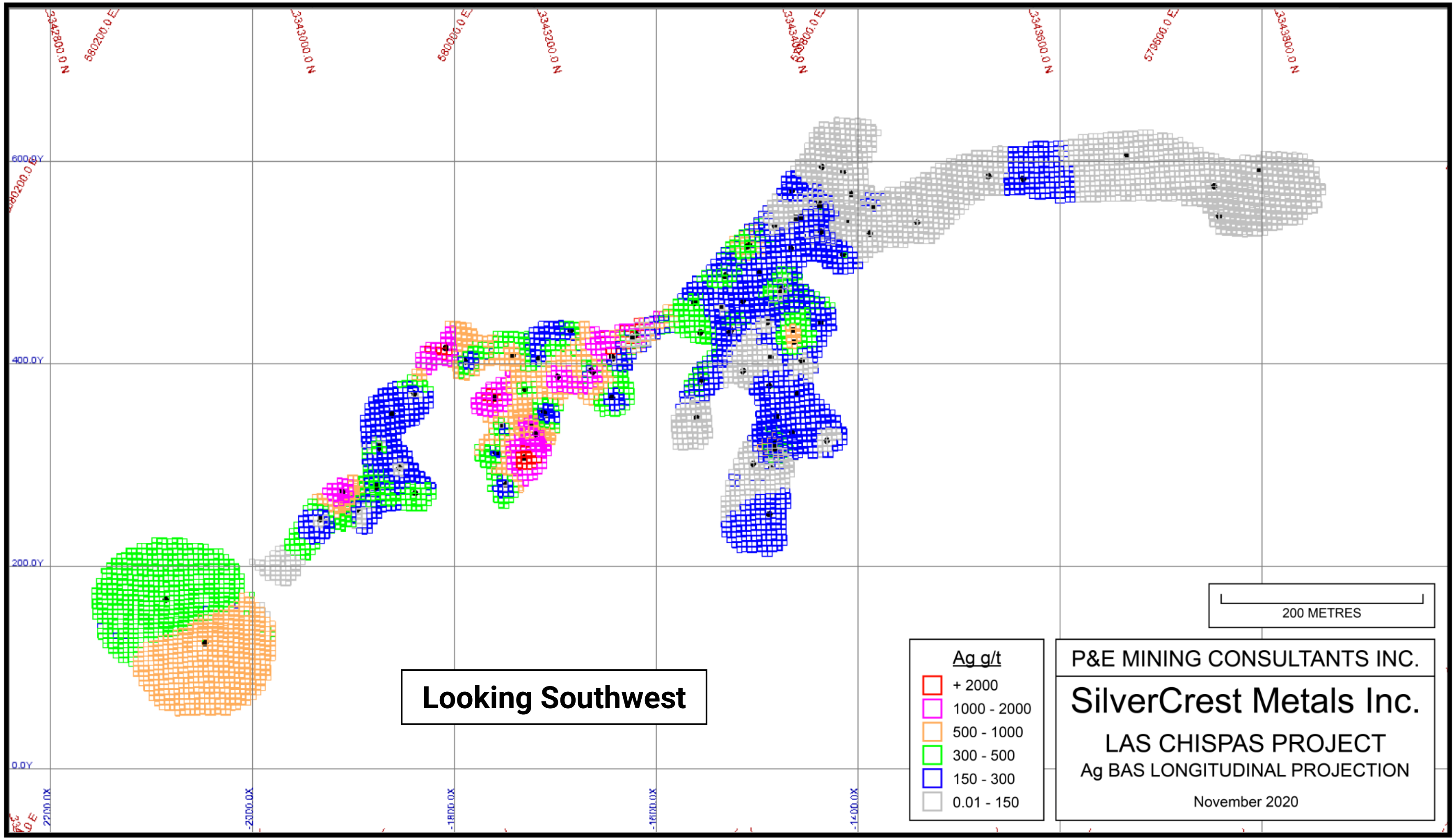

Luigi Ag major: 77.8394 -> 358.954 (45)
gamma(h)
distance
Normal
Variogram Model
Variogram Structures

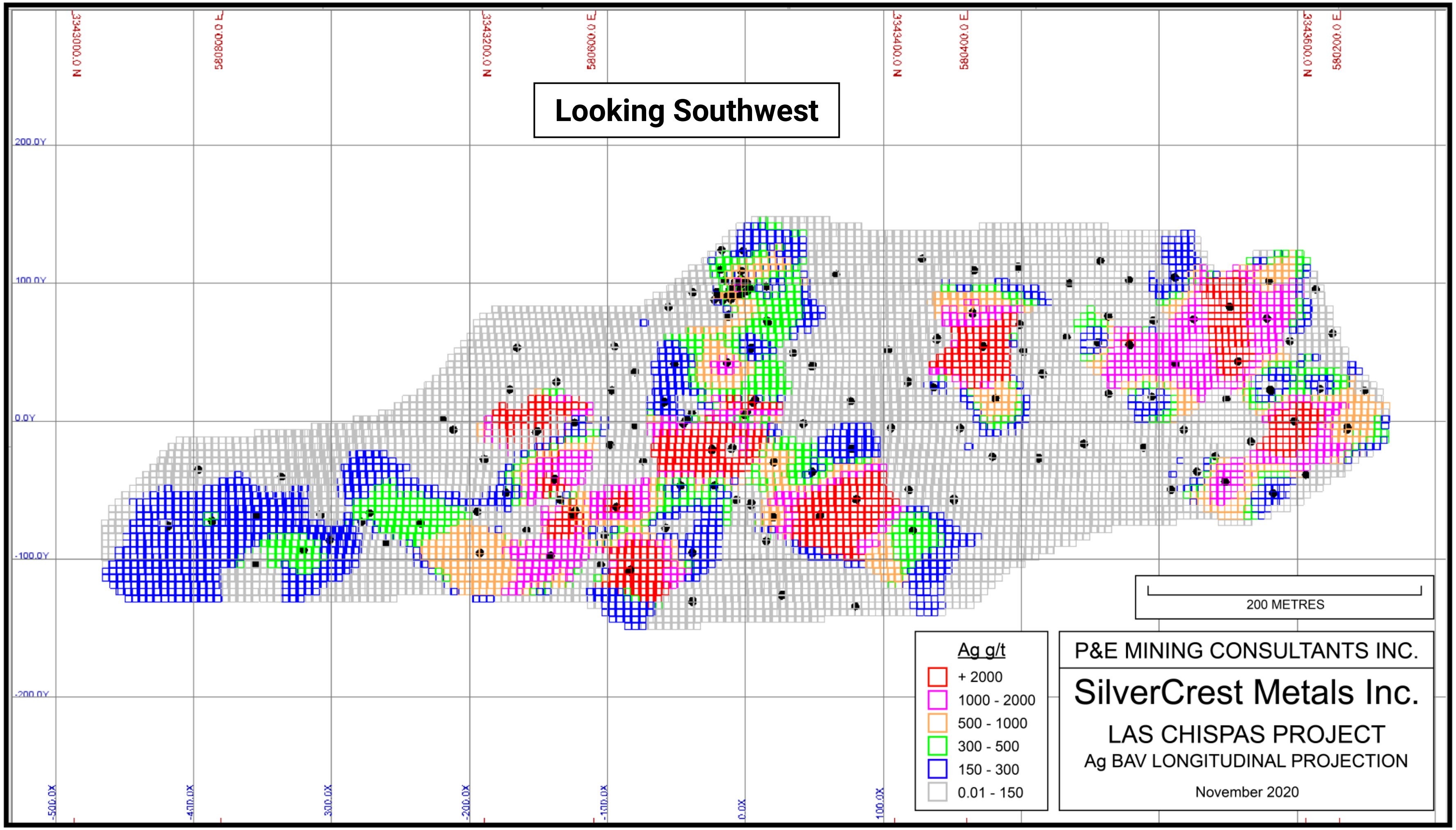
APPENDIX E AG LONGITUDINAL PROJECTIONS

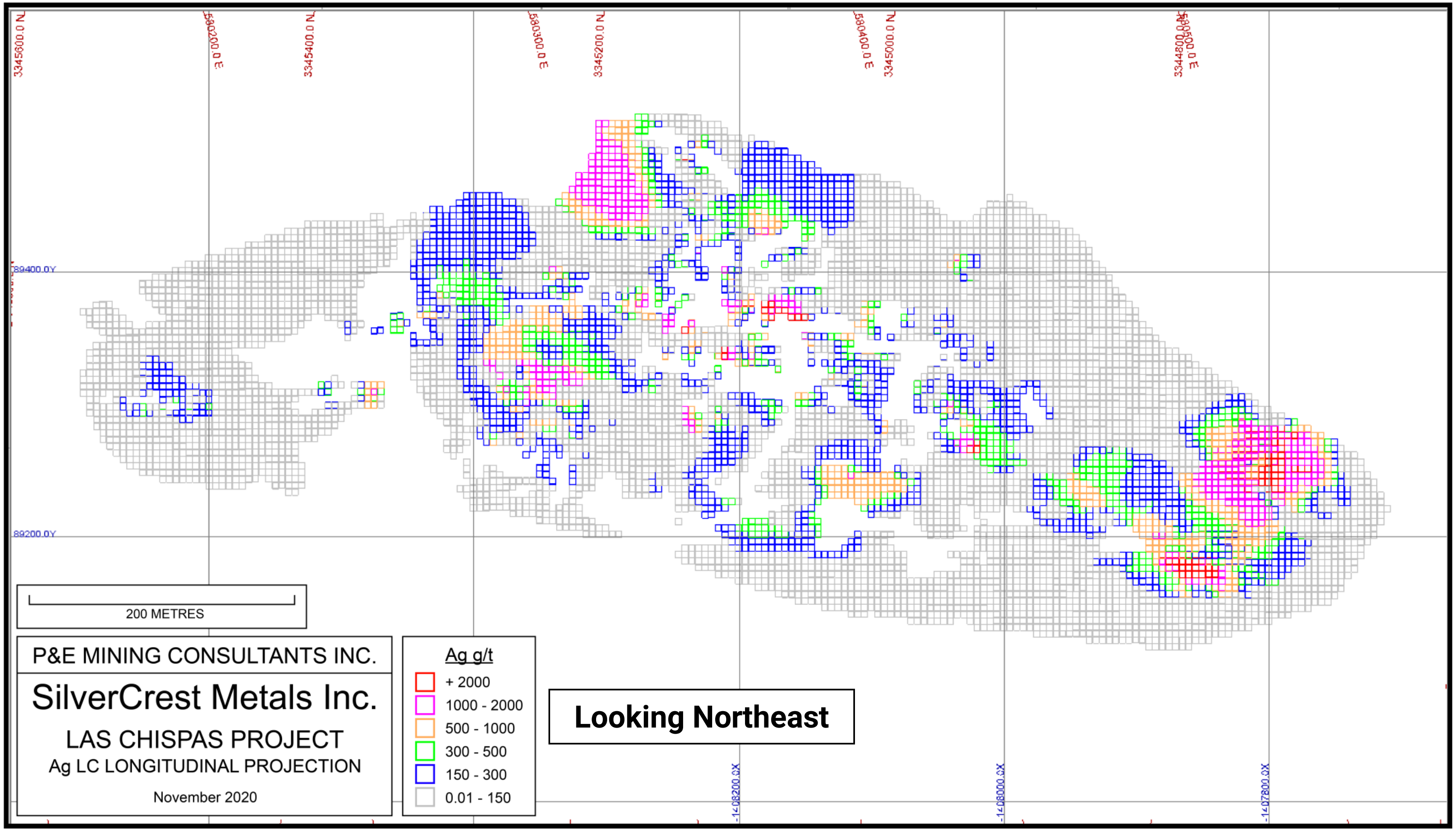


Looking Southwest
200 METRES
Ag g/t
+ 2000
1000 - 2000
500 - 1000
300 - 500
150 - 300
0.01 - 150
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
Ag BAM LONGITUDINAL PROJECTION
November 2020

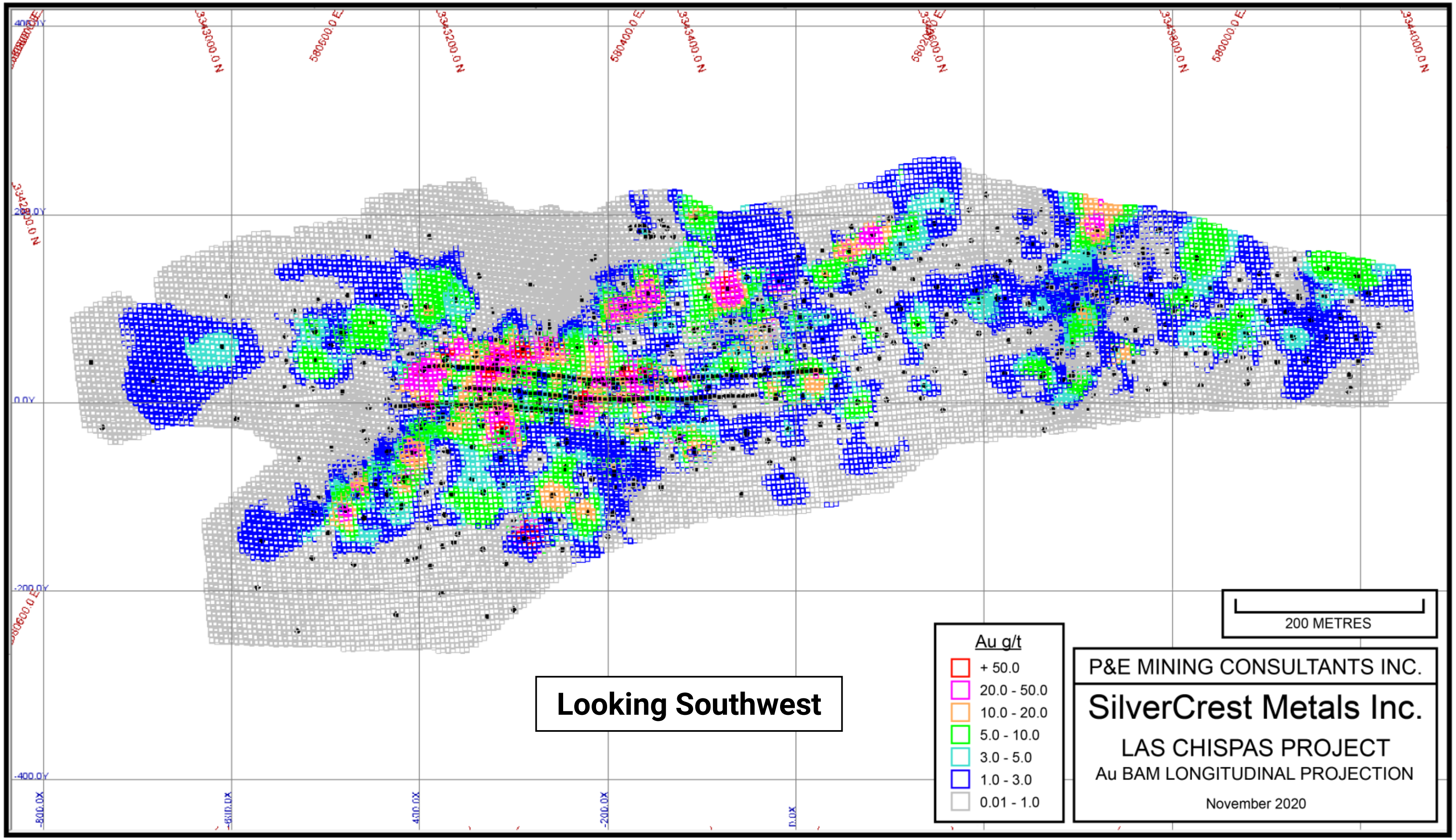

Looking Southwest
1400.0Y
1200.0Y
800.0Y
600.0Y
400.0Y
400 METRES
Ag g/t
+ 2000
1000 - 2000
500 - 1000
300 - 500
150 - 300
0.01 - 150
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
Ag BAN LONGITUDINAL PROJECTION
November 2020

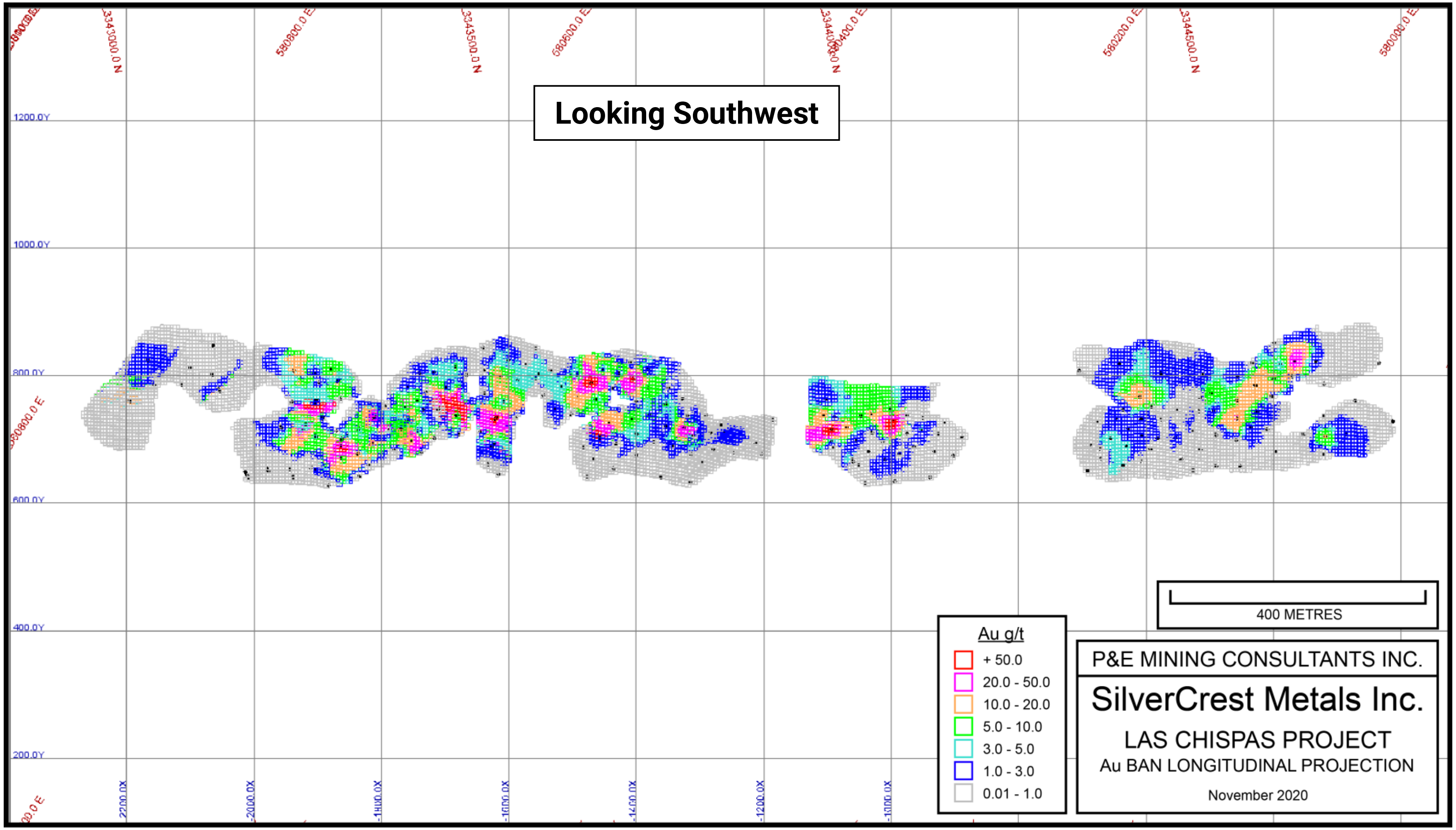

Looking Southwest
Ag g/t
+ 2000
1000 - 2000
500 - 1000
300 - 500
150 - 300
0.01 - 150
200 METRES
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
Ag BAS LONGITUDINAL PROJECTION
November 2020
600.0Y
400.0Y
200.0Y
0.0Y

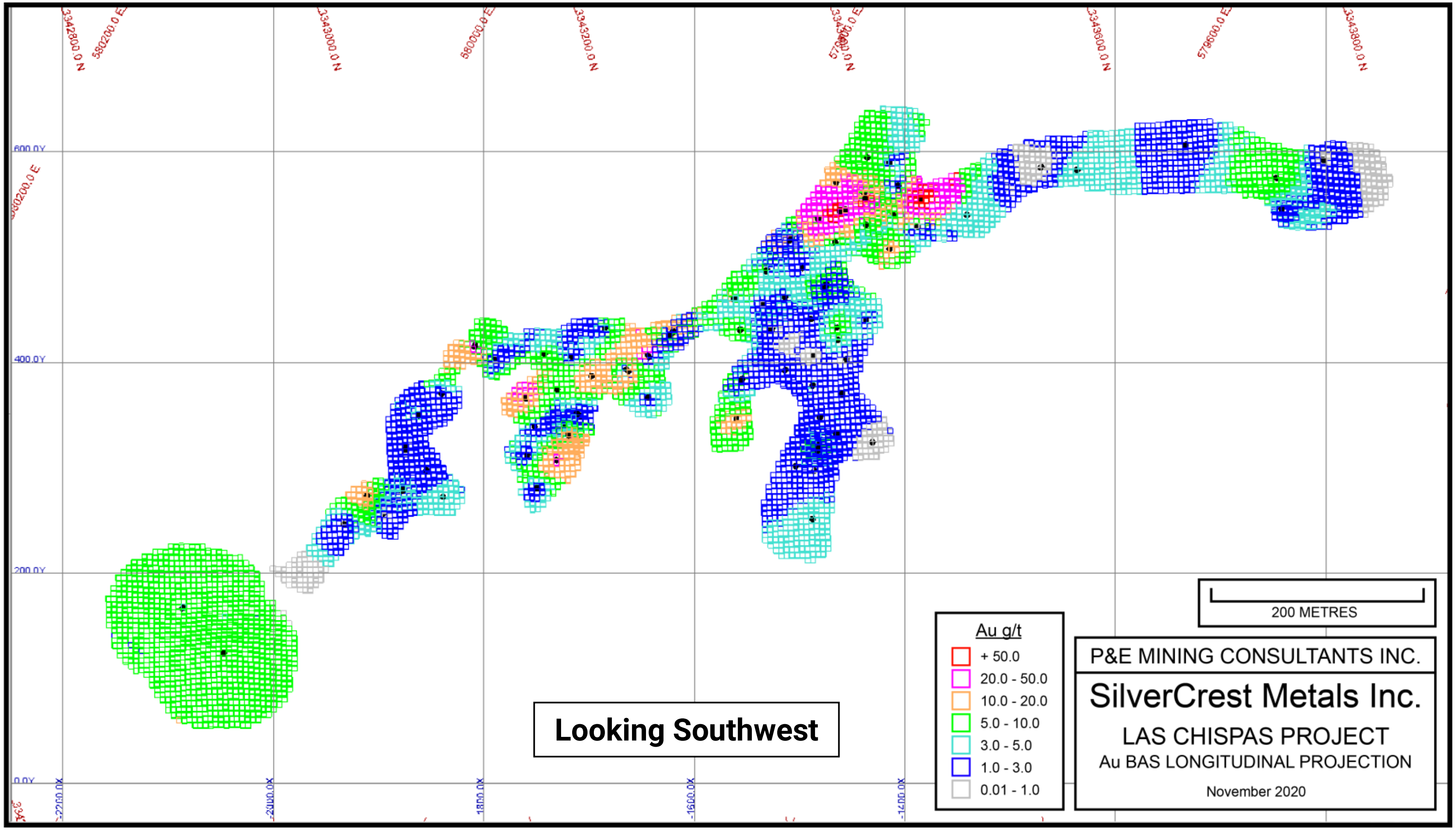

Looking Southwest
3343000.0 N
580800.0 E
3343200.0 N
581000.0 E
3343400.0 N
580400.0 E
3343600.0 N
580200.0 E
200.0Y
100.0Y
0.0Y
-100.0Y
-200.0Y
-500.0X
-400.0X
-300.0X
-200.0X
-100.0X
0.0X
100.0X
200 METRES
Ag g/t
+ 2000
1000 - 2000
500 - 1000
300 - 500
150 - 300
0.01 - 150
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
Ag BAV LONGITUDINAL PROJECTION
November 2020

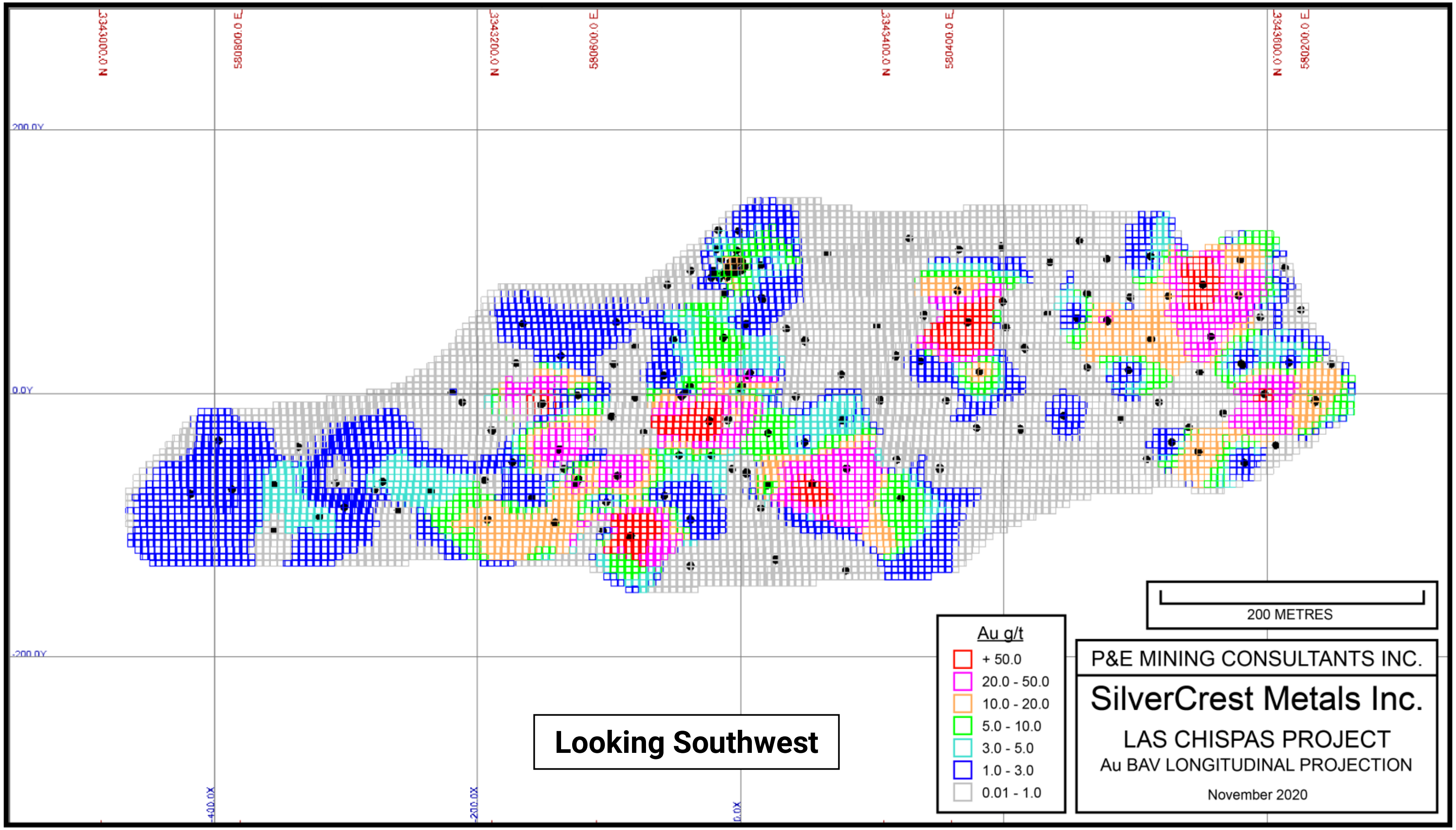

3345600.0 N
580200.0 E
3345400.0 N
580300.0 E
3345200.0 N
580400.0 E
3345000.0 N
3344800.0 N
580500.0 E
89400.0Y
89200.0Y
200 METRES
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
Ag LC LONGITUDINAL PROJECTION
November 2020
Ag g/t
+ 2000
1000 - 2000
500 - 1000
300 - 500
150 - 300
0.01 - 150
Looking Northeast
-108200.0X
-108000.0X
-107800.0X

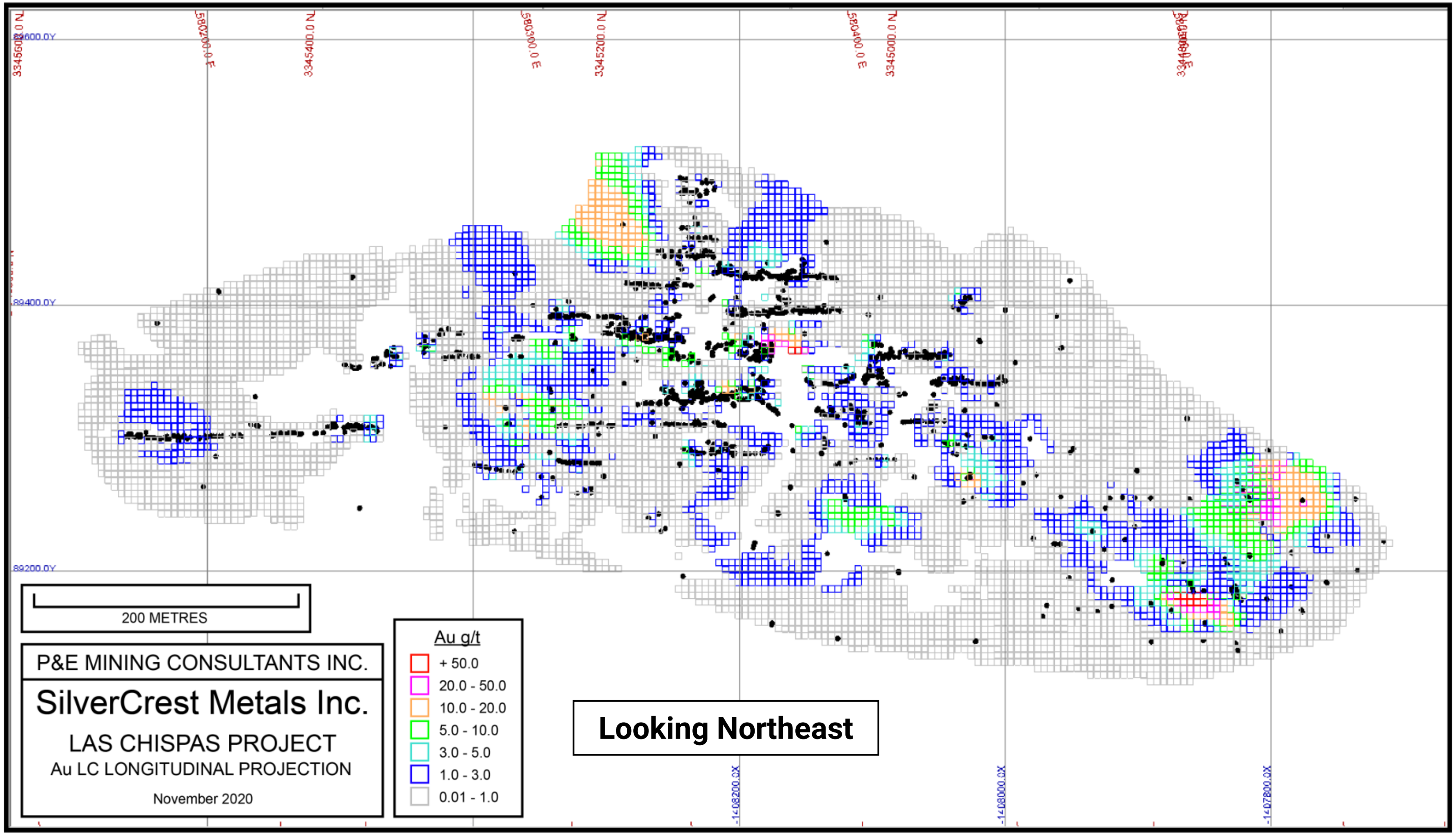
APPENDIX F AU LONGITUDINAL PROJECTIONS


Looking Southwest
Au g/t
+ 50.0
20.0 - 50.0
10.0 - 20.0
5.0 - 10.0
3.0 - 5.0
1.0 - 3.0
0.01 - 1.0
200 METRES
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
Au BAM LONGITUDINAL PROJECTION
November 2020

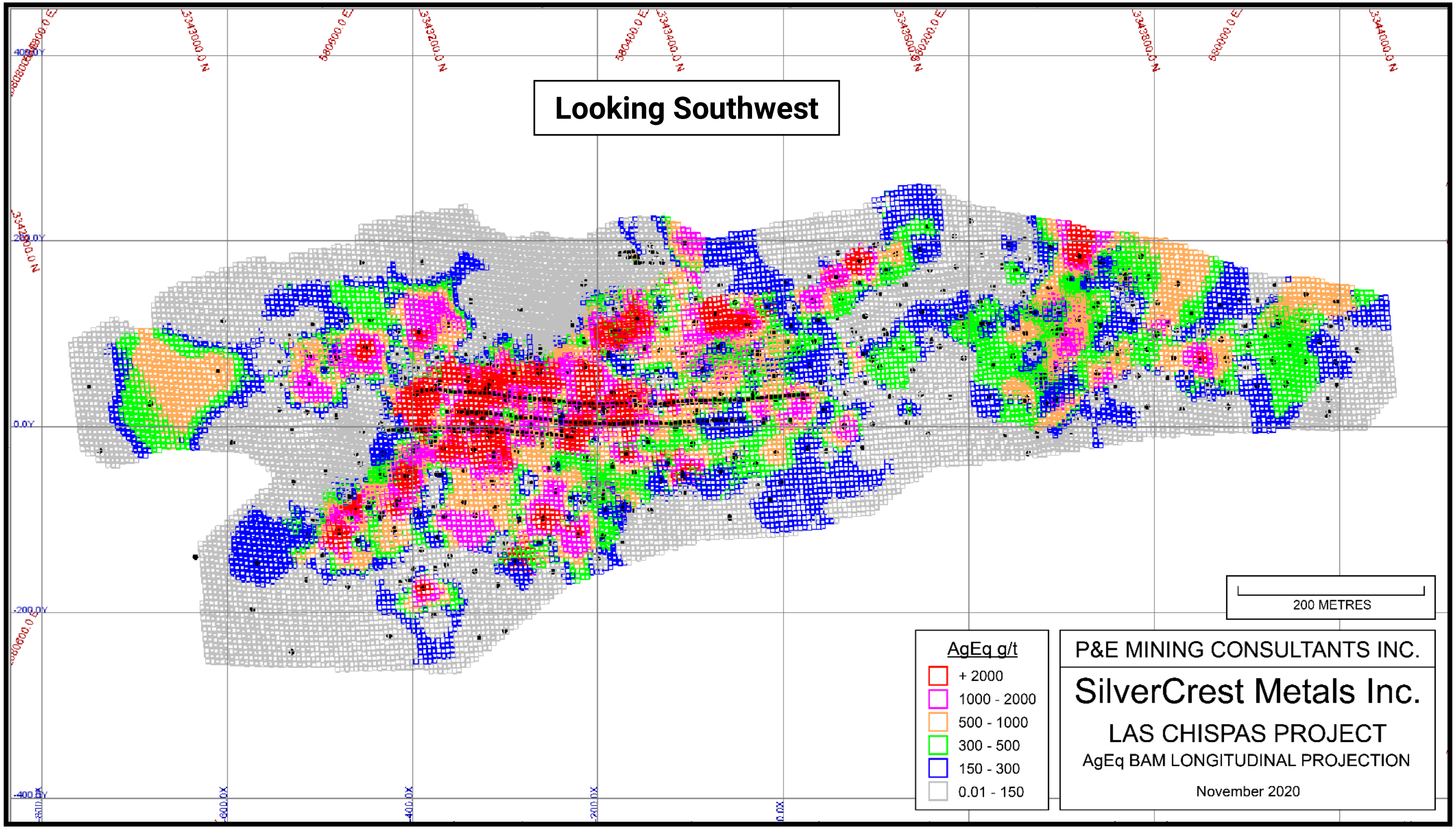

Looking Southwest
1200.0Y
1000.0Y
800.0Y
600.0Y
400.0Y
200.0Y
Au g/t
+ 50.0
20.0 - 50.0
10.0 - 20.0
5.0 - 10.0
3.0 - 5.0
1.0 - 3.0
0.01 - 1.0
400 METRES
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
Au BAN LONGITUDINAL PROJECTION
November 2020

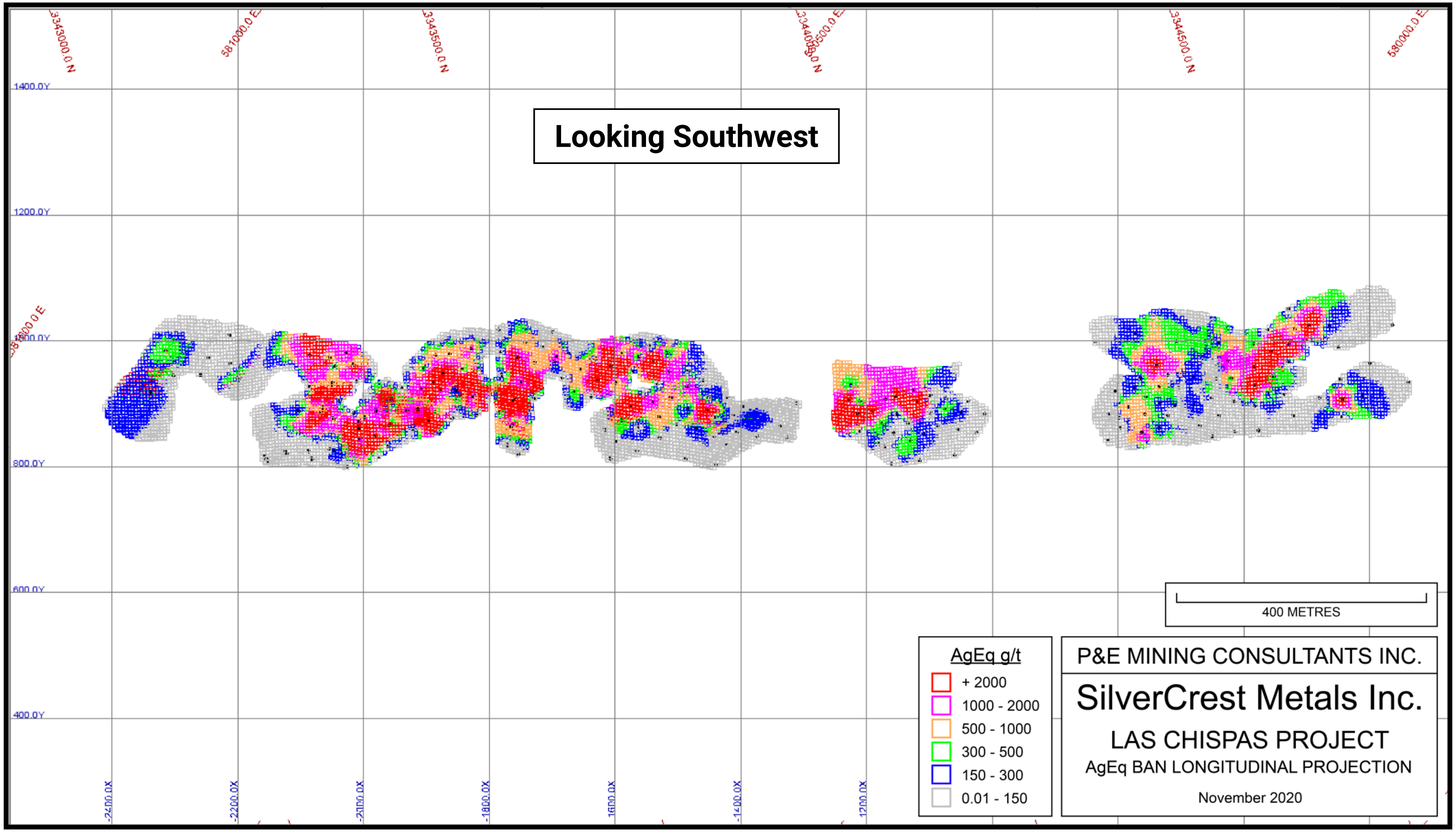

Looking Southwest
Au g/t
+ 50.0
20.0 - 50.0
10.0 - 20.0
5.0 - 10.0
3.0 - 5.0
1.0 - 3.0
0.01 - 1.0
200 METRES
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
Au BAS LONGITUDINAL PROJECTION
November 2020

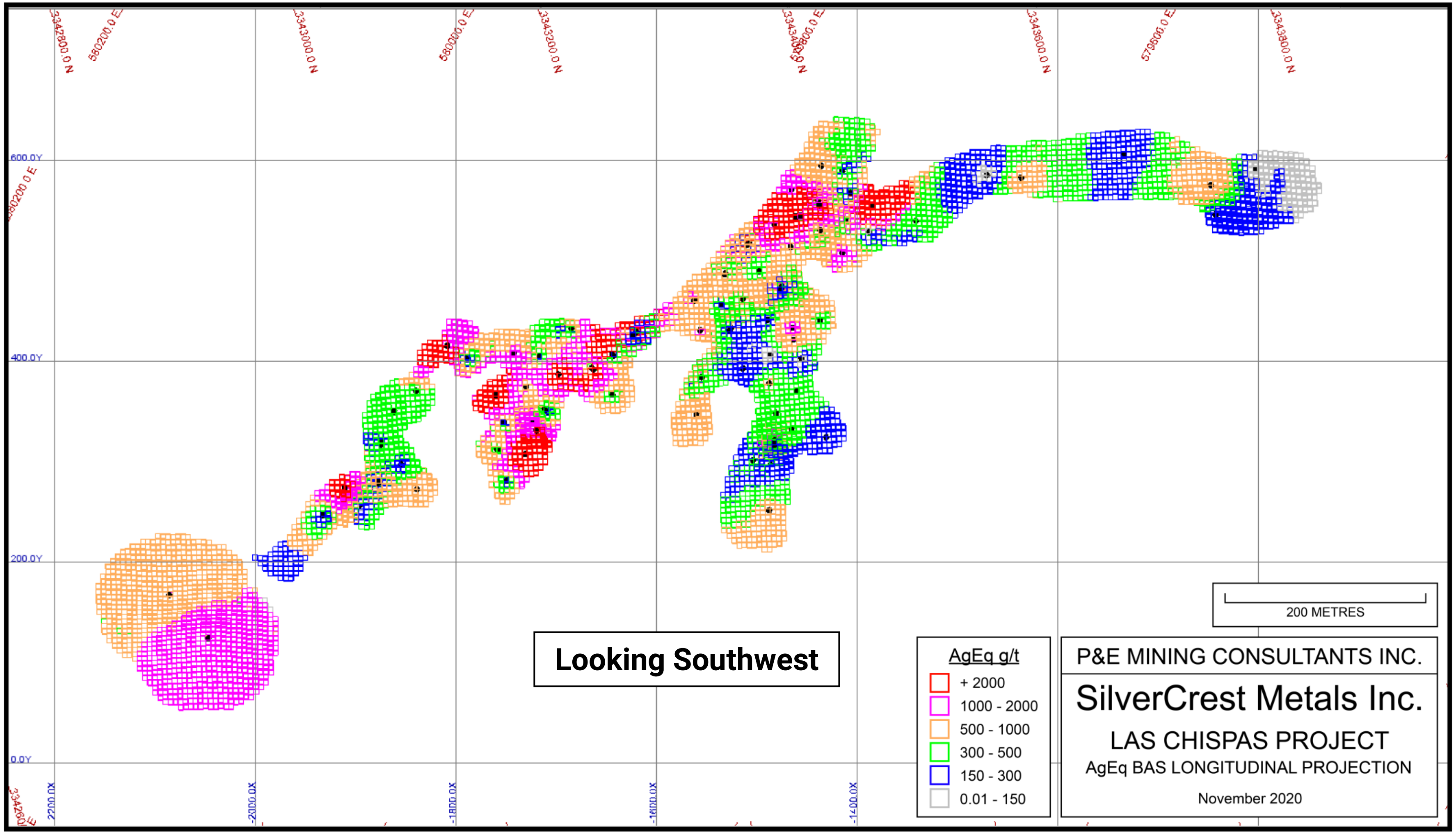

Looking Southwest
Au g/t
+ 50.0
20.0 - 50.0
10.0 - 20.0
5.0 - 10.0
3.0 - 5.0
1.0 - 3.0
0.01 - 1.0
200 METRES
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
Au BAV LONGITUDINAL PROJECTION
November 2020

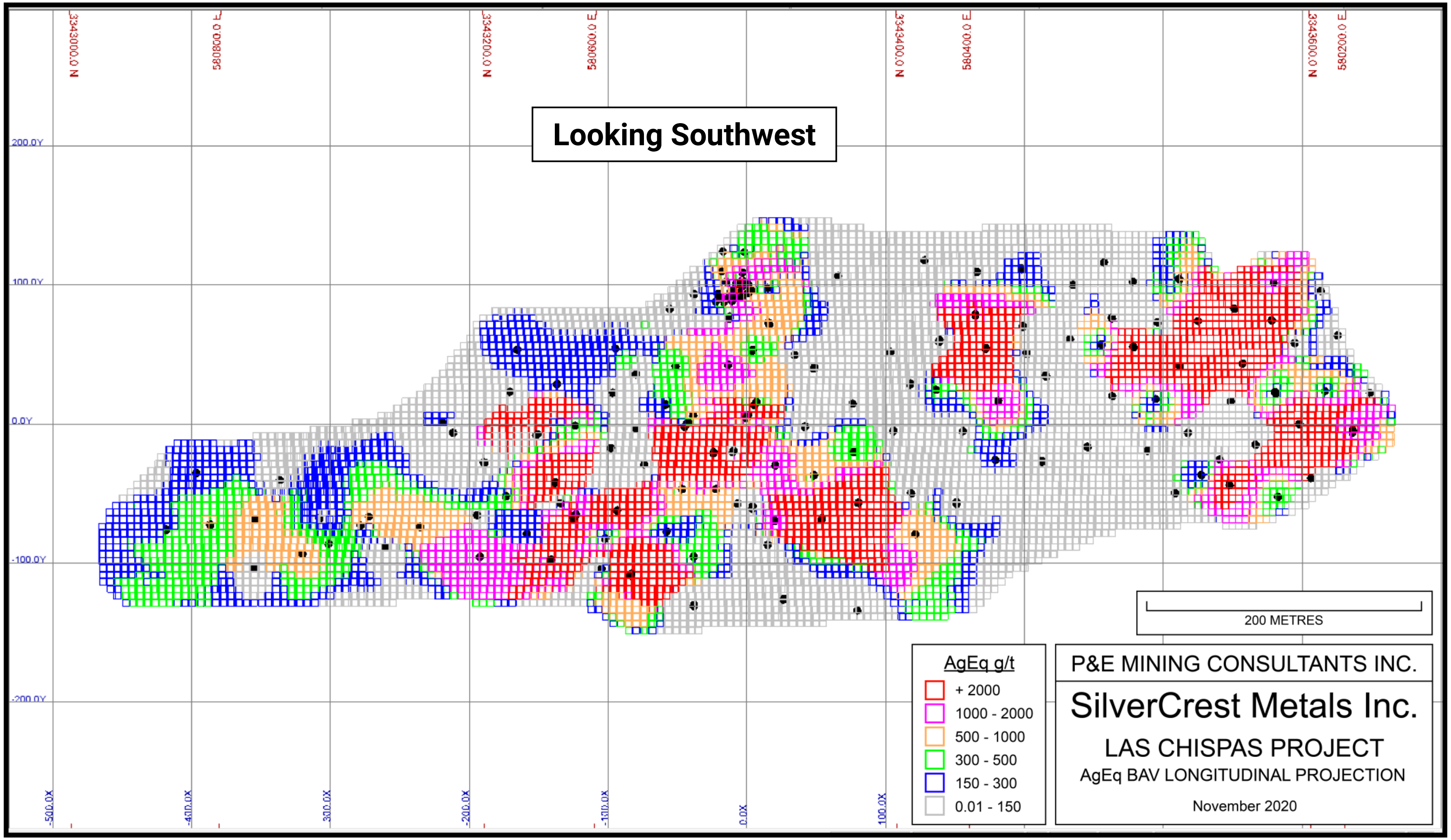

200 METRES
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
Au LC LONGITUDINAL PROJECTION
November 2020
Au g/t
+ 50.0
20.0 - 50.0
10.0 - 20.0
5.0 - 10.0
3.0 - 5.0
1.0 - 3.0
0.01 - 1.0
Looking Northeast

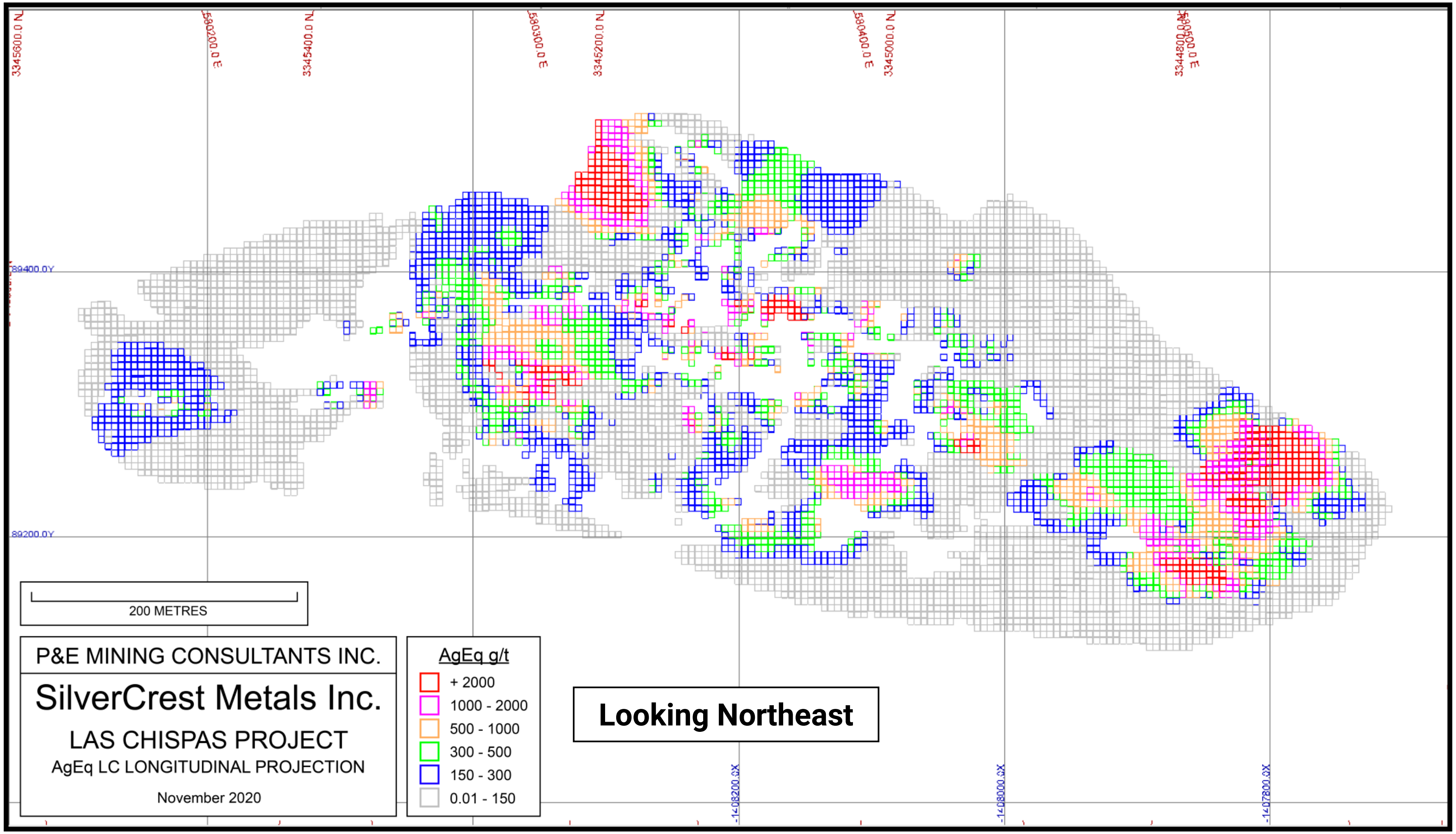
APPENDIX G AGEQ LONGITUDINAL PROJECTIONS


Looking Southwest
200 METRES
AgEq g/t
+ 2000
1000 - 2000
500 - 1000
300 - 500
150 - 300
0.01 - 150
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
AgEq BAM LONGITUDINAL PROJECTION
November 2020

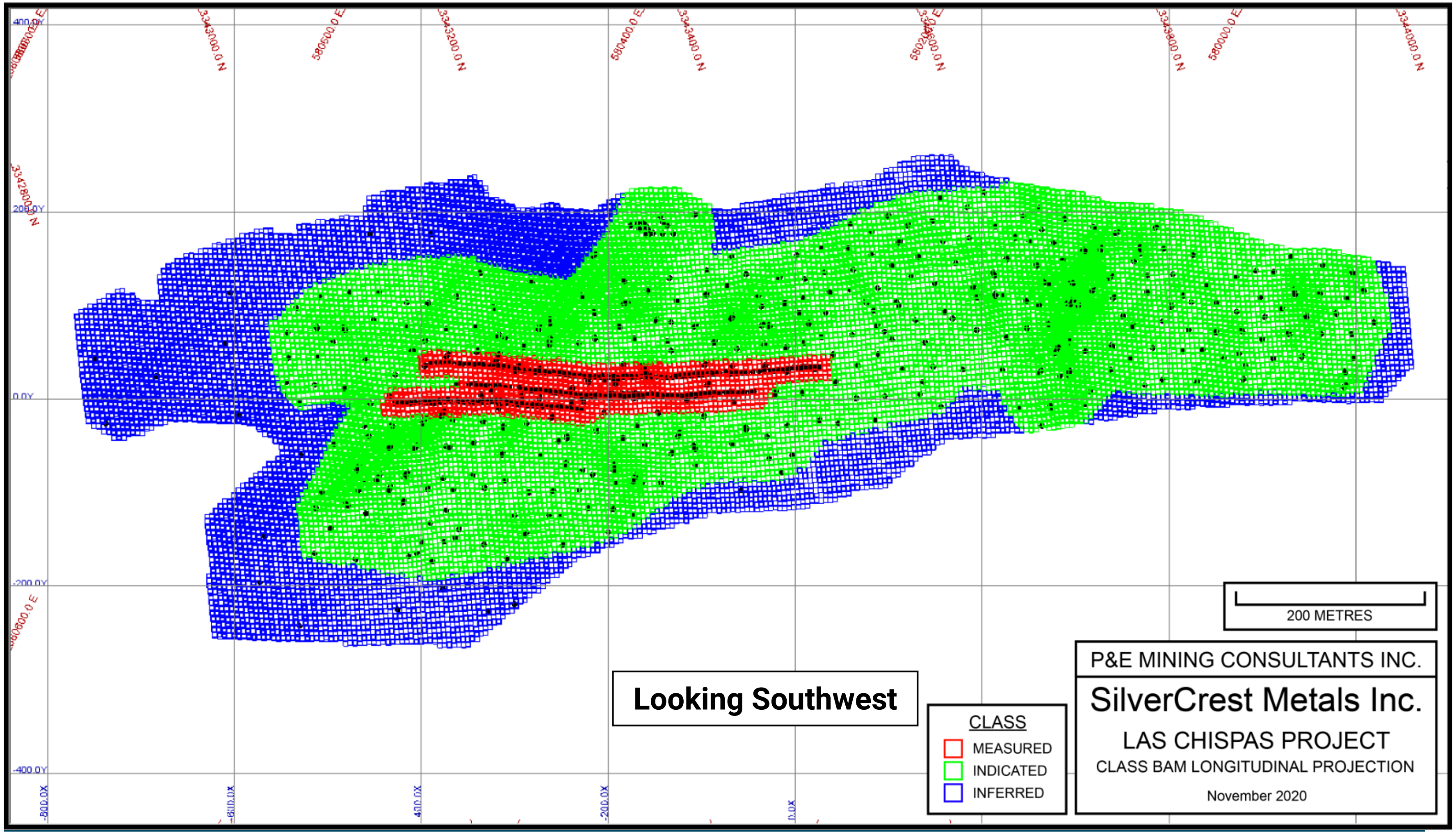

Looking Southwest
1400.0Y
1200.0Y
800.0Y
600.0Y
400.0Y
3343000.0 N
581000.0 E
3343500.0 N
3344500.0 N
580000.0 E
-2400.0X
-2200.0X
-2000.0X
-1800.0X
-1600.0X
-1400.0X
-1200.0X
400 METRES
AgEq g/t
+ 2000
1000 - 2000
500 - 1000
300 - 500
150 - 300
0.01 - 150
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
AgEq BAN LONGITUDINAL PROJECTION
November 2020

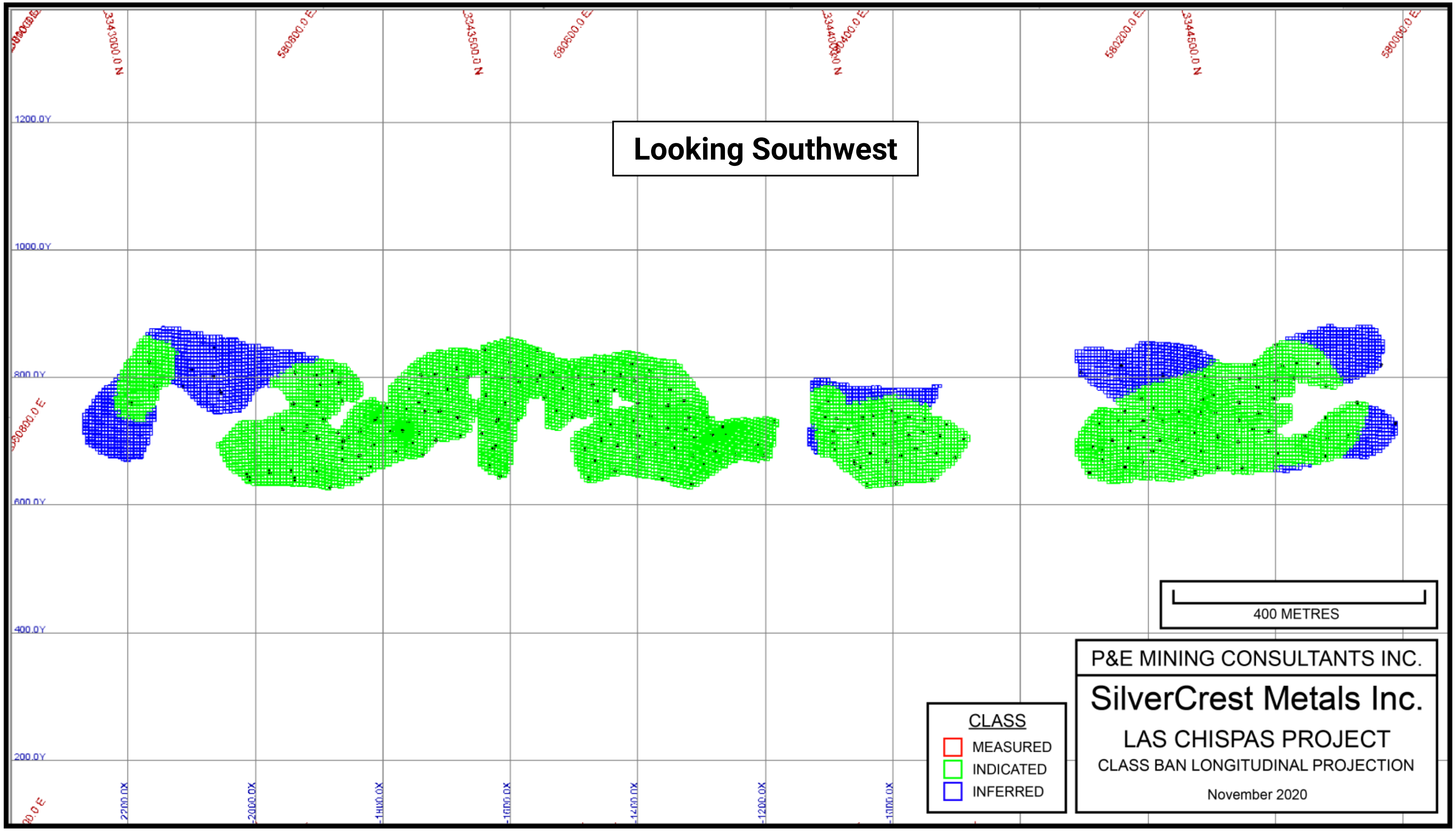

Looking Southwest
AgEq g/t
+ 2000
1000 - 2000
500 - 1000
300 - 500
150 - 300
0.01 - 150
200 METRES
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
AgEq BAS LONGITUDINAL PROJECTION
November 2020

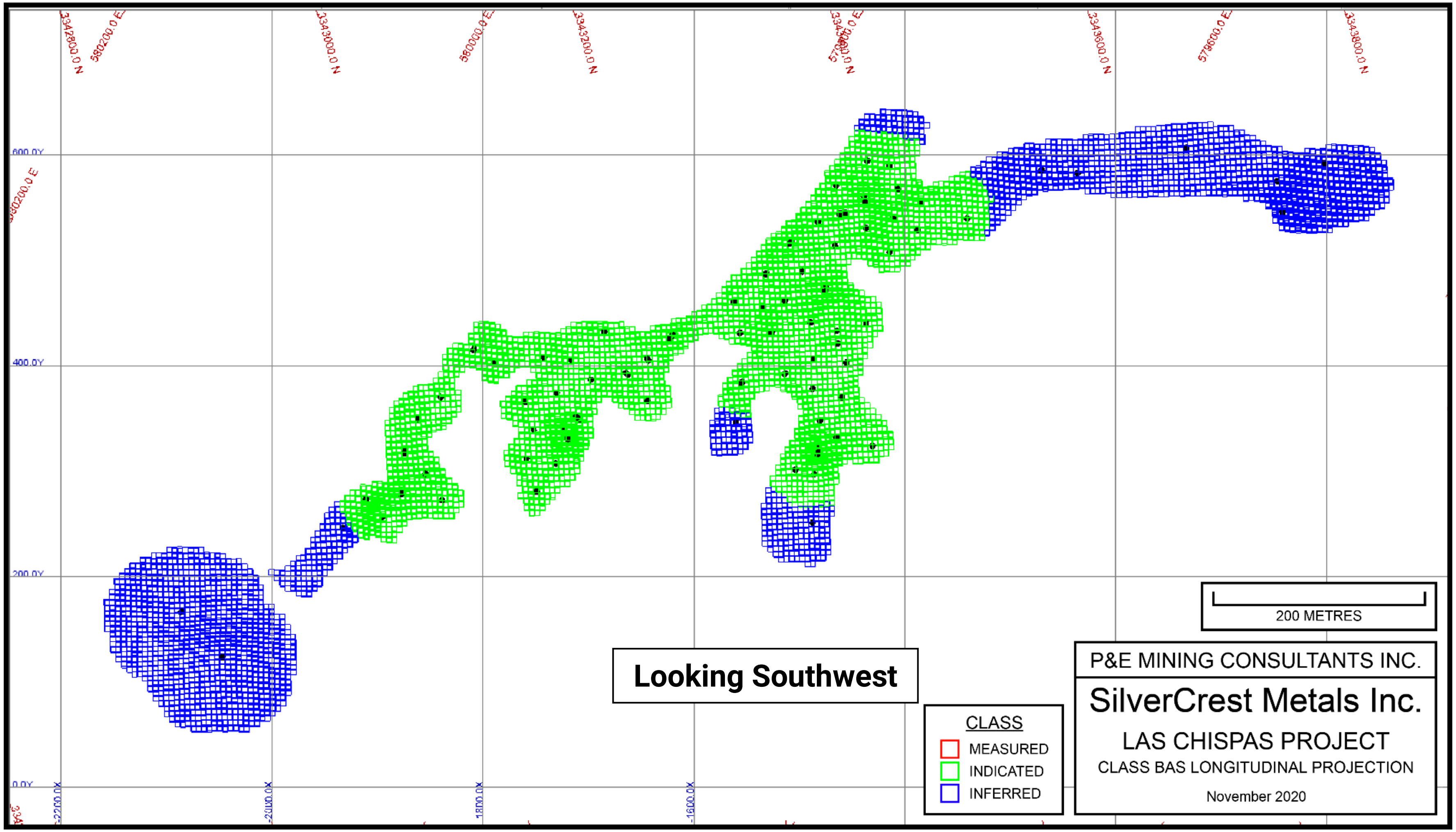

Looking Southwest
3343000.0 N
580800.0 E
3343200.0 N
580600.0 E
3343400.0 N
580400.0 E
3343600.0 N
580200.0 E
200.0Y
100.0Y
0.0Y
-100.0Y
-200.0Y
-500.0X
-400.0X
-300.0X
-200.0X
-100.0X
0.0X
100.0X
200 METRES
AgEq g/t
+ 2000
1000 - 2000
500 - 1000
300 - 500
150 - 300
0.01 - 150
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
AgEq BAV LONGITUDINAL PROJECTION
November 2020

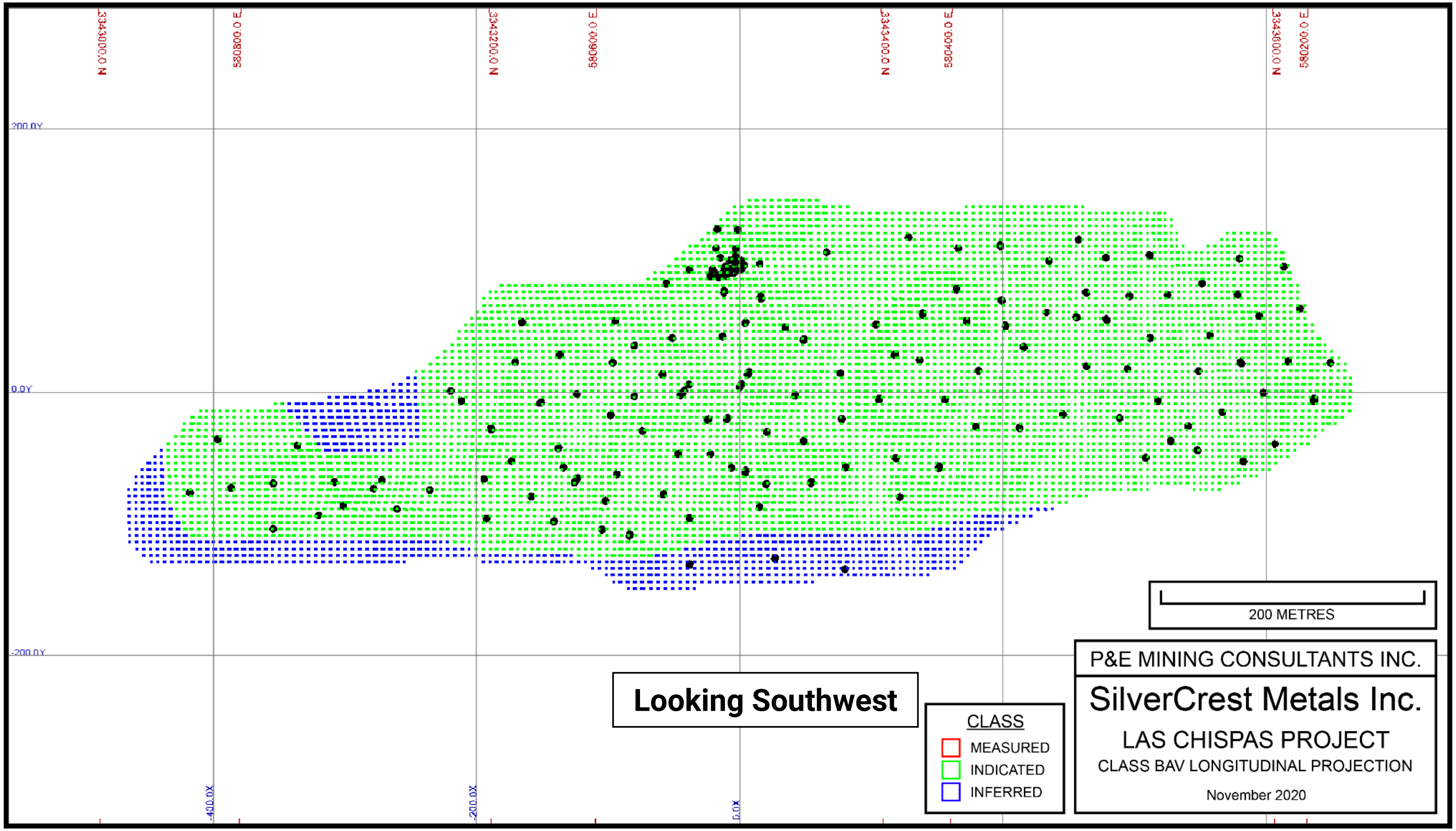

3345600.0 N
589200.0 E
3345400.0 N
589300.0 E
3345200.0 N
589400.0 E
3345000.0 N
3344800.0 N
589500.0 E
89400.0Y
89200.0Y
200 METRES
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
AgEq LC LONGITUDINAL PROJECTION
November 2020
AgEq g/t
+ 2000
1000 - 2000
500 - 1000
300 - 500
150 - 300
0.01 - 150
Looking Northeast
-1408200.0X
-1408000.0X
-1407800.0X

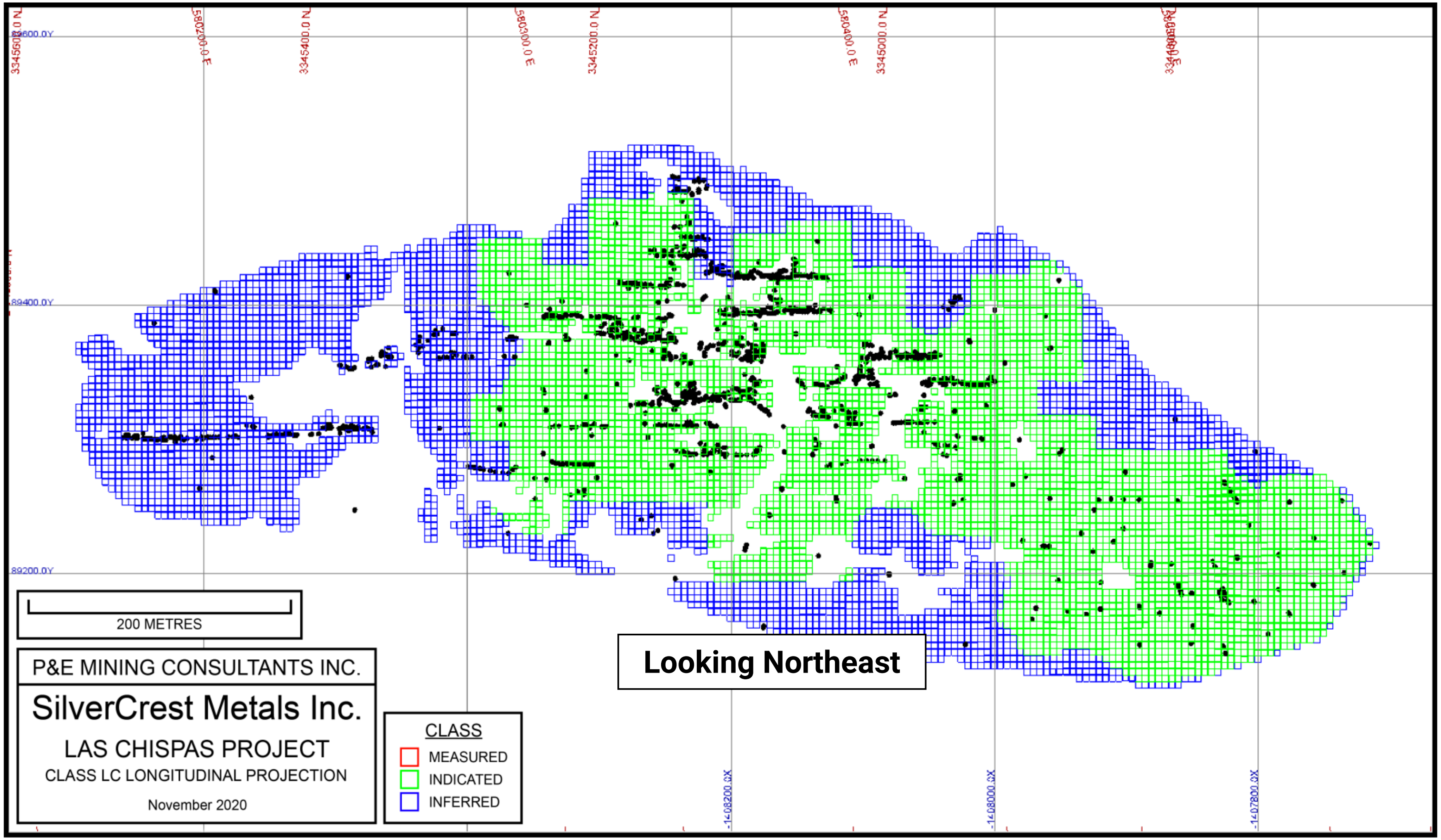
APPENDIX H CLASSIFICATION LONGITUDINAL PROJECTIONS


Looking Southwest
CLASS
MEASURED
INDICATED
INFERRED
200 METRES
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
CLASS BAM LONGITUDINAL PROJECTION
November 2020


Looking Southwest
1200.0Y
1000.0Y
800.0Y
600.0Y
400.0Y
200.0Y
3343000.0 N
3343500.0 N
3344500.0 N
580600.0 E
-2200.0X
-2000.0X
-1800.0X
-1600.0X
-1400.0X
-1200.0X
-1000.0X
400 METRES
CLASS
MEASURED
INDICATED
INFERRED
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
CLASS BAN LONGITUDINAL PROJECTION
November 2020


Looking Southwest
200 METRES
CLASS
MEASURED
INDICATED
INFERRED
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
CLASS BAS LONGITUDINAL PROJECTION
November 2020


3343000.0 N
580800.0 E
3343200.0 N
580600.0 E
3343400.0 N
580400.0 E
3343600.0 N
580200.0 E
200.0Y
0.0Y
-200.0Y
-400.0X
-200.0X
0.0X
200 METRES
Looking Southwest
CLASS
MEASURED
INDICATED
INFERRED
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
CLASS BAV LONGITUDINAL PROJECTION
November 2020


200 METRES
P&E MINING CONSULTANTS INC.
SilverCrest Metals Inc.
LAS CHISPAS PROJECT
CLASS LC LONGITUDINAL PROJECTION
November 2020
CLASS
MEASURED
INDICATED
INFERRED
Looking Northeast
3345400.0 N
3345200.0 N
3345000.0 N
89400.0Y
89200.0Y
-1408200.0X
-1408000.0X
-1407800.0X