N.I. 43-101 Technical Report on the Wasamac Feasibility Study Update

Québec, Canada

Effective Date: July 16, 2021

Prepared for: Yamana Gold Inc.

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Vincent Nadeau-Benoit, P.Geo. – InnovExplo

CERTIFICATE OF QUALIFIED PERSON
Tommaso Roberto Raponi

I, Tommaso Roberto Raponi, P. Eng., certify that I am employed as a Principal Metallurgist with Ausenco Engineering Canada Inc., with an office address of Suite 1550 - 11 King St West, Toronto, ONT M5H 4C7. This certificate applies to the technical report titled N.I. 43-101 Technical Report on the Wasamac Feasibility Study Update that has an effective date of July 16, 2021 (the "Technical Report").

I graduated from the University of Toronto with a Bachelor of Applied Science degree in Geological Engineering in 1984, with a specialization in Mineral Processing. I am a Professional Engineer registered with the Professional Engineers Ontario (No. 90225970), Engineers and Geoscientists British Columbia (No. 23536) and NWT and Nunavut Association of Professional Engineers and Geoscientists (No. L4508) and with OIQ (Temporary Permit No. 6043399). I have practiced my profession continuously for over 37 years with experience in the development, design, operation and commissioning of mineral processing plants, focusing on gold projects, both domestic and internationally. A summary of the more recent portion of my professional career is as follows: TR Raponi Consulting Ltd, Independent Consultant 2016-2021; Centerra Gold Inc., Director of Metallurgy 2005-2016; and SNC-Lavalin Inc., Senior Metallurgist, 1995-2005.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("N.I. 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.

I have not visited the Wasamac Project. I am responsible for Sections 1.1, 1.9, 1.13, 1.14.1, 1.14.2, 1.14.5, 1.16, 1.17, 1.18.3, 1.18.4, 2.1, 2.2, 2.5, 2.7, 3, 13, 16.7.1, 16.7.2, 16.7.3, 17, 18.1, 18.2, 18.3, 18.4, 18.5, 18.6, 18.9.2, 18.9.5, 18.10.6, 18.10.7, 18.13.1, 18.13.2, 18.13.5, 18.14, 19, 21.1, 21.2.1, 21.2.3, 21.2.4, 21.2.5, 21.2.6, 21.2.7, 21.2.8, 21.2.9, 21.2.10, 21.2.11, 21.3.2, 21.3.3, 21.3.4, 21.3.5, 21.3.6, 21.3.7, 21.4.1, 21.4.3, 21.4.4, 21.4.5, 21.4.6, 22, 24, 25.1, 25.6, 25.7, 25.8, 25.10, 25.11, 25.12, 25.13.2, 25.14.1, 25.14.2, 25.14.3, 26.1, 26.4 of the Technical Report.

I am independent of Yamana Gold Inc. as independence is defined in Section 1.5 of N.I. 43-101. I have had no previous involvement with the Wasamac Project.

I have read N.I. 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: September 14, 2021

"Original Signed and sealed"

Tommaso Roberto Raponi, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
Alain Carrier

I, Alain Carrier, P.Geo., M.Sc., certify that I am employed as Co-President Founder with InnovExplo Inc., with an office address of 560, 3e Avenue, Val-d'Or, Québec, Canada, J9P 1S4. This certificate applies to the technical report titled N.I. 43-101 Technical Report on the Wasamac Feasibility Study Update that has an effective date of July 16, 2021 (the "Technical Report").

I graduated from Cégep de l'Abitibi-Témiscamingue, Rouyn-Noranda, Québec, in 1989 with a mining technician degree in geology; from Université du Québec à Montréal, Montréal, Québec, in 1992, with a Bachelor's degree in Geology; and in 1994 with a Master's degree in Earth Sciences. I initiated a PhD in geology at INRS-Géoressources, Sainte-Foy, Québec, for which I completed the course program but not the thesis. I am a geologist of the Ordre des Géologues du Québec (OGQ licence No. 0281), the Association of Professional Geoscientists of Ontario (PGO licence No. 1719), the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (NAPEG No. L2701), the Canadian Institute of Mines, Metallurgy and Petroleum (CIM 91323), and of the Society of Economic Geologists (SEG 132243). I have practiced my profession for 28 years. I have been directly involved in mineral exploration, mine geology, ore control and resource modelling projects for gold, copper, zinc, silver, nickel, lithium, graphite and uranium properties in Canada and internationally.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("N.I. 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.

I visited the Wasamac Project on May 13, 2021 for 1 day. I am responsible for Sections 2.3.1 and 2.4 of the Technical Report and am the co-author of Sections 1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.10, 1.18.1, 4 to 12, 14, 23, 25.2, 25.3, 25.4, 26.3 of the Technical Report.

I am independent of Yamana Gold Inc. as independence is defined in Section 1.5 of N.I. 43-101. I have had no previous involvement with the Wasamac Project.

I have read N.I. 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: September 14, 2021

"Original Signed and sealed"

Alain Carrier, P.Geo., M.Sc.

CERTIFICATE OF QUALIFIED PERSON
Denis Gourde

I, Denis Gourde, P.Eng., certify that I am employed as a consulting mining engineer with InnovExplo Inc., with an office address of 560, 3e Avenue, Val-d'Or, Québec, Canada. This certificate applies to the technical report titled N.I. 43-101 Technical Report on the Wasamac Feasibility Study Update that has an effective date of July 16, 2021 (the "Technical Report").

I graduated from École Polytechnique de Montréal (Montréal, Québec) in 1987 with a Bachelor of Science in Mining Engineering. I am a member in good standing of the Ordre des Ingénieurs du Québec (No. 43860). I have practiced my profession for 34 years since my graduation from university. My mining expertise has been acquired in the Sleeping Giant, Kiena, Musselwhite, Meadowbank and Rosebel mines, whereas my engineering and management experience has been acquired with Agnico Eagle, Cambior, Iamgold and Ross Finlay Ltd. I have been a consulting mining engineer for InnovExplo Inc. since December 2013.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("N.I. 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.

I visited the Wasamac Project that is the subject of the Technical Report on May 12, 2021 for 1 day. I am the co-author of Sections 1.11, 1.12, 1.18.2, 2.2, 2.3.2, 15, 16.1, 16.2, 16.4, 16.5 (excluding 16.5.4.1), 16.6, 16.7.4, 16.7.5, 16.7.6. 16.7.7, 16.7.8, 16.9, 16.10, 21.2.2, 21.3.1, 21.4.2, 25.4, 25.5, 25.13.1, 25.14.4, 26.1 and 26.2 of the Technical Report.

I am independent of Yamana Gold as independence is defined in Section 1.5 of N.I. 43-101. I have had no previous involvement with the Wasamac Project.

I have read N.I. 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: September 14, 2021

"Original Signed and sealed"

Denis Gourde, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
Frank Palkovits

I, Frank Palkovits, P. Eng., certify that I am employed as senior technical director with Responsible Mining Solutions, with an office address of 83 Durham Street, Sudbury, Ontario, Canada. This certificate applies to the technical report titled N.I. 43-101 Technical Report on the Wasamac Feasibility Study Update that has an effective date of July 16, 2021 (the "Technical Report").

I graduated from Laurentian University (Sudbury, Ontario) in 1988 with a Bachelor's degree (B.Eng.) in mining engineering. I am a member in good standing of Professional Engineers Ontario (No: 90276379). I have practiced my profession for 33 years since my graduation from university. My mining expertise has been acquired in the Garson, Frood, Levack, Creighton, Crean Hill and Lupin mines, whereas my backfill and tailings experience has been acquired with Golder Paste Technology, Kovit Engineering, Mine Paste Engineering Ltd, Outotec Canada, and Responsible Mining Solutions over the past 21 years. I have been Senior Technical Director of Responsible Mining Solutions since August 2020.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("N.I. 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.

I have not visited the Wasamac Project that is the subject of the Technical Report. I am responsible for Sections 16.5.4.1, 18.13.3, 18.13.4 of the Technical Report.

I am independent of Yamana Gold as independence is defined in Section 1.5 of N.I. 43-101. I was involved previously with a backfill scoping study and testing campaign completed by Kovit Engineering for Richmont Mines Inc. in 2013.

I have read N.I. 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: September 14, 2021

"Original Signed and sealed"

Frank Palkovits, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
Luciano Piciacchia

I, Luciano Piciacchia, P.Eng., certify that I am employed as Managing Director Earth and Infrastructure with BBA Consultants, with an office address of 2020 Robert-Bourassa Blvd., Suite 300, Montréal, QC, H3A 2A5, Canada. This certificate applies to the technical report titled N.I. 43-101 Technical Report on the Wasamac Feasibility Study Update that has an effective date of July 16, 2021 (the "Technical Report").

I graduated in 1981 from McGill University (Montréal, Québec) in mining engineering, with a Masters' and Ph.D. focusing in soil and rock geotechnics, also from McGill in 1983 and 1988. I am a member of the order of engineers in, Quebec (OIQ No. 35912), Ontario (PEO No. 36633501), Newfoundland & Labrador (PEGNL No. 05606), British Columbia (EGBC No. 211083). I have over 32 years of experience in geotechnical engineering with a focus on mining. I have applied my geotechnical / civil background to mine waste management, including waste rock, tailings and water.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("N.I. 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.

I did not visit the Wasamac Project that is the subject of the Technical Report. I am responsible for Sections 1.14.3, 2.3.3, 18.9, 18.9.1, 18.9.3, 18.9.4 and 18.9.6 of the Technical Report

I am independent of Yamana Gold as independence is defined in Section 1.5 of N.I. 43-101. I have had no previous involvement with the Wasamac Project.

I have read N.I. 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: September 14, 2021

"Original Signed and sealed"

Luciano Piciacchia, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
Michael Verreault

I, Michael Verreault, P. Eng., certify that I am employed as a consulting mining hydrogeologist with Hydro-Ressources Inc. (HRI), with an office address of 1855, des Campanules, Jonquiere, Québec, Canada. This certificate applies to the technical report titled N.I. 43-101 Technical Report on the Wasamac Feasibility Study Update that has an effective date of July 16, 2021 (the "Technical Report").

I graduated from Universite du Quebec à Chicoutimi, (UQAC) in 2000 with a Bachelor's degree in geological engineering (B.Sc.), and a Master degree in Applied Sciences in 2003. I am a member in good standing of the Ordre des Ingénieurs du Québec (No. 125243). I have practiced my profession for 21 years since my graduation from university. I have been a consulting mining hydrogeologist for HRI since December 2011.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("N.I. 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.

I am responsible for the preparation of Section 16.3 of the Technical Report.

I am independent of Yamana Gold as independence is defined in Section 1.5 of N.I. 43-101. I have had no previous involvement with the Wasamac Project.

I have read N.I. 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: September 14, 2021

"Original Signed and sealed"

Michael Verreault, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
Charles Gagnon, ing

I, Charles Gagnon, ing., certify that I am employed as an independent consulting mining engineer with CGM, with an office address of 1155 avenue des Érables, Québec, Canada. This certificate applies to the technical report titled N.I. 43-101 Technical Report on the Wasamac Feasibility Study Update that has an effective date of July 16, 2021 (the "Technical Report").

I graduated from Université Laval (Québec, Québec) in 2001 with a Bachelor's degree in mining engineering (B.Sc.). I am a member in good standing of the Ordre des Ingénieurs du Québec (No. 130730). I have practiced my profession for 20 years since my graduation from university. My mining expertise has been acquired in the Eleonore Goldcorp operation and throughout consultant project (Genivar), whereas my underground ventilation engineering experience has been acquired with Genivar inc, Eleonore (Goldcorp inc.), Howden inc. I have been a consulting mining engineer for CGM Inc. since January 2019.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("N.I. 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.

I am responsible for the preparation of Section 16.8 and share responsibility for Sections 1, 2, and 21 of the Technical Report.

I am independent of Yamana Gold as independence is defined in Section 1.5 of N.I. 43-101. I have had no previous involvement with the Wasamac Project.

I have read N.I. 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: September 14, 2021

"Original Signed and sealed"

Charles Gagnon, ing.

CERTIFICATE OF QUALIFIED PERSON

Ali Hooshiar

I, Ali Hooshiar, P.Eng., certify that I am employed as Geotechnical Mining Engineer with Ausenco Engineering Canada Inc. ("Ausenco"), with an office address of 855 Homer Street, Vancouver, British Columbia, Canada, V6B 2W2. This certificate applies to the technical report titled N.I. 43-101 Technical Report on the Wasamac Feasibility Study Update that has an effective date of July 16, 2021 (the "Technical Report").

I graduated from Sharif University of Technology with BSc and MSc in Materials Science and Engineering in 2003 and 2006, respectively, and the University of Alberta in 2011 with a PhD in Materials Engineering. I am a Professional Engineer registered with the Engineers and Geoscientists British Columbia (No. 40965), Engineers Yukon, and OIQ (No. 6043599). I have practiced my profession for 18 years with experience in designing tailings and waste rock storage facilities as well as managing geotechnical field investigation and lab testing programs for mining projects across the globe. A summary of the more recent portion of my professional career is as follows:

- Geotechnical Mining Engineer, Ausenco, Canada 2018-present
- Geotechnical Mining Engineer, AECOM, Canada 2013-2017
- Senior Geotechnical Consultant, SRK Consulting Inc., Canada 2011-2013

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("N.I. 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.

I have not visited the Wasamac Project. I am responsible for Sections 1.14.4, 1.14.6, 1.18.5, 18.7, 18.8, 18.10.1, 18.10.2, 18.10.3, 18.10.4, 18.10.5, 18.10.8, 18.11, 18.12, 20.2.3, 20.2.4, and 26.5 of the Technical Report.

I am independent of Yamana Gold Inc. as independence is defined in Section 1.5 of N.I. 43-101. I have had no previous involvement with the Wasamac Project.

I have read N.I. 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: September 14, 2021

"Original Signed and sealed"

Ali Hooshiar, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Scott Weston

I, Scott Weston, P. Geo., certify that I am employed as Vice President, Business Development with Hemmera Envirochem Inc., a wholly-owned subsidiary of Ausenco Engineering Canada Inc., with an office address of 4730 Kingsway, 18th floor, Burnaby, BC, Vancouver, BC V5H 0C6. This certificate applies to the technical report titled N.I. 43-101 Technical Report on the Wasamac Feasibility Study Update that has an effective date of July 16, 2021 (the "Technical Report").

I graduated from the University of British Colombia with a Bachelors of Science in Physical Geography and from Royal Roads University with a Masters of Science, Environment and Management. I am a Professional Geoscientist of Engineers and Geoscientists, British Colombia (#124888). I have worked as a geoscientist continuously for 25 years.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("N.I. 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.

I have not visited the Wasamac Project. I am responsible for Sections 1.15, 1.18.6, 1.18.7, 20 (except 20.2.3 and 20.2.4), 25.9, 25.14.5, 26.6, 26.7, 26.8 of the Technical Report.

I am independent of Yamana Gold Inc. as independence is defined in Section 1.5 of N.I. 43-101. I have had no previous involvement with the Wasamac Project.

I have read N.I. 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: September 14, 2021

"Original Signed and sealed"

Scott Weston, P.Geo.

CERTIFICATE OF QUALIFIED PERSON
Sébastien Tanguay

I, Sébastien Tanguay, P. Eng., certify that I am employed as a consulting mining engineer with InnovExplo Inc., with an office address of 560, 3e Avenue, Val-d'Or, Québec, Canada. This certificate applies to the technical report titled N.I. 43-101 Technical Report on the Wasamac Feasibility Study Update that has an effective date of July 16, 2021 (the "Technical Report").

I graduated from École Polytechnique de Montréal (Montréal, Québec) in 2014 with a Bachelor's degree in mining engineering (B.Sc.) and in 2016 with a Master's in mineral engineering specialized in rock mechanics (M.Sc.A.). I am a member in good standing of the Ordre des Ingénieurs du Québec (No. 5074008). I have practiced my profession for five years since my graduation from university. My mine engineering expertise has been acquired with Graymont (one year) and through many projects since I have been a consulting mining engineer for InnovExplo Inc. since September 2016.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("N.I. 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.

I visited the Wasamac Project that is the subject of the Technical Report on June 2 to 3, 2021. I am the co-author of Sections 1.11, 1.12, 1.18.2, 2.2, 2.3.2, 15, 16.1, 16.2, 16.4, 16.5 (excluding 16.5.4.1), 16.6, 16.7.4, 16.7.5, 16.7.6. 16.7.7, 16.7.8, 16.9, 16.10, 21.2.2, 21.3.1, 21.4.2, 25.4, 25.5, 25.13.1, 25.14.4, 26.1 and 26.2 of the Technical Report.

I am independent of Yamana Gold as independence is defined in Section 1.5 of N.I. 43-101. I have had no previous involvement with the Wasamac Project.

I have read N.I. 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: September 14, 2021

"Original Signed and sealed"

Sébastien Tanguay, P. Eng. M.Sc.A

CERTIFICATE OF QUALIFIED PERSON
Vincent Nadeau-Benoit

I, Vincent Nadeau-Benoit, P. Geo., certify that I am employed as a project geologist with InnovExplo Inc., with an office address of 560, 3e Avenue, Val-d'Or, Québec, Canada, J9P 1S4. This certificate applies to the technical report titled N.I. 43-101 Technical Report on the Wasamac Feasibility Study Update that has an effective date of July 16, 2021 (the "Technical Report").

I graduated from Université du Québec à Montréal, Montréal, Québec, in 2010 with a Bachelor's degree in Earth and Atmosphere Science (Geology). I am a geologist of the Ordre des Géologues du Québec (OGQ No. 1535) and the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (NAPEG No. L4154). I have practiced my profession for 10 years. I have been directly involved in mineral exploration and mine geology projects for precious and base metal properties in Canada. I acquired my expertise with Royal Nickel Corporation and Glencore and through numerous projects for InnovExplo Inc. as a consulting geologist since August 2018.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("N.I. 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.

I visited the Wasamac Project on May 13, 2021 for a visit duration of 1 day. I am responsible for Section 2.6 of the Technical Report and am the co-author of Sections 1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.10, 1.18.1, 4 to 12, 14, 23, 25.2, 25.3, 25.4, 26.3 of the Technical Report.

I am independent of Yamana Gold Inc. as independence is defined in Section 1.5 of N.I. 43-101., I have had no previous involvement with the Wasamac Project.

I have read N.I. 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: September 14, 2021

"Original Signed and sealed"

Vincent Nadeau-Benoit, P. Geo

Important Notice

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1 SUMMARY

1.1 Overview

Yamana Gold Inc. (Yamana) commissioned Ausenco Engineering Canada Inc. (Ausenco) to update the previous feasibility study completed by Monarch Gold Corporation (Monarch) in 2018 and summarize the feasibility study update in this NI 43-101 Technical Report. Wasamac is supported by a feasibility study completed by the project's previous owner in 2018. As part of its technical diligence process relating to the acquisition of Wasamac in early 2021, Yamana identified several opportunities to optimize the mine design and process flowsheet. Post-acquisition, Yamana undertook several studies to evaluate these opportunities and to provide a level of confidence and accuracy to support Yamana's standards for feasibility studies and work. The results of these studies confirm the opportunities identified during the diligence process and provide for improved processing, production, cash flow and economics. As a result of the feasibility study update, Yamana announced a positive development decision on July 19, 2021.

The report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (N.I. 43-101) and in accordance with the requirements of Form 43-101 F1.

The responsibilities of the engineering and geological consultants and firms who are providing qualified persons are as follows:

- Ausenco was contracted by Yamana to manage and coordinate the work related to the report. Ausenco also developed the feasibility-level design and cost estimate for the process plant, general site infrastructure, and site water management infrastructure.

- InnovExplo Inc. (InnovExplo) was contracted by Yamana to complete the mineral resource estimate and the mineral reserve estimate for the project, and to design the underground mine plan, mine production schedule, and mine capital and operating costs.

- BBA Consultants (BBA) was contracted by Yamana to develop the feasibility-level design and cost estimate for the dry stack tailings storage facility.

The feasibility study is based only on mineral reserves within the Wasamac block and does not consider mineralization in the other blocks (Francoeur etc.). The report includes a summary of the consolidated land package but is focused on Wasamac.

1.2 Property Description and Ownership

The Wasamac property is located in the Abitibi-Témiscamingue administrative region of the Province of Québec, Canada, approximately 15 km west-southwest of the city of Rouyn-Noranda. The property covers an area of 10,268.56 ha, extending 20 km east-west and 15 km north-south. The coordinates of the approximate centre of the property are 48°12'08"N latitude, 79°14'30"W longitude, which corresponds to 630649E and 5340263N using NAD 83, Zone 17 UTM coordinates. The property underlies parts of Beauchastel and Dasserat townships.

The property is subdivided into six claim blocks: the Wasamac Block, Wasamac NE Block, Teck JV Block, R.M. Nickel Block, Consolidated Francoeur Block and Western Buff Block, which together comprise 6 mining concessions ("CM"), 281 mineral claims ("CDC", "CL", "CLD") and five mining leases ("BM"), for a total of 292 mineral titles.

The Issuer holds 100% ownership of the mineral titles for the property, except for the Teck JV Block in which the Issuer holds 60% ownership of five claims

On January 21, 2021, Yamana completed the acquisition of the Wasamac Block hosting the Wasamac deposit, the R.M. Nickel Block, the Teck JV Block, the Camflo property and the Camflo mill through the acquisition of all of the outstanding shares of Monarch not owned by Yamana. Yamana had previously announced that it had entered into a definitive agreement with Monarch on November 2, 2020 to acquire the properties under a plan of arrangement (the "Transaction"). Under the terms of the Transaction, Monarch shareholders received the following per Monarch share: 0.0376 of a Yamana share; C$0.192 in cash; and 0.2 of a share of Monarch Mining. Yamana issued 11,608,195 Yamana shares, paid US$46.9 million (C$59.3 million) in cash, and issued 383,764 replacement warrants, for total consideration paid of US$108.6 million. Yamana's consideration on close represented a value paid for the Wasamac asset of under US$67 per ounce of mineral reserves and under US$42 per ounce of mineral resources, based on mineral reserve estimate and mineral resource estimate in the 2018 Feasibility Study (Caumartin et al., 2018) and net of Yamana's existing Monarch interest in Wasamac. In connection with the Transaction, Monarch completed a spin-out to its shareholders of its other mineral properties and certain other assets and liabilities through a newly formed company, Monarch Mining Corporation (Monarch Mining). Yamana also acquired 6.7% of the outstanding shares of the newly formed Monarch Mining as part of the Transaction.

On June 14, 2021, Yamana announced that it agreed to acquire from Globex Mining Enterprises Inc. (Globex) the Francoeur, Arntfield and Lac Fortune gold properties (the "Consolidated Francoeur Block") adjoining the Wasamac Project, the Western Buff Block, as well as additional claims in the Beauchastel Township to the east of the Wasamac Project (the "Wasamac NE Block") (the "Agreement").

Pursuant to the terms of the Agreement, Yamana will pay an initial amount of C$4 million on closing, which at the direction of Globex will be paid in shares, with the remaining payment of C$11 million payable over four years in either cash or shares at the discretion of Globex. In addition, Globex will receive a 2% gross metal royalty from Yamana, of which 0.5% may be bought back at any time by Yamana for C$1.5 million following which the royalty would be reduced to a 1.5% gross metal royalty. The Globex gross metal royalty does not affect claims located on the Wasamac deposit.

The Transaction also included certain other claims in Malartic township adjacent to the Camflo property, which Yamana previously acquired and has transferred to the Canadian Malartic General Partnership (equally owned by Yamana and Agnico Eagle Mines Limited). These additional claims will be made available to the partnership. The closing of the Transaction took place on June 22, 2021.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The property is located approximately 15 km west-southwest of the city of Rouyn-Noranda, which is serviced by daily flights to Montreal. The property is accessed year-round from Provincial Highway 117 (the Trans-Canada Highway) that links Rouyn-Noranda and the community of Arntfield. A secondary road (rang Jacques-Paquin) leads directly to the Wasamac plant site area from Provincial Highway 117, crossing the Ontario Northland Railway close to the junction with the 117.

The climate of the Wasamac project area is marked by cold, dry winters and warm, humid summers. The temperature drops below freezing point an average of 204.9 days per year. Snow accumulates from mid-October to early- to mid-May. Freeze-up usually occurs in late December and break-up in March-April. Average annual precipitation indicates a mean rainfall of 918.4 mm, with the highest precipitation occurring in July.

The area is well served by existing infrastructure and human resources from Rouyn-Noranda, a well-established mining centre. Many suppliers and manufacturers in the mining industry are based in Rouyn-Noranda and other nearby communities. Skilled administrative personnel, technicians, geologists, mining engineers and experienced miners are available in the area. Hydro-Québec supplies electricity to the area.

The regional landscape is typical of the Abitibi Lowlands, with small rolling hills and widespread wetlands and swamps, and mixed broadleaf and conifer forests. The topography is relatively flat (average altitude is around 285 to 300 masl) with the exception of local areas of exposed outcrops, such as Mont Kekeko to the south and the northeastern part of the property.

1.4 History

Gold mineralization was first discovered on the property at Fortuna Lake in 1906 by prospectors Alphonse Ollier and Auguste Renaud.

Gold mineralization was originally discovered on the Wasamac Block in 1936 by Mine d'Or Champlain through surface trenching. Subsequent surface diamond-drilling intersected encouraging gold values but geological continuity seemed erratic. A 60-metre shaft (Wildcat shaft) was sunk, and one underground level was developed. Exploration and mining associated with the Wasamac deposit can be grouped into the following periods:

- 1936–1965: Initial gold discoveries and early exploration work (including historical drilling and trench sampling) by Mine d'Or Champlain, Wasa Lake Gold Mines, and Wasamac Mines Ltd.
- 1965–1971: Approximately 1.9 million tonnes of ore from the Wasamac deposit were treated by Wasamac Mines Ltd. and later by Wright-Hargreaves Mines Ltd. An average recovered grade of 4.16 g/t Au was recorded. Exploration work was also completed.
- 1971–1986: Exploration work (including drilling and ground geophysical surveys) and pre-feasibility work on surface pillars by Lac Minerals.
- 1986–2016: Exploration work, rehabilitation and reclamation of the Wasamac mine site by Richmont Mines Ltd. (Richmont). Pre-feasibility Study Report released in 2012.
- 2016–2019: 2018 Feasibility Study released by Monarch.

Mine production at the Wasamac mine was only between 1965 and 1971, approximately 254 000 oz Au (approximately 1.9 Mt at 4.16 g/t Au) were extracted.

At the Consolidated Francoeur Block, staking in the Francoeur area was done for the first time in 1923 following a gold discovery that later became Zone No.1. In 1932, Francoeur Gold Mines Ltd. sunk a 45° inclined shaft (No. 1) in the footwall of Zone No. 1. The Francoeur exploration and mining history can be grouped into the following periods:

- 1923 to 1938: Initial gold discoveries, early exploration work (including historical drilling and trench sampling) and preparation for commercial production by Francoeur Gold Mines Ltd.
- 1938 to 1947: Exploration and production of Francoeur mine by Francoeur Gold Mines Ltd.
- 1964 to 1968: Drilling and sinking of the Wasamac No. 2 vertical shaft (now called shaft No. 6).

- 1968 to 1971: Exploration and production of Francoeur mine by Wright-Hargreaves Mines Ltd.1971 to 1985: Exploration and underground drilling by Kerr Addison Mines Ltd., Noranda Exploration Company Ltd. and Long Lac Exploration Ltd.

- 1985 to 1991: Dewatering and rehabilitation of surface and underground mine facilities and levels. Underground drilling and extraction of a bulk sample by Ressources Minières Rouyn (RMR, now Richmont).

- 1991 to 2001: Exploration and production at Francoeur mine by RMR.

- 2001 to 2012: Exploration drilling programs focusing on the West Zone, internal resource calculations and N.I. 43-101 reports, dewatering of the mine and rehabilitation of underground levels for exploration drilling by Richmont.

- 2015 to 2019: Surface exploration work (mapping, sampling, trenching, geophysical surveys) by Globex.

During these three periods of mine production at the Francoeur mine, a total of 508,642 oz Au (2.60 Mt at 6.1 g/t Au) were extracted.

In addition, within the limits of the Consolidated Francoeur Block, the Arntfield mine, which comprises the No. 1, No. 2, and No. 3 deposits, produced 480,804 tonnes grading 3.98 g/t Au and 0.93 g/t Ag from 1935-1942.

1.5 Geology and Mineralization

The Wasamac property is underlain by the Blake River Group within the Rouyn-Noranda mining district in the southern Abitibi Greenstone Belt (Abitibi greenstone belt) of the Superior Province of the Canadian Shield. The southern boundary of the Abitibi greenstone belt is along the Cadillac–Larder Lake Fault Zone, a major structural break marking the contact with younger metasedimentary rocks of the Pontiac Subprovince.

The Wasamac, Consolidated Francoeur, and Western Buff blocks are along the Francoeur-Wasa Shear Zone, a second-order fault that is parallel and 2.5 km north of the Cadillac–Larder Lake fault zone; whereas the R.M. Nickel area is 1.6 km north of the Francoeur-Wasa shear zone. In the Wasamac deposit, the Francoeur-Wasa shear zone crosscuts the meta-volcanic units of the Blake River Group. Along this break, the units of the Blake River Group have been tilted toward the North and follow the same east-west trend. The Francoeur-Wasa shear zone shows strong structural similarities with the Cadillac–Larder Lake fault zone, presenting a thick intense ductile shearing of the volcanic units, associated with strong metasomatic alteration.

Gold mineralization at Wasamac is typically associated with finely disseminated pyrite and stockwork of pyrite-rich micro-veinlets hosted in albite-sericite-ankerite alteration zones confined within the shear zone. The albite-sericite-ankerite alteration related to gold mineralization is typically beige-brown and visually distinguishable from the surrounding sheared rocks. Quartz veins are not common and do not significantly contribute to the gold endowment of the system.

The defined Wasamac deposit is continuous over 900 metres vertically and 2.7 kilometres along strike and remains open at depth and on its lateral extensions. The deposit contains five mineralized areas, from west to east: Main Area, Area 1-2, Area 3-4, Wildcat Zone, and MacWin Zone. Areas 1-2, 3-4, MacWin Zone, and most of the Main Area are contained within the Francoeur-Wasa shear zone. The Wildcat Zone occurs within a different structure.

The Main Area is located on the western part of the deposit where it is constrained to the west by the Horne Creek Fault. The upper part of the Main Area was mined during historic underground operations between 1965 and 1971. Gold mineralization in the Main Area occurs within several discrete sub-parallel zones that are generally 5 to 15 m (locally up to 25 m) true thickness. They include the Main Zone, Main Zone 2, Main Zone 3, Stockwork Zone, Footwall Zone, and Footwall Zone 2. The Main Zone, Main Zone 2, Main Zone 3 are confined to the Francoeur-Wasa shear zone and are associated with disseminated pyrite and stockwork of pyrite-rich micro-veinlets hosted in albite-sericite-ankerite alteration zones.

The Stockwork Zone, Footwall Zone, and Footwall Zone 2 occur below the Francoeur-Wasa shear zone and are primarily associated with relatively undeformed pyrite stockwork with minor albite alteration. They are locally associated with higher gold grades.

Areas 1-2 and 3-4 both contain one single mineralized zone characterized by albite-sericite-ankerite-pyrite alteration. It consists of a continuous structure interpreted as the eastern extension of the Main Zone. Area 3-4 is slightly offset relative to Area 1-2, which is interpreted to be related to a crosscutting structure that offset the shear zone and stratigraphy. Historically, the zone in Area 1-2 and Area 3-4 was separated into four zones (Zone 1, Zone 2, Zone 3, and Zone 4).

A small portion of Area 1-2 was mined during the last phase of production; however, only limited tonnage was extracted (approximately 100,000 tonnes of ore was mined). Area 3-4 was first intersected during the 2002-2004 drilling programs and was better defined during the 2011 drilling.

The MacWin Zone, formerly known as the Wingate Zone, was discovered in 1945 along the Francoeur-Wasa shear zone approximately 300 m east of Zone 3. Gold mineralization occurs both within the shear zone and in the hanging wall rhyolite. A small shaft was completed on the zone; however, this area was included in the mineral resources, but voluntary excluded from the mineral reserves defined in the present report.

1.6 Deposit Types

The Wasamac deposit is an example of an Archean greenstone belt gold deposit hosted by the Francoeur-Wasa shear zone, a second-order brittle-ductile shear zone of the Cadillac–Larder Lake fault zone. Gold mineralization is constrained to the altered and sheared part of the Francoeur-Wasa shear zone. Regionally, the Wasamac deposit lies at the boundary between the orogenic gold district of Noranda and the dominantly intrusion-related gold systems of Kirkland Lake. The Wasamac deposit shares similarities characteristics of both alkaline syenite intrusion-related gold deposits and orogenic gold deposits.

1.7 Status of Exploration and Drilling

An ongoing drilling program by Yamana is in progress at the time of writing. No final results are available for the mineral resources update presented in this report. The issuer has not completed any other drilling since acquiring the property.

Drilling is currently being conducted for further rock mechanic studies and exploration purposes, and other drilling campaigns are at the planning and/or permitting stage. The results from the ongoing and proposed campaigns will dictate future approaches. Ongoing and preliminary plans are in line with recommendations in Chapter 26 and include 120,000 metres of drilling in 2021 and 2022. The main objective of future drilling is infill drilling to better delineate areas expected to be developed in the first three years of production is expected to include 30,000 metres in 2021, with a further 38,000 metres in 2022 to provide further delineation of the remaining mineral resources. This work is expected to increase confidence in grade, improve mine planning, and provide further geotechnical and metallurgical data.

Exploration on the broader property is expected to include 10,000 metres in 2021 in an effort to delineate secondary zones (such as Wildcat) and test high-priority extensions of the Francoeur-Wasa shear zone.

1.8 Sample Preparation, Analyses and Security

From 2002 to 2016, core logging was performed by Richmont geological staff using industry standard procedures. For the drill campaigns from 2010 to 2012, 2015 and 2016, samples were sent to Laboratoire Expert Inc. The QA/QC program consisted of the insertion of CRM and blank samples into the sample stream for every 20 samples, and check assays at a secondary umpire laboratory for analysis of pulps and/or coarse rejects.

It is the qualified person's opinion that the procedures followed on the Wasamac Project conform to the industry practices and the quality of the assay data is adequate and acceptable to support a mineral resource and mineral reserve estimation.

1.9 Mineral Processing and Metallurgical Testing

Historical testwork data from the previous feasibility study in 2018 and other testwork data from 2012 to 2013 was heavily relied upon in this feasibility study update. Ausenco managed metallurgical testwork in the following areas as part of the feasibility study update:

- whole ore leach
- flotation with flotation concentrate and tailings leach
- oxygen uptake testing
- SMC test feed size analysis

Although the 2018 testwork was sufficient for a feasibility level of study, Ausenco conducted additional testwork in 2021 to evaluate opportunities for improved recoveries, reduced capital cost, and reduced operating cost. As a result of this testwork, the changes described below were made.

Leaching testwork showed that a leach residence time of 35 hours is sufficient for the Wasamac ore, compared to 48 hours in the 2018 feasibility study. However, additional testwork may optimize and reduce leach retention time. The testing program should include testing to better define the optimum leaching time by conducting fixed duration leach tests for 8, 12, 16, 24 and 36 hours on representative samples. Furthermore, previous leach testwork results were analyzed along with the 2021 results summarized here to determine the optimum leach conditions:

- grind size K_{80} of 60 µm
- sodium cyanide (NaCN) addition rate of 0.6 kg/t of leach feed (design)
- CaO addition rate of 1.0 kg/t of leach feed (design) added to SAG mill feed and maintained at pH 10.5-11.0 in the leach circuit

Oxygen uptake testing showed the samples to have moderate to high oxygen demand. However, additional testwork is recommended on a wider range of samples to confirm oxygen supply requirements.

Feed size analysis on the SMC tests results showed that a less conservative Axb value of 39.3 can be used for grinding circuit design, compared to 32.2 in the 2018 feasibility study. This allows for the implementation of a smaller SAG mill in the grinding circuit. There is further opportunity to confirm the grinding testwork and circuit sizing in the future.

Additional testwork validated the previous recovery assumptions and further testwork is recommended to better understand the underlying reasons for the low recoveries from Main Zone Central and Main Zone East (formerly Zone 1 and Zone 2), and evaluate opportunities to increase recoveries in these zones, such as flotation. Additional samples should be collected to represent grade ranges, and spatial and lithological variability in these zones specifically. Incorporating diagnostic leach test protocols may assist with determining the underlying poor recoveries.

1.10 Mineral Resource Estimate

The mineral resource estimate for the Wasamac deposit presented in this report is based on Yamana's independently validated internal mineral resource estimate and is effective as of June 30, 2021. The QPs from InnovExplo audited and validated Yamana's internal mineral resource estimate.

The main steps in the methodology were as follows:

- compile and validate the databases used for the 3D modelling and for the mineral resource estimation
- ensure availability and accuracy of the void model (inclusions of historical underground openings), assess the use of historical information versus more recent drilling information
- validate the 3D lithological model and interpretation of the mineralized zones based on lithological units, alteration zones, mineralization, structural information, gold assay values and the general geometry of historical stopes
- validate and review the drill hole intercepts database, conduct independent sensitivities on composite length and approaches, conduct independent sensitivities on capping values and the resulting estimation databases for the purposes of geostatistical analysis and variography including independent tests in Snowden Supervisor
- review and validate all key assumptions, including the requirement for additional in-situ density measurements for supporting tonnage estimates
- validate the block model key parameters, interpolation methods, strategies and parameters, conduct independent sensitivities with other interpolation methods in Leapfrog Edge and check Datamine final grade interpolation results
- establish classification criteria, produce clipping areas for mineral resource classification and apply them to the block model
- assess the mineral resources with "reasonable prospects for potential economic extraction" by selecting the appropriate cut-off grades and produce "resources-level" optimized underground mineable shapes
- ensure adequate subtraction of the historical voids and exclusion of the mineral reserves (from this study) from the mineral resource estimate
- generate the final mineral resource estimate

Following the validation, the qualified person is of the opinion that the mineral resource estimate can be categorized as indicated and inferred mineral resources. The qualified person considers the mineral resource estimate reliable based on quality data, reasonable hypotheses and parameters that follow CIM Best Practices Guidelines (2019). The mineral resource estimate also conforms to the CIM Definition Standards on Mineral Resources and Reserves (2014). Table 1-1 provides an estimate of mineral resources for the Wasamac deposit as of June 30, 2021.

Table 1-1: Wasamac Statement of Mineral Resources as of June 30, 2021 (Exclusive of Mineral Reserves)

Category	Tonnage (kt)	Grade Au (g/t)	Contained Gold (koz)
Indicated	5,769	1.76	326
Inferred	3,984	2.01	258

Notes: 1. The qualified persons for the current mineral resource estimates are Mr. Vincent Nadeau-Benoit, P.Geo. and Alain Carrier, M.Sc., P.Geo. (InnovExplo). Mineral resources have been estimated by Yamana and independently audited and validated by InnovExplo. The mineral resource estimate conforms to the 2014 CIM Definition Standards on Mineral Resources and Reserves and follows 2019 CIM definitions and guidelines for mineral resources and are reported exclusive of mineral reserves. 2. The mineral resource estimate has an effective date of June 30, 2021. 3. Mineral resources were evaluated using the ordinary kriging weighting algorithm informed by capped composites and constrained by three-dimensional mineralization wireframes. Mineral resource categories were assigned using clipping boundaries. Indicated category resources were established for blocks interpolated during the first two passes within 40 m closest distance from a drill hole composite within the same mineralized zone. Inferred category resources were established for the remaining interpolated blocks inside the mineralization wireframes. A bulk density of 2.80 g/cm3 was used to convert volume to tonnage. 4. Cut-off grades, which corresponds to 75% of the cut-off grades used to estimate the mineral reserves, are variable based on the metallurgical recoveries and range from 1.10 to 1.30 g/t Au. 5. Mineral resources are below a 32 m surface crown pillar and outside a 5 m minimum buffer around historical underground infrastructures and constrained by potentially mineable shapes based on a minimum mining width of 2 m and considering internal waste and dilution. 6. All figures are rounded to reflect the relative accuracy of the estimate. Sum totals may not add up due to rounding.

The qualified persons responsible for this section of the technical report are not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that could materially affect the mineral resource estimate.

1.11 Mineral Reserve Estimate

Mineral reserves were classified in compliance with the CIM "Definition Standards for Mineral Resources and Mineral Reserves". As such, the mineral reserves are based on measured and indicated mineral resources and do not include any inferred mineral resources. Measured and indicated mineral resources are inclusive of proven and probable mineral reserves. Mineral reserves are the estimated tonnage and grade of ore that is considered economically viable for extraction. Mineral reserves for the Wasamac deposit incorporate dilution and mining recovery factors based on the selected mining method and design. Also, economic analyses were completed to validate the profitability of particular areas of the mineral reserves.

The mineral resource block model was used as the basis for estimating the mineable tonnage considered in the mine plan. Cut-off grades for the different mining areas were first estimated, then the stope shapes were optimized according to various parameters, such as geometry and dilution. The final mineral reserve estimate was obtained after completing the stope and underground mine designs, including the economic validation and considering additional factors, such as mine recovery.

Table 1-2 provides an estimate of mineral reserves for the Wasamac deposit as of June 30, 2021.

Table 1-2: Wasamac Estimate of Mineral Reserves as of June 30, 2021(1)

Category	Tonnage (kt)	Grade Au (g/t)	Contained Gold (koz)
Probable	23,168	2.56	1,910

Notes: 1. The QPs for the mineral reserve estimate are Mr. Denis Gourde, P.Eng. and Sébastien Tanguay, P.Eng. (InnovExplo). The mineral reserve estimate conforms to the 2014 CIM Definition Standards on Mineral Resources and Reserves and follows 2019 CIM definitions and guidelines. 2. Mineral reserve estimate has an effective date of June 30, 2021. 3. The metallurgical recoveries varies with the metallurgical domain: 92.0% for the eastern part of the main zone; 81.6% for the central part of the main zone; 86.2% for the western part of the main zone; 92.7% for zones 3 and 4. 4. Estimated at US$1,250/oz Au using an exchange rate of US$1.32:C$1.00, variable cut-off value related to the metallurgical domain: 1.45 g/t for the eastern part of the main zone; 1.68 g/t Au for the central part of the main zone; 1.63 g/t for the western part of the main zone; 1.62 g/t for zones 3 and 4. 5. Mineral reserve tonnage and mined metal have been rounded to reflect the accuracy of the estimate and numbers may not add due to rounding. 6. The total refining, processing, mining, tailing management and general and administration cost is estimated at 63.21$/t. 7. Mineral reserves include both internal and external dilution. The internal mining dilution varies with the metallurgical domain: 11.9% for the eastern part of the main zone; 15.1% for the central part of the main zone; 17.4% for the western part of the main zone; 25.9% for zones 3 and 4. The external dilution is estimated to be 11%. the average dilution of the project is estimated to be 13%. 8. The estimated mining recoveries for the site range from 86% for the sills to 95% for the stopes. The average mining recovery factor was set at 93.6% to account for mineralized material left in each block in the margins of the deposit. For the purpose of the COG calculation, mine recovery was set at 95%. 9. The qualified person responsible for this section of the technical report is not aware of any mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the mineral reserve estimate.

1.12 Mining Method

The current project is focused below and east of the historical Wasamac mine. Geomechanical engineering design criteria were developed to support the underground mining production sequence and design. The geomechanical assessments for the project evaluated rock mass domains and characterization, ground support requirements, anticipated rock behaviour, stope sizing and dilution, backfill strength assessment, and hydrogeology.

The proposed mining method for Wasamac comprises long-hole stoping optimized for the local mineralization width and dip. Approximately 75% of stopes are planned to be mined in a longitudinal direction and 25% in a transverse direction. The transverse stoping is mostly concentrated in mining area E29, the widest zone of the mine. This approach results in average external dilution of less than 11%. Level spacing is set to 25 metres, whereas stope spans are variable depending on geometry and local ground conditions. Stopes will be filled using a combination of paste fill (80%), delivered from an underground paste fill plant, cemented rock fill (2%), and rock fill (18%).

To reduce the project's surface footprint, most mining infrastructures will be installed underground. Additionally, maximum tailings and waste rock will be deposited underground as paste fill and rockfill. A full paste fill plant is also planned for underground, using cement delivered from a surface borehole. A main hub located underground will contain the main crusher and an underground conveyor to surface. This setup will ensure ore output from the mine, while minimizing equipment activities on surface. The optimized material handling system utilizes an ore pass and 60-tonne haul trucks to transport ore from the production levels to a central underground primary crusher. Waste material will be hauled by the 60-tonne trucks to closest backfilling activities or to a waste pad at surface via the service ramp. The haul trucks will be automated to allow haulage to continue between shifts.

The underground mine will be accessed by two main ramps starting north of Highway 117, close to the mill. One of these ramps will house a 3-km-long conveyor; the other will be used for equipment, personnel, and services.

Modern equipment and methodologies will be used to optimize the drill and blast patterns and to minimize ground vibration. To minimize additional development, uppers are drilled when required most notably at the apex of the zone, and for sill pillar recovery to avoid development only required for drilling purposes.

Fresh air will be supplied to the mine through two ventilation raises by high-efficiency fans installed underground. The two access ramps will serve as exhaust and an internal network of raises and regulators will be used to supply fresh air to every level. Heating systems will be installed to prevent freezing during winter months. Before development of the main intake raises, ventilation will be provided from the conveyor ramp and optimized by the creation of ventilation loops connected to the service ramp.

Total primary and secondary underground development is 107 km, considering 2.3 km of raisebore. The ratio of ore tonnes to development metre is 221:1. Underground development is schedule to start on October 1, 2024. The first development in the ore is projected during the last quarter of 2026. The commercial production period is slated to start in Q4 2027 when the mine reaches 7,000 t/d. To reach this objective, an average of 8,700 m of horizontal development per year, with a maximum of 15,800 m in 2027, should be completed. During the pre-production period, major infrastructure like the services bay, underground crusher, paste plant and main raises are to be excavated and all associated equipment installed and commissioned.

The life-of-mine plan shows a rapid ramp-up in production in the first year, with production rising to approximately 200,000 ounces after mill recovery per year for the subsequent four years. Average gold production is expected to be 169,000 ounces after mill recovery per year over the life of mine of 10 years. Based on current mineral reserves, Wasamac has a mine life to October 2036, but potential conversion of mineral resources and exploration potential could possibly extend the mine life.

The former Wasamac mine will be fully dewatered before the end of 2025 utilizing the old shaft, so as not to interfere with project development. Two water samples were taken in June of 2021 to begin the characterization of the historic void water. The majority of constituents of concern based on analysis of the two void water samples occurred at concentrations that were lower than compliance levels; therefore, traditional treatment methods (dosing, reverse osmosis, ion exchange) are not needed provided the water chemistry data on which this analysis is based provides an accurate characterization of the void water.

Permanent project dewatering will be conducted with a system of drain holes and submersible pumps. Muddy water is treated by a mudwizard system.

The mine will operate seven days per week, night and day. A maximum of 71 pieces of mobile equipment will be bought by Yamana and used underground (includes seven 63-tonne trucks, ten production LHDs, four jumbos and three production drills). Up to 307 personnel will be employed by Yamana for the underground mine, including 216 in operations, 66 in maintenance, and 25 in technical services. To maximize the efficiency of main equipment and production activities, automation is planned between shifts from surface.

1.13 Recovery Methods

The process plant was designed using conventional processing unit operations to treat up to 7,500 t/d (2.74 Mt/a) based on an availability of 8,059 hours per annum or 92%. The crushing plant section design is set at 70% availability and the gold room availability is set at 52 weeks per year. The plant will operate two shifts per day, 365 days per year, and will produce doré bars. The life-of-mine plan used for the feasibility study assumes a processing rate of 7,000 t/d, representing potential production upside.

The plant feed will be hauled underground from the mine to the underground primary crushing facility (that includes a jaw crusher) and then to surface where it will be stockpiled. The crushed ore will be ground by a SAG mill in closed circuit with a pebble crusher followed by a closed-circuit ball mill with hydrocyclone classification. The hydrocyclone overflow with a final grind size of 80% passing 60 µm will flow to a thickener to increase the slurry density before the conventional leach and carbon-in-pulp (CIP) circuit.

Gold and silver adsorbed in the CIP circuit will be recovered onto activated carbon and eluted using a pressure Zadra elution circuit. It will then be recovered by electrowinning in the gold room. The gold-silver precipitate will be dried in an oven and mixed with fluxes and smelted in a furnace to pour gold doré bars.

Carbon will be reactivated in a carbon regeneration kiln before being returned to the CIP circuit. CIP tails slurry will be treated in cyanide destruction using an SO2/O2 circuit with sodium metabisulphite (SMBS) before reporting to a final tails thickener.

The tails thickener underflow can be pumped to either the filtration plant or paste plant, depending on mine demand for paste. The filtered tailings from the filtration plant will be dry stacked.

Reagents will include pebble lime, sodium cyanide, hydrated lime, oxygen, sodium hydroxide, copper sulphate, hydrochloric acid, sodium metabisulphite, activated carbon and flocculant.

Compared to the 2018 feasibility study, the following changes to the flowsheet have been implemented:

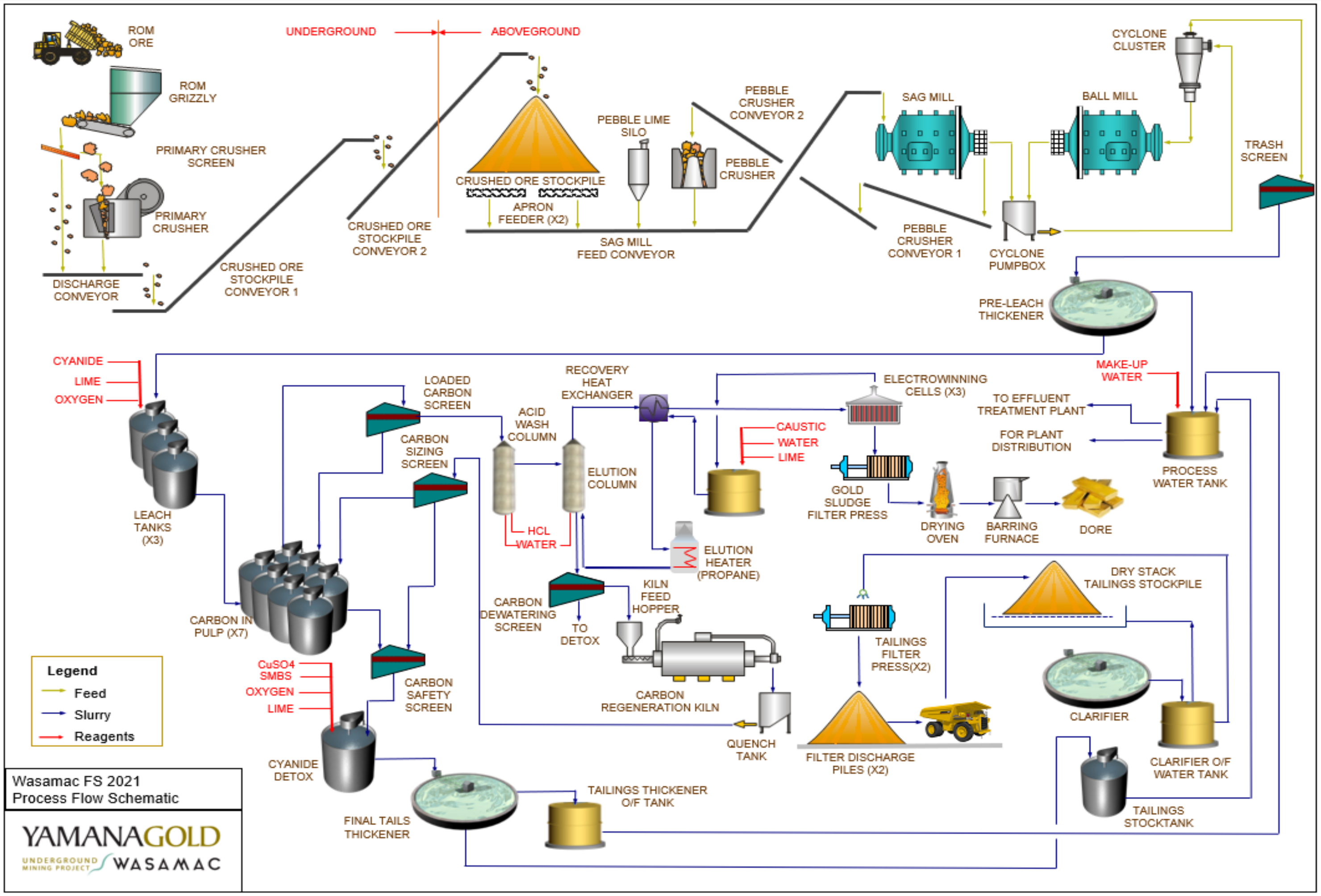
- The primary crusher has been moved underground for noise and dust mitigation for the surrounding residents.
- The quantity of leach tanks has decreased from five to three.
- The eight pumpcell CIP tanks have been decreased to seven conventional CIP tanks.
- The pre-detoxification thickener has been removed.
- The quantity of detoxification tanks has decreased from two to one.

Key process design criteria listed in Table 1-3 were derived from testwork. Figure 1-1 presents the process flow diagram for the plant.

Table 1-3: Key Design Criteria

Design Parameter	Units	Value
Plant Throughput, Design	t/d	7,500
Gold Head Grade – Maximum for Design	g/t Au	3.67
Crushing Plant Availability	%	70
Mill Availability	%	92
Filter Plant Availability	%	82.5
Material Specific Gravity	-	2.80
Bond Crusher Work Index (CWi), 75th percentile	kWh/t	19.5
Bond Rod Mill Work Index (BWi), 75th percentile	kWh/t	16.9
Bond Ball Mill Work Index (BWi), 75th percentile	kWh/t	15.0
SMC Axb, 25^{th} percentile	-	35.9
Bond Abrasion Index (Ai)	g	0.243
Primary Crusher		jaw crusher, 1150 mm x 760 mm
SAG Mill Dimensions		7.6 m dia. X 4.0 m EGL
SAG Mill Installed Power	MW	4.1, with VSD
Ball Mill Dimensions		6.1 m dia. X 8.8 m EGL
Ball Mill Installed Power	MW	6.0
Primary Grind size (P_{80})	µm	60
Leach + CIP Residence Time	h	35
Leach Extraction	% Au	90
Leach-CIP Operating Density	%wt	50
Leach Sodium Cyanide Addition	kg/t	0.6
Leach Hydrated Lime Addition	kg/t	1.0
Leach pH target	-	10.5
Leach Dissolved Oxygen Target	ppm	20
Leach & CIP Tanks	no.	3 + 7
Tonnes of Carbon per Elution Column	t	7.0
Loaded Carbon Grade, Average	g/t Au	2970
Detoxification Residence Time	min	70
Detoxification Tanks	#	1
Detoxification Feed CN_{WAD} Concentration, Maximum (Design)	mg/L	150
Detoxification Discharge CN_{WAD} Concentration, Design	mg/L	<10
Detoxification SO_2 addition	SO_2:CN_{WAD} ratio	5.0
Detoxification lime addition	$Ca(OH)_2$:SO_2	1.0
Final Tails Thickener Underflow Density	%wt	63
Tailings Filtration – Type		horizontal fast-opening, 62 to 74 chambers, 3.5 m x 2.5 m plates, filter cloth
Filter Feed Pulp Density	%wt	63
Filter Cake Moisture	%wt H_2O	10-12

Figure 1-1: Overall Process Flow Diagram

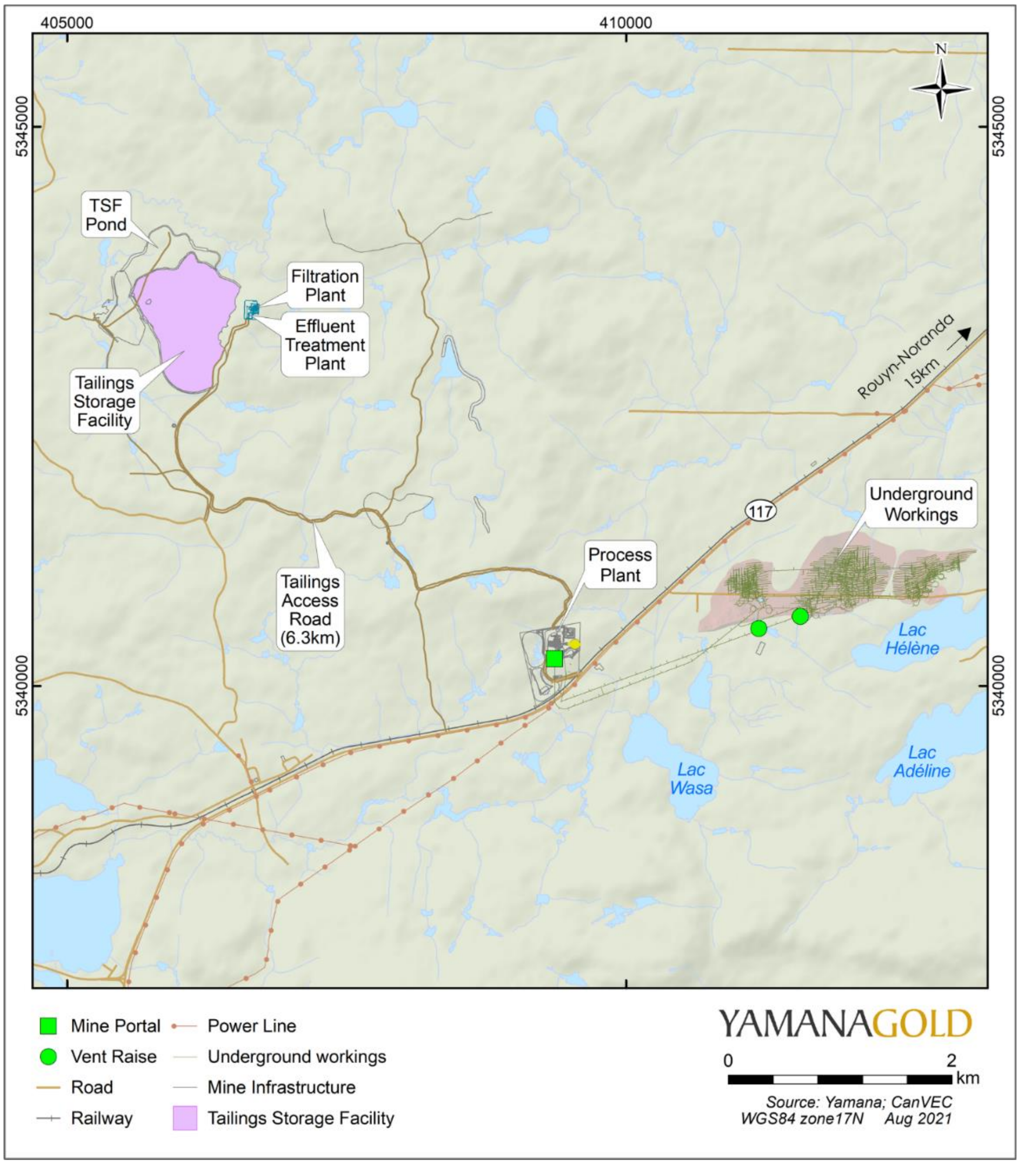


Source: Ausenco, 2021

1.14 Infrastructure

The overall site plan in Figure 1-2 shows the major surface project facilities including the access road, process plant, waste rock storage facility, dry stack tailings storage facility (TSF), the TSF pond, other infrastructure for tailings management, the mill basin, TSF, the ore stockpile and the effluent treatment plant. Support facilities also displayed include the gold room, the assay and metallurgical laboratory, truckshop, the maintenance shop and warehouse, the office complex, and the security gatehouse.

Figure 1-2: Overall Site Plan



Source: Yamana, 2021

1.14.1 Access

The property is located approximately 15 km west-southwest of the city of Rouyn-Noranda, which is serviced by daily flights to Montreal. The property is accessed year-round from Provincial Highway 117 (the Trans-Canada Highway) that links Rouyn-Noranda and the community of Arntfield. A secondary road (rang Jacques-Paquin) leads directly to the Wasamac deposit (above the underground mine) approximately 1 km east-northeast of the main project site along Highway 117.

New access roads will be built from rang Jacques-Paquin to the main plant site (520 m), and from the main plant site to the tailings storage facility and filtration plant (6.3 km).

1.14.2 Utility Power Supply

Primary power will be supplied to the Wasamac site by the Hydro-Quebec utility via a 120 kV overhead transmission line that terminates at the plant's 120 kV outdoor substation. This overhead line will be constructed by the Hydro-Quebec utility; however, discussions are ongoing, so the details of the utility terminal substation, faults levels, and line model are yet to be determined.

1.14.3 Tailings Storage Facility

The Wasamac project will host a dry stack tailings storage facility (TSF) situated 6.3 km northwest of the process plant, proximal to the filtration plant. The TSF is designed to receive 60% of the total production of tailings. The ultimate capacity of the facility is estimated to be approximately 8,570,280 m^3 (14.1Mt) of dry stack tailings with an additional surface layer of approximately 518,779 m^3 (1.04 Mt) of development waste rock. As the stack deposition advances, the TSF will be progressively reclaimed with the waste rock to reduce dust and to return the landscape to a natural state as quickly as possible.

Stability analysis has been performed in both static and pseudo-static conditions for three critical sections of the Wasamac TSF. It was assumed the TSF would be constructed in stages to allow the clay foundation to gain strength through consolidation of the clay. Staged construction allows the use of the "stress history and normalized soil engineering properties" (SHANSEP) method for stability analysis. The obtained factor of safety for the assumed degree of consolidation (40%) during the staged construction shows that the satiability of the TSF in proposed configurations meets the design criteria specified in MERN (2017) and Directive 019 (MDDEP 2012); however, the validity of these assumptions needs to be addressed by more detailed geotechnical tests during detailed engineering. It is important to note that Yamana has planned additional geotechnical tests (CPT).

1.14.4 Waste Rock Storage Facility

Mine development material will be placed in a waste rock storage facility (WRSF) west of the mill and adjacent to the plant pad. The WRSF will be accessed by trucks across an intersection at the main site entrance, and will be traffic-controlled with boom gates.

Stacking of this facility will commence in Year -1 (pre-production) and be completed in Year 2 of operations. The facility is designed to accommodate 1.83 Mt (914,982 m^3) of waste rock. The current understanding from a limited acid base accounting (ABA) testing program is the waste rock is classified as non-acid-generating due to the ratio of the neutralizing potential verses the acid-generating potential of the minerals in the waste rock. However, testwork is underway to characterize the waste rock.

The stability of the WRSF was assessed using the limit-equilibrium modelling software SLIDE 2, (RocScience, 2021). Analyses were undertaken for both static and pseudo-static (earthquake loading) conditions with the calculated factors of safety (FOS) higher than the minimum required values in accordance with design of waste rock facility best practice. The WRSF stability exceeded both static and pseudo-static best practice guidelines.

1.14.5 Buildings

The buildings for the Wasamac Project consist of either pre-engineered or modular-type structures. Pre-engineered buildings will be supported by reinforced concrete footings with concrete slabs and pedestals, and fully enclosed with metal cladding. The process plant consists of two main buildings, the mill building (grinding/elution) and the reagents storage building. The plant is located west of the domed ore stockpile. The gold room is located immediately east of the reagents storage area.

The truck shop, washbay, maintenance shop, and warehouse are all pre-engineered and located south of the process plant. Modular buildings in the plant site area include the assay and metallurgy laboratory, the security and gatehouse, and the office complex. The office complex will be two storeys high and will consist of modules for the administration office, the mine office, the mill office, and the mine dry.

The filtration plant area located 6.3 km northwest of the main plant site consists of two pre-engineered buildings for the filtration plant and the maintenance and warehouse. A modular office and control room is located adjacent to the filtration plant. The effluent treatment plant is located south-southwest of the filtration plant and is also a pre-engineered building.

1.14.6 Water Management

Water management facilities of the Wasamac Project consist of two main ponds, the mill basin and the TSF ponds. The ponds are located southwest of the plant pad and west of the TSF, respectively. The ponds are designed to collect surface water runoff and seepage from the waste rock storage facility (mill basin) and the TSF. The mill basin water is pumped to the process plant for make-up water and the TSF pond water is pumped to the effluent treatment plant. The process plant pad includes collection ditches which ultimately report to the mill basin. Additional water management features for the site include berms, drainage ditches, and pumps.

Process water and contact water report to the effluent treatment plant (200 m^3/h capacity) located next to the filtration plant for treatment prior to discharge to the environment, west of the TSF pond.

1.15 Environmental Studies, Permitting and Social or Community Impact

The Wasamac property is contained within the Dense Softwood Stands and Light Hardwood Stands subzone of the Softwood Dominated Boreal Vegetation Zone. The project site is made up of mostly deciduous stands, but mixed stands and coniferous stands are also present. Deciduous stands are dominated by Trembling Aspen (*Populus tremuloides*) and trees of the genus Betula. Coniferous stands in the area are dominated by Balsam Fir (*Abies balsamea*) and Black Spruce (*Picea mariana*).

The Centre du patrimoine naturel du Québec (CPNDQ) indicated the presence of 16 fish species within the project area. During the 2019 Englobe inventory, the presence of seven of these species was confirmed between six lake and ten stream survey stations. Avifauna surveys in 2013 revealed the presence of 79 species, 26 of which were confirmed in the complementary 2019 inventory. Mammals present at the project site include both large, small, and micro mammals. The 2019 mammalian inventory efforts were primarily concentrated on chiropterans. In total, 31 bat calls were recorded.

The hydrography of the project site is characterized by the presence of several lakes, meandering streams and wetlands, as well as the absence of rivers. Most streams are affected by beaver activity, leading to the formation of ponds and, subsequently, wetlands. Flow speeds within streams are slow Major lakes within the area are Arnoux, Mackay, Adéline, Wasa, Chat Sauvage and Hélène. In terms of groundwater, elevated levels of naturally occurring iron and manganese are present along with mercury, lead, and selenium. Elevated levels of chlorine found in groundwater samples could be caused by the salt application along Rideau Boulevard (Highway 117). Calcium and sulphate levels were found to be higher in wells proximal to the old TSF. PAH exceedances also are likely to have anthropogenic origin.

A single soil quality characterization study was carried out by AECOM (2013). The A to B criteria (low to medium concentrations) of the MELCC policy was exceeded for concentrations of:

- arsenic at 2 stations
- cobalt at 10 stations
- copper at 22 stations
- nickel at 3 stations
- zinc at 2 stations

The B to C criteria range was exceeded for concentrations of molybdenum at 13 stations. Cadmium, mercury, lead and total cyanides did not exceed MELCC criteria at any sampling station.

There are sufficient tailings samples to adequately characterize the acidification risk of the tailings as 'low'; however, depending on the tailings material distribution in situ and local environmental conditions, there may be variable responses observed.

1.15.1 Closure and Reclamation Considerations

Under the *Mining Act*, anyone who engages in mining exploration work or mining operations determined by regulation must submit a rehabilitation and restoration plan (subsequently referred to as "closure plan") for approval by the Ministère de l'Énergie et des ressources naturelles (MERN). Approval is conditional upon a favourable opinion from the Ministère de l'Environnement et de la Lutte contre les changements climatiques (MELCC).

Yamana is currently preparing a conceptual closure plan and cost estimate for the Wasamac Project, concurrently with the EIA. The conceptual closure plan will meet the requirements of the guide and applicable legislation.

1.15.2 Permitting Considerations

Yamana relies on a collaborative approach to ensure the success of Wasamac. In this regard, its environmental assessment process is conducted in collaboration with neighbors and First Nations. A community relations office will soon be opening its doors to ensure constant dialogue and accessibility to the Yamana team as well as to information on the project. A campaign of baseline monitoring and testing is currently underway with the objective of completing the EIA submissions at the provincial and federal levels by the second quarter of 2022.

Yamana expects to receive all permits and certificates of authorization required for project construction by the third quarter of 2024.

The project is subject to the federal Environmental Assessment process. The project entered the planning phase with the submission of an Initial Project Description (IPD) on November 16, 2020. During the planning phase, multiple deliverables are produced, some by the proponent and some by the Impact Assessment Agency (the Agency), in consultation with other federal and provincial entities as well as with First Nations.

The Agency published the Notice of Impact Assessment Decision on November 26, 2020, which confirms that the project is subject to the Federal IA process as per section 16(1) of the Act; the Public Participation Plan, the Cooperation Plan, the Indigenous Engagement and Partnership Plan, and the Permitting Plan were released by the agency on March 26, 2021.

The planning phase ended with the publication of the Tailored Impact Statement Guidelines (TISG) document, which covers the content required for the Environmental Impact Statement (EIS). The TISG were published on March 26, 2021. The project is subject to the Provincial EIA and review procedure provided for in Subdivision 4 of Division II of Chapter IV of title I of the *Environmental Quality Act* (EQA), and must obtain an authorization from the provincial government.

Under the Provincial EIA process, a Project Notice for Wasamac was submitted on November 19, 2019.

In addition to this, the Wasamac Project was selected as a pilot project by the government, under the authority of an interministerial table composed of the five following ministries: Ministry of Energy and Natural Resources, Ministry of Forests, Wildlife and Parks, Minister of the MEFCC, Ministryr of Municipal Affairs and Housing, and the Ministry of Economy and Innovation. The primary objective of this initiative is to establish a viable interaction system with stakeholders and, in particular, to promote the social acceptability of mining projects. The first meeting took place on December 19, 2019. Sporadic follow-ups are made with the Ministère de l'Énergie et des ressources naturelles (MERN), leader of the table.

In addition to the Environmental Assessment processes, the Wasamac Project will need multiple permits at various instances such as a federal Fisheries Act authorization for impacts to fish habitat and a provincial Wetland Compensation plan for impacts to wetland habitat. These will be clarified following the environmental assessment process.

Following completion of the environmental assessment processes, Yamana will proceed to the authorization requests for the construction and the exploitation of the project with provincial and municipal authorities.

1.15.3 Social Considerations

The Wasamac property lies within the administrative region of Abitibi-Témiscamingue and within the municipality of Rouyn-Noranda. The Wasamac property is situated in a rural environment with isolated residential, recreational, and agricultural structures. No protected areas either at the federal level or the provincial level under the *Natural Heritage Conservation Act* are located within 5 km of the Wasamac property.

At the Project Notice stage and considering the results of the feasibility study, prior consultations with stakeholders and the inventories carried out by the proponent, the main anticipated social issues identified include, without being limited to:

- protecting and maintaining quality of life for residents
- reconciling land uses
- maintaining the quality of landscape
- minimizing technological and geotechnical risks

- implementing responsible mining development
- facilitating the social acceptability of the project

As per the detailed Project Description (WSP, 2020), various means of communication have been put in place to establish and maintain dialogue with the community and the various stakeholders, as follows:

- written communications to citizens (notification of upcoming activities and work)
- newsletter (published twice a year) distributed by mail to residents of the area and by email to stakeholders
- online discussion forum (tool to be redefined)
- dedicated community relations email address (administered daily)
- individual meetings with the project's neighbours (mitigation measures and corrections to work done prior to the acquisition by Monarch)
- personalized letters and regular exchanges with the city of Rouyn-Noranda (urban planning department); holding four coffee meetings for the project's neighbours and the City's elected officials (January 22, 2018, October 24, 2018, October 3, 2019 and February 11, 2020)
- mailing coffee shop reports to project neighbours; presentations on the project to various municipal, para-municipal and community organizations.

In September 2019, Yamana began a consultation process prior to the launch of the IA process with certain stakeholders potentially affected by the project.

With respect to First Nations, a first meeting with the management of the Abitibiwinni First Nation Council was held in October 2018 during which Monarch representatives presented the project and its progress, prior to Yamana acquisition.

The Abitibiwinni First Nation Council expressed its interest in being informed and involved in the next steps of the project. A second meeting was held on December 5, 2019. No concerns were raised during these meetings.

As a result of contacts made with these First Nations by mail during the spring and summer of 2020, three Indigenous groups (Timiskaming First Nation, Abitibiwinni First Nation Council and Wahgoshig First Nation) directly expressed their interest to Monarch to be met and involved in the development of the project due to the potential impacts on their traditional territory. Meetings are underway, or planned, by Yamana with these groups to better identify their interests and expectations as to how to involve them in preparing the EIA submissions.

1.16 Capital and Operating Costs

The capital cost estimate conforms to Class 3 guidelines for a feasibility study level estimate with a ±15% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The operating cost estimate was developed to have an accuracy of ±15%. Both estimates are provided in Q2 2021 Canadian dollars.

1.16.1 Capital Costs

The capital cost estimate summarized in Table 1-4 includes costs for mining, site preparation, process plant, tailings storage facility, power infrastructure, Owner's costs, spares, first fills, buildings, roadworks, and off-site infrastructure. The estimate is based on the EPCM execution approach outlined in Chapter 24. The following parameters and qualifications were considered:

- For material sourced in US dollars (8.8% of initial capital cost), an exchange rate of 1.23 Canadian dollars per US dollar was assumed, as per the exchange spot rate at the time of bid solicitation.
- No allowance has been made for exchange rate fluctuations.
- There is no escalation added to the estimate.
- A growth allowance was included.
- Data for the estimates have been obtained from numerous sources, as outlined in Chapter 21.

Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs, and Owner's costs) were identified and analyzed. Percentage of contingency was allocated to each of these categories on a line-item basis based on the accuracy of the data. An overall contingency amount was derived in this fashion.

The total initial capital cost for the Wasamac Project is C$533 million and life-of-mine sustaining capital costs are C$432 million. The sustaining capital cost includes closure costs of C$22 million.

Table 1-4: Capital Cost Summary

WBS	Description	Initial Capital Costs (C$M)	Sustaining Capital Costs (C$M)
1100	Old Wasamac Dewatering & Rehabilitation	1.8	0
1200	Mine Portals	1.2	0
1300	Mine Development	81	150
1400	Underground Mobile Equipment	40	84.5
1500	Underground Infrastructure & Construction	5.3	0.1
1600	Underground Services	36	31
1700	Backfill Plant & Network	0	52
1800	Technical Services & Instrumentation	0.1	0
1900	Waste Rock Storage Facility	2.9	4.1
2800	Underground Plant & System Ore Receival & Crushing	24	0
3100	Crushed Ore Storage & Reclaim Tunnel	5.7	0
3200	Grinding	30	0
3300	Leaching	18	0
3400	Elution and Gold Room	10	0
3500	Tailings Disposal & Pipeline	9.4	0
3600	Reagents	4.2	0
3700	Tailings Filtration Plant	12	0
3800	Process & Tailings Air & Water	4.9	0
3900	Process Buildings	13	0

WBS	Description	Initial Capital Costs (C$M)	Sustaining Capital Costs (C$M)
4100	Bulk Earthworks	7.5	0
4200	Hight-Voltage Power Switchyard & Power Distribution	12	0
4300	Communications	1.6	0.5
4400	Truck Shop & Fuel Storage	5.5	0
4500	Buildings	8	0.8
4600	Site Services	9.7	6.4
4700	Industrial Water Management	1	0
4800	Tailings Storage Management Facilities	15	48
5100	Main Access Road	0.7	0
5300	Mine Services Access Roads (to Vent Collars)	0.6	0
6100	Temporary Construction Facilities & Services	21	0
6200	Commissioning Representatives & Assistance	1.1	0
6300	Spares (Commissioning, Initial & Insurance)	0.7	0
6400	First Fills & Initial Charges	1.8	0
6500	Freight & Logistics	0.1	0
7100	Engineering & Construction Management Services	25	0
7200	Underground Mining & Engineering	14	0
8100	Project Management & Home Office	6.3	0
8200	Construction Labour	6.9	0
8300	Pre-Production Labour	4	0
8400	Pre-Production Programs	2.5	0
8500	Finance, Legal, Insurance	3.4	0
8600	Closure Costs	0	22
8700	Pre-Production Mining Operating Cost	25	0
8800	Pre-Production Process Operating Cost	2	0
8900	Pre-Production Technology & Innovation	3.7	0
	Subtotal	477	401
9100	Project Contingency	56	31
	Total Project Costs	533	432

1.16.2 Operating Costs

The operating cost estimate includes mining, processing, and general and administration (G&A) costs. The operating cost estimates for a typical year of production are provided in Table 1-5. Note that the life-of-mine all-in process operating cost is presented in Section 1.7 and includes plant ramp-up.

Table 1-5: Typical Annual Operating Cost Summary

Cost Centre	C$M/a	C$/t
Processing Operating Cost	38.1	14.9
G&A Costs	13.2	5.2
Mining Costs	69.7	36.1
Total	121.0	56.2

Source: Ausenco for Plant Milling and G&A - Expenses Cost Centres, 2021. InnovExplo for Mining Cost Centre, 2021

The operating cost estimates are based on the following assumptions:

- Cost estimates are based on Q2 2021 pricing without allowances for inflation.
- For material sourced in US dollars, an exchange rate of 1.28 Canadian dollar per US dollar was assumed.
- Fuel costs and associated taxes were taken as the historical average for the previous six years at nearby Val d'Or, Québec. Estimated retail costs are C$1.158/L for petroleum diesel.
- The annual power costs were calculated using a unit price of C$0.051/kWh. This value was assumed to be the same as in the previous feasibility study completed by BBA.
- Labour is assumed to come from the local area of highly skilled workers in Rouyn-Noranda.

1.17 Economic Analysis

The economic analysis was performed assuming a 5% discount rate and based only in mineral reserves. Cash flows have been discounted to the start of construction (December 1, 2024), assuming that the project execution decision will be made and major project financing will be carried out at this time.

1.17.1 Forward-Looking Information Cautionary Statements

The results of the economic analyses discussed in this chapter represent forward-looking information as defined under Canadian securities law. This report has included certain non-IFRS performance measures, such as: Cash cost and All-in sustaining cost ("AISC"). These non-IFRS performance measures do not have a standardized meaning, and therefore may not be comparable to similar measures employed by other issuers. Cash cost is calculated by summing mining cost, processing cost, G&A, refining charges, and royalties, and dividing it by payable gold ounces. AISC is calculated by summing Cash cost, sustaining capital, closure cost, and salvage value and dividing it by payable gold ounces. The results of the economic analysis are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Forward-looking information includes the following:

- mineral reserve estimates
- assumed commodity price and exchange rates
- proposed mine production plan
- projected mining and process recovery rates
- assumptions about mining dilution and the ability to mine in areas previously exploited using underground mining methods as envisaged
- sustaining costs and proposed operating costs
- interpretations and assumptions regarding joint venture and agreement terms
- assumptions as to closure costs and closure requirements
- assumptions about environmental, permitting, and social risks

Additional risks to the forward-looking information include:

- changes to costs of production from what is assumed
- changes in the estimated timing and quantity of production
- unrecognized environmental risks
- unanticipated reclamation expenses
- unexpected variations in quantity of mineralized material, grade or recovery rates
- geotechnical or hydrogeological considerations during mining being different from what was assumed
- failure of mining methods to operate as anticipated
- failure of plant, equipment, or processes to operate as anticipated
- changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis
- ability to maintain the social license to operate
- accidents, labour disputes, and other risks of the mining industry
- changes to interest rates
- changes to tax rates
- changes in government regulation of mining operations
- potential delays in the issuance of permits and any conditions imposed with the permits that are granted

1.17.2 Financial Model Parameter

A base case gold price of US$1,550/oz is based on consensus analyst estimates and recently published economic studies. The forecasts are meant to reflect the average metal price expectation over the life of the project. No price inflation or escalation factors were taken into account.

The economic analysis was performed using the following key assumptions:

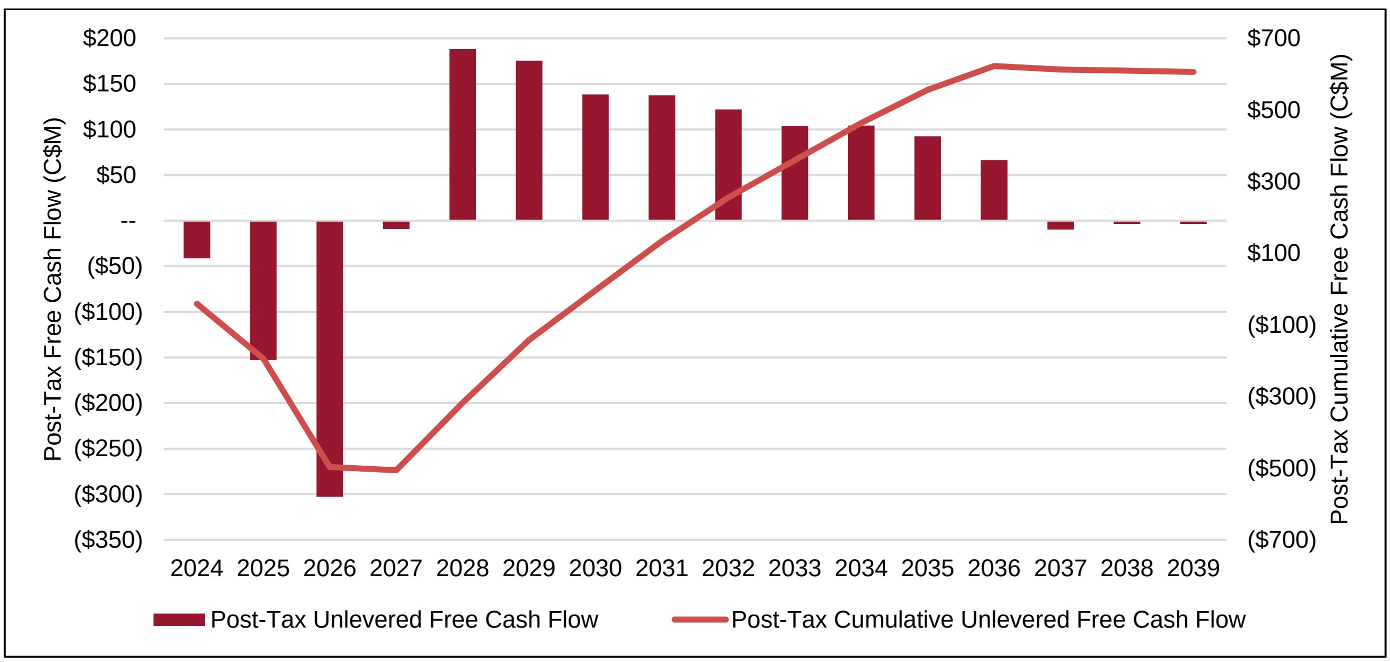
- construction starting December 1, 2024
- commercial production starting on October 1, 2027
- mine life of 9.7 years
- exchange rate of 1.28 (USD:CAD)
- cost estimates in constant Q3 2021 Canadian dollars with no inflation or escalation
- 100% ownership with 1.5% NSR
- capital costs funded with 100% equity (no financing costs assumed)

1.17.3 Economic Analysis

The pre-tax net present value (NPV) discounted at 5% ($NPV_{5\%}$) is C$610 million; the IRR is 21.7%, and payback period is 3.6 years. On a post-tax basis, the $NPV_{5\%}$ C$326 million, the IRR is 16.1%, and the payback period is 4.0 years.

A summary of project economics shown graphically in Figure 1-3 and listed in Table 1-6.

Figure 1-3: Project LOM Cash Flow



Source: Ausenco, 2021.

Table 1-6: Summary of Project Economics

Description		LOM Total / Avg.
General		
Gold Price (US$/oz)		$1,550
Mine Life (years)		9.7
Total Mill Feed Tonnes (kt)		23,168
Production		
Mill Head Grade (g/t)		2.56
Mill Recovery Rate (%)		88.7%
Total Mill Ounces Recovered (koz)		1,694
Total Average Annual Production (koz)		169
Operating Costs		
Mining Cost (C$/t Milled)		$36.08
Processing Cost (C$/t Milled)		$15.70
G&A Cost (C$/t Milled)		$5.75
Refining & Transport Cost (C$/oz)		$2.00
Total Operating Costs (C$/t Milled)		$57.53
Cash Costs (US$/oz Au)		$640
AISC (US$/oz Au)		$828
Capital Costs		
Initial Capital (C$M)		$533
Sustaining Capital (C$M)		$406
Closure Costs (C$M)		$25
Salvage Costs (C$M)		($25)
Financials	**Pre-Tax**	**Post-Tax**
NPV (5%) (C$M)	$610	$326
IRR (%)	21.7%	16.1%
Payback (years)	3.6	4.0

Note: Cash cost, and AISC are non-IFRS performance measures. Refer to Forward-Looking Cautionary Statements.

1.17.4 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV, and IRR of the project using the following variables: gold price, discount rate, foreign exchange, operating cost, initial capital cost, and head grade. The analysis revealed that the project is most sensitive to changes in gold price, foreign exchange, and head grade and less sensitive to discount rate, operating cost, and initial capital cost. Table 1-7 and Table 1-8 summarizes the pre-tax and post-tax sensitivity analysis results. respectively.

Table 1-7: Pre-Tax Sensitivity

Pre-Tax NPV Sensitivity to Discount Rate

Discount Rate	Gold Price (US$/oz) $1,300	$1,400	$1,550	$1,700	$1,800
1.0%	$434	$633	$933	$1,232	$1,431
3.0%	$320	$495	$757	$1,020	$1,195
5.0%	$225	$379	$610	$842	$996
8.0%	$111	$240	$433	$626	$754
10.0%	$50	$165	$337	$509	$623

Pre-Tax IRR Sensitivity to Discount Rate

Discount Rate	Gold Price (US$/oz) $1,300	$1,400	$1,550	$1,700	$1,800
1.0%	11.9%	16.1%	21.7%	26.7%	29.9%
3.0%	11.9%	16.1%	21.7%	26.7%	29.9%
5.0%	11.9%	16.1%	21.7%	26.7%	29.9%
8.0%	11.9%	16.1%	21.7%	26.7%	29.9%
10.0%	11.9%	16.1%	21.7%	26.7%	29.9%

Pre-Tax NPV Sensitivity to FX

FX	Gold Price (US$/oz) $1,300	$1,400	$1,550	$1,700	$1,800
1.36	$350	$514	$760	$1,005	$1,169
1.32	$288	$447	$685	$924	$1,083
1.28	$225	$379	$610	$842	$996
1.24	$163	$312	$536	$760	$909
1.20	$100	$244	$461	$678	$822

Pre-Tax IRR Sensitivity to FX

FX	Gold Price (US$/oz) $1,300	$1,400	$1,550	$1,700	$1,800
1.36	15.3%	19.4%	25.0%	30.1%	33.3%
1.32	13.7%	17.8%	23.3%	28.4%	31.6%
1.28	11.9%	16.1%	21.7%	26.7%	29.9%
1.24	10.1%	14.3%	19.9%	25.0%	28.1%
1.20	8.2%	12.5%	18.1%	23.2%	26.3%

Pre-Tax NPV Sensitivity to Opex

Opex	Gold Price (US$/oz) $1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	$416	$571	$802	$1,033	$1,187
(10.0%)	$321	$475	$706	$937	$1,091
--	$225	$379	$610	$842	$996
10.0%	$129	$284	$515	$746	$900
20.0%	$34	$188	$419	$650	$805

Pre-Tax IRR Sensitivity to Opex

Opex	Gold Price (US$/oz) $1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	17.0%	20.7%	25.9%	30.6%	33.6%
(10.0%)	14.5%	18.4%	23.8%	28.7%	31.8%
--	11.9%	16.1%	21.7%	26.7%	29.9%
10.0%	9.2%	13.6%	19.4%	24.7%	27.9%
20.0%	6.1%	10.9%	17.1%	22.6%	26.0%

Pre-Tax NPV Sensitivity to Initial Capex

Initial Capex	Gold Price (US$/oz) $1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	$326	$480	$711	$943	$1,097
(10.0%)	$276	$430	$661	$892	$1,046
--	$225	$379	$610	$842	$996
10.0%	$175	$329	$560	$791	$945
20.0%	$124	$278	$510	$741	$895

Pre-Tax IRR Sensitivity to Initial Capex

Initial Capex	Gold Price (US$/oz) $1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	16.5%	21.0%	27.2%	32.8%	36.4%
(10.0%)	14.0%	18.4%	24.2%	29.5%	32.9%
--	11.9%	16.1%	21.7%	26.7%	29.9%
10.0%	10.1%	14.0%	19.4%	24.3%	27.3%
20.0%	8.4%	12.3%	17.4%	22.1%	25.0%

Pre-Tax NPV Sensitivity to Head Grade

Head Grade	Gold Price (US$/oz) $1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	($174)	($50)	$135	$320	$444
(10.0%)	$26	$165	$373	$581	$720
--	$225	$379	$610	$842	$996
10.0%	$424	$594	$848	$1,102	$1,272
20.0%	$624	$809	$1,086	$1,363	$1,548

Pre-Tax IRR Sensitivity to Head Grade

Head Grade	Gold Price (US$/oz) $1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	-%	3.2%	9.3%	14.5%	17.7%
(10.0%)	5.9%	10.2%	15.9%	21.0%	24.1%
--	11.9%	16.1%	21.7%	26.7%	29.9%
10.0%	17.2%	21.3%	26.8%	32.0%	35.2%
20.0%	21.9%	26.0%	31.6%	36.8%	40.1%

Table 1-8: Post-Tax Sensitivity

Post-Tax NPV Sensitivity to Discount Rate — Gold Price (US$/oz)

Discount Rate	$1,300	$1,400	$1,550	$1,700	$1,800
1.0%	$238	$360	$540	$718	$835
3.0%	$157	$265	$423	$580	$683
5.0%	$89	$185	$326	$465	$556
8.0%	$8	$88	$207	$324	$401
10.0%	($36)	$36	$143	$248	$317

Post-Tax IRR Sensitivity to Discount Rate — Gold Price (US$/oz)

Discount Rate	$1,300	$1,400	$1,550	$1,700	$1,800
1.0%	8.3%	11.6%	16.1%	20.2%	22.8%
3.0%	8.3%	11.6%	16.1%	20.2%	22.8%
5.0%	8.3%	11.6%	16.1%	20.2%	22.8%
8.0%	8.3%	11.6%	16.1%	20.2%	22.8%
10.0%	8.3%	11.6%	16.1%	20.2%	22.8%

Post-Tax NPV Sensitivity to FX — Gold Price (US$/oz)

FX	$1,300	$1,400	$1,550	$1,700	$1,800
1.36	$167	$267	$416	$561	$657
1.32	$128	$226	$371	$513	$607
1.28	$89	$185	$326	$465	$556
1.24	$50	$143	$280	$416	$505
1.20	$11	$101	$235	$366	$453

Post-Tax IRR Sensitivity to FX — Gold Price (US$/oz)

FX	$1,300	$1,400	$1,550	$1,700	$1,800
1.36	11.0%	14.3%	18.8%	22.9%	25.5%
1.32	9.7%	13.0%	17.5%	21.6%	24.2%
1.28	8.3%	11.6%	16.1%	20.2%	22.8%
1.24	6.9%	10.2%	14.7%	18.8%	21.4%
1.20	5.4%	8.8%	13.3%	17.4%	19.9%

Post-Tax NPV Sensitivity to Opex — Gold Price (US$/oz)

Opex	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	$206	$300	$437	$573	$663
(10.0%)	$148	$243	$382	$519	$610
--	$89	$185	$326	$465	$556
10.0%	$29	$126	$268	$408	$501
20.0%	($32)	$66	$210	$351	$444

Post-Tax IRR Sensitivity to Opex — Gold Price (US$/oz)

Opex	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	12.4%	15.4%	19.5%	23.3%	25.7%
(10.0%)	10.4%	13.5%	17.9%	21.8%	24.3%
--	8.3%	11.6%	16.1%	20.2%	22.8%
10.0%	6.1%	9.6%	14.4%	18.6%	21.2%
20.0%	3.7%	7.5%	12.5%	16.9%	19.6%

Post-Tax NPV Sensitivity to Initial Capex — Gold Price (US$/oz)

Initial Capex	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	$190	$286	$426	$565	$657
(10.0%)	$140	$235	$376	$515	$606
--	$89	$185	$326	$465	$556
10.0%	$39	$134	$275	$414	$505
20.0%	($12)	$84	$225	$364	$455

Post-Tax IRR Sensitivity to Initial Capex — Gold Price (US$/oz)

Initial Capex	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	13.2%	16.9%	22.0%	26.6%	29.5%
(10.0%)	10.6%	14.1%	18.8%	23.1%	25.8%
--	8.3%	11.6%	16.1%	20.2%	22.8%
10.0%	6.4%	9.5%	13.8%	17.7%	20.1%
20.0%	4.6%	7.7%	11.8%	15.5%	17.8%

Post-Tax NPV Sensitivity to Head Grade — Gold Price (US$/oz)

Head Grade	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	($165)	($85)	$33	$149	$225
(10.0%)	($36)	$51	$181	$308	$392
--	$89	$185	$326	$465	$556
10.0%	$212	$315	$468	$618	$717
20.0%	$333	$445	$608	$770	$877

Post-Tax IRR Sensitivity to Head Grade — Gold Price (US$/oz)

Head Grade	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	-%	1.6%	6.3%	10.4%	13.0%
(10.0%)	3.6%	6.9%	11.5%	15.6%	18.1%
--	8.3%	11.6%	16.1%	20.2%	22.8%
10.0%	12.5%	15.8%	20.3%	24.4%	27.0%
20.0%	16.4%	19.6%	24.2%	28.4%	31.0%

1.18 Recommendations

1.18.1 Drilling & Geology

The results presented herein demonstrate that the Wasamac Project is technically and economically viable. In light of these results, InnovExplo recommends a bulk sample program for Yamana to de-risk the investment. This bulk sample program would target specific mining areas for geomechanical and metallurgical assessments. Also, the geological continuity and grade would be confirmed with development and stope excavations.

Based on the information presented in this technical report including the results of the updated mineral resource estimate, the qualified persons recommend that drilling and exploration programs should continue to be carried out with the objectives described below and summarized in Table 1-9.

Table 1-9: Drilling & Geology – Proposed Recommendations and Estimated Costs

Item	Cost (C$)
Infill Drilling	11,000,000
"Near-Deposit" Exploration	1,500,000
Francoeur Deposit Confirmation Drilling and Near-Mine Exploration	1,500,000
Greenfield and Brownfield Exploration on the Wasamac Property	1,000,000
Total	**15,000,000**

1.18.1.1 Infill Drilling

Infill drilling is recommended to better delineate the mineral resources and increase confidence in grade, improve mine planning, and provide further geotechnical and metallurgical data. The delineation drilling program should focus first on the areas expected to be developed in the first three years of production and subsequently on the remaining mineral resource. A budget of C$11 million to complete 68,000 metres of drilling is recommended for this program.

1.18.1.2 "Near-Deposit" Exploration

Drilling is recommended to expand the current mineral resource envelopes of the Deposit to depths below the established mineral resource and to test for mineralization in the poorly explored gaps between defined zones. The focus of this exploration effort is to delineate secondary zones, such as Wildcat, and test high-priority extensions of the Francoeur-Wasa Shear Zone at depth, and to the east and west of the Horne Creek Fault. Approximately 10,000 metres of drilling corresponding to C$1.5 million is recommended for this program.

1.18.1.3 Francoeur Deposit Confirmation Drilling and Near-Mine Exploration

Drilling to confirm the current mineral resource at the Francoeur deposit and expand it by testing high-potential targets adjacent to, and down-dip from, historical mining operations and along the Arntfield-Francoeur segment of the Francoeur-Wasa Shear Zone. A projected 10,000 metres of drilling corresponding to C$1.5 million is recommended for this program.

1.18.1.4 Greenfield and Brownfield Exploration on the Wasamac Property

Development of a long-term pipeline of greenfield and brownfields exploration discoveries through geophysical surveying and testing of exploration targets is recommended, focusing first on the secondary gold-bearing shear zones to the Arntfield-Francoeur segment of the western Francoeur-Wasa Shear Zone (which includes Lac Fortune) and concentrating subsequently on other known occurrences. A budget of 6,000 metres of drilling corresponding to C$1 million is recommended for this program.

1.18.2 Underground Mining

The activities, optimizations and reviews outlined in Table 1-10 are recommended to ensure an optimal mine design considering all aspects of a sustainable project.

Table 1-10: Underground Mining – Proposed Recommendations and Estimated Costs

Section	Item	Cost (C$)
26.2.1	Bulk Sample Program	90,000,000
26.2.2	Geomechanical Detailed Assessment	300,000[1]
26.2.3	Crown Pillar Monitoring during Dewatering	65,000
26.2.4	Hydrogeological Investigations	80,000
26.2.5	Fully Electric Fleet – Trucks & LHDs	30,000
26.2.6	Optimize Underground Paste Plant Location & Network – Surface vs. Underground	300,000
26.2.7	Blasting Optimization & Vibration Modelling	150,000
26.2.8	Ventilation Network Optimization	50,000
26.2.9	Replace Propane by Natural Gas[2]	100,000
26.2.10	Additional Engineering	300,000
	Total	**91,375,000**

Notes: 1. Considering the utilization of exploration drill holes for the geomechanical campaign. 2. The replacement of propane with natural gas should be studied in combination of with the utilization of natural gas for surface infrastructures.

1.18.2.1 Bulk Sample Program

The bulk sample will consist of excavating approximately 10,500 m of access ramps and development to access the top portion of both metallurgical main zone central and east (mining area E38) with low recoveries to test with more material (25,000 t) and validate or improve the recovery. This could have a material impact on the finance of the project. Geological continuity and geomechanics properties will also be assess with this program. A minimal surface infrastructure will be required to support these 30 months like temporary offices, dry, shop, water supply, power and water treatment facilities.

1.18.2.2 Geomechanical Input for Mine Design

To validate ground conditions, dilution, and recovery factors (and other parameters such as efficiencies of equipment, drilling patterns and location of major infrastructures), a two-phase geomechanical program has been established. Phase 1 has been completed. An analysis of Phase 1 data and the completion of Phase 2 is recommended for each of the main mining areas.

The geomechanical program targets major lithologies affecting the infrastructure and production centres of project. Results will be used to validate the properties of geomechanical domains in all principal mining areas. This program will also allow for detailed engineering of the portals and crown pillars for both ramps underneath Highway 117 and the railway parallel to Highway 117. The geomechanical field campaign includes complete joint-set and rock-mass logging, laboratory testwork, and updated rock mechanics analysis of the project, etc. Major structures such as faults will also be surveyed.

1.18.2.3 Hydrogeologic Input for Mine Design

A two-phase hydrogeological testing program has been established. Phase 1 work has been completed, but the data has yet to be analysed. The tested drill holes are mainly in undisturbed areas, with no obvious faulting. The main shear zones (Francoeur-Wasa shear zone) and other transversal faults have not yet been tested to validate preferential flow zones. Phase 2 will require testing of additional drill holes to be drilled in the Francoeur-Wasa shear zone and transverse faults. The same testing procedures as for phase 1 is required, and should include slug tests, injection tests, and profile tracer tests, among others. The drill holes to be tested could be the same as future geomechanical drill holes if they are located in appropriate geological structures. The central and the eastern parts of the Project area should be prioritized for this work, as these are the least understood due to a lack of historic data.

Once completed, the groundwater flow model should be updated accordingly, and new inflow and environmental impact predictions carried out. This model should be built on an unstructured mesh system so the faults can be properly represented in 3D.

1.18.3 Metallurgical Testwork & Recovery Methods

The following activities are recommended to support the development of a detailed design of the processing facility beyond the feasibility study stage:

- Conduct diagnostic leach test protocols to assist with determining underlying reasons for low recoveries in Zones 1 and 2
- carry out leach testing to better define the optimum leach time by conducting fixed duration leach tests for 8, 12, 16, 24 and 36 hours on representative samples
- generate additional leached and cyanide detoxified slurry to provide samples for vendor thickener and filtration tests (thickening and tailings filtration testing completed to date is not vendor specific and may not be suitable for equipment selection with process guarantees)
- conduct tests on samples generated from the proposed test mining program to validate the selected flowsheet (a more detailed financial analysis may show that the flotation flowsheet economics are superior to the whole ore leach flowsheet. If so, it is recommended to conduct the required testing to support any engineering design required)
- conduct tests on samples generated from the proposed bulk sampling program to validate the selected flowsheet

The metallurgical testwork outlined above is estimated at C$512,000, which includes laboratory testwork costs, management, and interpretation of the results.

1.18.4 Site Infrastructure

It is recommended to optimize the storage of waste rock underground in the paste backfill versus on the surface in the waste rock storage facility (WRSF) to reduce the surface expression of mine waste product. It is also recommended to consider alternative locations for the waste rock, given that the study location impacts wetlands and as a result will require a costlier design and possibly more lengthy permitting.

The classification of the waste rock tested to date indicates tendency for acid consumption. However, it is recommended that a more comprehensive program of geochemical waste rock and tailings characterization be performed, including kinetic testwork (which is in progress with WSP) to confirm the geochemical behaviour of waste rock and tailings.

1.18.5 Geotechnical Input for Site Infrastructure

It is recommended to pursue further data collection of the subsurface in the areas of major infrastructure. A seismic cone penetration test (SCPT) program has been designed and is planned for execution in the first half of 2022. The SCPT program will investigate the shallow subsurface in the area of the TSF, WRSF, plant pad area, mill basin and TSF pond. Data collected will investigate variability in soil horizons and confirm foundation loading. Targets were selected with aerial photography. The program has been designed to leverage winter conditions that will allow access to wetland areas. The program is quoted to cost C$48,900.

1.18.6 Tailings Storage Facility

There are sufficient tailings samples to adequately classify the tailings as having a ‘low’ risk of acidification (Price, 2009); however, depending on the tailings material distribution in situ and local environmental conditions, there may be variable responses observed. Preparation of tailings samples for kinetic testwork is in progress for four samples; the program is quoted to cost C$64,056, which includes laboratory testwork costs, management, and data interpretation.

1.18.7 Environment, Permitting, and Community Relations

Recommendations for environment, permitting, and community relations are as follows:

- complete additional baseline studies in the summer of 2021
- complete the EIS for submission to the province and the federal Impact Assessment Agency
- continue with public and First Nation consultation activities, addressing and documenting concerns
- complete a Closure Plan for the project

2 INTRODUCTION

This report was prepared by Ausenco Engineering Canada Inc. (Ausenco) for Yamana Gold Inc. (Yamana or the Issuer) to summarize the results of the Wasamac Feasibility Study Update. The report was prepared in compliance with the Canadian disclosure requirements of National Instrument 43-101 (N.I. 43-101) and in accordance with the requirements of Form 43-101 F1.

The responsibilities of the engineering consultants and firms who are providing qualified persons are as follows:

- Ausenco was contracted by Yamana to manage and coordinate the work related to the report. Ausenco also developed the feasibility-level design and cost estimate for the process plant, general site infrastructure, and site water management infrastructure.
- InnovExplo Inc. (InnovExplo) was contracted by Yamana to complete the mineral resource and mineral reserve estimates for the project, and to design the underground mine plan, mine production schedule, and mine capital and operating costs.
- BBA Consultants (BBA) was contracted by Yamana to develop the feasibility-level design and cost estimate for the dry stack tailings storage facility.

2.1 Terms of Reference

The report supports disclosures by Yamana in a news release dated July 19, 2021 entitled, “Yamana Gold Announces Positive Development Decision on its Wholly-Owned Wasamac Project Based on Positive Results from Several Studies Showing Higher Average Daily Throughput, Increased Mineral Reserves, Increased Average Annual Production and Strong, Increased Cash Flows”.

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” (2014) and “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” (2019).

2.2 Qualified Persons

This report was prepared by the following qualified persons:

- Tommaso Roberto Raponi, P.Eng., Principal Metallurgist, Ausenco
- Alain Carrier, P.Geo., Co-President Founder, InnovExplo
- Denis Gourde, P.Eng., Vice-President Engineering and Sustainable Development, InnovExplo
- Frank Palkovits P.Eng., Senior Technical Director; Responsible Mining Solutions
- Luciano Piciacchia, P.Eng., Managing Director Earth and Infrastructure, BBA

- Michael Verreault, P.Eng., Hydrogeologist, President, Hydro-Ressources Inc.
- Charles Gagnon, P.Eng., Senior Mining Engineer, CGM Expert
- Ali Hooshiar Fard, P.Eng., Senior Geotechnical Engineer, Ausenco
- Scott Weston, P.Geo., Vice President, Business Development, Ausenco
- Sébastien Tanguay, P.Eng., Mining Engineer, InnovExplo
- Vincent Nadeau-Benoit, P.Geo., Project Geologist, InnovExplo

By virtue of their education, experience, and professional association, the individuals presented in Table 2-2 (overleaf) are considered “qualified persons” (QPs) as defined by N.I. 43-101. Report sections for which each QP is responsible are also provided in Table 2-2. The QPs meet the requirement of independence defined in N.I. 43-101.

2.3 Site Visits and Scope of Personal Inspection

A summary of the site visits completed by the QPs is presented in Table 2-1 below. Antonio Vides, P.Eng., performed the site visit on behalf of Luciano Piciacchia, P.Eng., on July 12, 2021, for 1 day.

Table 2-1: Qualified Person Site Visits

Qualified Person	Date of Site Visit	Days on Site
Alain Carrier, P.Geo.	May 12, 2021	1
Vincent Nadeau-Benoit, P.Geo.	May 12, 2021	1
Denis Gourde, P.Eng.	May 12, 2021	1
Sébastien Tanguay, P.Eng.	June 2, 2021	2

2.3.1 Geology

Alain Carrier and Vincent Nadeau-Benoit visited the Wasamac property to review the property geology, drill hole collar locations and core library. During the site visit, Alain Carrier and Vincent Nadeau-Benoit were accompanied by Sébastien Bernier, Jean-François Ravenelle, Dominic Chartier, Manon Garant, Luc Turcotte and Marcel St-Pierre of Yamana.

2.3.2 Mining

Denis Gourde visited the Wasamac property to review the area of the proposed ramp declines, proposed mill and service building area. The south side of the property was also visited to review the proposed area of both fresh air raise intakes, the cement borehole and propane storage areas. During the site visit, Denis Gourde was accompanied by Sébastien Bernier, Jean-François Ravenelle, Dominic Chartier, Manon Garant, Luc Turcotte and Marcel St-Pierre of Yamana.

Sébastien Tanguay visited the Wasamac property to review the area of the proposed ramp declines. The south side of the property was also visited to review the proposed area of both fresh air raise intakes, the cement borehole and propane storage areas. During the site visit, Sébastien Tanguay was accompanied by Manon Garant and Marcel St-Pierre of Yamana.

Table 2-2: Report Contributors

Qualified Person	Professional Designation	Position	Employer	Independent of Yamana	Report Sections
Tommaso Roberto Raponi	P.Eng.	Senior Mineral Processing Specialist	Ausenco Engineering Canada	Yes	1.1, 1.9, 1.13, 1.14.1, 1.14.2, 1.14.5, 1.16, 1.17, 1.18.3, 1.18.4, 2.1, 2.2, 2.5, 2.7, 3, 13, 16.7.1, 16.7.2, 16.7.3, 17, 18.1 to 18.6, 18.9.2, 18.9.5, 18.10.6, 18.10.7, 18.13.1, 18.13.2, 18.13.5,18.14, 19, 21 except 21.2.2 and 21.3.1 and 21.4.2, 22, 24, 25.1, 25.6, 25.7, 25.8, 25.10, 25.11, 25.12, 25.13.2, 25.14.1 to 25.14.3, 26.1, 26.4
Alain Carrier	P.Geo.	Co-President Founder	InnovExplo Inc.	Yes	1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.10, 1.18.1, 2.3.1, 2.4, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, 25.2, 25.3, 25.4, 26.3
Denis Gourde	P.Eng.	Vice President Engineering	InnovExplo Inc.	Yes	1.11, 1.12, 1.18.2, 2.2, 2.3.2, 15, 16.1, 16.2, 16.4, 16.5 excluding 16.5.4.1, 16.6, 16.7.4 to 16.7.8, 16.9, 16.10, 21.2.2, 21.3.1, 21.4.2, 25.4, 25.5, 25.13.1, 25.14.4, 26.1 and 26.2
Frank Palkovits	P.Eng.	Senior Technical Director	Responsible Mining Solutions	Yes	16.5.4.1, 18.13.3, 18.13.4
Luciano Piciacchia	P.Eng.	Managing Director Earth and Infrastructure	BBA Consultants	Yes	1.14.3, 2.3.3, 18.9, 18.9.1, 18.9.3, 18.9.4, 18.9.6
Michael Verreault	P.Eng.	Hydrogeologist, President	Hydro-Ressources Inc.	Yes	16.3
Charles Gagnon	P.Eng.	Senior Mine Engineer	CGM	Yes	16.8
Ali Hooshiar	P.Eng.	Senior Geotechnical Engineer	Ausenco Engineering Canada	Yes	1.14.4, 1.14.6, 1.18.5, 18.7, 18.8, 18.10.1 to 18.10.5, 18.10.8, 18.11, 18.12, 20.2.3, 20.2.4, 26.5
Scott Weston	P.Geo.	Vice President, Business Development Hemmera	Ausenco Engineering Canada	Yes	1.15, 1.18.6, 1.18.7, 20 except 20.2.3 and 20.2.4, 25.9, 25.14.5, 26.6, 26.7, 26.8
Sébastien Tanguay	P.Eng.	Mining Engineer	InnovExplo. Inc.	Yes	1.11, 1.12, 1.18.2, 2.2, 2.3.2, 15, 16.1, 16.2, 16.4, 16.5 excluding 16.5.4.1, 16.6, 16.7.4 to 16.7.8, 16.9, 16.10, 21.2.2, 21.3.1, 21.4.2, 25.4, 25.5, 25.13.1, 25.14.4, 26.1 and 26.2.
Vincent Nadeau Benoit	P.Geo.	Project Geologist	InnovExplo Inc.	Yes	1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8,1.10, 1.18.1, 2.6, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, 25.2, 25.3, 25.4, 26.3

2.3.3 Tailings

Antonio Vides, ing., visited the Wasamac Project to review the area of the proposed dry stack tailings facility. During the site visit, Antonio Vides was accompanied by Marcel St-Pierre of Yamana.

Analyzing the findings of the visit, and considering the updated TSF layout and footprint, BBA validated the suitability of the chosen TSF location. The visit also allowed BBA to validate the following:

- the need for tree cutting over the TSF footprint
- the need for organic stripping over a portion of the TSF footprint
- the presence of major constraints on some areas of the TSF footprint
- confirmation of the need of additional geotechnical information and the location of suitable areas to investigate

2.4 Effective Dates

The effective date of this report, as per the financial analysis, is July 16, 2021. The effective date of the Wasamac mineral resource and mineral reserve estimates are June 30, 2021.

2.5 Information Sources & References

This technical report is based on internal company reports, maps, published government reports, and public information, as listed in Chapter 27. Additionally, it is based on information cited in Chapter 3.

2.6 Previous Technical Reports

The Wasamac Project has been the subject of previous technical reports, as summarized in Table 2-3.

Table 2-3: Summary of Previous Technical Reports

Reference	Company	Study	Name
(Caumartin et al., 2018)	Monarch Gold Corp.	FS	N.I. 43-101 Technical Report, Feasibility Study of the Wasamac Project, Rouyn-Noranda, Québec, Canada
(Gauthier et al., 2012)	Richmont Mines Inc.	PEA	Technical report on the Wasamac Project, Rouyn-Noranda, Québec, Canada

2.7 Units of Measurement

All units of measurement in this report are metric and all currencies are expressed in Canadian dollars (CAD, C$) unless otherwise stated. Contained gold metal is expressed as troy ounces (oz), where 1 oz = 31.1035 g. All material tonnes are expressed as dry tonnes (t) unless stated otherwise. A list of abbreviations is provided in Table 2-4.

Table 2-4: Acronyms and Abbreviations

Acronym/Abbreviation	Definition	Acronym/Abbreviation	Definition
ABA	Acid base accounting	Agency	Impact Assessment Agency of Canada
AMD	Acid mine drainage	ANFO	Ammonium nitrate fuel oil
AP	Acidity potential	BAPE	Bureau d'audiences publiques sur l'environnement
CCME	Canadian Council of Ministers of the Environment	CND	Contaminated neutral drainage
CNWA	*Canadian Navigable Waters Act*	CPNDQ	Centre du patrimoine naturel du Québec
DFO	Fisheries and Oceans Canada	ECCC	Environment and Climate Change Canada
EIA	Environmental Impact Assessment	EIARP	Environmental Impact Assessment and Review Process
EIS	Environmental Impact Statement	EQA	*Environmental Quality Act*
IA	impact assessment	IAA	*Impact Assessment Act, 2019*
LQE	Loi sur la qualité de l'environnement	MDMER	Metal and Diamond Mining Effluent Regulations
MEFCC	Ministry of Environment and Fight Against Climate Change	MELCC	Ministère de l'Environnement et de la Lutte contre les changements climatiques
MERN	Ministère de l'Énergie et des ressources naturelles	MFFP	Ministère de la Faune, des Forêts et des Parcs
NP	neutralization potential	NPR	Neutralization potential ratio
ON	Ontario	PCA	Parks Canada Agency
PM2.5	fine particulate matter	PM_{10}	Inhalable particulate matter
Project	Wasamac Gold Mine Project	QC	Québec
SARA	*Species at Risk Act*	TC	Transport Canada
TISG	Tailored Impact Statement Guidelines	TSF	Tailings storage facility
µm	Micron	km	Kilometre
°C	Degrees Celsius	km^2	Square kilometre
°F	Degrees Fahrenheit	L	Litre
°	Azimuth/dip in degrees	m	Metre
µg	Microgram	M	Mega (million)
a	Annum	m^2	Square metre
Au	Gold	m^3	Cubic metre
C$ or CAD	Canadian dollars	min	Minute
cal	Calorie	masl	Metres above sea level
cm	Centimetre	mm	Millimetre
d	Day	NO_x	Nitrogen oxide gases produced by diesel vehicles
ft	Foot or feet	oz/t, oz/st	Ounce per short ton
g	Gram	oz	Troy ounce (31.1035 g)
G	Giga (billion)	ppb	Parts per billion
g/L	Grams per litre	ppm	Part per million
g/t	Grams per tonne	%	Percent
ha	Hectare	s	Second
hp	Horse power	ton, st	Short ton
in or ″	Inch or inches	t, tonne	metric tonne
kg	Kilogram	US$ or USD	United States dollar
km	Kilometre	y	year
km^2	Square kilometre	P_{80}	80% passing product size

Source: Ausenco (2021).

3 RELIANCE ON OTHER EXPERTS

3.1 Property Agreements, Mineral Tenure, Surface Rights and Royalties

The QPs have not independently reviewed the ownership of the project area and any underlying property agreements, mineral tenure, surface rights, or royalties. The QPs have fully relied upon, and disclaim responsibility for, information derived from Yamana and legal experts retained by Yamana for this information through the following documents: various e-mail exchanges with Yamana representatives, excel spreadsheets and documents filed on SEDAR by Yamana.

This information is used in Chapter 4 of the report. The information is also used in support of the cut-off grade assumptions (royalties) for the mineral resource estimate (Section 14), mineral reserve estimate (Section 15), and economic analysis (Section 22).

3.2 Taxation

The QPs have fully relied upon, and disclaim responsibility for, information supplied by experts retained by Yamana for information related to taxation as applied to the financial model, received by email from Yamana on July 16, 2021. This information is used in Chapter 22 of the report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The property is located in the Abitibi-Témiscamingue administrative region of the Province of Québec, Canada, approximately 15 km west-southwest of the city of Rouyn-Noranda (see Figure 4-1on the following page).

The property covers an area of 10,268.56 ha, extending 20 km east-west and 15 km north-south. The coordinates of the approximate centre of the property are 48°12'08"N latitude, 79°14'30"W longitude, which corresponds to 630649E and 5340263N using NAD 83, Zone 17 UTM coordinates. The property underlies parts of Beauchastel and Dasserat townships on NTS map sheets 32D/03 and 32D/06.

4.2 Mining Title Status

Mineral title status was supplied by Manon Garant, Senior Project Geologist for the Issuer. The QP verified the status of all mining titles using GESTIM, the Government of Québec's online claim management system (gestim.mines.gouv.qc.ca). The QPs have not independently reviewed the ownership of the mineral tenure (see Section 3.2).

The property is subdivided into six claim blocks: the Wasamac Block, Wasamac NE Block, Teck JV Block, R.M. Nickel Block, Consolidated Francoeur Block and Western Buff Block, which together comprise 6 mining concessions ("CM"), 281 mineral claims ("CDC", "CL", "CLD") and five mining leases ("BM"), for a total of 292 mineral titles.

The Issuer holds 100% ownership of the mineral titles for the property, except for the Teck JV Block in which the Issuer holds 60% ownership of five claims (CDC 2450991 to 2450995). All claims and leases are in good standing as at June 30, 2021. A mineral tenure table for the property with the expiration dates of the leases and claims titles is provided in Table 4-1.

A claim map of the property is depicted in Figure 4-2; Figure 4-3 and Figure 4-4 show a claim map of the eastern and western parts of the property, respectively.

Figure 4-1: Location of the Wasamac Project

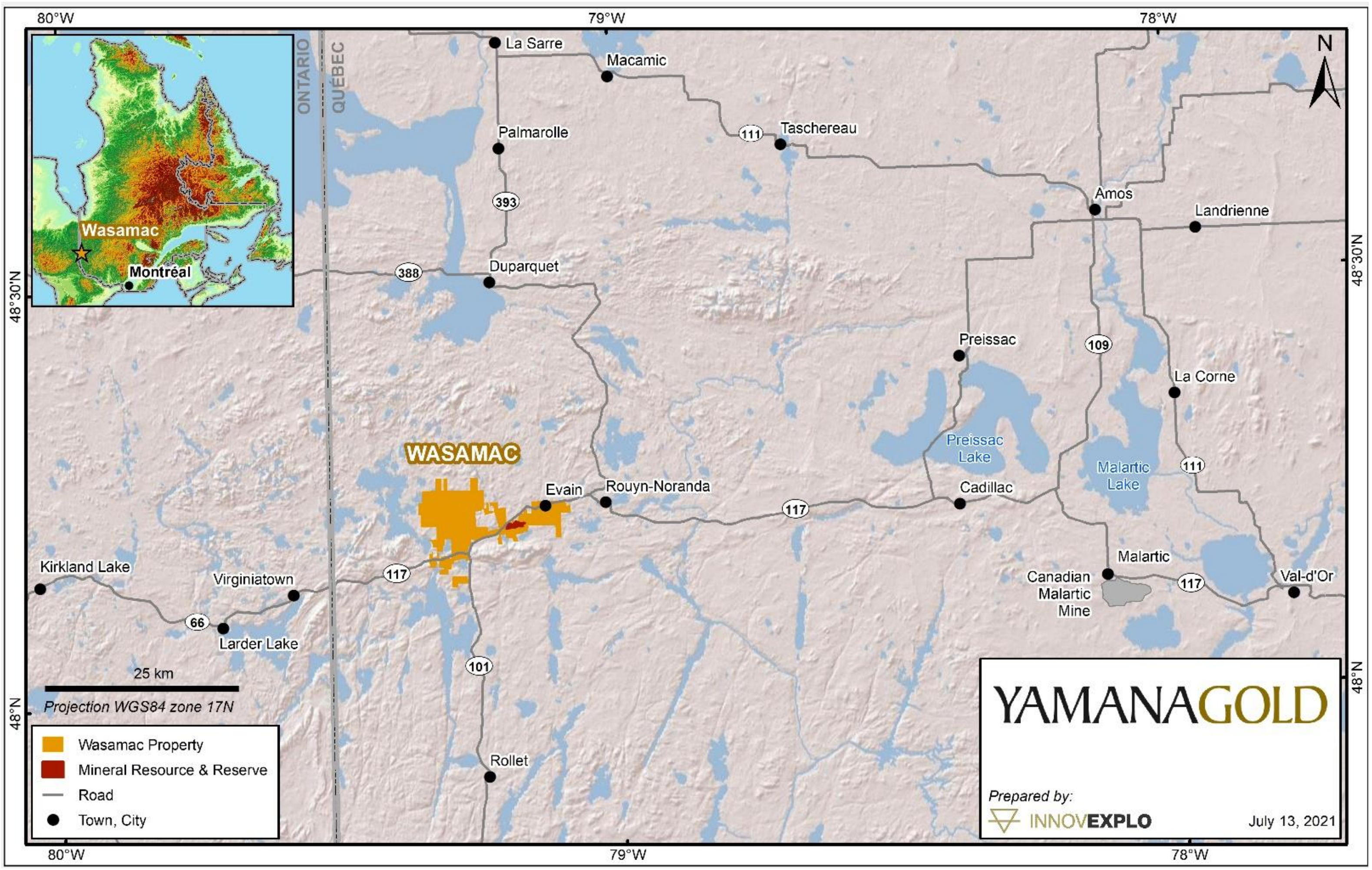


Table 4-1: Wasamac Property Mineral Tenure Table

Title Type	Title Number	Township	Issued Date yyyy-mm-dd	Expiry Date yyyy/mm/dd	Surface (ha)	Ownership (% Responsible)
Wasamac Block						
CM	364	Beauchastel	1948-01-13		349.65	Yamana 100 %
CM	370	Beauchastel	1948-11-26		101.98	Yamana 100 %
CM	349	Beauchastel	1946-11-20		306.02	Yamana 100 %
CLD	6001253	Beauchastel	2000-04-26	2023-04-25	70.41	Yamana 100 %
CL	604891	Beauchastel	1952-05-30	2022-05-14	20	Yamana 100 %
CL	3818441	Beauchastel	1979-08-07	2022-07-10	40	Yamana 100 %
CL	3606841	Beauchastel	1976-07-20	2022-07-02	20.4	Yamana 100 %
CL	3606844	Beauchastel	1976-07-20	2022-07-02	20.4	Yamana 100 %
CL	3606822	Beauchastel	1976-07-20	2022-07-03	40	Yamana 100 %
CL	3817001	Beauchastel	1979-08-07	2022-06-26	40	Yamana 100 %
CL	3606821	Beauchastel	1976-07-20	2022-07-03	40	Yamana 100 %
CL	604903	Beauchastel	1952-05-30	2022-05-14	18.8	Yamana 100 %
CL	604895	Beauchastel	1952-05-30	2022-05-14	12.8	Yamana 100 %
CL	604893	Beauchastel	1952-05-30	2022-05-14	16	Yamana 100 %
CL	3606843	Beauchastel	1976-07-20	2022-07-02	19.6	Yamana 100 %
CL	3723921	Beauchastel	1978-08-14	2022-07-27	40	Yamana 100 %
CL	3816991	Beauchastel	1979-08-07	2022-06-26	40	Yamana 100 %
CL	3723931	Beauchastel	1978-08-14	2022-07-27	40	Yamana 100 %
CL	604892	Beauchastel	1952-05-30	2022-05-14	20	Yamana 100 %
CL	2021987	Beauchastel	1963-04-16	2022-03-28	29	Yamana 100 %
CL	2021981	Beauchastel	1963-04-16	2022-03-28	2.2	Yamana 100 %
CL	653764	Beauchastel	1953-04-30	2022-04-12	2	Yamana 100 %
CL	3818452	Beauchastel	1979-08-07	2022-07-10	40	Yamana 100 %
CL	3731111	Beauchastel	1979-07-06	2022-06-19	40	Yamana 100 %
CL	3806312	Beauchastel	1979-07-06	2022-06-19	40	Yamana 100 %
CL	3816992	Beauchastel	1979-08-07	2022-06-26	40	Yamana 100 %
CL	3806311	Beauchastel	1979-07-06	2022-06-19	40	Yamana 100 %
CL	3818451	Beauchastel	1979-08-07	2022-07-10	40	Yamana 100 %
CL	24292	Beauchastel	1941-04-07	2022-03-01	15.3	Yamana 100 %
CL	653762	Beauchastel	1953-04-30	2022-04-12	6	Yamana 100 %
CL	653763	Beauchastel	1953-04-30	2022-04-12	3.7	Yamana 100 %
CL	3723922	Beauchastel	1978-08-14	2022-07-27	40	Yamana 100 %
CL	653761	Beauchastel	1953-04-30	2022-04-12	7.2	Yamana 100 %
CL	24291	Beauchastel	1941-04-07	2022-03-01	16	Yamana 100 %
CL	3731112	Beauchastel	1979-07-06	2022-06-19	40	Yamana 100 %
CL	604902	Beauchastel	1952-05-30	2022-05-14	32	Yamana 100 %
CL	3606832	Beauchastel	1976-07-20	2022-07-03	23.2	Yamana 100 %
CL	3817002	Beauchastel	1979-08-07	2022-06-26	40	Yamana 100 %
CL	3819151	Beauchastel	1979-08-07	2022-07-10	40	Yamana 100 %
CL	3818442	Beauchastel	1979-08-07	2022-07-10	40	Yamana 100 %
CL	2021972	Beauchastel	1963-04-16	2022-03-28	40	Yamana 100 %
CL	2021971	Beauchastel	1963-04-16	2022-03-28	40	Yamana 100 %
CL	604894	Beauchastel	1952-05-30	2022-05-14	12	Yamana 100 %
CL	604901	Beauchastel	1952-05-30	2022-05-14	30	Yamana 100 %
CL	3606842	Beauchastel	1976-07-20	2022-07-02	19.6	Yamana 100 %
CL	3606831	Beauchastel	1976-07-20	2022-07-03	38.8	Yamana 100 %
CL	C003441	Beauchastel	1930-05-05	2022-04-08	35.56	Yamana 100 %
CDC	1098768	Beauchastel	2002-07-24	2023-07-23	42.45	Yamana 100 %
CDC	20098	Beauchastel	2004-05-20	2023-05-19	1.71	Yamana 100 %
CDC	1098767	Beauchastel	2002-07-24	2023-07-23	42.42	Yamana 100 %
CDC	2175990	Beauchastel	2008-12-19	2021-12-18	16.97	Yamana 100 %
CDC	2210574	Beauchastel	2010-03-15	2023-03-14	14.95	Yamana 100 %
CDC	2210575	Beauchastel	2010-03-15	2023-03-14	15.02	Yamana 100 %
CDC	2287840	Beauchastel	2011-04-26	2022-04-25	1.19	Yamana 100 %
Teck JV Block						
CDC	2385385	Beauchastel	2013-05-15	2022-05-14	16.85	Yamana 100 %
CDC	2450991	N/A	2016-08-31	2022-03-30	1.85	Teck Resources 40 %; Yamana 60 %
CDC	2450992	N/A	2016-08-31	2022-03-30	29.09	Teck Resources 40 %; Yamana 60 %
CDC	2450993	N/A	2016-08-31	2022-03-30	19.53	Teck Resources 40 %; Yamana 60 %
CDC	2450994	N/A	2016-08-31	2022-03-30	4.07	Teck Resources 40 %; Yamana 60 %
CDC	2450995	N/A	2016-08-31	2022-03-30	29.73	Teck Resources 40 %; Yamana 60 %
Wasamac NE Block						
CL	3719852	Beauchastel	1978-07-03	2022-06-13	40	Yamana 100%
CL	3577172	Beauchastel	1976-05-25	2022-05-07	36	Yamana 100%
CL	3577181	Beauchastel	1976-05-25	2022-05-07	34	Yamana 100%
CL	3680522	Beauchastel	1977-07-20	2022-07-03	40	Yamana 100%
CL	3572971	Beauchastel	1976-05-18	2022-05-01	40	Yamana 100%
CL	3680521	Beauchastel	1977-07-20	2022-07-03	40	Yamana 100%
CL	3719862	Beauchastel	1978-07-03	2022-06-13	20	Yamana 100%
CL	3572961	Beauchastel	1976-05-18	2022-05-01	40	Yamana 100%
CL	3572962	Beauchastel	1976-05-18	2022-05-01	40	Yamana 100%
CL	3606862	Beauchastel	1976-07-20	2022-07-01	23.2	Yamana 100%
CL	3606863	Beauchastel	1976-07-20	2022-07-01	22	Yamana 100%

Title Type	Title Number	Township	Issued Date yyyy-mm-dd	Expiry Date yyyy/mm/dd	Surface (ha)	Ownership (% Responsible)
CL	3719851	Beauchastel	1978-07-03	2022-06-13	40	Yamana 100%
CL	3719861	Beauchastel	1978-07-03	2022-06-13	20	Yamana 100%
CL	3577171	Beauchastel	1976-05-25	2022-05-07	38.8	Yamana 100%
CL	3606852	Beauchastel	1976-07-20	2022-07-02	20	Yamana 100%
CL	3577182	Beauchastel	1976-05-25	2022-05-07	30.8	Yamana 100%
CL	3572972	Beauchastel	1976-05-18	2022-05-01	40	Yamana 100%
CDC	2175991	Beauchastel	2008-12-19	2021-12-18	42.38	Yamana 100%
CDC	2179110	Beauchastel	2009-02-10	2022-02-09	3.94	Yamana 100%
CDC	2179111	Beauchastel	2009-02-10	2022-02-09	7.06	Yamana 100%
CDC	2179112	Beauchastel	2009-02-10	2022-02-09	10.02	Yamana 100%
CDC	2225583	Beauchastel	2010-05-03	2023-05-02	20.99	Yamana 100%
CDC	2225584	Beauchastel	2010-05-03	2023-05-02	21.09	Yamana 100%
CDC	2225585	Beauchastel	2010-05-03	2023-05-02	21.07	Yamana 100%
CDC	2225586	Beauchastel	2010-05-03	2023-05-02	20.07	Yamana 100%
CDC	2225587	Beauchastel	2010-05-03	2023-05-02	19.89	Yamana 100%
CDC	2225588	Beauchastel	2010-05-03	2023-05-02	22.44	Yamana 100%
CDC	2225589	Beauchastel	2010-05-03	2023-05-02	14.59	Yamana 100%
CDC	2303323	Beauchastel	2011-07-22	2022-07-21	0.61	Yamana 100%
CDC	2303324	Beauchastel	2011-07-22	2022-07-21	1.61	Yamana 100%
R.M. Nickel						
CDC	2146958	N/A	2008-04-21	2023-04-20	0.05	Yamana 100 %
CDC	2146905	N/A	2008-04-21	2023-04-20	3.18	Yamana 100 %
CDC	2146906	N/A	2008-04-21	2023-04-20	14.2	Yamana 100 %
CDC	2147355	N/A	2008-04-25	2023-04-24	4.75	Yamana 100 %
CDC	2184709	N/A	2009-07-08	2022-07-07	32.11	Yamana 100 %
CDC	2184710	N/A	2009-07-08	2022-07-07	9.56	Yamana 100 %
CDC	2450241	N/A	2017-07-17	2022-04-19	57.32	Yamana 100 %
CDC	2450242	N/A	2017-07-17	2022-04-19	57.22	Yamana 100 %
CDC	2450243	N/A	2017-07-17	2022-04-19	57.32	Yamana 100 %
CDC	2450245	N/A	2017-07-17	2022-04-19	57.34	Yamana 100 %
CDC	2450246	N/A	2017-07-17	2022-04-19	57.33	Yamana 100 %
CDC	2450254	N/A	2017-07-17	2022-04-19	0.05	Yamana 100 %
CDC	2450258	N/A	2017-07-17	2022-04-19	27.56	Yamana 100 %
CDC	2450261	N/A	2017-07-17	2022-04-19	57.32	Yamana 100 %
CDC	2450265	N/A	2017-07-17	2022-04-19	2.16	Yamana 100 %
CDC	2450267	N/A	2017-07-17	2022-04-19	57.33	Yamana 100 %
CDC	2450268	N/A	2017-07-17	2022-04-19	0.03	Yamana 100 %
CDC	2450269	N/A	2017-07-17	2022-04-19	49.76	Yamana 100 %
CDC	2450272	N/A	2017-07-17	2022-04-19	42.16	Yamana 100 %
CDC	2450279	N/A	2017-07-17	2022-04-19	57.32	Yamana 100 %
CDC	2450283	N/A	2017-07-17	2022-04-19	57.33	Yamana 100 %
CDC	2450284	N/A	2017-07-17	2022-04-19	57.32	Yamana 100 %
CDC	2450286	N/A	2017-07-17	2022-04-19	18.77	Yamana 100 %
CDC	2450290	N/A	2017-07-17	2022-04-19	57.33	Yamana 100 %
CDC	2450300	N/A	2017-07-17	2022-04-19	57.34	Yamana 100 %
CDC	2450302	N/A	2017-07-17	2022-04-19	31.63	Yamana 100 %
CDC	2450314	N/A	2017-07-17	2022-04-19	57.33	Yamana 100 %
CDC	2450315	N/A	2017-07-17	2022-04-19	29.87	Yamana 100 %
CDC	2450316	N/A	2017-07-17	2022-04-19	41.07	Yamana 100 %
CDC	2450327	N/A	2017-07-17	2022-04-19	51.71	Yamana 100 %
CDC	2450336	N/A	2017-07-17	2022-04-19	57.33	Yamana 100 %
CDC	2450341	N/A	2017-07-17	2022-04-19	57.33	Yamana 100 %
CDC	2450343	N/A	2017-07-17	2022-04-19	57.31	Yamana 100 %
CDC	2450347	N/A	2017-07-17	2022-04-19	0.01	Yamana 100 %
CDC	2450350	N/A	2017-07-17	2022-04-19	57.34	Yamana 100 %
CDC	2450355	N/A	2017-07-17	2022-04-19	57.32	Yamana 100 %
CDC	2450356	N/A	2017-07-17	2022-04-19	57.34	Yamana 100 %
CDC	2496271	N/A	2017-07-17	2022-04-19	18.12	Yamana 100 %
CDC	2496277	N/A	2017-07-17	2022-04-19	28.79	Yamana 100 %
CDC	2496279	N/A	2017-07-17	2022-04-19	28.82	Yamana 100 %
CDC	2496280	N/A	2017-07-17	2022-04-19	19.55	Yamana 100 %
CDC	1101738	Dasserat	2002-09-23	2023-10-09	71.84	Yamana 100 %
CDC	1100366	Dasserat	2002-08-22	2023-10-09	43.05	Yamana 100 %
CDC	30278	Dasserat	2004-07-26	2023-07-25	42.07	Yamana 100 %
CDC	30279	Dasserat	2004-07-26	2023-07-25	42.21	Yamana 100 %
CDC	1100369	Dasserat	2002-08-22	2023-10-09	36.61	Yamana 100 %
CDC	1100365	Dasserat	2002-08-22	2023-10-09	43.66	Yamana 100 %
CDC	1100364	Dasserat	2002-08-22	2023-10-09	41.25	Yamana 100 %
CDC	1100367	Dasserat	2002-08-22	2023-10-09	58.78	Yamana 100 %
CDC	1100370	Dasserat	2002-08-22	2023-10-09	40.89	Yamana 100 %
CDC	1100368	Dasserat	2002-08-22	2023-10-09	34.25	Yamana 100 %
CDC	1101739	Dasserat	2002-09-23	2023-10-09	52.3	Yamana 100 %
CDC	1101740	Dasserat	2002-09-23	2023-10-09	118.57	Yamana 100 %
CDC	2146908	Dasserat	2008-04-21	2023-04-20	48.8	Yamana 100 %
CDC	2146909	Dasserat	2008-04-21	2023-04-20	42.28	Yamana 100 %
CDC	2146910	Dasserat	2008-04-21	2023-04-20	43.32	Yamana 100 %
CDC	2146911	Dasserat	2008-04-21	2023-04-20	37.71	Yamana 100 %

Title Type	Title Number	Township	Issued Date yyyy-mm-dd	Expiry Date yyyy/mm/dd	Surface (ha)	Ownership (% Responsible)
CDC	2146912	Dasserat	2008-04-21	2023-04-20	53.09	Yamana 100 %
CDC	2146913	Dasserat	2008-04-21	2023-04-20	54.95	Yamana 100 %
CDC	2194152	Dasserat	2009-11-10	2022-11-09	42.49	Yamana 100 %
CDC	2184729	Dasserat	2009-07-09	2022-07-08	41.06	Yamana 100 %
CDC	2184730	Dasserat	2009-07-09	2022-07-08	2.77	Yamana 100 %
CDC	2184731	Dasserat	2009-07-09	2022-07-08	2.77	Yamana 100 %
CDC	2184732	Dasserat	2009-07-09	2022-07-08	30.53	Yamana 100 %
CDC	2184733	Dasserat	2009-07-09	2022-07-08	42.65	Yamana 100 %
CDC	2184734	Dasserat	2009-07-09	2022-07-08	29.57	Yamana 100 %
CDC	2241667	Dasserat	2010-07-21	2023-07-20	42.68	Yamana 100 %
CDC	2241668	Dasserat	2010-07-21	2023-07-20	42.7	Yamana 100 %
CDC	2241669	Dasserat	2010-07-21	2023-07-20	42.64	Yamana 100 %
CDC	2241670	Dasserat	2010-07-21	2023-07-20	42.63	Yamana 100 %
CDC	2241671	Dasserat	2010-07-21	2023-07-20	42.68	Yamana 100 %
CDC	2384642	Dasserat	2013-04-26	2022-04-25	42.6	Yamana 100 %
CDC	2384643	Dasserat	2013-04-26	2022-04-25	42.61	Yamana 100 %
CDC	2385384	Beauchastel	2013-05-15	2022-05-14	21.6	Yamana 100 %
Francoeur/Lac Fortune/Arntfield Block						
BM	1006	Beauchastel	2012-06-27	2032-06-26	44.59	Yamana 100%
BM	776	Beauchastel	1988-07-14	2028-07-13	46.18	Yamana 100%
BM	825	Beauchastel	1995-09-08	2025-09-07	1.67	Yamana 100%
BM	826	Beauchastel	1995-09-08	2025-09-07	2.15	Yamana 100%
BM	849	Beauchastel	2000-04-17	2030-04-16	9.36	Yamana 100%
CM	194	Beauchastel	1926-12-20		82	Yamana 100%
CM	322	Beauchastel	1941-10-17		61.66	Yamana 100%
CM	326	Beauchastel	1942-04-30		90.1	Yamana 100%
CDC	1100346	Beauchastel	2002-08-22	2023-10-09	41.81	Yamana 100%
CDC	1100347	Beauchastel	2002-08-22	2023-10-09	41.82	Yamana 100%
CDC	1100348	Beauchastel	2002-08-22	2023-10-09	41.8	Yamana 100%
CDC	1100349	Beauchastel	2002-08-22	2023-10-09	41.78	Yamana 100%
CDC	1100350	Beauchastel	2002-08-22	2023-10-09	41.77	Yamana 100%
CDC	2450231	N/A	2017-07-17	2022-04-19	57.41	Yamana 100%
CDC	2450232	N/A	2017-07-17	2022-04-19	57.41	Yamana 100%
CDC	2450233	N/A	2017-07-17	2022-04-19	0.6	Yamana 100%
CDC	2450240	N/A	2017-07-17	2022-04-19	57.35	Yamana 100%
CDC	2450248	N/A	2017-07-17	2022-04-19	34.63	Yamana 100%
CDC	2450250	N/A	2017-07-17	2022-04-19	49.49	Yamana 100%
CDC	2450251	N/A	2017-07-17	2022-04-19	34.65	Yamana 100%
CDC	2450253	N/A	2017-07-17	2022-04-19	6.71	Yamana 100%
CDC	2450260	N/A	2017-07-17	2022-04-19	30.69	Yamana 100%
CDC	2450262	N/A	2017-07-17	2022-04-19	57.35	Yamana 100%
CDC	2450264	N/A	2017-07-17	2022-04-19	30.74	Yamana 100%
CDC	2450270	N/A	2017-07-17	2022-04-19	4.58	Yamana 100%
CDC	2450273	N/A	2017-07-17	2022-04-19	21.09	Yamana 100%
CDC	2450282	N/A	2017-07-17	2022-04-19	49.17	Yamana 100%
CDC	2450289	N/A	2017-07-17	2022-04-19	4.4	Yamana 100%
CDC	2450292	N/A	2017-07-17	2022-04-19	2.84	Yamana 100%
CDC	2450293	N/A	2017-07-17	2022-04-19	57.34	Yamana 100%
CDC	2450294	N/A	2017-07-17	2022-04-19	38.69	Yamana 100%
CDC	2450296	N/A	2017-07-17	2022-04-19	1.1	Yamana 100%
CDC	2450305	N/A	2017-07-17	2022-04-19	56.87	Yamana 100%
CDC	2450307	N/A	2017-07-17	2022-04-19	31.16	Yamana 100%
CDC	2450309	N/A	2017-07-17	2022-04-19	0.18	Yamana 100%
CDC	2450312	N/A	2017-07-17	2022-04-19	2.59	Yamana 100%
CDC	2450313	N/A	2017-07-17	2022-04-19	6.02	Yamana 100%
CDC	2450318	N/A	2017-07-17	2022-04-19	48.5	Yamana 100%
CDC	2450321	N/A	2017-07-17	2022-04-19	2.79	Yamana 100%
CDC	2450322	N/A	2017-07-17	2022-04-19	24.39	Yamana 100%
CDC	2450323	N/A	2017-07-17	2022-04-19	35.64	Yamana 100%
CDC	2450325	N/A	2017-07-17	2022-04-19	57.35	Yamana 100%
CDC	2450329	N/A	2017-07-17	2022-04-19	7.1	Yamana 100%
CDC	2450331	N/A	2017-07-17	2022-04-19	29.6	Yamana 100%
CDC	2450334	N/A	2017-07-17	2022-04-19	23.2	Yamana 100%
CDC	2450339	N/A	2017-07-17	2022-04-19	53.93	Yamana 100%
CDC	2450344	N/A	2017-07-17	2022-04-19	30.79	Yamana 100%
CDC	2450349	N/A	2017-07-17	2022-04-19	52.28	Yamana 100%
CDC	2450353	N/A	2017-07-17	2022-04-19	26.8	Yamana 100%
CDC	2450354	N/A	2017-07-17	2022-04-19	0.17	Yamana 100%
CDC	2450358	N/A	2017-07-17	2022-04-19	13.67	Yamana 100%
CDC	2496263	N/A	2017-07-17	2022-04-19	20.89	Yamana 100%
CDC	2496264	N/A	2017-07-17	2022-04-19	54.46	Yamana 100%
CDC	2496265	N/A	2017-07-17	2022-04-19	1.78	Yamana 100%
CDC	2496266	N/A	2017-07-17	2022-04-19	0.36	Yamana 100%
CDC	2496267	N/A	2017-07-17	2022-04-19	4.5	Yamana 100%
CDC	2496561	N/A	2017-07-18	2022-07-22	7.79	Yamana 100%
CDC	2499489	N/A	2017-08-08	2022-08-07	20.97	Yamana 100%
CDC	2501595	N/A	2017-10-23	2022-02-20	0.75	Yamana 100%

Title Type	Title Number	Township	Issued Date yyyy-mm-dd	Expiry Date yyyy/mm/dd	Surface (ha)	Ownership (% Responsible)
CDC	2501596	N/A	2017-10-23	2022-02-20	7.87	Yamana 100%
CDC	2501597	N/A	2017-10-23	2022-02-20	9.99	Yamana 100%
CDC	2501601	N/A	2017-10-23	2022-02-20	57.37	Yamana 100%
CDC	2501602	N/A	2017-10-23	2022-02-20	0.65	Yamana 100%
CDC	2501603	N/A	2017-10-23	2022-02-20	1.71	Yamana 100%
CDC	2501604	N/A	2017-10-23	2022-02-20	1.81	Yamana 100%
CDC	2501605	N/A	2017-10-23	2022-02-20	35.68	Yamana 100%
CDC	2501606	N/A	2017-10-23	2022-02-20	14.42	Yamana 100%
CDC	2501607	N/A	2017-10-23	2022-02-20	56.32	Yamana 100%
CDC	2501608	N/A	2017-10-23	2022-02-20	57.36	Yamana 100%
CDC	2501609	N/A	2017-10-23	2022-02-20	0.12	Yamana 100%
CDC	2501610	N/A	2017-10-23	2022-02-20	57.38	Yamana 100%
CDC	2501611	N/A	2017-10-23	2022-02-20	19.23	Yamana 100%
CDC	2501612	N/A	2017-10-23	2022-02-20	51.63	Yamana 100%
CDC	2501613	N/A	2017-10-23	2022-02-20	28.04	Yamana 100%
CDC	2501614	N/A	2017-10-23	2022-02-20	57.36	Yamana 100%
CDC	2501615	N/A	2017-10-23	2022-02-20	55.87	Yamana 100%
CDC	2501616	N/A	2017-10-23	2022-02-20	27.45	Yamana 100%
CDC	2501617	N/A	2017-10-23	2022-02-20	2.15	Yamana 100%
CDC	2501618	N/A	2017-10-23	2022-02-20	56.17	Yamana 100%
CDC	2501619	N/A	2017-10-23	2022-02-20	14.63	Yamana 100%
CDC	2501620	N/A	2017-10-23	2022-02-20	17.23	Yamana 100%
CDC	2501621	N/A	2017-10-23	2022-02-20	8.36	Yamana 100%
CDC	2501622	N/A	2017-10-23	2022-02-20	57.36	Yamana 100%
CDC	2501623	N/A	2017-10-23	2022-02-20	10.2	Yamana 100%
CDC	2501624	N/A	2017-10-23	2022-02-20	2.77	Yamana 100%
CDC	2501625	N/A	2017-10-23	2022-02-20	24.21	Yamana 100%
CDC	2501626	N/A	2017-10-23	2022-02-20	44.68	Yamana 100%
CDC	2501627	N/A	2017-10-23	2022-02-20	57.35	Yamana 100%
CDC	2501628	N/A	2017-10-23	2022-02-20	43	Yamana 100%
CDC	2501629	N/A	2017-10-23	2022-02-20	34.26	Yamana 100%
CDC	2501631	N/A	2017-10-23	2022-02-20	37.79	Yamana 100%
CDC	2501632	N/A	2017-10-23	2022-02-20	57.35	Yamana 100%
CDC	2501633	N/A	2017-10-23	2022-02-20	39.23	Yamana 100%
CDC	2501634	N/A	2017-10-23	2022-02-20	28.53	Yamana 100%
CDC	2501635	N/A	2017-10-23	2022-02-20	25.67	Yamana 100%
CDC	2501637	N/A	2017-10-23	2022-02-20	26.83	Yamana 100%
CDC	2501638	N/A	2017-10-23	2022-02-20	57.37	Yamana 100%
CDC	2501639	N/A	2017-10-23	2022-02-20	57.37	Yamana 100%
CDC	2501640	N/A	2017-10-23	2022-02-20	9.16	Yamana 100%
CDC	2501641	N/A	2017-10-23	2022-02-20	26.92	Yamana 100%
CDC	2501644	N/A	2017-10-23	2022-02-20	1.48	Yamana 100%
CDC	2501649	N/A	2017-10-23	2022-02-20	0.07	Yamana 100%
CDC	2560899	N/A	2020-03-27	2023-03-26	57.34	Yamana 100%
CDC	2587958	N/A	2020-12-03	2022-11-06	29.63	Yamana 100%
CDC	2587959	N/A	2020-12-03	2022-11-06	36.19	Yamana 100%
CDC	2587960	N/A	2020-12-03	2022-11-06	42.43	Yamana 100%
CDC	2587961	N/A	2020-12-03	2022-11-06	53.04	Yamana 100%
CDC	2608278	N/A	2021-05-18	2022-03-21	53.64	Yamana 100%
CDC	2608279	N/A	2021-05-18	2022-03-21	56.61	Yamana 100%
CDC	2608280	N/A	2021-05-18	2022-03-21	29.34	Yamana 100%
CDC	2608281	N/A	2021-05-18	2022-03-21	57.36	Yamana 100%
CDC	2608282	N/A	2021-05-18	2022-03-21	57.36	Yamana 100%
CDC	2608283	N/A	2021-05-18	2022-03-21	33.34	Yamana 100%
CDC	2608284	N/A	2021-05-18	2022-05-07	16.07	Yamana 100%
CDC	2608285	N/A	2021-05-18	2022-05-07	57.38	Yamana 100%
CDC	2608286	N/A	2021-05-18	2022-05-07	57.38	Yamana 100%
CDC	2608287	N/A	2021-05-18	2022-09-26	5.01	Yamana 100%
CDC	2608294	N/A	2021-05-18	2022-06-05	57.39	Yamana 100%
CDC	2608295	N/A	2021-05-18	2022-06-05	57.39	Yamana 100%
CDC	2608296	N/A	2021-05-18	2022-06-05	57.39	Yamana 100%
CDC	2608482	N/A	2021-05-20	2022-06-05	16.05	Yamana 100%
Western Buff Block						
CDC	2134905	Beauchastel	2007-10-31	2022-10-30	51.61	Yamana 100%
CDC	2134906	Beauchastel	2007-10-31	2022-10-30	39.61	Yamana 100%
CDC	2134907	Beauchastel	2007-10-31	2022-10-30	39.45	Yamana 100%
CDC	2134908	Beauchastel	2007-10-31	2022-10-30	36.09	Yamana 100%
CDC	2134909	Beauchastel	2007-10-31	2022-10-30	37.19	Yamana 100%
CDC	2134910	Beauchastel	2007-10-31	2022-10-30	33.31	Yamana 100%
CDC	2134911	Beauchastel	2007-10-31	2022-10-30	23.63	Yamana 100%

Figure 4-2: Claim Map of the Property

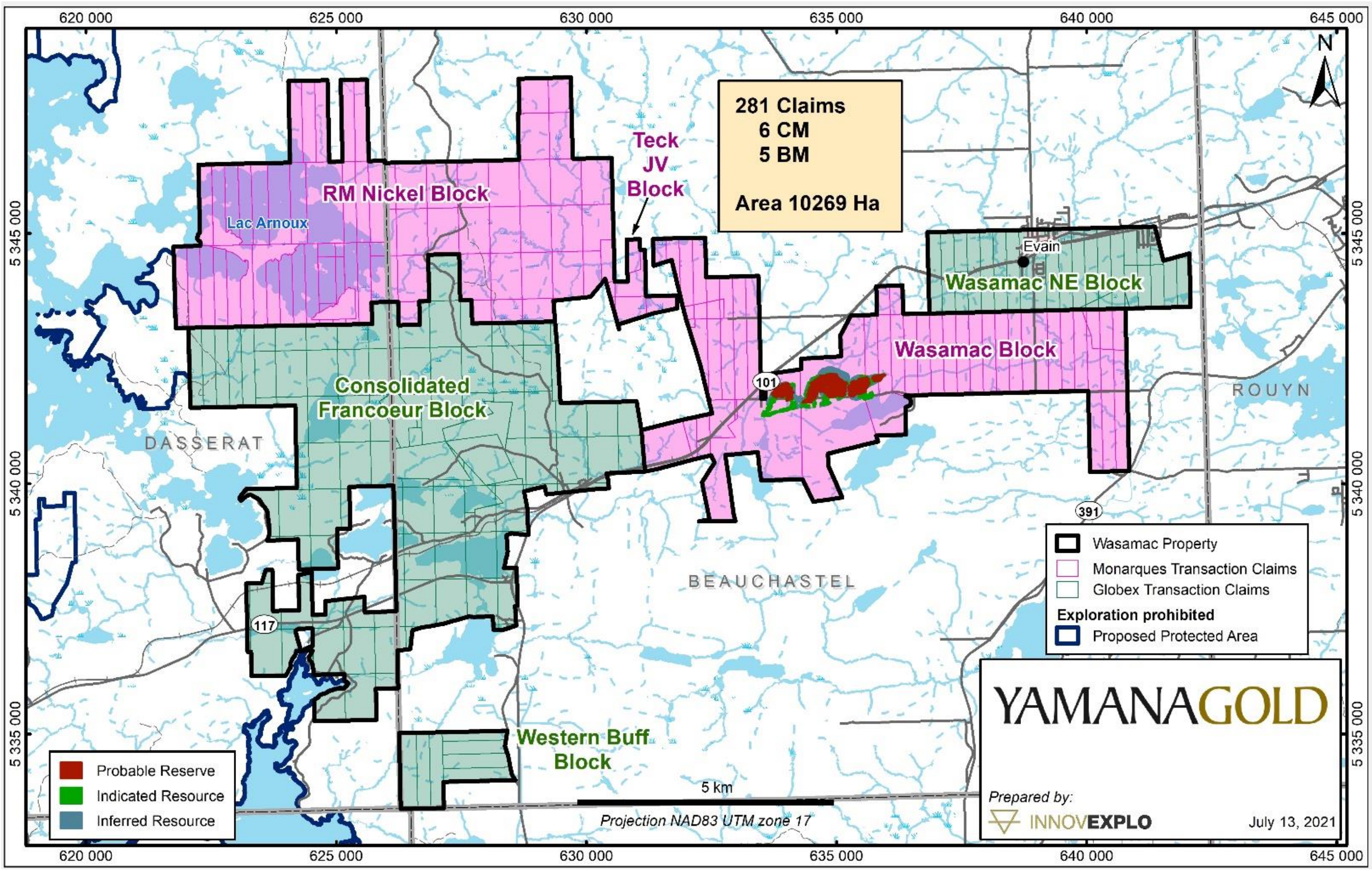
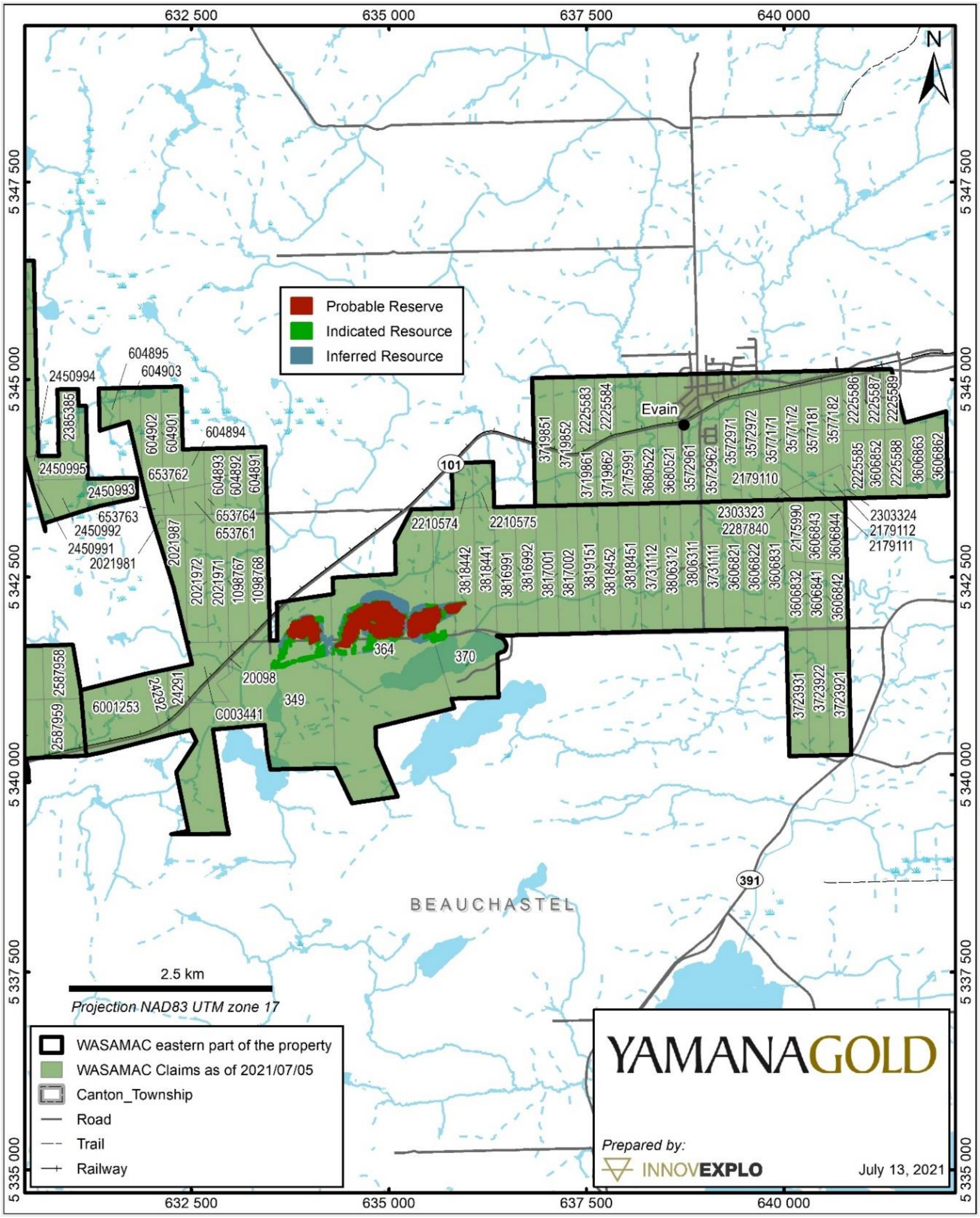
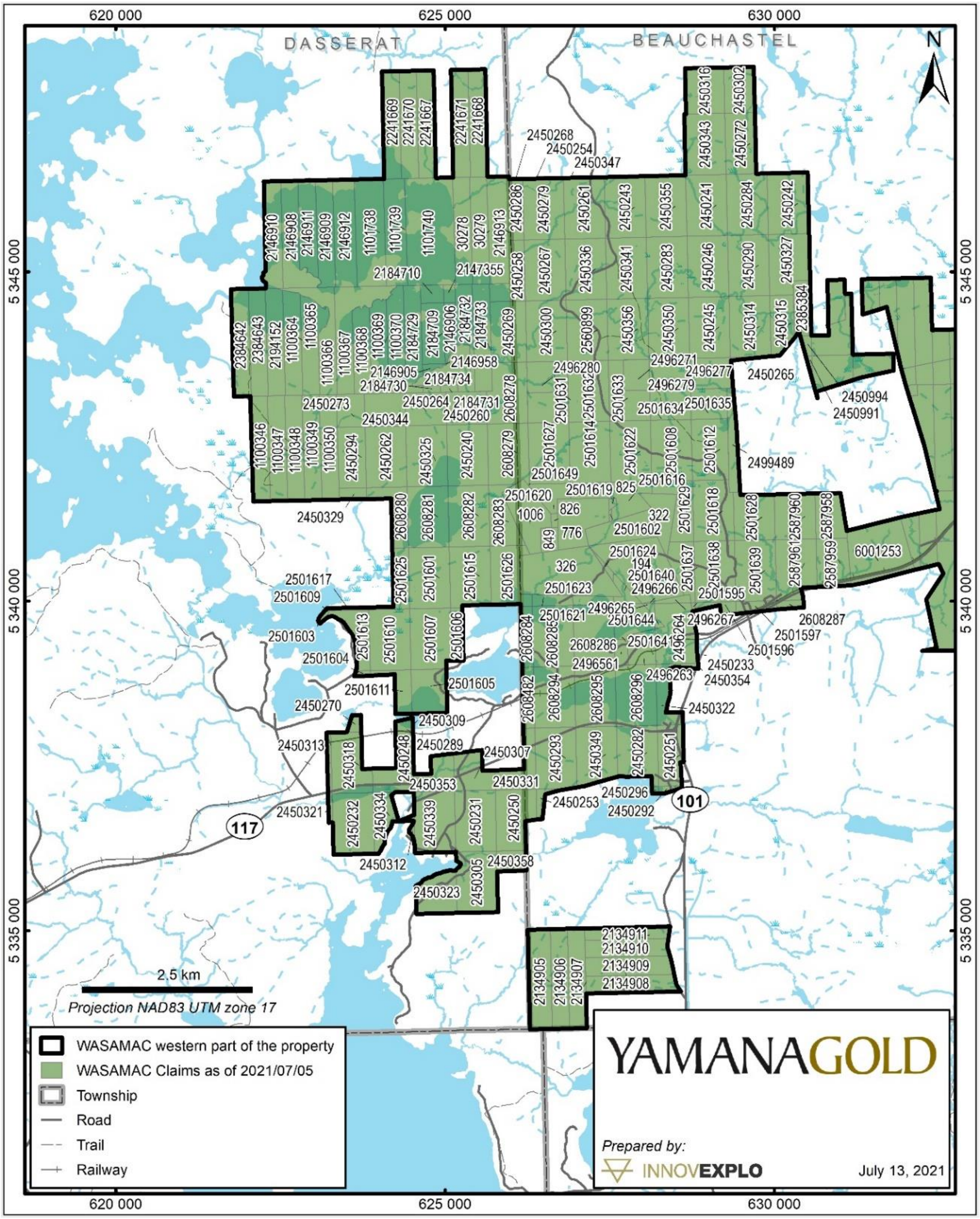


Figure 4-3: Claim Map of the Eastern Part of the Property



Note: Includes Wasamac and Wasamac NE Block.

Figure 4-4: Claim Map of the Western Part of the Property



Note: Includes R.M. Nickel, Francoeur/Lac Fortune/Arntfield and Western Buff Block.

4.3 Acquisition of the Property Agreements

4.3.1 Acquisition of the Wasamac, R.M. Nickel and Teck JV Blocks

On January 21, 2021, Yamana completed the acquisition of the Wasamac Block hosting the Wasamac deposit, the R.M. Nickel Block, the Teck JV Block, the Camflo property and the Camflo mill through the acquisition of all of the outstanding shares of Monarch not owned by Yamana. Yamana had previously announced that it had entered into a definitive agreement with Monarch on November 2, 2020 to acquire the properties under a plan of arrangement (the “Transaction”). Under the terms of the Transaction, Monarch shareholders received the following per Monarch share: 0.0376 of a Yamana share; C$0.192 in cash; and 0.2 of a share of Monarch Mining. Yamana issued 11,608,195 Yamana shares, paid US$46.9 million (C$59.3 million) in cash, and issued 383,764 replacement warrants, for total consideration paid of US$108.6 million. Yamana’s consideration on close represented a value paid for the Wasamac asset of under US$67 per ounce of mineral reserves and under US$42 per ounce of mineral resources, based on the mineral reserves and mineral resources estimates in the 2018 Feasibility Study (Caumartin et al., 2018) and net of Yamana’s existing Monarch interest in Wasamac. In connection with the Transaction, Monarch completed a spin-out to its shareholders of its other mineral properties and certain other assets and liabilities through a newly formed company, Monarch Mining Corporation (Monarch Mining). Yamana also acquired 6.7% of the outstanding shares of the newly formed Monarch Mining as part of the Transaction.

4.3.2 Acquisition of the Consolidated Francoeur, Wasamac NE and Western Buff Blocks

On June 14, 2021, Yamana announced that it agreed to acquire from Globex Mining Enterprises Inc. (Globex) the Francoeur, Arntfield and Lac Fortune gold properties (the “Consolidated Francoeur Block”) adjoining the Wasamac Project, the Western Buff Block, as well as additional claims in the Beauchastel Township to the east of the Wasamac Project (the “Wasamac NE Block”) (the “Agreement”).

Pursuant to the terms of the Agreement, Yamana will pay an initial amount of C$4 million on closing, which at the direction of Globex will be paid in shares, with the remaining payment of C$11 million payable over four years in either cash or shares at the discretion of Globex. In addition, Globex will receive a 2% gross metal royalty from Yamana, of which 0.5% may be bought back at any time by Yamana for C$1.5 million following which the royalty would be reduced to a 1.5% gross metal royalty.

The Transaction also included certain other claims in Malartic township adjacent to the Camflo property, which Yamana previously acquired and has transferred to the Canadian Malartic General Partnership. These additional claims will be made available to the partnership. The closing of the Transaction took place on June 22, 2021.

4.4 Mineral Tenure

In the Province of Québec, the management of mineral resources and the granting of exploration and mining rights for mineral substances and their use are regulated by the Québec Mining Act, which is administered by the Ministry of Energy and Natural Resources (Ministère de l’Énergie et des Ressources Naturelles or MERN). Mineral rights are owned by the Crown and are distinct from surface rights.

The six mining concessions (CM349, CM364, CM370, C322, C326, C194) that comprise part of the property are in good standing.

In Québec, a map-designated claim is valid for two years and can be renewed indefinitely subject to the completion of necessary expenditure requirements and payment of renewal fees. Each claim gives the holder an exclusive right to search for mineral substances, except sand, gravel, clay, and other unconsolidated deposits on the land subjected to the claim. The claim also guarantees the holder's right to obtain an extraction permit upon discovery of a mineral deposit. Ownership of the mining rights confers the right to acquire the surface rights.

4.5 Surface Rights

Yamana currently has all the surface rights for its mining leases and mining concessions, including the mineral resource and mineral reserve area of the Wasamac deposit (this study). Other exploration claims included in the property are either located on Crown land or on private land. Yamana has the first right to acquire the surface rights to the claims by taking it to the mining lease status. Under Québec Mining Legislation, the owner of the mining rights can make use of the timber on the leased property by paying a nominal fee if such timber is deemed to be of commercial value.

The QPs have not independently reviewed the ownership of the surface rights (see Section 3.2).

4.6 Royalties and Encumbrances

Yamana holds a 100% interest in the property except for the six claims comprising the Teck JV Block in which it holds a 60% interest. With respect to its former properties, the Consolidated Francoeur, Wasamac NE and Western Buff Blocks, Globex will receive a 2% gross metal royalty from Yamana, of which 0.5% may be bought back at any time by Yamana for C$1.5 million, following which the royalty would be reduced to a 1.5% gross metal royalty.

A 1.5% net smelter return (NSR) royalty is payable to Metalla Royalty & Streaming Ltd. upon commercial production of the Wasamac deposit (three mining concessions covering 757.65 ha and 11 mining claims of the Wasamac Block for a total area of 1,149.33 ha). A buyback right of 0.5% for C$7.5 million is retained by Yamana.

The QPs are not aware of any other royalties (except those mentioned above), back-in rights, or other obligations.

4.7 Permitting Considerations

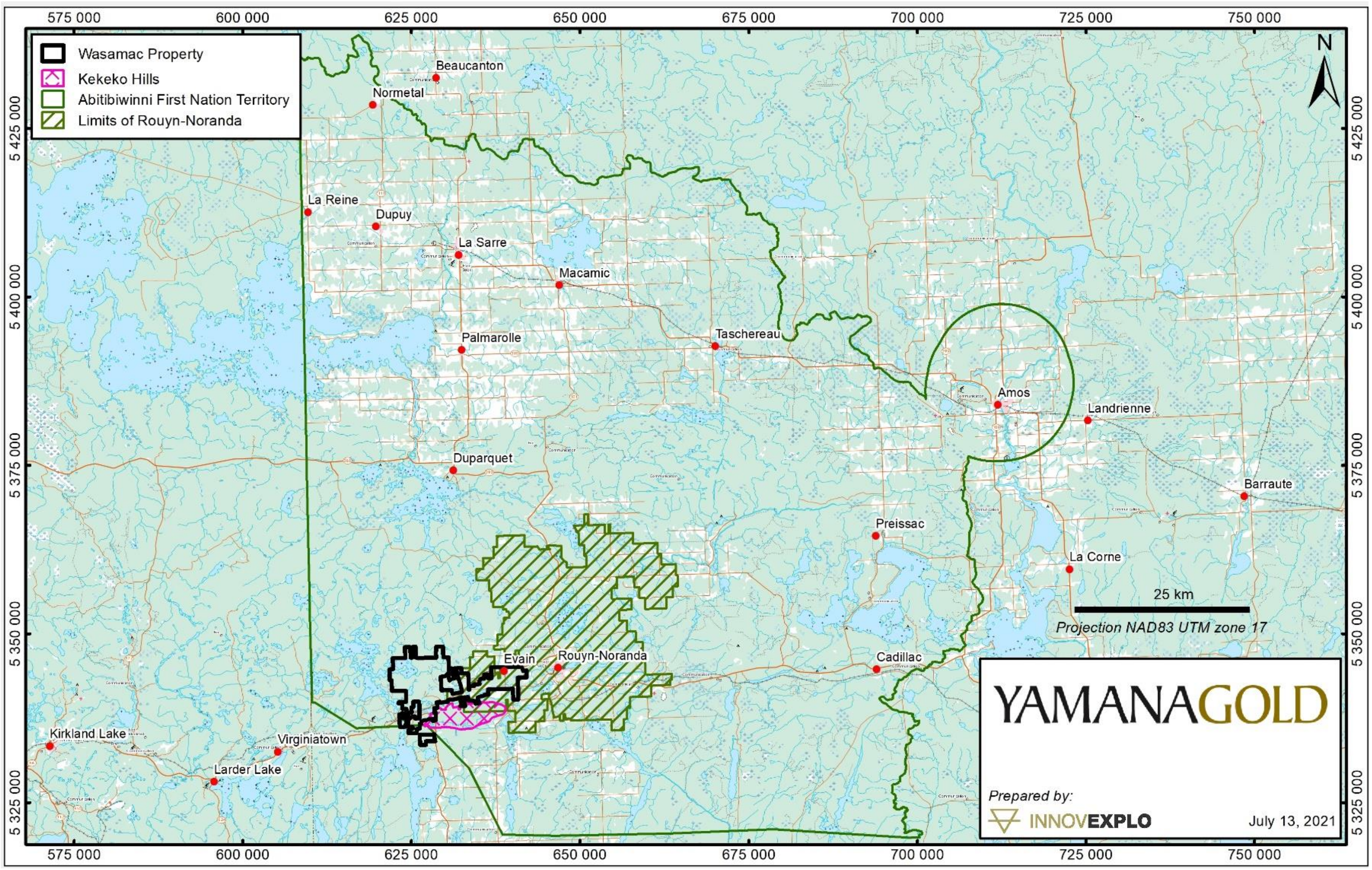
The property is located within the Abitibiwinni First Nation Territory (see Figure 4-5). Holders of claims located within the area of application of the *Agreement on Consultation and Accommodation* between the Abitibiwinni First Nation Council and The Government of Québec are asked to contact the "Secrétariat aux ressources naturelles".

The Wasamac and Wasamac NE blocks fall within a "Referred to Minister" designation in lands surrounding towns, in this case the municipal limits of Rouyn-Noranda. Currently, exploration and mining activities are allowed but this designation states that if the claims lapse, exploration is prohibited/withdrawn from mining and could then be withdrawn from future staking.

The southwestern part of the Wasamac Block, and southeastern part of the Consolidated Francoeur Block fall within a proposed Protected Area; the Kekeko Hills ("Collines de Kekeko") where exploration activities are allowed under certain conditions.

In general, minimal permitting is required to carry out the work program considered in this report. For drilling, however, Yamana will have to obtain certain permits and certification from relevant governmental agencies, including a timber permit (Autorisation de coupe de bois sur un territoire du domaine de l'État où s'exerce un droit minier) from MERN. Please refer to Section 20 for required permitting to carry out the work program considered in this report.

Figure 4-5: Implication of Various Levels of Restriction of the Property



Source: MERN, 2021

4.8 Environmental Considerations

The QPs are not aware of any environmental liabilities associated with the property.

The QPs are not aware of any other significant factors or risks that could affect access, title, or the right or ability to estimate the mineral resources or mineral reserves or perform work on the property.

4.9 Social License Considerations

Maintaining and strengthening the social license to operate is a critical part of Yamana's mining operations. The property is located within the Abitibiwinni First Nation (other First Nations have also provided maps of their territory covering the area of the property) and is partly within the municipal limit of Rouyn-Noranda and is thus socially sensitive. Social acceptance will be of crucial importance for the success of the project.

Yamana has a variety of mechanisms that help guide them to industry best practice and to fully understand the concerns of the stakeholders, mainly their Health and Safety, Environment and Community (HSEC) Framework and the Social License to Operate (SLO) Index. The framework is based on three pillars of social risk management: (1) stakeholder engagement, (2) impact management and (3) benefit management.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

The property is located approximately 15 km west-southwest of the city of Rouyn-Noranda, which is serviced by daily flights to Montreal. The property is accessed from Provincial Highway 117 (the Trans-Canada Highway) that links Rouyn-Noranda and the community of Arntfield (see Figure 4-1). A secondary road (Rang des Cavaliers) leads directly to the Wasamac deposit from Provincial Highway 117.

5.2 Climate

The Wasamac property is under the influence of a typical continental-style climate marked by cold, dry winters and warm, humid summers. According to Environment Canada1 climate data at the nearest weather station (Amos), the average temperatures are +17.2°C in July and -17.3°C in January. The mean annual temperature is +1.2°C, slightly above freezing. The lowest recorded temperature was -52.8°C, and the highest was +37.2°C. In this area, the temperature drops below freezing point an average of 204.9 days per year. Snow accumulates from mid-October or November to early/mid-May. Freeze-up usually occurs in late December and break-up in March and April. Average annual precipitation indicates a mean rainfall of 918.4 mm, with the highest precipitation occurring in July (112.8 mm).

Climate conditions do not have a significant impact on mining and exploration activities; they can be performed year round. However, it is preferable to conduct drilling on the few marshy areas on the Property during winter when the ground is frozen. In addition, geological and geochemical prospecting, mapping and surveys are generally restricted to the months from May to October.

5.3 Local Resources and Infrastructure

The area is well served by existing infrastructure and human resources from Rouyn-Noranda, a well-established mining centre with a population of approximately 42,000. The area has a network of paved provincial roads, including highways, and commercial airline companies service the local airport. Many suppliers and manufacturers in the mining industry are based in Rouyn-Noranda and other nearby communities. Hydro-Québec supplies electricity to the area. Skilled administrative personnel, technicians, geologists, mining engineers and experienced miners are available in the area.

The Ontario Northland Railway runs north of the property, parallel to Provincial Highway 117.

[1] Environment Canada: Historical Weather and Climate

5.3.1 Wasamac Mine

In the past, the Wasamac mine had an inclined shaft dipping to the north in the footwall of the Main Zone to depth of approximately 1,400 ft (420 m). Drifting was done on seven main levels (every 200 vertical ft) to approximately 1,310 ft (400 m) below surface. Two lateral drifts accessed Zone 1 and Zone 2 towards the east (at the 400 ft and 800 ft levels). Immediately to the south, the Wildcat shaft was used as a ventilation raise and connected to the Wasamac mine by a drift at the 200 ft level.

The mine was closed in 1971 and is currently flooded. All infrastructure was dismantled, and equipment was removed from site. As ore has been processed at the mine site, there is a legacy non-acid generating tailings pond in the centre of the property.

Access to the mine site is fenced and secondary access points are closed (obstructed). Surveillance cameras and alarm systems are installed on the site and in the various buildings. A Hydro-Québec power line goes to the mine site.

The surface rights covering the area of the old infrastructure and of the tailings pond are owned by Yamana.

5.3.2 Francoeur Mine

Access to the historical Francoeur mine site is fenced, and secondary access points are closed (obstructed). Surveillance cameras and alarm systems are installed on the site and in the various buildings. A Hydro-Québec power line goes to the mine site.

Some of the main infrastructure still in place includes: the headframe and underground galleries of the mine, administrative buildings, core shack, garages and warehouses, and wells 6 and 7. A complete list of infrastructures present on the site is found in the internal report entitled "*Plan de restauration pour la mise en production de la mine Francoeur en vertu de l 'article 232.2 de la Loi sur les mines*" (Restoration plan for the putting into production of the Francoeur mine under section 232.2 of the Mining Act) (Genivar, 2012).

The surface rights covering the area of the historic infrastructure and of the tailings pond are owned by Yamana.

5.4 Physiography

The regional landscape is typical of the Abitibi Lowlands, with small rolling hills and widespread wetlands and swamps, and mixed broadleaf and conifer forests. The topography is relatively flat (average altitude is around 285 to 300 masl). Exception are local areas of exposed outcrops, such as Mont Kekeko to the south and the northeastern part of the property.

The region provides habitat for various large mammal species: moose, black bear, lynx, snowshoe hare, beaver, wolf, and coyote. Bird species include sharp-tailed grouse, black duck, wood duck, hooded merganser, and pileated woodpecker.

Further information on the flora and fauna present at the property is presented in Section 20.

5.5 Seismicity

A site-specific seismic hazard analysis carried out by the Geological Survey of Canada (latitude/longitude coordinates 48.2095°N/79.2189°W), using the 2015 National Building Code of Canada seismic hazard calculator, determined that the peak ground horizontal accelerations for a return period of 2% / 50 years is equal to 0.085 g (see Table 5-1).

Table 5-1: 2% / 50 Years (0.000404 per annum) Probability

Sa (0.05)	Sa (0.1)	Sa (0.2)	Sa (0.30	Sa (0.5)	Sa (1.0)	Sa (2.0)	Sa (5.0)	Sa (10.0)	PGA	PGV
0.112	0.149	0.139	0.115	0.090	0.052	0.027	0.0069	0.0030	0.085	0.074

Note: *Spectral (Sa(T)), where T is in seconds; peak ground acceleration (PGA) values are given in units of g (9.81 m/s^2); peak ground velocity (PGV) is given in m/s.

5.6 Comments on Accessibility, Climate, Local Resources, Infrastructure and Physiography

The QP is of the opinion that, to the extent relevant to the project, there is a sufficiency of surface rights and water.

Surface rights are discussed in Section 4.5.

6 HISTORY

This chapter is divided and discussed for each of the six claim blocks that makes up the property: the Wasamac Block, the Wasamac NE Block, the Teck JV Block, the R.M. Nickel Block, the Consolidated Francoeur Block and the Western Buff Block.

6.1 Wasamac Block

The Wasamac Block has been the object of extensive exploration work in the past. Gold mineralization was originally discovered in 1936 by Mine d'Or Champlain through surface trenching work. Subsequent surface diamond drilling intersected encouraging gold values but geological continuity was erratic. A 60 m shaft (Wildcat shaft) was sunk, and one underground level was developed. The Wasamac Block exploration history is summarized in Table 6-1 (overleaf) and can be grouped into the following periods:

- 1936–1965: Initial gold discoveries and early exploration work (including historical drilling and trench sampling) by Mine d'Or Champlain, Wasa Lake Gold Mines, and Wasamac Mines Ltd.
- 1965–1971: Exploration and production of Wasamac mine by Wasamac Mines Ltd.
- 1971–1986: Exploration work (including drilling and ground geophysical surveys) and pre-feasibility work on surface pillars by Lac Minerals.
- 1986–2016: Exploration work, rehabilitation and reclamation work at the Wasamac mine site by Richmont Mines Ltd. ("Richmont"). Pre-feasibility study report released in 2012.

Mine production at the Wasamac mine was only between 1965 and 1971, approximately 254 000 oz Au (approximately 1.9 Mt at 4.16 g/t Au) were extracted.

6.2 Wasamac NE Block

The Wasamac NE Block is located directly north of the Wasamac Block. Exploration work in and around the Wasamac NE Block started when early prospection discovered gold and other metal and non-metal showings in the 1930s, as well as the discovery of the Horne deposit in 1925. Prior to acquisition by Yamana, parts of this claim block were extensively worked by Horne Fault Mines Ltd., Noranda Explorations Ltd, Eider Resources Inc, and Inmet Mining Corp.

Exploration work, including field surveys and drilling campaigns in the 1940s and 1960s, lead to the discovery of the "84 Showing" in 1963. Diamond-drill hole (84 intersected 4.1 g/t Au / 1.1 m and 2.0 g/t / 0.9 m (downhole interval not found) in an altered lava.

Extensive drilling over a five-phase drilling program (41 drill holes) in 1986-1987 discovered the Peltier Showing in the southcentral area of the claim block with 2.23 g/t Au / 0.60 m (from 219.50 to 220.10 m) and 2.09 g/t / 0.80 m (from 223.5 to 224.30 m) (DDH 86-RB-20, GM45594).

In 1989, grab samples from a mineralized gabbro outcrop returned 0.97% Cu (sample #19994) and 0.69% Cu (#19992) associated with quartz veins; leading to the naming of the Caron-SO showing (Giovenazzo, 1990a and 1990b).

Table 6-1: Historical Ownership Changes and Historical Work on the Wasamac Project from 1945 to 2016

Year	Company	Work	Results
1944	Wasa Lake Gold Mines (formerly Mine d'Or Champlain)	Field exploration	New gold showing discovery: the Main Zone, located approximately 300 m north of the Wildcat Zone
1945 - 1948		Inclined shaft sunk (55°) to 1,000 ft level	Significant underground work on 5 level. Ore reserves* established at the time were approximately 2 Mt at an average grade of 5.28 g/t Au
1960	Wasamac Mines Ltd.	Acquisition of Wasa Lake Gold Mines by Barant Mines Ltd. in association with Little Long Lac Gold Mines	Name change from Wasa Lake Gold Mines to Wasamac Mines Ltd.
1964 - 1965		Underground workings were dewatered and rehabilitated	Production officially commenced on April 1, 1965
1965 - 1971		Producing Mine	Approximately 1.9 Mt of ore was treated by Wasamac Mines Ltd., and later by Wright-Hargreaves Mines Ltd. An average grade of 4.16 g/t Au was recorded (Karpoff, 1986b)
1974	Lac Minerals Ltd. (Lac Minerals)	Diamond Drilling on MacWin Zone and diamond drilling on the Main Zone	Better definition of the Main Zone
1980 -1983		Field exploration work: ground geophysics HLEM, MAG and VLF surveys), followed by geological mapping and diamond drill program (64 DDH totalling 7,357 m)	
1983		Diamond drilling program following pre-feasibility work on the surface pillar recovery	33 DDH totalling 1,880 m at 15 m spacing to upgrade the level of confidence of this surface zone
1983 -1986		Many open-pit studies prepared for the surface pillars	Low gold prices at the time prevented the company from commencing production
1986	Ressources Minières Rouyn (RMR, who changed its name for Richmont in 1991)	Option agreement with Lac Minerals	
		Field exploration work	11 surface drill holes totalling 3,710 m further evaluating the surface pillar zone along with the Zone 1 and Main Zone down-dip extensions.
1987 -1988		Dewatered the mine to a depth of 975 ft and rehabilitated the 400 ft and 800 ft levels to explore the down-dip extension of Zone 1 through underground drilling.	Again, low gold prices at the time prevented the company from commencing production
1994	Richmont	Restored Wasamac Mine site	All surface installations were dismantled, the shaft was capped, and the tailings pond was revegetated
1989 - 2002		Diamond drilling to keep mining leases in good standings	Eight surface DDH, totalling just over 4,500 m. The main geological target was the Wasa shear zone at depth (Zones 1 and 2)
2002		Exploration program (down-dip extension of Zone 1 and 2 at depth)	WS-02-01 intersected 4.15 g/t Au over a true width of 6.8 m
2003		15-drill holes (9,475 m) surface drilling program	All drill holes intersected the Francoeur-Wasa shear zone at depth. Nine drill holes returned assay values greater than 4.0 g/t Au and six drill holes returned grades greater than 4.5 g/t Au in Zones 2 and 3
2004		Additional 3,859 m drill program	Results from the 2002 - 2004 drilling supported an internal inferred mineral resource estimate for Zones 2 and 3 (Guay, 2004)
2005		One DDH (745 m) drilled west of Zone 1	WS-05-21 intersected 0.91 g/t Au over 4.1 m
2007		Two DDH targeting West extension of the Wildcat Zone	WS-07-22: 1.39 g/t Au over 6.6 m
2008-2009		Three exploration drill holes targeted geophysical anomalies that could indicate parallel structure to the WSZ	Alteration zones were intersected but returned no significant gold values
2010		29 DDH (19,853 m) with the goal of reassessing mineral resources using a lower cut-off grade in order to evaluate the potential for an underground bulk mining operation.	Favorable results in several DDH including: 6.14 g/t Au over 6.47 m (WS-10-37 – from 534.23 to 541.70 m), 3.12 g/t Au over 14.04 m (WS-10-42 – from 283.60 to 300.40 m), and 3.17 g/t Au over 7.75 m (WS-10-38 – from 643.42 to 651.17 m)
		Mineral resource estimate of the project	Updated Mineral Resources
2011		Exploration drilling targeting the Main Zone and Zones 1, 2 and 3 in 78 DDH (52,000 m)	A 35% increase in the measured and indicated mineral resource estimate and a 111% increase in the Inferred mineral resource estimate on the property
2012		16-drill hole (11,803 m) to test the Francoeur Wasa Shear Zone between the vertical depths of 200m and 1,000 m across claims optioned from Globex Mining Enterprises Inc. (Globex)	All drill holes intersected the the Francoeur Wasa Shear Zone; however, where intersected, the structure appeared to be less highly deformed and not as highly altered as elsewhere. Some of the best intersections include: WG-480-02: 4.07 g/t Au over 5.60 m (from 533.40 to 539.00m), WG-480-04: 2.25 g/t Au over 6.70 m (from 589.40 to 602.10m).
		Mineral Resource Estimate of the project	As of December 31, 2011, Mineral Resources were updated (part of a PEA) (Gauthier et al., 2012) *
2015-2016		Two DDH (600m) targeting the east and west extensions of the Wildcat Zone	WC-15-01: 2.55 g/t Au over 2.00 m (from 90.00 to 92.00 m); and WC-16-01: 1.78 g/t Au over 2 m (from 87.00 to 89.00 m), 2.71 g/t Au over 3.5 m (from 131.00 to 134.50 m), 1.53 g/t Au over 4.5 m (from 164.50 to 169.00 m) and 14.02 g/t Au over 1.5 m (from 237.00 to 238.50 m) (west of the zone)

Work carried out by Minnova Inc. (later Inmet Mining Corp.) during 1992 to 1994 on the "Flag" area of the project consisted of detailed geological mapping and litho-geochemical sampling, minor geophysical surveys of VLF-EM and surface DEEP-EM, induced polarization (IP) and stratigraphic diamond drilling supported by downhole pulse electromagnetic (PEM) surveys. No significant mineralization was intersected; somme drill holes confirmed lithologies and structures were at depth (Pearson, 1992).

In 2000, an airborne Aerotem survey was completed by Aurogin Resources Ltd. and Globex. Several bedrock EM conductors were identified (Zalnieriunas and Fiset, 2000).

6.3 Consolidated Francoeur Block

6.3.1 Francoeur

The history of the Consolidated Francoeur Block was obtained from the 2020 Globex Exploration Report (Garant and Mougin, 2021). A detailed resume of the historical ownership changes and historical work on the Francoeur mine and area is shown in Table 6-2.

The Francoeur area was staked for the first time in 1923 following a gold discovery that later became Zone No. 1. In 1932, Francoeur Gold Mines Ltd. sunk a 45° inclined shaft (No. 1) of approximately 740 feet with four levels (95-, 191-, 290- and 488-foot levels) in the footwall of Zone No. 1.

The Francoeur area exploration history can be grouped into the following periods:

- 1923 to 1938: Initial gold discoveries, early exploration work (including historical drilling and trench sampling) and preparation for commercial production by Francoeur Gold Mines Ltd.
- 1938 to 1947: Exploration and production of Francoeur mine by Francoeur Gold Mines Ltd.
- 1964 to 1968: Drilling and sinking of the Wasamac No. 2 vertical shaft (now called shaft No. 6).
- 1968 to 1971: Exploration and production of Francoeur mine by Wright-Hargreaves Mines Ltd.1971 to 1985: Exploration and underground drilling by Kerr Addison Mines Ltd., Noranda Exploration Company Ltd. and Long Lac Exploration Ltd.
- 1985 to 1991: Dewatering and rehabilitation of surface and underground mine facilities and levels. Underground drilling and extraction of a bulk sample by Ressources Minières Rouyn (RMR, now Richmont).
- 1991 to 2001: Exploration and production at Francoeur mine by Richmont.
- 2001 to 2012: Exploration drilling programs focusing on the West Zone, internal mineral resource calculations and N.I. 43-101 reports, dewatering of the mine and rehabilitation of underground levels for exploration drilling by Richmont.
- 2015 to 2019: Surface exploration work (mapping, sampling, trenching, geophysical surveys) by Globex.

Table 6-2: Historical Ownership Changes and Historical Work on the Francoeur Mine and Area from 1936 to 2020

Year	Company	Work	Results
1936	Francoeur Gold Mines Ltd	Underground drilling. Another 45° incline shaft (No. 2) was sunk to a depth of 600 feet with four development levels (132-, 221-, 311- and 399-foot levels) in the lower footwall of Zone No. 2.	Discovery of Zones No. 2 and 3.
1938	Francoeur Gold Mines Ltd	A concentrator capable of handling 150 short tons per day was built.	Production of No. 1 and No. 2 deposits.
1939-1940	Francoeur Gold Mines Ltd	Drilling and creating a transverse gallery starting from the 2nd level of Zone No. 2. From the level 3, ROM level 3, the drift was continued for 914.4 m westward in order to reach and mine the ore of the No. 3 zone.	Zone No. 8 was discovered approximately 245 m north of the Francoeur-Wasa shear zone.
1938-1947	Francoeur Gold Mines Ltd	Producing mine.	94,303 oz of gold from 520,363 tonnes of ore grading 5.6 g/t of gold (Brown, 1962).
1964	Wright-Hargraves Mines Ltd (Wasamac Division)	Acquisition of the Francoeur property from Francoeur Gold Mines Ltd.	
1965	Wright-Hargraves Mines Ltd	Further drilling and Wasamac No. 2 vertical shaft (now called shaft No. 6) was sunk to a depth of 477 m in order to mine the No. 3 zone from levels 4 to 11.	
1968-1971	Wright-Hargraves Mines Ltd	Producing Mine. The ore was milled at the Wasamac No1 mine concentrator located 6 km to the east.	69,227 oz of gold from 385,292 tonnes grading 5.6 g/t Au (Karpoff, 1986b).
1973- 1974	Kerr Addison Mines Ltd. and Noranda Exploration Company Ltd	Acquisition of the Francoeur property from Wright-Hargraves Mines Ltd. Drilled five drill holes along the Francoeur-Wasa shear zone.	Little success (GM 29431, GM 30512).
1980-1984	Long Lac Exploration Ltd	Drilling: three deep drill holes (FR-82-1, 2 and 17).	Confirmed that the Francoeur Shear is present below the 11th level. (GM 40041, GM 41362).
1985	Long Lac Exploration Ltd and RMR	RMR (now Richmont Mines) signed an option agreement with Lac Minerals for the acquisition of a 50% interest on the Francoeur property. An assessment of the property was done.	Evaluation by Karpoff (1986b) evaluated remaining tonnes still accessible by the Wasamac shaft No. 2 (shaft No. 6).
1986	RMR	Dewatering and the rehabilitation of the underground infrastructures.	
1986-1987	RMR	75 DDH (27,799 m) on zones No. 1, No. 2, No. 8, and below the 11th level of zone No. 3. surface facilities were installed and underground diamond drilling.	The surface diamond drilling program delineated more than 1 Mt of possible mineral reserves* under the 11th level of Zone No. 3.
1988	RMR	Extraction of 23,111 tonne bulk sample grading 6.8 g/t Au from the No. 3 deposit. And operations from the No. 6 shaft started.	
1989	RMR	Jean-Guy Rivard (No. 7) shaft was sunk to 818.1 mm.	
1991-2001	Richmont	Producing mine.	1,701,892 tonnes of ore at a grade of 6.31 g/t (for 345,112 ounces of gold).
1992	Richmont	Richmont acquired Lac Minerals for 50% share of the Francoeur and Wasamac properties.	
1993	Richmont	Camflo Mill was purchased and started processing Francoeur's ore (ore was previously milled at Lac's East Malartic Mill and Deak's Virginiatown Mill). Underground diamond drilling program (3,433 m).	
1996	Richmont	Underground exploration program: 847 m of drifting and 6,540 m of underground drilling.	
1997-1998	Richmont	Exploration Program at Francoeur Mine, follow-up by a development program, rehabilitation work of shaft No. 6 and infill drilling on levels 4,6, and 7.	Exploration program led to the discovery of Zone 7.
1999	Richmont	Development work on levels 4, 6 and 7 and related sublevels were completed.	
2000	Richmont	Commercial production of Zone 7.	
2001	Richmont	Exploration work in West part of mine. However due to low gold prices, the development of the new zone was uneconomic and subsequently, the mine was closed in November 2001.	Discovery of a new zone, the West Zone.
2002-2003	Richmont	Exploration programs (7,801 m). Subsequently to a low profitable margin despite good results at the West zone, the mine was flooded, and restoration of the site began.	Minereal Resousrces update increasing the West Zone mineral resources. However, the feasibility study demonstrated that any effort to resume production by deepening the No. 7 shaft would not be profitable at the time.
2005-2007	Richmont	Following the closure of the mine, minimal exploration work was conducted to keep the mining concessions and leases in good standing: four exploration drill holes targeting auriferous shear subsidiary to the Francoeur-Wasa shear zone.	
2008	Richmont	New mineral resource calculations were done internally by Richmont's geologist in May 2008 and by Geopointcom in August (D'Amours, C. 2008).	New mineral resource calculations
2009	Richmont	Short drilling program followed by a Richmont published NI 43-101 technical report detailing a mineral resource and mineral reserve calculation for the West Zone (2009, amended in 2010).	The mineral resources was updated
2009-2010	Richmont	Dewatering of the mine and major repairs to surface infrastructure due to vandalism.	
2010-2012	Richmont	501 DDH (33,423 m) drilled to define the West and North zones.	
2012	Richmont	A potable water treatment plant and an acid treatment plant were built. Rehabilitation and access drifts to the west zone on levels 12 to 17. The excavation for the infrastructure of the ramp was completed and the ramp between the level 16 and level 17 is underway.	
2015	Globex Mining Enterprises Inc	Globex signed an agreement with Richmont Mines to acquire the assets of the Francoeur property. The same year, Globex. performs for Richmont. a 216 m deep drill hole, targeting the western extension of the West zone.	Average grades of 318 ppb Au / 3.0 m and 615 ppb Au / 1.5 m were obtained in a shear zone altered in silica-albite-hematite and carbonate.
2016	Globex Mining Enterprises Inc	100% Acquisition of Francoeur property from Richmont including the historic Francoeur and Arntfield mines. One DDH on Zone No. 7.	Drilling results: a large low-grade envelope close to surface grading 1.19 g/t Au over 74 m (FS-16-35; from 37.00 to 111.00 m).
2017-2020	Globex Mining Enterprises Inc	Exploration work (Compilation, stripping, sampling, geophysical surveys and drilling) carried out on the Francoeur/Arntfield/Lac Fortune property.	Grab samples returning significant Au values.

6.3.2 Arntfield

The history for the Arntfield Block was parsed from Garant and Mougin (2021), after Pearson (1988). A resume of the historical ownership changes and historical work on the Arntfield area is included as Table 6-3. Mining claims in the Arntfield area were first staked in 1923 by F.S. Arntfield.

Table 6-3: Historical Ownership Changes and Historical Work on the Arntfield mine and area from 1923 to 2019

Year	Company	Work	Results
1923-1935	Various prospectors and companies	Gold mineralization was discovered on the western part of the property and Shaft #1 was sunk. Shaft #2 and Shaft #3 were sunk after further exploration and the discovery of new gold mineralization.	
1935-1942	Arntfield Gold Mines	Production of Arntfield #1 Mine.	480,804 tonnes grading 3.98 g/t Au and 0.93 g/t Ag (Adam, D. et al., 2012)*
1944-1947	Arntfield Gold Mines	Exploration and development.	Mine remained inactive
1947-1977	Several companies	Property owned by several companies with several work programs carried out.	No additional mineral resources were calculated
1977-1984	Noranda Exploration	Surface exploration work (mapping, IP, EM, and magnetic geophysical surveys, and diamond drilling).	
1985-1987	Noranda Exploration and RMR	A total of 83 DDH (29,407.41 m). A historical mineral resource calculation completed (not in accordance with NI 43-101) for Zone #4 and #5 (Pearson, 1988).	Discovery and later delineating of Zone #4 and Zone #5 and mineral resource completed
2004	Globex Mining Enterprises	Exploration work on claims east of Arntfield #2 (compilation, prospecting, 3 DDH totalling 300.64 m).	BEA-01: 3.46 g/t Au over 2.84 m (from 17.71 to 20.55 m). BEA-02: 1.63 g/t Au over 1.74 m (from 12.74 to 14.48 m).; BEA-03: 1.81 g/t Au over 2.0 m (from 21.00 to 23.00 m).
2016	Globex Mining Enterprises	100% Acquisition of Francoeur property from Richmont including the historic Francoeur and Arntfield mines.	
2017-2020	Globex Mining Enterprises	Exploration work (Compilation, stripping, sampling, geophysical surveys and drilling) carried out on the Francoeur/Arntfield/Lac Fortune property.	Grab samples returning significant Au values

6.3.3 Lac Fortune

The discovery of gold mineralization at Fortuna Lake was in 1906 by prospectors Alphonse Ollier and Auguste Renaud (Sohier, 1908). A resume of the historical ownership and work on the Lac Fortune is found is included as Table 6-4.

Table 6-4: Historical Ownership Changes and Historical Work on the Lac Fortune area from 1908 to 2020

Year	Company	Work	Results
1908	Pontiac and Abitibi Mining Company	M. Ollier et M. Renault founded the Pontiac and Abitibi Mining Company after their gold discovery close to Lac Fortune in 1906.	
1910	Abitibi Mining Company	Acquisition of Lac Fortune property by Abitibi Mining Company. Built a mill and sinking of Shaft #1 at 55° N to 46 m. The Company ceased activities at the beginning of WW1 and was liquidated afterwards.	
1922-1924	Lake Fortune Mining Company	Dewatering of shaft # 1 and sinking of shaft # 2 up to 38 m.	
1924	Alderson and Mackay	Geological mapping of Lac Fortune.	
1926-1933	Towagmac Exploration Company	Acquisition of Lac Fortune property by Towagmac Exploration Company. 14 total DDH in 1926, 1927 and 1933 (2,035 m) with 7 drill holes targeting the carbonate banding between levels 76 and 152, trenching and stripping.	Karpoff (1986a) references a 1934 document that mentions an average grade of 8.57 g/t Au over 1.25m was obtained. The company's 1927 annual report indicates an average of 18.36 g/t Au for what appears to be the same area.
1934-1935	Towagmac Exploration Company	Sinking of shaft #3 to 149 m.	
1935	Lake Fortune Gold Mines, Ltd	Report on underground developments carried out in 1935 and development underground plans and levels 108 and 142 m (355 and 465 ft).	
1944?	Toburn gold mines Ltd	Geological and drilling works, trenches of underground developments.	
1944-1946	Renfort Gold Mines Ltd.	55 DDH (8,091 m).	DDH near Shaft 1 and 2 confirmed good gold grades neat the Lac Fortune structure.
1967-1968	Albert Coutu & Associates	3 DDH (DDH #1, #2, #3 totalling 432.51 m) and EM and magnetic geophysical surveys.	Little findings, shear zone with quartz and pyrite in DDH#1.
1973	Quain-Renfort	1 DDH (73-1) of 47.55 m.	
1975-1978	Bédard Exploration	5 DDH (75-1,2, 77-1,2,3, 1-78) totalling 160.93 m.	No significant results.
1979	Noranda Exploration company	Compilation Report.	No further work was recommended.
1980	Par G.M. Hogg & Associate	Appraisal report for the Lac Fortune property of Bédard Exploration.	
1984-1985	RMR	1)11 DDH (827.82 m), digging a 189 m ramp to level 30, from a 64-metre crossbench, bulk sampling of 425.7 t, dewatering of well #1. 2) 35 surface DDH (85-15 to 85-49) totalling 4,862 m. 3) Surface installations were enlarged and improved. Shaft #3 was dewatered; a temporary headframe and winch were installed; a new access road was built.	The average of the analysis results of the sampling of 425.7 tonnes was 5.75 g/t Au (Karpoff, 1986a). DDH 85-49: 4.45 g/t Au over 6.44 m intersected in a gold zone in pillowed andesites under the carbonate shear zone.
1986	RMR	Surface drill program: LFS86-50 to LFS86-57 (2,439.6 m) and underground drill program: LF-1 to LF-18 (459.93 m) from level 1 and 3.	DDH LFS86-50: 8.06 g/t Au over 1.21 m (from 359.26 to 360.58 m) (open) intersected in the same gold zone as DDH 85-49.
1987	Coopers & Lybrand Consulting & RMR	Assessment report for the Lac Fortune property. Calculation of historical mineral resources (Karpoff, 1986a).	Calculation of historical mineral resources (Karpoff, 1986a).
1988	RMR	2 DDH: LFS-88-58, LFS-88-59: (979.63 m).	
1997-1998	Richmont	9 DDH (1,227.5 m) and extension of drift 03-200 by 22 m exploration shaft. Bulk sample of 5,002 tonnes collected + 3,223 tonnes historical (Daigneault, 1998).	The average of the analysis results of the sampling of 8,225 tonnes is 3.36 g/t Au (Daigneault, 1998).
2016	Globex Mining Enterprises	100% Acquisition of Francoeur property from Richmont including the historic Francoeur and Arntfield mines.	
2020	Globex Mining Enterprises	Exploration work (prospection, mapping and sampling). The purpose of the fieldwork was to verify whether this mineralized zone found in in the basalts of DDH 85-49 and LFS86-50 is visible in outcrop.	The gold zone contained in the basalts identified in historic drilling to the south of the carbonate zone was located at the surface. Samples collected from this stockwork of quartz veins returned up to 31.47 g/t Au. 225 m to the southwest, pyrite clusters and bands of up to 15% per year with traces of chalcopyrite and malachite were observed. The samples returned anomalous values for Cu (4,350 ppm) and Au (0.42 g/t) and up to 138.3 g/t Ag. Follow-up work in the area is recommended.

6.3.4 Western Buff Block

The information in Table 6-5 is taken from Pilote (2009). The discovery of a mineralized outcrops on the Western Buff Block was in 1929 (Vein #2) and in 1930 (Vein #1) by prospection.

Table 6-5: Historical Ownership Changes and Historical Work on the Western Buff Block from 1930 to 2009

Year	Company	Work	Results
1930*-1931	Albion Cooper-Gold Mines	Exploration work: trenching and two drill holes on Vein #1 and Vein #2.	
1941*	Chicades Mines	7 DDH (167.7 m).	
1945*	Kenikonda Mining Corp	Drill program (1,166.8 m) and surface trench extension.	
1951	Crangold Mines Ltd	Exploration work: geophysical spontaneous polarization (SP) survey, stripping (2,000 m) and channel sampling and drilling campaign: 23 DDH (1,362 m; 22 under Vein #1 and one exploration).	SP survey: five weak anomalies, one was in response to mineralization of Vein. 1 and another that of mineral occurrence No. 7. Channel sampling: southern part of Vein #1: avg. 3.45% Cu / 1.70 m (width) and 42.7 m (length). 4 mineralized zones identified on Vein #2. Drilling: 17 DDH intercepted the zone at depth (including K-1: 28.2% Cu, 7.8 g/t Ag / 0.3 m; K-3: 2.8% Cu / 1.9 m; and K-5: 3.7% Cu / 1 m), giving an average of 4.44% Cu / 0.76 m, a length of 70 m and a minimum vertical extension of 61 m, Hole K-34 intercepted Vein # 1 at a vertical depth of 126 m (1.69% Cu / 0.5 m).
1952		Exploration work: geophysical (Mag and IP), Geological prospecting and trenching (1,525 m total), and drilling campaign: 11 DDH (493 m on Vein #2 and 367 m extension of Vein #1; no drill logs available). Stripping of the Gauthier mineral occurrence (30.5 m).	Geophysical surveys: no significant results Geological survey: identified several veins oriented parallel to the zones (NNE), 2 contains galena (Gn) and sphalerite (Sp). Drilling: No significant results. Gauthier Mineral occurrence: 0.15 m wide by 9.1 m long Gn-Sp vein (selected sample: 2.68 oz/t Ag, 38.11% Pb and 8.8% Zn). Vein # 4 composed of massive Gp and 0.15 m wide, gave 2.08 oz/t Ag and 41.18% Pb (grab sample).
1953-1954	Cran-Kor Metals Mines Ltd.	Work preparations to put Vein #1 into production.	
1956	Valray Expls Ltd.	Study to develop the Vein #1.	
1957	Cran-Kor Metals Mines Ltd.	Delineation of Vein #1 between 200 and 400 ft deep by 11 drill holes (1,313 m).	Drilling: confirmed the continuity of the vein over a minimum vertical depth of 61 m at 3.78% Cu / 0.6 m (true thickness). Despite some good Cu intersections in 7 drill holes (including no. 35: 1.3% Cu / 2.1 m; no. 37: 4.1% Cu / 1.2 m; and no. 46: 4.4% Cu over 1.1 m), the results showed that the vein is very narrow (about 0.6 m).

Year	Company	Work	Results
1962	Valray Expls Ltd.	Geophysical: partial IP and resistivity survey; Geoscientific compilation of each showing and previous work (veins #1 to 4 and showings 6 to 9).	
1962-1963	MRN	Well colonization, 13 boreholes (wells).	No boring and analysis logs are available.
1968	Colleen Copper Mines Ltd.	9 DDH (1,001 m).	C-1: 6.3% Cu + Tr. Au / 0.9 m (from 59.50 to 60.40 m); C-2: 3.91% Cu / 3.8 m (from 67.10 to 70.90 m); C-4: 2.08% Cu / 0.45 m (from 50.00 to 50.45 m); C-5: 2.14% Cu / 0.8 m (from 71.00 to 71.80 m); C-6. 1.24% Cu / 1.1 m (from 56.80 to 57.90 m); C-8: 5.8% / 0.4 m (from 92.40 to 92.80 m).
1970	Western-Buff Mines & Oils Ltd.	Geophysical EM and partial IP surveys: E-W lines (interval of 61 m and a few lines of details at 30.5 m). Follow-up drilling campaign: 10 DDH (740.3 m) including 6 on Seam #2.	Geophysical surveys: Detected the already known Vein #1 and Vein # 2 Drilling: several intersections of cpy in silicified meta-sediments and in Qtz (WB-1: 1.3% Cu / 3.7 m (from 118.00 to 121.70 m); WB-3: 4.98% Cu / 1. 2 m (from 119.20 to 120.40 m); WB-8 2.27% Cu / 0.8 m (from 43.80 to 44.60 m); WB-10: 3.09% Cu / 2.9 m (from 34.30 to 37.20 m)).
1971	Eldona Gold Mines Ltd., Western-Buff Mines & Oils Ltd	14 DDH on Vein #2 to test mineralization depth.	Several small quartz veins were intercepted; however, the report concluded that these veins are sterile.
1981-1982	Silver Scepter Mines Ltd.	Geophysical EM, very low frequency (TBF) ground survey to identify Au-bearing shear zones. Follow-up drilling campaign: 2 DDH (216 m)	Geophysical surveys: several small conductive zones trending E-W (parallel to the bedding). Drilling: No significant results.
1987		3 surface samples collected from Vein #2.	Samples showed trace Au (only analysed for Au).
1988	Kimex Resources Inc.	2 DDH (95 m) under Vein #1.	
1989	Exploration Brex Inc.	Geophysical MAG (11.6 km) EM-VLF (11.6 km) and IP (15.4 km) surveys on E-W lines every 100 m. Geological compilation and surveys: mapping, stripping, and sampling. Follow-up drilling campaign: 12 DDH (1,370.6 m).	Geophysical surveys: MAG survey only revealed small anomalies (small shallow pyrrhotite boudins). The EM-VLF survey revealed a main NE structure in the NE part of the property. The IP survey revealed 2 noteworthy anomalies that could come from SMS. Drilling: No significant results (Trace Au and Ag)
2009	Richmont	Geological compilation, a summary mapping of the sector and sampling of existing mineralized showings.	Au potential of veins Vein #1 and Vein # 2 is negligible. Their Cu potential is noteworthy but the veins are narrow and there does not appear to be any significant mineralization in the wall rock. Field mapping revealed a new showing Ag, Pb, Zn, similar to Vein #3 and Showing 6, further north.

* No documents are available for this work, but the information has been summarized in more recent documents (Pearson, 1989).

6.4 R.M. Nickel Block

The history of the R.M. Nickel Block is taken from Kelly (2000), unless otherwise stated. A resume of the historical ownership changes and historical work on the R.M. Nickel property is included as Table 6-6.

Table 6-6: Historical Ownership Changes and Historical Work on the R.M. Nickel Deposit and Claim Block from 1947 to 2000

Year	Company	Work	Results
1947-1956	Various prospectors and mining compagnies	Exploration work (prospecting, mapping, sampling, and drilling).	
1957	R.M. Nickel Co.	110 DDH (over 6,096 m) to define the ore body	
1959		Exploration work (geophysical).	
1974	Yvanex Developments	Exploration work including geophysics survey and 6 DDH.	
1980	Falconbridge Copper (now Minnova Inc.)	Exploration work (geological mapping, lithogeochemical survey, and stripping and trenching at the R.M. Nickel site).	Based on earlier exploration data, a mineral reserve estimate was completed
1981-1983		Exploration work: Mapping (1:5,000) and systematic rock sampling at 25 m (lines per 100 m): 661 rock samples in 1981, 232 rock samples in 1982 and 12 rock samples in 1983. Geophysical Mag and EM surveys. The "Spotted Ore" zone, associated with the R.M. Nickel deposit was stripped and mapped at 1:2,000 and 1:200 scale. Sectors of the "First Zone" and the "Pyrite Zone" have also been cleared.	Lithogeochemical sampling on the property identified four strong anomalies, which correspond with four known showings - Gan Copper, West Gan, Provencher and R.M. Nickel.
1987	Equinox Resources Ltd. and Technigen Corp. under an agreement with Minnova Inc.	8 DDH to evaluate the Pt and Pd content of the R.M. Nickel deposit.	
1988		Due to increase in nickel prices, 20 DDH were drilled to complete an evaluation of the deposit for purposes of a preliminary feasibility study (GM48379).	Completed an evaluation of the deposit for purposes of a preliminary feasibility study (GM48379).
1999	Dasserat Ressources Inc.	14 DDH (471.22 m);.	Re- calculated the probable mineral reserves of the richest sector of the R.M. Nickel deposit
2000		3 DDH (368 m) and mapping of an outcrop close to in the sector of the "First Zone".	DK-03: 6.18 g/t Au over 2.8 m (from 22.00 to 24.80 m), incl. 15.26 g/t Au over 0.7 m (from 24.10 to 24.80 m).
2007	Radisson Mining	Conditional sale/purchase agreement for the purchase of 48 mining claims (R.M. Nickel) from a prospector.	
	Radisson Mining	519.5 m drill program to test the high- and low-grade mineralized zones and provided enough material from the high-grade zone to conduct metallurgical tests.	RM07-01: 4.89 %Cu (from 90.50 to 100.50 m). A composite sample of 225 kg of core averaged 2.72% Cu, 3.25% Ni, 1.12 g/t Pt, 5.05 g/t Pd and 0.22 g/t Au.
2008	Radisson Mining	Received results from 2007 metallurgical test.	Results determined the recoveries of Ni and Cu would be uneconomic. The property was part of the La Reine property agreement, it was returned to the vendors in June 2008

Massive sulphide mineralization adjacent to the R.M. Nickel deposit was originally discovered by prospecting, and geophysical survey work in 1947.

6.5 Teck JV Block

No known historical showing or deposit is located on the Teck JV property and limited exploration information was found on the six claims that form the JV with Teck Resources, acquired through the acquisition package from Monarch Mining.

One DDH (totalling 183 m) was completed in 1956 in the northeastern part of the Teck JV Block. Syenite and diorite were noted in the drill logs, as well as a pyrite-mineralized syenite zone from 239 to 291 ft down hole; however, no samples were collected (GM04265B).

6.6 Production History and Historical and Prior Mineral Resource Estimates

6.6.1 Wasamac Mine

Although several historical mineral resource estimates and mineral reserve estimates have been prepared for the Wasamac Mine since its discovery, none of these estimates are currently regarded as significant. Only the prior mineral resource estimate and reserve estimate of the 2018 feasibility study (Caumartin et al., 2018), for the Wasamac Deposit is presented below; it is superseded by the Mineral Resource and Mineral Reserve estimate presented in Section 14 and Section 15.

From 1965 to 1971, approximately 1.9 Mt of ore from the Wasamac deposit were treated by Wasamac Mines Ltd. and later by Wright-Hargreaves Mines Ltd. An average recovered grade of 4.16 g/t Au was recorded (Karpoff, 1986b) (see Table 6-7).

In 1981, Exploration Long Lac Limitée, after the completion of a surface diamond drilling campaign, re-assessed the mineral resources of the Wasamac mine, including the surface pillar of the Main Zone (Bugnon, 1981).

In 1983, Lac Minerals completed another drilling campaign in the surface pillar of the Main Zone and re-assessed the mineral resource (Bugnon, 1983).

Table 6-7: Summary of Wasamac Mine Production History

Year	Tonnes Milled	Gold Grade (g/t)	Ounces Recovered (Au)
1965	222,422	3.94	28,189
1966	368,986	4.42	52,451
1967	369,914	4.17	49,531
1968	376,236	4.41	53,280
1969	305,142	3.67	35,982
1970	212,660	4.11	28,123
1971	37,088	4.50	5,367
Total	1,892,448	4.16	252,923

Source: After Karpoff, 1986b.

Karpoff (1986b) updated the mineral resource estimates for all zones.

Following the 2002-2003 drilling campaigns, Richmont completed an internal MRE for the projected downward extension of Zone 1 and Zone 2 (Guay, 2004). This MRE was updated using the 2004 drilling results..

Surface drilling in 2010 by Bradley Bros. Limited targeted parts of Zone 1 and Zone 2 that were identified during the 2002 to 2004 drilling. The 2010 results enabled Richmont to estimate a measured and indicated mineral resource.

Following the 2010 and 2011 drilling campaigns, mineral resources were updated in 2012 as part of a preliminary economic assessment (PEA) prepared by RPA Inc. (RPA) mining consultants and filed on SEDAR on May 11, 2012 (Gauthier et al., 2012).

A feasibility study of the Wasamac Project was submitted in December 2018 by BBA Engineering Consultants ("BBA"), (the ("2018 Feasibility Study") based on the 2017 mineral resources estimate prepared by RPA.

The Mineral Resource Estimate had an effective date of October 20, 2017 and was based on the assumption that the deposit would potentially be developed and mined using underground mining methods. Estimation was completed using Geovia GEMS 6.7 and using a percent block model approach, constrained within interpolation domains (mineralized zone) with resource classification following 2014 CIM Definition Standards. The cut-off grade of 1.0 g/t Au was used and was based on a gold price of US$1,500 per ounce of gold and assumed operation costs. RPA estimated 3.99Mt at an average grade of 2.52 g/t Au, containing 323,300 ounces in the Measured category, and 25.87Mt at an average grade of 2.72 g/t Au, containing 2,264,500 ounces in the Indicated category. An additional 4.16Mt at an average grade of 2.20 g/t Au, containing 293,900 ounces were estimated in the Inferred category. The resource was inclusive of the reserve.

The Mineral Reserve Estimate had an effective date of December 1, 2018. BBA estimated 1,028,000t at an average grade of 2.66 g/t Au, containing 88,000 ounces as Proven reserves, and 20,427,000t at an average grade of 2.56 g/t Au, containing 1,679,000 ounces as Probable reserves. The cut-off grade of 1.0 g/t Au was based on a gold price of US$1,500 per ounce of gold and assumed operation costs. The above reserves contained an average of 16.2% mine dilution and 86.4% mine recovery.

The mineral resource and mineral reserve estimate referred above is the prior estimate and should not be relied upon. It is included in this section for illustrative purposes only. The Issuer is not treating this prior estimate as current mineral resources or mineral reserves; they are superseded by the mineral resource and mineral reserve estimates presented in Section 14 and Section 15.

6.6.2 Francoeur Mine

Although several historical mineral resource estimates and mineral reserve estimates have been prepared for the Francoeur Mine since its discovery, none of these estimates are currently regarded as significant. Only the latest mineral resource estimate for the Francoeur Mine is presented below (Adam, 2014); it is considered as historical.

Francoeur mine was in production from 1938 to 1947, from 1968 to 1971, and from 1991 to 2001. During these three past periods of mine production, a total of 508,642 oz Au (2.60 Mt at 6.1 g/t Au) were recorded (Table 6-8).

After the success of the 2002-2003 exploration program, the West Zone mineral resource was calculated, however, re-opening of the mine was not profitable at the time.

In August 2008, following the increase of the gold price, new mineral resource estimations were completed by Richmont's geologist in May (Guay, 2008) and by D'Amours (2008). After a short drilling program from surface, Richmont published a NI 43-101 technical report detailing a mineral resource and mineral reserve calculation for the West Zone (2009, amended in 2010). In the NI 43-101 technical report conducted by Richmont in 2012, for the West and North zones, probable mineral reserves were estimated and measured and indicated mineral resources for the West Zone were estimated.

Table 6-8: Francoeur Mine Production History

Production Years	Tonnes Milled	Gold Grade (g/t)	Ounces Recovered (Au)
1938-1947	520,363	5.6	94,303
1968-1971	385,292	5.6	69,227
1991-2001	1,701,892	6.31	345,112
Total	2,607,547	5.84	508,642

The last mineral resource estimate completed of the Francoeur Mine was conducted by Richmont in 2014 using Gemcom and using a block model approach, constrained within interpolation domains (mineralized zone) with resource classification following 2005 CIM Definition Standards (in effect at that time). The cut-off grade for the resource was of 4.30 g/t Au for a gold price of C$1,450. A measured mineral resource of 40,000 tonnes grading 5.89 g/t Au (7,600 ounces Au), an indicated mineral resource of 280,000 tonnes grading 6.55 g/t Au (59,000 ounces Au) and an inferred mineral resource 18,000 tonnes at 7.17 g/t Au (18,000 ounces Au) was identified by Richmont in the West Zone of the Francoeur mine (Adam, 2014). Work required to bring this historical mineral resource estimate to a current one includes: personal inspection and review by a QP of the drill hole core (mineralization), data validation, geological model review, review all resource key parameters, complete a new estimate with constraining volume as required in 2019 CIM MRMR Best Practice Guidelines.

The mineral resource estimate referred above is historical in nature and should not be relied upon. The QPs has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves. It is included in this section for illustrative purposes only and the Issuer is not treating the historical estimate as current mineral resources.

6.6.3 Arntfield Mine

The Arntfield mine comprises the No. 1, No. 2, and No. 3 deposits. The Arntfield mine was in production from 1935 to 1942, and produced 480,804 tonnes grading 3.98 g/t Au and 0.93 g/t Ag. (Adam et al., 2012).

6.6.4 Lac Fortune

Located less than 5 km south of the Francoeur mine, the Lac Fortune Block contains three veins, including the "Central" Zone, from which a historical mineral resource estimate, regarded as not significant, was calculated by Coopers and Lybrand (1986). The Lac Fortune deposit has never been a producing mine.

6.6.5 R.M. Nickel Block

Although a few historical mineral resource estimates and mineral reserve estimates have been prepared for the R.M. Nickel deposit since its discovery, none of these estimates are currently regarded as significant. The R.M. Nickel deposit is northwest of the property and has never been a producing mine.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Wasamac deposit is the focus of this report and will be described in greater detail in Section 7.3 and Chapter 1.

The Wasamac property is underlain by the Blake River Group within the Rouyn-Noranda mining district, in the Abitibi greenstone belt of the Superior province of the Canadian Shield (see Figure 7-1). The following overview of the regional geology section is from Monecke et al. (2017), unless indicated otherwise.

The Archean Superior Province forms the core of the North American continent and is surrounded by Paleoproterozoic provinces to the west, north and east, and by the Mesoproterozoic Grenville Province to the southeast.

Tectonic stability has prevailed since approximately 2.6 Ga in large parts of the Superior Province. Proterozoic and younger activity is limited to rifting of the margins, emplacement of numerous mafic dyke swarms (Buchan and Ernst, 2004), compressional reactivation, large-scale rotation at approximately 1.9 Ga, and failed rifting at approximately 1.1 Ga. With the exception of the northwest and northeast Superior margins, which were pervasively deformed and metamorphosed from 1.9 to 1.8 Ga, the craton has escaped ductile deformation.

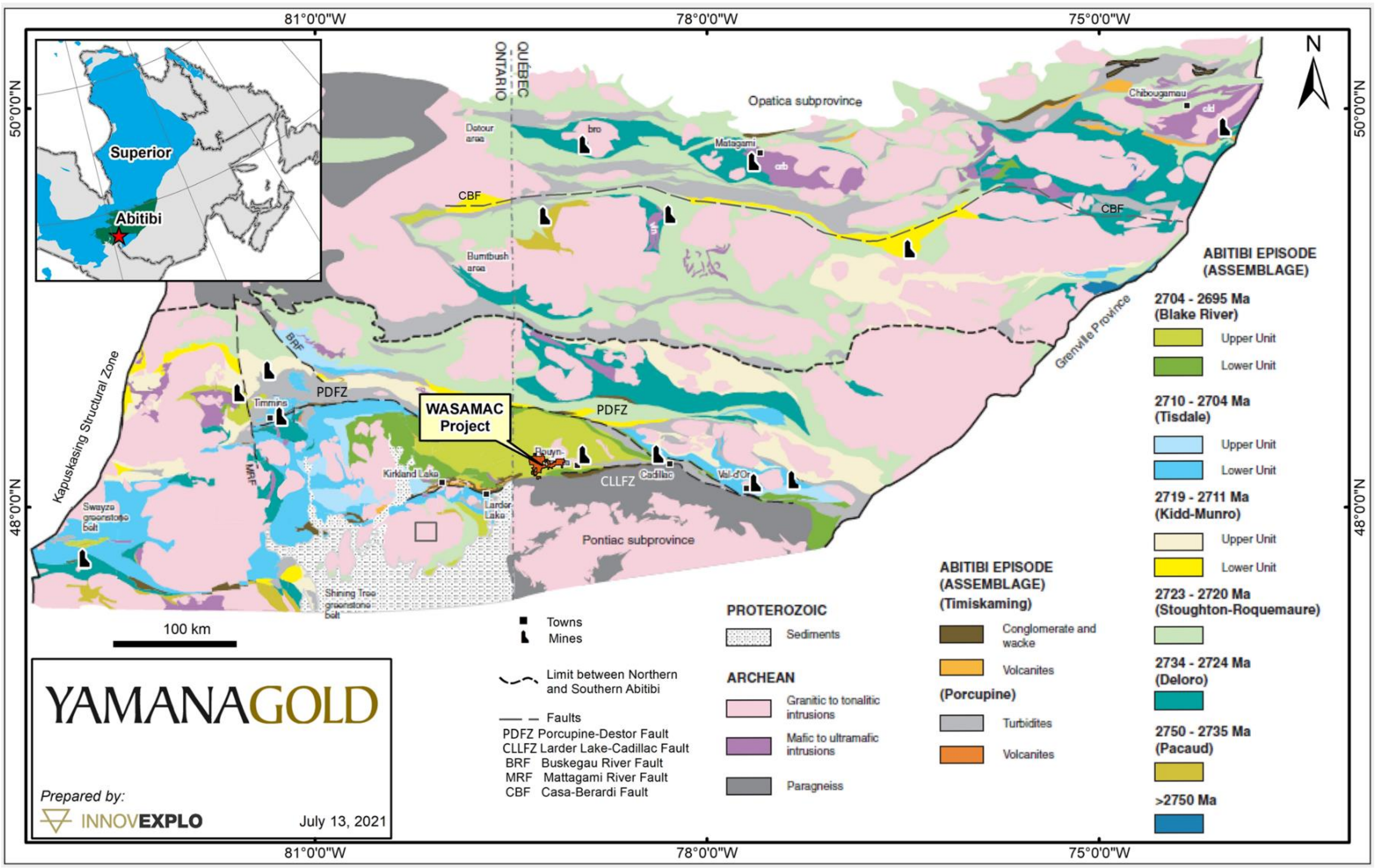
The Abitibi greenstone belt is bounded to the west by the Kapuskasing Structural Zone and to the east, by the Grenville Province. To the north, it is in structural contact with the plutonic Opatica Subprovince. The southern boundary of the Abitibi greenstone belt is marked by the Cadillac–Larder Lake fault zone, a major structural break marking the contact with younger metasedimentary rocks of the Pontiac Subprovince. The Cadillac–Larder Lake fault zone hosts numerous gold deposits and mines that have produced several million ounces of gold. The Kirkland Lake, Rouyn-Noranda, Cadillac, Malartic and Val-d'Or mining camps are located along this deformation zone. The Francoeur/Arntfield/Lac Fortune, Beauchastel and Wasamac mine areas are located along the Francoeur-Wasa shear zone, a second-order kilometric-scale fault of the Cadillac–Larder Lake fault zone, crosscutting meta-volcanic units of the Blake River Group.

The Abitibi greenstone belt was formed over a period that spans approximately 150 Ma and is composed of volcanic rocks and synvolcanic and syntectonic plutonic rocks (gabbro-diorite, tonalite and granite) alternating with east-trending sequences of turbiditic sedimentary rocks (Ayer et al., 2002; Daigneault et al., 2004; Goutier and Melançon, 2007; Monecke et al., 2017). Most of the volcanic and sedimentary strata have a subvertical dip.

According to Monecke et al. (2017) and references therein, the Abitibi greenstone belt is subdivided into eight discrete stratigraphic episodes or assemblages (see Figure 7-2), based on U-Pb zircon ages. Submarine volcanism mostly occurred between 2795 and 2695 Ma and was followed by sedimentation in large deep basins and then by large-scale thin-skin folding and thrusting. New U-Pb zircon ages and recent mapping by the Ontario Geological Survey and Géologie Québec clearly shows similarity in the timing of volcanic episodes and ages of plutonic activity between the northern and southern Abitibi greenstone belt.

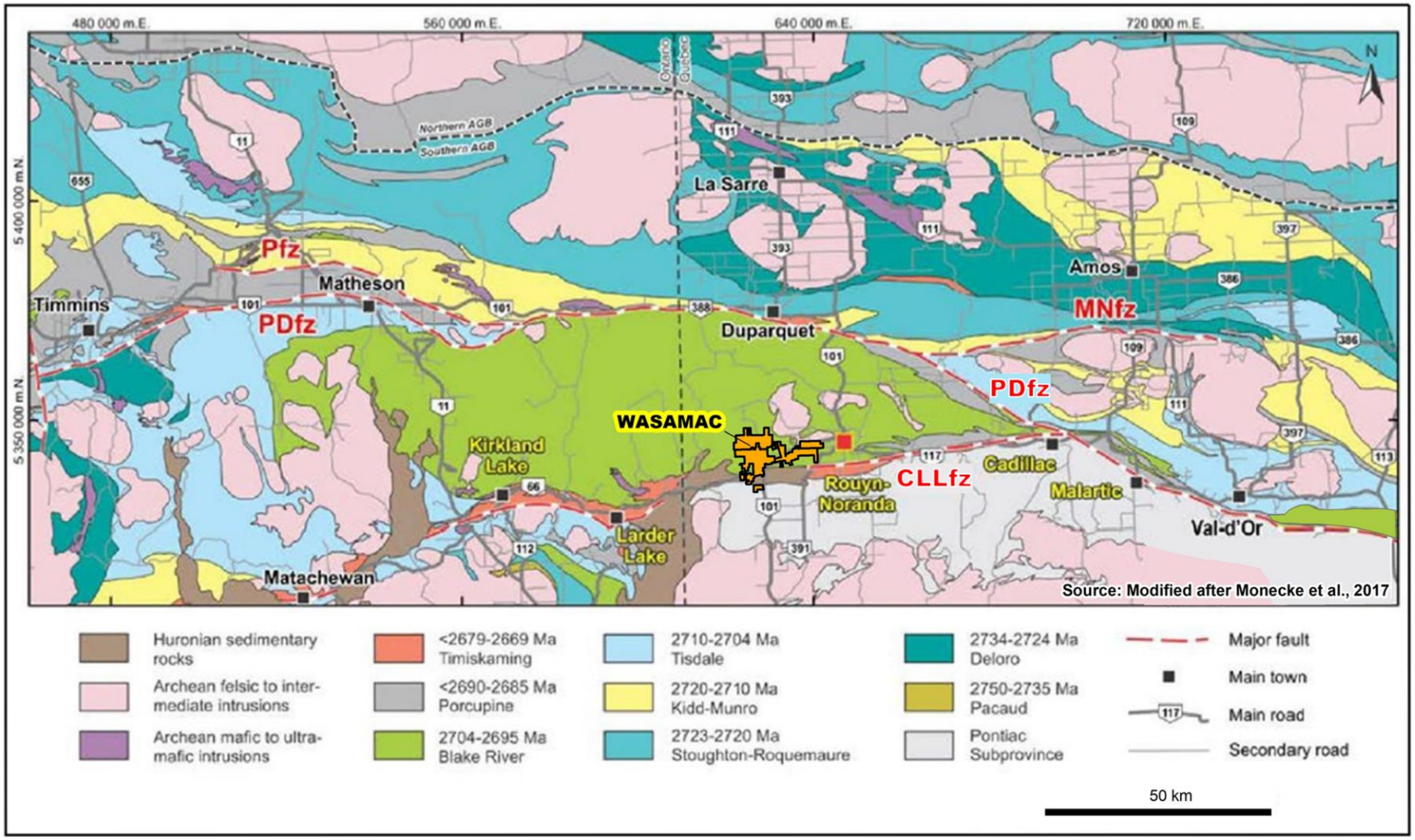
Two periods of sedimentary basin development are recognized: early, widespread and laterally extensive Porcupine-style basins of fine-grained clastic rocks (turbidites); and later Timiskaming-style basins of coarser aerial clastic sediment deposits with minor volcanic rocks that are mainly found in close proximity to major strike-slip faults (Thurston and Chivers, 1990; Mueller et al., 1992; Ayer et al., 2002; Goutier and Melançon, 2007).

Figure 7-1: Geological Map of the Abitibi Greenstone Belt



Source: Modified after Monecke et al., 2017.

Figure 7-2: Geological Map of the southern Abitibi Greenstone Belt



Notes: Pfz= Pipestone fault zone, PDfz= Porcupine-Destor-fault zone, MNfz= Manneville-North fault zone, CLLFZ= Cadillac-Larder-Lake fault zone. Source: Modified after Monecke et al., 2017.

The episodes are listed below from oldest to youngest:

- Pacaud Assemblage (2750-2735 Ma)
- Deloro Assemblage (2734-2724 Ma)
- Stoughton-Roquemaure Assemblage (2723-2720 Ma)
- Kidd-Munro Assemblage (2720-2710 Ma)
- Tisdale Assemblage (2710-2704 Ma)
- Blake River Assemblage (2704-2695 Ma)
- Porcupine Assemblage (<2690-2685 Ma)
- Timiskaming Assemblage (<2679-2669 Ma)

The Abitibi greenstone belt has been intruded by numerous late-tectonic plutons ranging in composition from gabbro to granite, with lesser dykes or plugs of syenite, lamprophyre and carbonatite. Regional metamorphism varies from prehnite-pumpellyite to greenschist facies and may exhibit amphibolite facies near large intrusions.

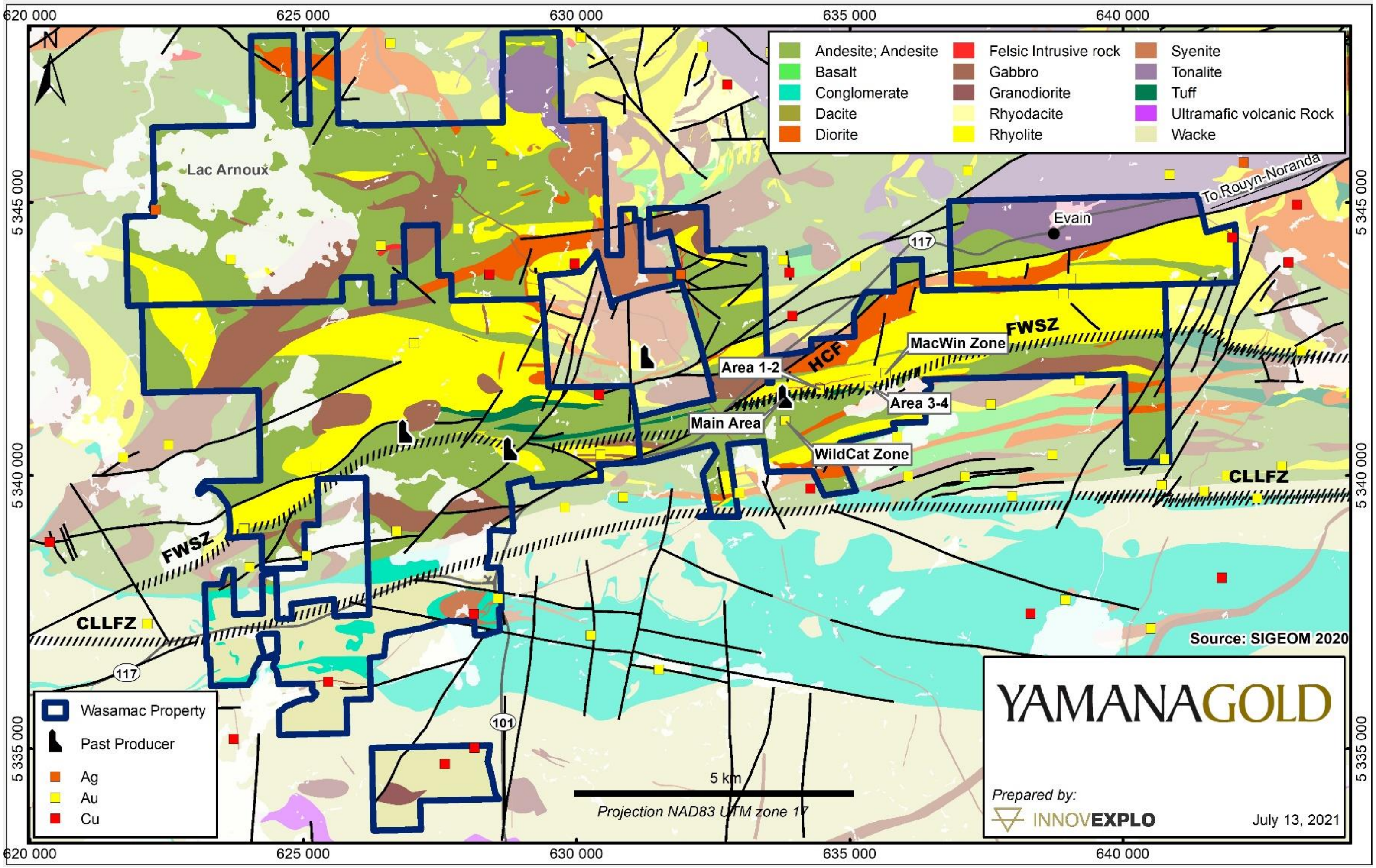
7.2 Local Geology

The following is taken from Monecke et al. (2008) and Mériaud, (2015) unless otherwise indicated.

The Blake River Group hosts the Wasamac property. The surface expression of the Blake River Group in this part of the Abitibi greenstone belt is lenticular, approximately 120 km by 35 km in east-west aspect, limited to the north by the Porcupine-Destor fault zone and Parfouru fault, and to the south by the Cadillac–Larder Lake fault zone (Dion and Rhéaume, 2007). It is composed of bimodal volcanic strata of predominantly tholeiitic to mildly transitional affinity. It comprises numerous alternating mafic and felsic units intruded by mafic to felsic and alkaline plutons, dykes and sills (Legault and Rabeau, 2006). Just north of the property, there are two large granitic intrusions: the Flavrian and the Powell batholiths. These two bodies intrude volcanic rocks and are located within the general axis of the Blake River Group syncline.

The Cadillac–Larder Lake fault zone section between the Rouyn-Noranda and Kirkland Lake districts exhibits two metallogenic environments associated with gold deposits. Gold deposits of the Kirkland Lake area are spatially linked with syenitic intrusions (Robert, 2001; Zhang et al., 2014), whereas in the Rouyn-Noranda area, the Cadillac–Larder Lake fault zone shows disseminated mineralization associated with diversified metasomatic footprints (Couture and Pilote, 1993; Legault and Rabeau, 2007). The property is between these two districts. The Wasamac, Consolidated Francoeur, and Western Buff blocks are along the Francoeur-Wasa shear zone, a second-order fault that is parallel and 2.5 km north of the Cadillac–Larder Lake fault zone; whereas the R.M. Nickel area is 1.6 km north of the Francoeur-Wasa shear zone (see Figure 7-3).

Figure 7-3: Geological Map showing the Traces of Major Fault Zones Underlying the Property



Notes: HCF=Horne Creek fault, FWSZ= Francoeur-Wasa shear zone, CLLFZ= Cadillac-Larder-Lake fault zone. Source: after Mériaud, N., 2015.

7.3 Wasamac Deposit

The Wasamac deposit can be subdivided into two distinct volcanic sequences: (1) a northern part comprising intercalated mafic flows and felsic tuffs and breccias; and (2) a southern sequence dominated mainly by massive mafic to intermediate flows. The contact between these two sequences is the Francoeur-Wasa shear zone, an important metallotect in the sector. The Francoeur-Wasa shear zone is a secondary dislocation that transects the entire E-W extent of the property from approximately 1.6 to 2.5 km north of the Cadillac–Larder Lake fault zone. In the southern part of the property, Proterozoic sedimentary rocks of the Cobalt Group lie unconformably on mafic volcanic rocks.

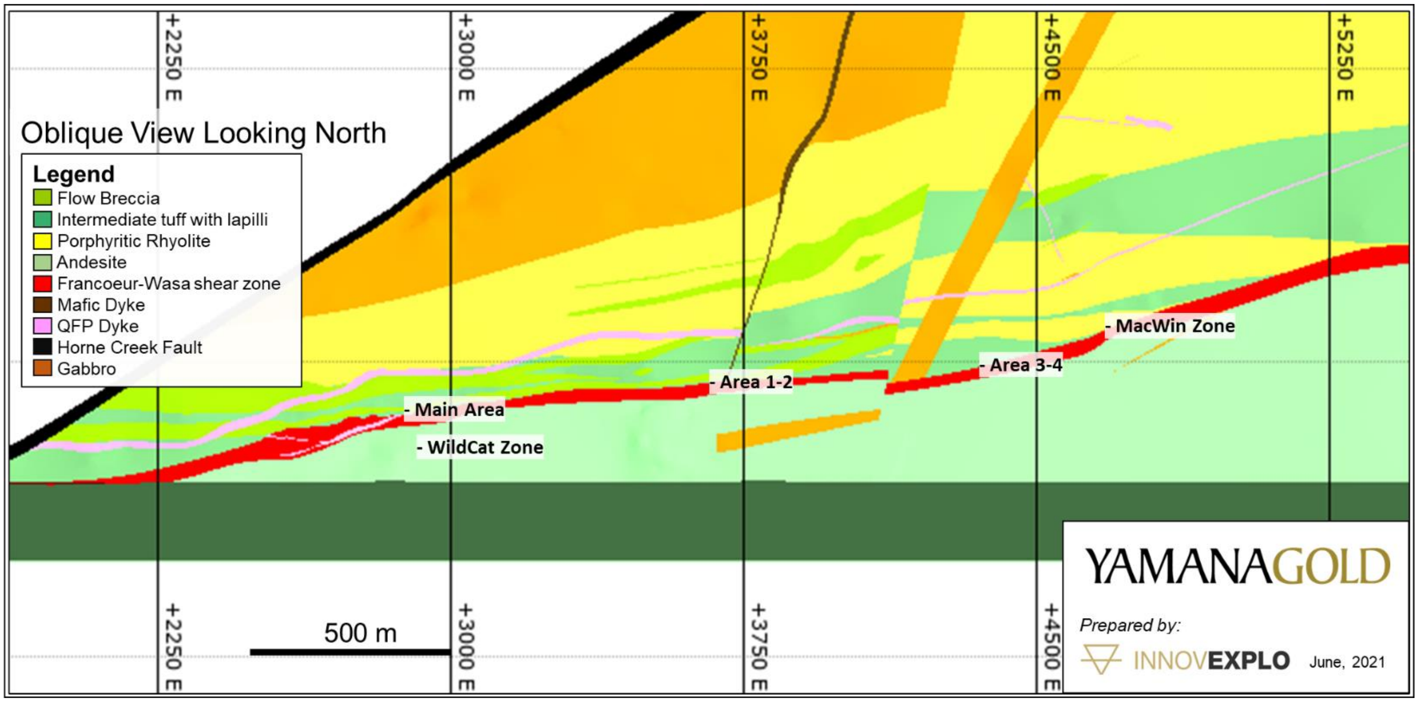
Two generations of intrusive units crosscut the metavolcanic units of the Wasamac property, as intercepted by drilling: gabbroic intrusions and quartz and/or feldspar porphyry dykes. The quartz and/or feldspar porphyry dykes are younger than the gabbro units, but they are both affected by the Francoeur-Wasa shear zone. Elsewhere on the property, Proterozoic diabase and lamprophyre dykes are present.

The volcanic stratigraphy and the Francoeur-Wasa shear zone underlying the Wasamac property follow an east-west trend and dip roughly 55° northward. The Francoeur-Wasa shear zone is a reverse shear zone that control most of the gold mineralization discovered on the property to date.

Gold mineralization at Wasamac is typically associated with finely disseminated pyrite and stockwork of pyrite-rich micro-veinlets hosted in albite-sericite-ankerite alteration zones confined within the shear zone. The albite-sericite-ankerite alteration related to gold mineralization is typically beige-brown and visually distinguishable from the surrounding sheared rocks. Quartz veins are not common and do not significantly contribute to the gold endowment of the system. High continuity and regular geometry, combined with a relatively simple structural setting and consistent mineralized widths of 5 to 30 metres.

The defined Wasamac deposit is characterized by high continuity and regular geometry and combined with a relatively simple structural setting and consistent mineralized widths of 5 to 30 metres. It is relatively shallow compared to other mines in the region; it has a maximum depth of 845 metres below surface and 2.7 kilometres along strike although the deposit is still open at depth and on its lateral extensions. It comprises five mineralized areas, from west to east: Main Area, Area 1-2, Area 3-4, Wildcat Zone, and MacWin Zone. Areas 1-2, 3-4, MacWin Zone, and most of the Main Area are contained within the Francoeur-Wasa shear zone. The Wildcat Zone occurs within a different structure (see Figure 7-3 and Figure 7-4).

Figure 7-4: Lithological Model of the Wasamac Deposit



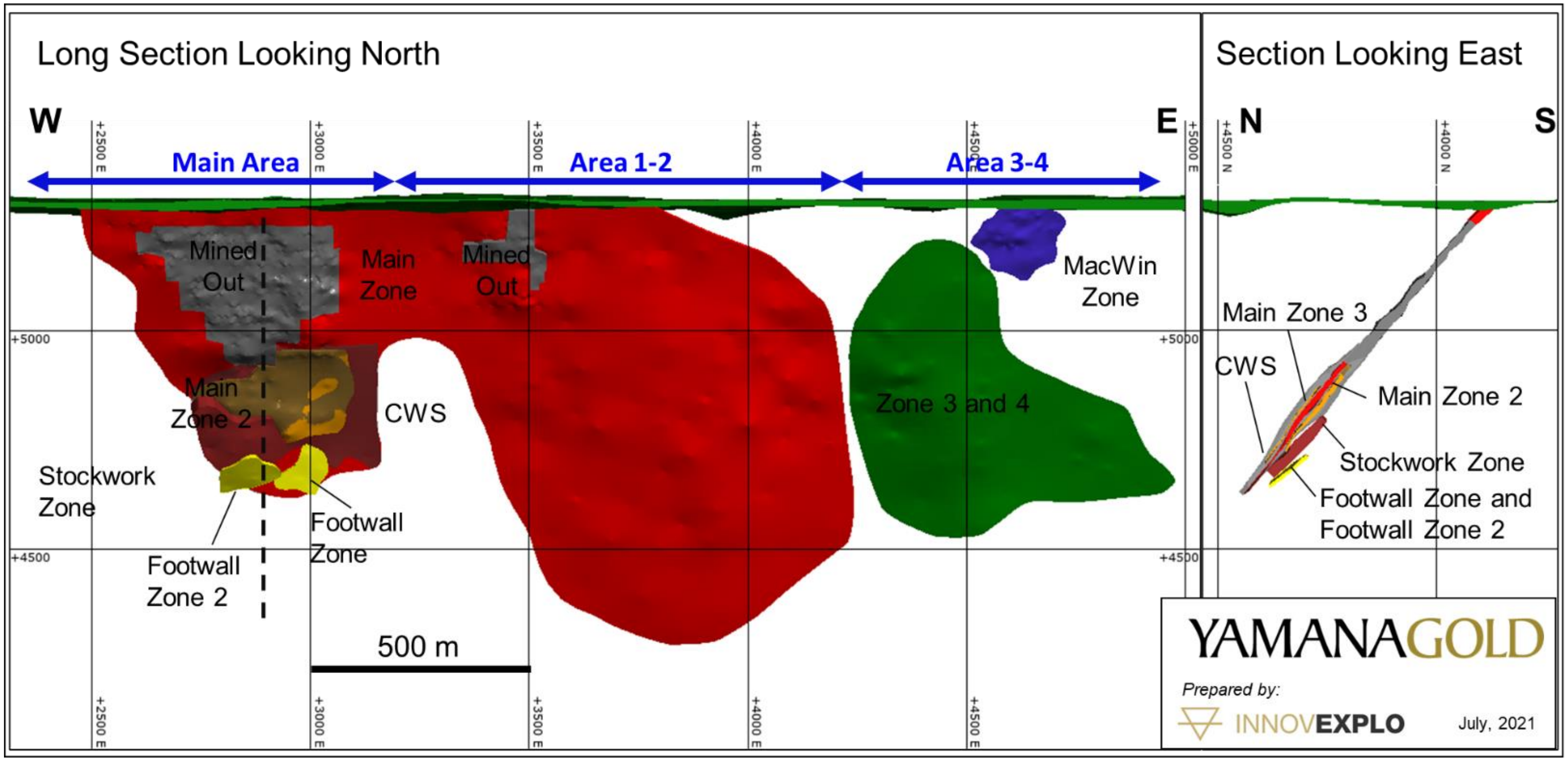
Note: Local mine grid coordinate system.

7.3.1 Main Area

The Main Area is located on the western part of the deposit where it is constrained to the west by the Horne Creek Fault. The upper part of the Main Area was mined during historic underground operations between 1965 and 1971. Gold mineralization in the Main Area occurs within several discrete sub-parallel zones that are generally 5 to 15 m (locally up to 25 m) true thickness. They include the Main Zone, Main Zone 2, Main Zone 3, Stockwork Zone, Footwall Zone, and Footwall Zone 2. The Main Zone, Main Zone 2, Main Zone 3 are confined to the Francoeur-Wasa shear zone and are associated with disseminated pyrite and stockwork of pyrite-rich micro-veinlets hosted in albite-sericite-ankerite alteration zones (see Figure 7-6). Hematized felsic dykes locally occur but further work is required to determine their control on the gold mineralization.

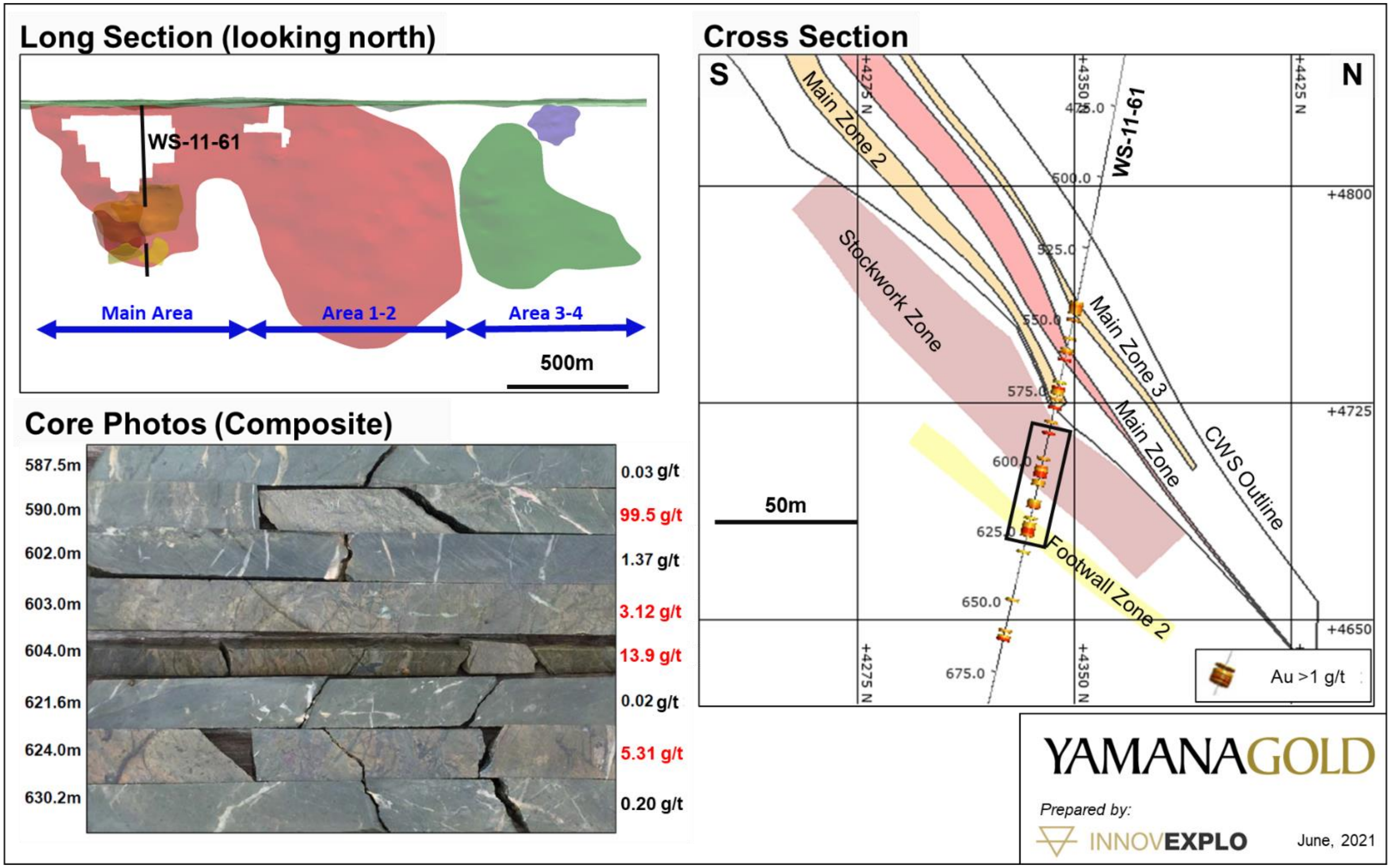
The Stockwork Zone, Footwall Zone, and Footwall Zone 2 occur below the Francoeur-Wasa shear zone and are primarily associated with relatively undeformed pyrite stockwork with minor albite alteration. They are locally associated with higher gold grades (> 10 g/t).

Figure 7-5: Mineralized Areas and Model Completed by Yamana in 2021



Note: Local mine grid coordinate system.

Figure 7-6: Mineralized Zones within the Main Area (Example of Drill Hole WS-11-61)

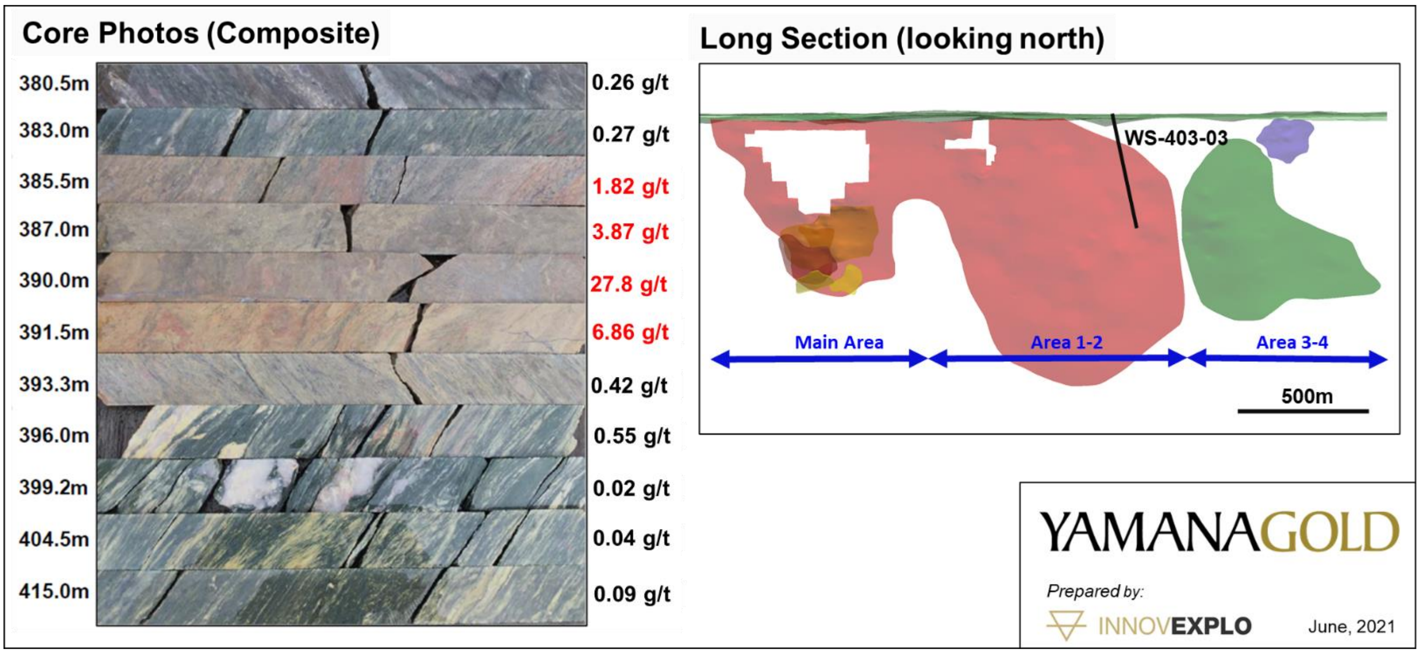


Note: Local mine grid coordinate system

7.3.2 Area 1-2, Area 3-4, and MacWin Zone

Areas 1-2 and 3-4 both contain one single mineralized zone characterized by albite-sericite-ankerite-pyrite alteration (see Figure 7-7). It consists of a continuous structure interpreted as the eastern extension of the Main Zone. Area 3-4 is slightly offset relative to Area 1-2, which is interpreted to be related to a crosscutting structure that offset the shear zone and stratigraphy. Historically, the zone in Area 1-2 and Area 3-4 was separated into four zones (Zone 1, Zone 2, Zone 3, and Zone 4).

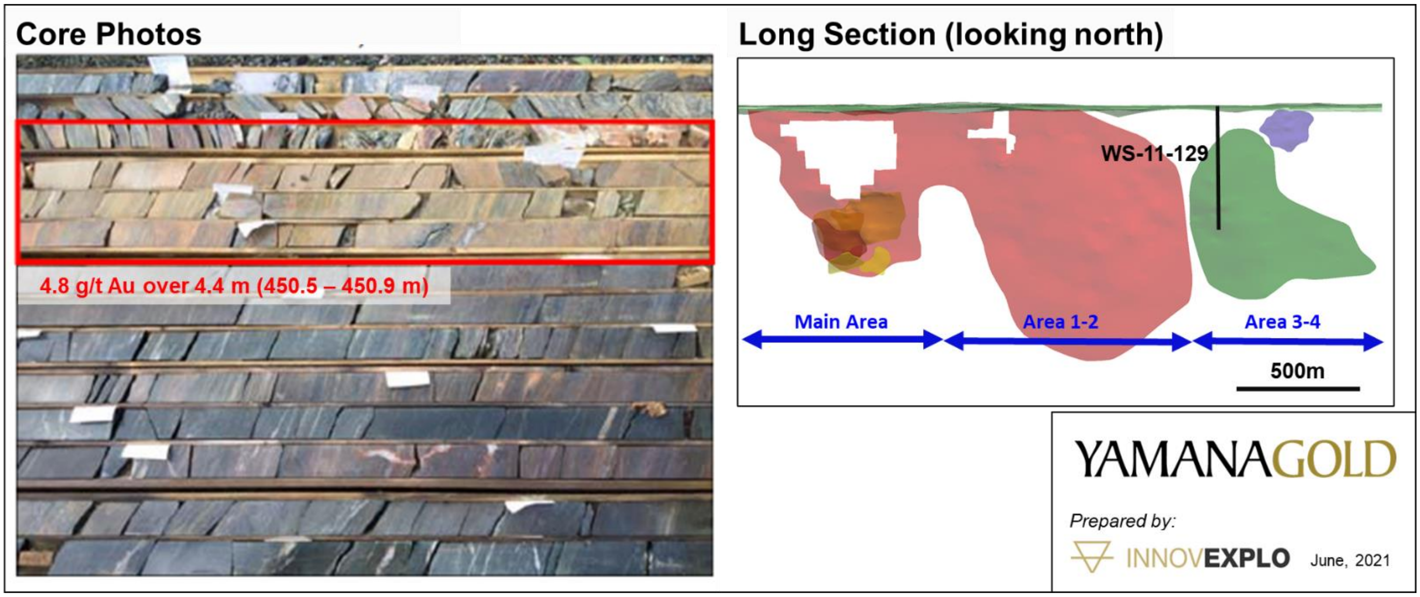
Figure 7-7: Characteristics of Area 1-2 (Example of Drill Hole WS-403-03)



A small portion of Area 1-2 was mined during the last phase of production; however, only limited tonnage was extracted (approximately 100,000 tonnes of ore was mined). Area 3-4 was first intersected during the 2002-2004 drilling programs and was better defined during the 2011 drilling. The zone is around 5m of true thickness over a 750 m strike length (see Figure 7-8).

The MacWin Zone, formerly known as the Wingate Zone, was discovered in 1945 along the Francoeur-Wasa shear zone approximately 300 m east of Zone 3. Gold mineralization occurs both within the shear zone and in the hanging wall rhyolite. A small shaft was completed on the zone; however, this area was included in the mineral resources but voluntarily excluded from the mineral reserves defined in the present report.

Figure 7-8: Characteristics of Area 3-4 (Example of Drill Hole WS-403-03)



7.3.3 Wildcat Zone

The Wildcat Zone, discovered in 1936, was the first gold showing discovered on the property. It is located approximately 300 m south of the Main Zone, and therefore not directly along the Francoeur-Wasa shear zone. Gold mineralization in this zone is related to a carbonate-altered zone at the margins of a gabbroic unit. The gold mineralization is associated with fine-grained pyrite in quartz carbonate veinlets and is described as being erratic. This zone was investigated through underground development work in 1937; however, operations ceased a year later due to lower than expected gold grades. Further surface drilling was completed in 1944, but efforts failed to improve the grade. Limited tonnage was extracted from this zone. The Wildcat shaft was later used as a ventilation raise and was connected to the Wasamac mine by a drift on the 200 ft level.

In 1981, Exploration Long Lac completed 18 drill holes for 1,562 m. These vertical drill holes showed a possible extension of the mineralization to the southeast. Furthermore, drilling by Richmont in this area also indicated a possible extension of the mineralized structure to the southwest. The Wildcat Zone is not included in the mineral resources defined in the present report.

7.4 Wasamac NE Block

The northern part of the Wasamac NE Block is underlain by the tonalitic Powell Pluton and the southern part is underlain by the same NE-SW trending felsic volcanic unit of the Blake River Group that hosts the Wasamac deposit. The Wasamac NE Block hosts the Peltier showing, which is found in a contact zone with strong silicification and hematization between massive rhyolitic lava flows and pyroclastic rhyolites of the Blake River Group. The mineralization consists of varying amounts of pyrite, 1% to 2%, chalcopyrite, and trace magnetite.

7.5 Consolidated Francoeur Block

The following is taken from Adam et al. (2009) unless indicated otherwise.

There are four main rock types underlying the Consolidated Francoeur Block: Blake River Group volcanic rocks, sedimentary rocks of the Timiskaming and Cobalt groups, and late tectonic and synvolcanic intrusions. In the Francoeur/Arntfield/Lac Fortune deposit areas, all rock units face to the north and have variable trends from west to north-west, with moderate to steep dips (Bugnon, 1982).

Volcanic rocks of the Blake River Group that host the gold deposits are the principal rocks exposed on the Consolidated Francoeur Block. The Blake River Group volcanic rocks are underlain by sedimentary rocks of the Timiskaming Group, which are themselves overlain by weakly deformed Proterozoic sedimentary rocks of the Cobalt Group along the south boundary. The Blake River Group volcanic rocks are intruded by mafic gabbro-diorite sills and stocks that are either syn-volcanic or clearly post-tectonic. All lithological unit, except for syenites, are deformed and metamorphosed.

Below the Proterozoic Cobalt sediments, about one kilometre south of the Lac Fortune deposit, the LLCFZ transects Archean rocks and separates rocks of the Blake River Group to the north from sedimentary rocks of the Timiskaming Group to the south. Besides this major structure, the Archean rocks are also affected by two families of very different faults, one of which is related to the zones of Lac Fortune and Francoeur-Wasa shears, and the other to the Beauchastel and Ruisseau Horne faults. Like the regional structures, these faults and shear zones strike E-W. The Lac Fortune and the Francoeur-Wasa shear zone are composed of a series of reverse faults with moderate north dips and are strongly hydrothermally altered.

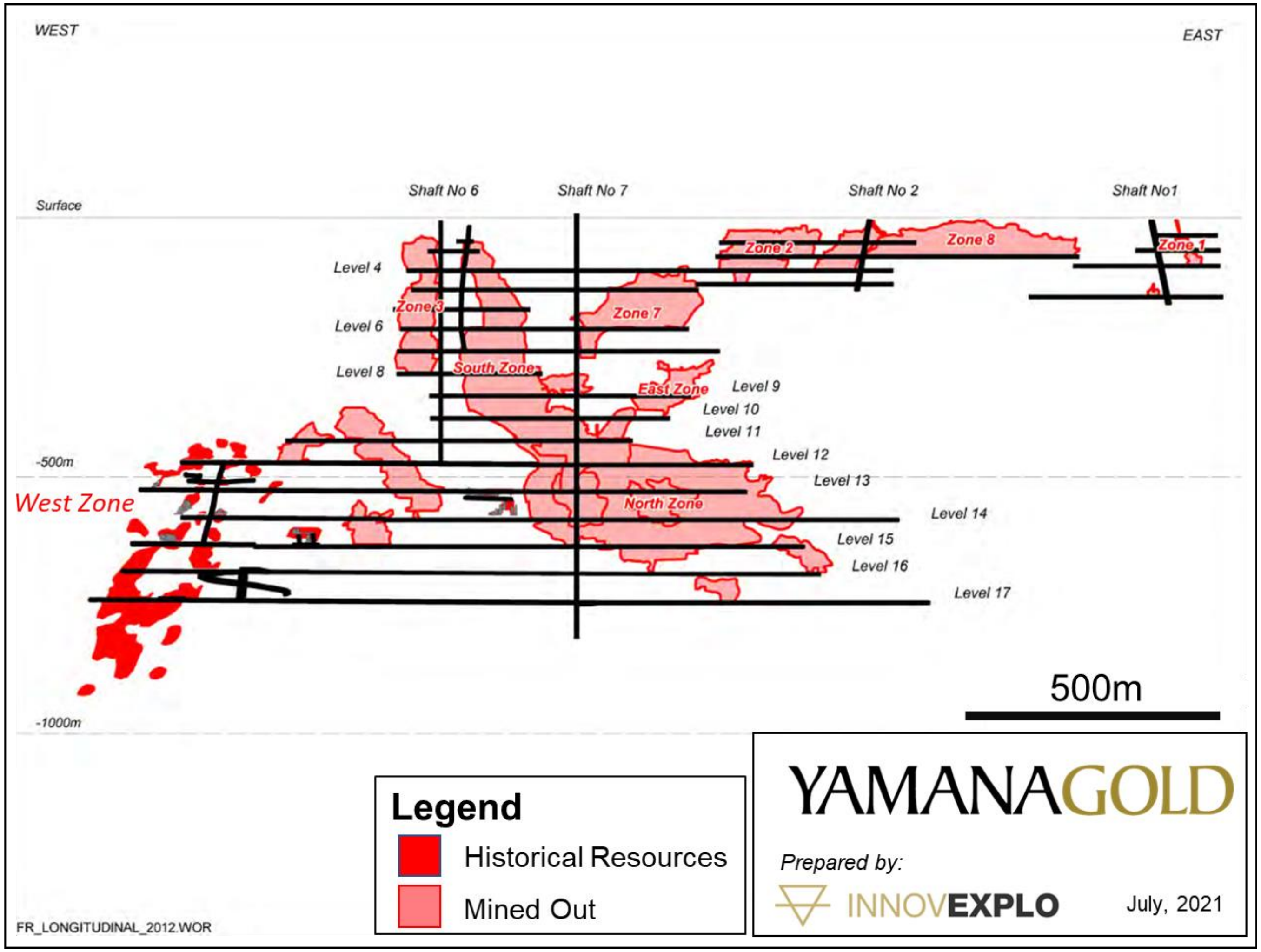
7.5.1 Francoeur Mine

The Francoeur project includes the Francoeur No. 1, No. 2 and No. 3 deposits (see Figure 7-9), which occur along the Francoeur-Wasa shear zone, together with the Arntfield No. 1, No. 2, and No. 3 deposits. Despite some local differences, these deposits are very similar geologically and mineralogically.

Mineralization at the Francoeur deposit is a gold replacement type with coexistence of gold and pyrite disseminated in the altered shear zones. Hydrothermal alteration is well developed, and alteration minerals have distinct zonation from orebody outward: albite-pyrite to carbonate-hematite to muscovite-chlorite. Gold mineralization is closely associated with these alteration haloes, especially albite-pyrite alteration. The Francoeur No. 3 deposit constitutes the main ore zone of the Francoeur project. It was mined until 2001 by Richmont. The No. 3 deposit is hosted in metavolcanic rocks of the Blake River Group. Gold mineralization mainly developed in the ductile Francoeur-Wasa shear zone and is in contact with the southern margin of a gabbro-diorite stock. The mineralized zone extends for at least 1,200 metres down dip from surface to beyond the 17th level. It is a composite orebody consisting of four distinct ore zones, three of which occur within the Francoeur-Wasa shear zone.

The “West Zone” is located to the west of the No. 3 deposit. It is composed of two zones—the main zone (West) and the footwall zone (FW)—both located in the Francoeur-Wasa shear zone and dipping northward at about 30° to 40°. Gold-bearing mineralization is closely associated with albite-pyrite alteration. The West Zone plunges to the northwest, rather than to the general northeast plunge observed in most of the other deposits in the Francoeur project.

Figure 7-9: Vertical Longitudinal Section of the Francoeur Mine



Note: 2012 mineral resources are included for illustrative purposes only; the Issuer is not treating the historical estimate as current mineral resources. Source: Modified from Adam D. et al., 2012.

7.5.2 Arntfield Mine

The Arntfield mine consists of the No. 1, No. 2 and No. 3 deposits. It occurs along the Francoeur-Wasa shear zone and is linked to the Francoeur mine and to the Wasamac mine, showing similar geology and type of mineralization. The Arntfield mine is reported to have produced 480,804 tonnes grading 3.98 g/t Au and 0.93 g/t Ag between 1935 and 1942.

7.5.3 Lac Fortune

The Lac Fortune project is located less than 5 km south of the Francoeur mine. Gold mineralization is mainly found in a highly strained shear zone with chert and quartz in clusters and veins that crosscut a carbonated andesite unit. This andesite is part of the Blake River Group, and the Lac Fortune shear zone is likely associated with the CLLSZ.

Gold is primarily associated with quartz veins and pyrite, but also can be found as visible gold, and as tellurium. Gold mineralization has also been observed in quartz veins contained in pillow basalt south of the carbonate band and in weathering bands bordering porphyry (Karpoff, 1986a).

7.5.4 Western Buff Block

The information for this section is summarized from Pilote (2009), unless otherwise stated.

The Western Buff Block is located approximately 25 km west of Rouyn-Noranda, just north of Lac Evain. It is at the southwestern limit of the area underlain by the Blake River Group, and approximately 2 km south of the Cadillac–Larder Lake fault zone in meta-sedimentary rocks of the Archean Pontian Group. The main lithology of the area is an east-west trending biotite schist metamorphosed to amphibolite facies. Small felsic intrusions and a series of NNE trending faults cross-cut the sediments.

Two types of mineralization were observed to underlie the Western Buff Block:

1. Vein-type characterized by a set of quartz veins with chalcopyrite, galena and sphalerite and minor pyrite). Veins #1 and #2 are of this type.

2. Disseminated type corresponding to disseminated native copper mineralization parallel to the bedding in the meta-sedimentary rocks and/or associated with mafic rocks (dyke or volcanic rock). This type has been less studied.

All zones described below are hosted in the meta-sedimentary rocks of the Pontiac Group (see Figure 7-10).

7.5.4.1 Vein #1 Western Buff I Mineral Occurrence – Cu (Ag, Pb, Zn)

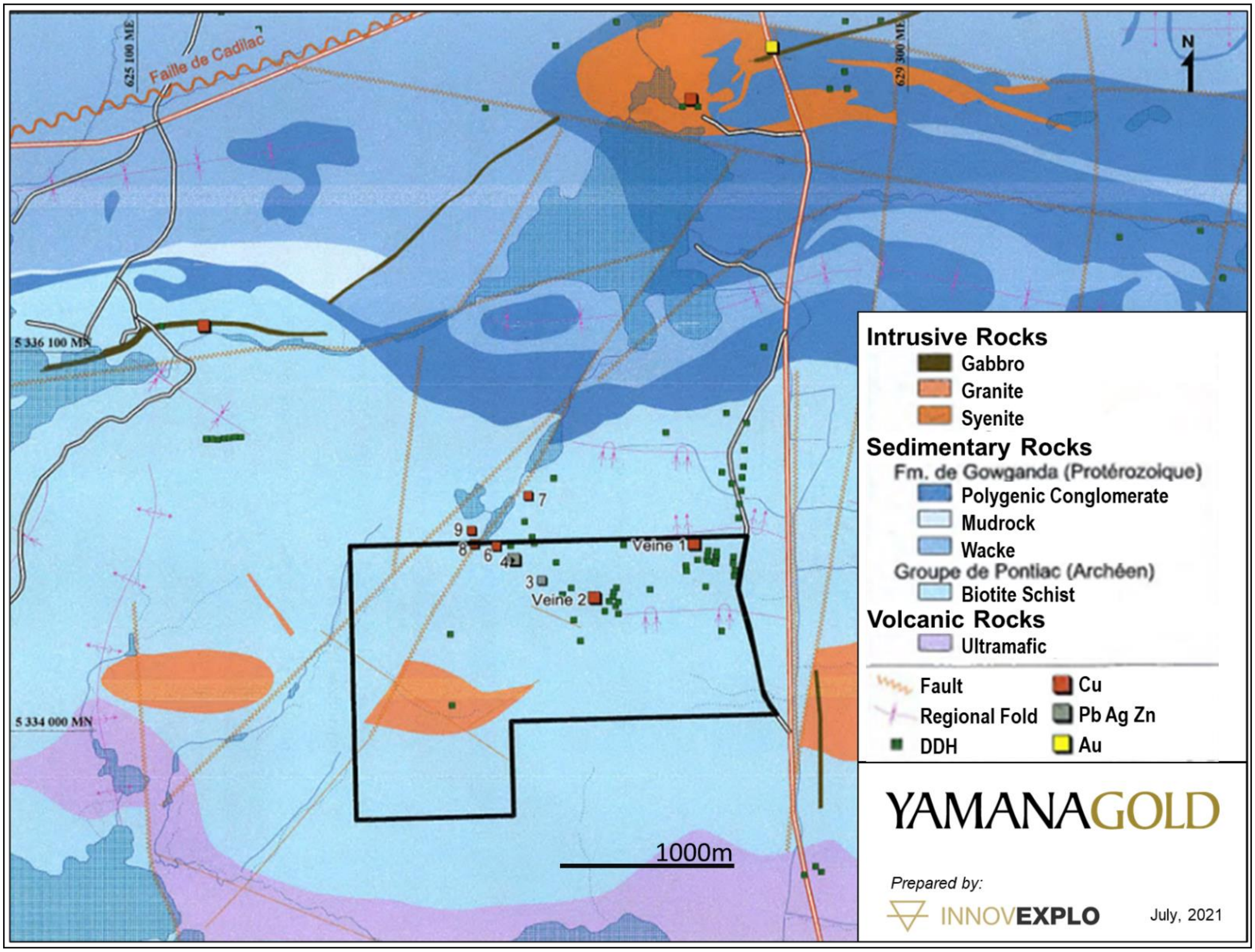
Vein #1 is orientated 190° and dipping 80° east. Quartz veins host decimetric clusters of chalcopyrite, pyrite and traces of sphalerite, galena and bornite and Ag. The occurrence has been stripped and intersected by 45 drill holes. GM48825 details the best intersections for this zone.

7.5.4.2 Vein #2 Western Buff II Mineral Occurrence – Cu (Au)

Vein #2 is orientated 215°, dipping 80° east and is similar to Vein #1. In total, 20 drill holes were drilled in the area, following the vein over 300 m and defining four mineralized zones of interest (1.74% Cu / 3.16 x 35 ft, 1.73% Cu / 3.50 x 25 ft, 1.99%

Cu / 4.08 x 100 ft, 1.48% Cu / 2.33 x 15 ft). The chalcopyrite and pyrite mineralization is massive and disseminated with quartz veins associated with a fault.

Figure 7-10: Locations of Vein #1, #2, and Mineralized Mineral Occurrences 6 to 9 on the Western Buff Block



Source: Modified from Pilote, 2009.

7.5.4.3 Vein #3 and #4 (Pb, Ag, Zn)

Both are quartz veins mineralized with galena ± silver and Zn. Vein #3 is orientated NNE with selected samples yielding 2.68 oz/t Ag, 38.1% Pb and 8.8% Zn. Vein #3 is approximately 0.15 m wide and more than 9 m long. Vein #4 is located approximately 200 m northwest of Vein #3 and is over 0.15 m wide and 4.5 m long. A selected sample graded 2.08 oz/t Ag and 41.18% Pb.

7.5.4.4 Mineral Occurrences #6 to #9

Mineral occurrence #6 comprises massive galena, chalcopyrite and disseminated sulphides in quartz veins associated with mafic volcanic rocks. Mineral occurrence #7 is a disseminated quartz veined chalcopyrite mineralization orientated parallel to bedding near an NNE major fault. Mineral occurrences #8 and #9 are Cu-Pb showings located in association with the same NNE fault zone as Mineral occurrence #7.

7.6 R.M. Nickel Block

The information for this section is taken from Kelly (2020) and Pilote (2010) unless otherwise stated.

The R.M. Nickel Block is located nearly 6 km north of the Cadillac–Larder Lake fault zone (see Figure 7-3). and nearly 2.5 km south of the regional Hunter Creek Fault The western part of the block is covered by Arnoux Lake. Several mineralized showings were observed in the block, including gold, copper, and platinum group elements (PGE).

The block is dominantly underlain by mafic to felsic volcanic rocks and the Horseshoe gabbro-diorite intrusion. The volcanic sequence consists of an interstratification of felsic lavas (massive to brecciated with laminar flow and autoclastic breccias) and mafic to andesitic lavas (massive, pillowed, brecciated with hyaloclastites), with polarity towards the south. The sequence is intersected by a series of Proterozoic dykes and several NE trending faults.

Centrally located within the R.M. Nickel Block is the R.M. Nickel deposit, which is characterized by two types of mineralization:

1. The first is associated with the contact between the Horseshoe gabbro and andesitic flows. These are bands of massive, locally marbled sulphides, up to 2.4 m thick, which mainly contain pyrite, chalcopyrite, millerite and magnetite. The gangue observed at this location is generally silicified and chloritized. This type of mineralization constitutes the bulk of the economic mineralization accessible through an open pit.

2. The second type of mineralization is mainly observed within the Horseshoe gabbro, above the massive sulphide bands. Originally identified as "spotted ore", these are coarse tonsils that appear to have replaced crystals of mafic composition along fine fractures. These concentrations of sulphides are typically accompanied by calcite and/or epidote.

7.7 Teck JV Block

The Teck JV Block is located between the southeastern-most corner of the R.M. Nickel Block and the northwestern-most corner of the Wasamac Block. It lies directly between the Gan Cooper (Cu-Zn-Ag-Au) showing to the west and C-1 (Zn-Ag) showing to the east.

The Gan Copper showing was discovered in 1938 and was described as altered basic and intermediate lavas of the Blake River Group that are crosscut by narrow quartz veinlets with massive and disseminated chalcopyrite and pyrite mineralization and the present of dalmatianite alteration (1960 Copper in Québec Report: ES004-A). The C-1 Zn-Ag showing, discovered through diamond drilling in 1988, intersected a 6.1 m section of epidotized massive volcanic rock with 5% disseminated pyrite.

The Aldermac Pluton (syenite), which intrudes felsic volcanic units of the Blake River Group, is the dominant rock type underlying the Teck JV Block. A similar geological setting hosts the Aldermac mine, 2 km to the south. The Aldermac mine is a massive Cu-Zn-Ag deposit that produced 1.87 Mt of Cu-Zn ore that averaged 1.47% Cu (Zn was not recovered).

8 DEPOSIT TYPES

8.1 Wasamac Deposit

The Wasamac deposit is an example of an Archean greenstone belt gold deposit hosted by the Francoeur-Wasa shear zone, a second-order brittle-ductile shear zone of the Cadillac–Larder Lake fault zone. Gold mineralization is constrained to the altered and sheared part of the Francoeur-Wasa shear zone. Regionally, the Wasamac deposit lies at the boundary between the orogenic gold district of Noranda and the dominantly intrusion-related gold systems of Kirkland Lake. The Wasamac deposit shares similar characteristics of both alkaline syenite intrusion-related gold deposits and orogenic gold deposits (see Table 8-1).

There are several examples of orogenic or intrusion-related deposits along the Cadillac–Larder Lake fault zone including: the Francoeur deposit (500 koz produced), Kirkland Lake deposits (21 Moz produced), Kerr-Addison (9.5 Moz produced), and Lapa (700 koz mineral resource) (Mériaud, 2015). One difference between Wasamac and these other deposits is that they all have mineralization related to veining, a characteristic that is missing at Wasamac, which exhibits disseminated mineralization only.

The Wasamac deposit has most similarities with the Francoeur deposit, for which the geology and the alteration have been described by Couture and Pilote (1991, 1993) and Gao (1994).

Table 8-1: Wasamac Deposit Characteristics

Characteristic	Description
Host lithology	Meta-andesite
Host structure(s)	- Second order Francoeur-Wasa shear zone, 2.5km north of the Cadillac–Larder Lake fault zone - Disseminated gold-bearing sulphides and free native gold grains in altered mylonite
Alteration mineralogical characteristics	Two gold-rich alteration assemblages: 1) K-feldspar, sericite, carbonate; quartz; hematite and pyrite 2) albite, sericite. carbonate, chlorite, and pyrite—the alteration is visually distinguishable from the surrounding sheared rocks.
Mineralization mineralogical characteristics	1) Telluride gold minerals associated with As-free native gold 2) Disseminated free native Au
Gold resources and historical production	- 276,536 oz Au produced - 1,125,727 oz Au mineral resource
Genetic interpretation	- Magmatic-hydrothermal genetic model - Bi-phased alkaline hydrothermal circulation within orogenic structure possibly intrusion related

Source: Modified from Mériaud (2015), after Groves et al., 1998; Lang and Baker, 2001; Robert, 2001; Goldfarb et al., 2005

8.2 Francoeur and Arntfield Deposits

The Blake River Group rocks underlying the Francoeur and Arntfield gold deposits are intruded by diorite and gabbro masses. The Francoeur No. 3 deposit is the largest of a series of deposits located within the Francoeur-Wasa shear zone over a distance of more than 10 km. These deposits are, from west to east, the Cutting zone and the No. 1, 2 and 3 deposits of the Francoeur and Arntfield mine, the Wasamac Principal, No. 1 and 2 deposits as well as the Wingate deposit.

Mineralization at the Francoeur deposit is a gold replacement type with close coexistence of gold and pyrite disseminated within and peripheral to altered shear zones. Hydrothermal alteration is well developed, and alteration minerals have distinct zonation from orebody outward: albite-pyrite to carbonate-hematite to muscovite-chlorite. Gold mineralization is closely associated with these alterations, especially albite-pyrite alteration.

The quantity of gold metal (production and mineral reserves) of these deposits is estimated at approximately 45 tonnes (Pilote, 1994). Although their local location differs, the morphology and nature of all these deposits are very similar. Typical mineralization consists of disseminations of pyrite and gold in sheared and strongly carbonatized rock.

8.3 Additional Mineral Showings

The Abitibi greenstone belt is known for its prolific and diverse rich mineral deposits. Table 8-2 summarizes the principal metallic showings and deposits underlying the property.

Table 8-2: Additional Property Showings and Deposits

Claim Block	Showing	Discovery Date	Commodity	Deposit Type	Description
Wasamac NE	84	1963	Au	Insufficient Information	DDH 84 intersected 4.1 g/t Au / 1.1 m and 2.0 g/t / 0.9 m in an altered lava
	Peltier	1986 (DDH 86-RB-20)	Au	Exhalative and synvolcanic	Strong Sil and Hem at contact zone between rhyolite units of the Blake River Group. DDH 86-RB-20 (2.23 g/t Au / 0.60 m and 2.09 g/t / 0.80 m) GM45594
	Caron-SO	1989	Cu	Fractured host intrusion in-filled with mineralized veins (lode)	Grab sample from mineralized gabbro: 0.97%Cu (sample #19994) and 0.69%Cu (#19992) associated with quartz veins
Wasamac	C-1	1988	Zn-Ag	Insufficient Information	Epidotized volcanic rock with 5% Py disseminations/fracture-filling: 12.5 g/t Ag and 0.12% Zn / 6.1 m (DDH C-1, GM48680)
	McDonough	1935	Au	Mesothermal; Fractured host rock with quartz-gold fluid	Sheared Blake River Group Andesite with Qtz-Cb ± Au veins: 41.14 g/t Au / 0.50 m (GM 43824)
Consolidated Francoeur Block	O'Leary Malartic	1949	Cu	Sedimentary hosted (lode)	Qtz contact zone between diabase dyke and biotite-gneiss (Pontiac Group) with Cpy-Bo mineralization
	Guinard	1935	Cu-Au	Intrusion-related (lode)	Qtz-Cpy-Py veins in a syenite within the Timiskaming Group sediments. Grab samples: 1.28 g/t Au (GM 06217); 1.09 % Cu (2005043335); 0.88 % Cu (2005043323)
	Lac Renaud-Sud	1986	Au	Intrusion-related (lode)	Qtz-Py veins in a syenite in Timiskaming Group sediments. 94 g/t Au (Sample G-86-15, GM 43707)
	Lac Fortune	1906	Cu-Au	Hydrothermal replacement	Qtz-Tour-Ser-Cb veins with Py ± Cpy in a strongly sheared andesite
	Lac Saines Zones East	1940	Cu-Au	Volcanic-related (lode)	Massive and disseminated Py ± Cpy Qtz-veined contact between a sheared andesite and quartz-diorite (Blake River Group). 34.29 g/t Au (grab sample)
	Lac Saines Zones West (Bédard)	1943	Cu-Au-Ag-Mo	Vein and shear zone hosted (volcanic)	Disseminated Py-Mo-Cpy-Bn-Au-Ag in a shear zone within an Andesite (Blake River Group)
	Arncoeur	1936	Au-Cu	Linked to Arntfield and Francoeur mineralization	Andesitic tuff and rhyolite crosscut by massive to disseminated Py-Cpy-Au in Qtz-Cb veins. 3.43 to 5.83 g/t Au / 0.84 to 5.2 m (DDH ES-002)
	As-87-86	1987	Au	Fracture/fault-hosted	Gold fracture-hosted mineralized andesite (Blake River Group); 2.056 g/t Au / 1.52 m' GM47310)
	Dickenson	1988	Au	Fracture/fault-hosted	Shear-hosted gold in mylonitic QFP. Chl-Ser-Sil-Hem alteration associated (GM48409: 3.37 g/t Au / 9.10 m (DDH D-88-05)
	Chance	1988	Zn-Cu	Volcanogenic massive sulphide	Cpy and Sp MS mineralization in silica and chlorite altered volcanics rocks (1.7% Cu and 1.6% Cu (surface samples, GM-48381)
	Gryffondor	2006	Ni-Cu-Au	Magmatic and hydrothermal Cu-Ni-PGE	Clusters of Py-Cpy-Pn in Horseshoe gabbro intersecting volcanics rocks of the Blake River Group
	Morgan-Piche	1938	Au	Volcanic and intrusive rocks (lode) + fracture/fault-hosted	Shear-hosted Au-Py-Hem-Qtz ± Cpy-Po mineralized rhyolite, syenite and andesite (Blake River Group). 0.07 to 32.9 g/t Au (Grab samples)
Western Buff Block	Western Buff II (Vein #1)	1930	Cu (Ag, Pb, Zn)	Sedimentary-hosted (lode) (Fracture-filling hydrothermal mineralization)	Meta-sediment of Pontiac Group host rock: set of Qtz veins with Cpy, Gn and Sp (rare Py)
	Western Buff II (Vein #2)	1929	Cu (Au)		Similar to Vein #1. Cpy and Py mineralization is massive and disseminated.
	Western Buff II (Vein #3 and 4)	1929	Pb, Ag, Zn		Both veins are Gn ± Ag and Zn mineralized quartz veins. Vein #3: 2.68 oz/t Ag, 38.1% Pb and 8.8% Zn and Vein #4: 2.08 oz/t Ag and 41.18% Pb
R.M. Nickel Block	R.M. Nickel Deposit	1947	Ni-Cu-Ag-Au	Magmatic	Mineralization within Horseshoe gabbro and at contact with Blake River Group volcanics with Horseshoe gabbro
	Lac Arnoux	1948	Au-Ag-Pb-Ni-Cu	Fractured host rock in-filled with quartz-gold fluid	Py-Cpy-Gl-Qtz veins in a shear zone intersecting gabbro and volcanic rocks
	Baribeau	1948	Au-Cu	Hydrothermal and exhalative	Massive and disseminated gold and copper-bearing Qtz-Cb veins cross-cutting andesite (Blake River Group): 35.2 g/t Au (sample, GM 02461)
	Provencher	1940	Au	Hydrothermal replacement	Gold mineralization (disseminated pyrite) hosted in strongly fractured and altered intrusive rocks and porphyritic rhyolites of the Blake River Group. 3.54 g/t Au / 0/55 m (DDH CM-06-16; GM 64143)
	B-2-81	1981	Au-Ag	Insufficient Information	DDH B-2-81 (GM38073) with 3-12% fine Py disseminated in a felsic tuff: 2.03 g/t Au and 4.07 g/t Ag / 0.95 m (sample 36740, DDH B-2-81)
	Rivière Arnoux	1950	Au	Fractured host rock in-filled with quartz-gold fluid	Qtz-Py (Au)-Cb-Ep veins in andesite; selected samples (GM 05879): 4.46 g/t Au; 5.4 g/t Au
	Lac Arnous East	1948	Au-Ag-Pb-Ni-Cu	Shear and fractured host rocks (gabbro and volcanics) in-filled with quartz-gold fluid	Gold-bearing (Py-Cpy-Gl mineralized) quartz veins contain disseminated to massive sulphides (15 to 80%) within gabbros and volcanics
	West Gan	1926	Zn-Cu-Au	Magmatic	5-10% Po and Cpy in quartz veins cross-cutting rhyolites and diorites of the Blake River Group
	Gan Copper	1938	Cu-Zn	Magmatic	Disseminated to SMS/MS (Cpy and Py) associated with a brecciated rhyolite and andesite of Blake River Group; 1.46% Cu / 0.75 m (DDH B-5, GM35502)

9 EXPLORATION

9.1 Exploration Completed by the Issuer

9.1.1 Very High-Resolution Helicopter-borne Magnetic Survey

Novatem Inc. (Novatem) was mandated by Yamana to carry out a very high-resolution helicopter-borne magnetic survey covering the whole Wasamac (22.8km^2) and R.M. Nickel (32.3km^2) claim blocks. Novatem carried out the survey between April 11th and April 19th, 2021. The instrumentations mounted on a Guimbal G2 light helicopter (supplied by Synergy Aviation based in Edmonton pilots and mechanics) for its very-high resolution helicopter-borne system were as follows:

- Two very high-resolution laser optically pumped scalar magnetic sensors, mounted at the front of the magnetometer stinger.
- A real-time multi-frequency GNSS and RTK sensor positioning system capable of receiving the GPS, Glonass, Galileo and BeiDou constellations.
- A very high-resolution fluxgate vector magnetic sensor, manufactured by Billingsley, also mounted on the end of the magnetometer pole.
- An attitude angle measurement system (Inertial Measurement Unit), manufactured by Microstrain, for magnetic compensation;
- A "draped" acquisition and navigation system (SAMM) developed by Novatem, making it possible to follow a continuous flight surface, calculated in advance, and therefore to minimize deviations at intersections of lines and tie-lines;
- A compensation system developed by Novatem for very high resolution using jointly the components provided by the fluxgate vector magnetometer, the angles measured by the attitude center, and inversion algorithms optimized for the calculation of the coefficients.

The flight lines spacing was 25m (direction of N0°) and the tie-lines spacing was 250m (direction of N90); the survey totaled 2,892 linear km.

The sampling rate was reduced to 10Hz (10 measurements per second) for all the measurements (Attitude angles, components of the magnetic field and the total magnetic field measurements)

All instruments, including spare ones, were tested and calibrated prior to mobilization. The configuration was then tested at the location of the survey. A magnetic and GPS base station was installed far from the roadhouse, isolated from human disturbances. The technical manager of Novatem collected the data every day. The location was chosen according to Novatem's specifications: location far from anthropogenic disturbances and a weak local gradient in particular. These validation measurements were performed using the station. Throughout the project, thorough quality checks were carried to ensure the quality of the data and, on the other hand, that all the expected data was present.

The magnetic measurements were corrected for disturbances due to the helicopter. The compensated measurements were then corrected for variations in the external magnetic field (mainly diurnal variations and pulsations) using measurements from the magnetometric base station. The residual intrinsic directional error of each magnetometer ("heading error") was recalculated and subtracted from the measurements for each direction of flights. Finally, an iterative leveling procedure was applied, to eliminate the residual errors caused mainly by the variations in height. The final data including derivative maps (Gradients, Tilt derivative, Analytical Signal, Reduction to Pole) was merged into a Geosoft format file as requested by Yamana.

The magnetic survey was used and will be used to refine the geological models and geological interpretations of those two claim blocks; magnetics survey help identify contacts between units with magnetic contrasts and breaks or shifts within lithological units caused by faulting and shearing.

9.2 Exploration Potential

In addition to the mineral reserves used as the basis for the mine plan in the feasibility study and the currently define mineral resources, Wasamac expansion potential at depth and in other areas of the property. Furthermore, there are additional opportunities to increase conversion of mineral resources to mineral reserves, especially close to previously mined areas of the property. Yamana has commenced an infill and exploration drilling campaign to generate additional mineral resources and mineral reserves.

Data compilation and drill planning combined with a recent high resolution airborne magnetic survey have established numerous exploration targets within the property. Preliminary plans include 120,000 metres of drilling in 2021 and 2022 with a budget of C$15 million over a two-year period.

Infill drilling to better delineate areas expected to be developed in the first three years of production is expected to include 30,000 metres in 2021, with a further 38,000 metres in 2022 to provide further delineation of the remaining mineral resource. This work is expected to increase confidence in grade, improve mine planning, and provide further geotechnical and metallurgical data.

A concurrent exploration effort will focus on expanding the current mineral resource envelopes to depths below the established mineral resource and with testing for mineralization targeting the poorly explored gaps between zones (Figure 9-1). Exploration on the broader Wasamac property is expected to include 10,000 metres in 2021 with an effort to delineate secondary zones such as Wildcat and test high priority extensions of the Wasa Shear (see Figure 9-2).

East of the defined Wasamac deposit, recent magnetic survey and historic drilling indicate strong potential to trace and test the Wasa Shear for a further 3.2 kilometres. West of the Wasamac main zone the shear is displaced along a north-east trending post-mineral fault and the Horne Creek fault. Geological and geophysical information as well as significant gold intercepts in historic drilling have delineated a high priority target along 1.5 kilometres from the Horne Creek fault to the Francoeur project boundary that is expected to be tested during the third quarter of 2021.

Yamana's recent acquisition of the Francoeur, Arntfield, and Lac Fortune gold deposits, which are located just six kilometres from the planned Wasamac milling facilities, represents additional potential exploration upside. Mineralization on the Francoeur property as well as mineralization exposed in recent trenching at Arntfield by the property's previous owner consists of mylotinized, albite-carbonate altered rocks with pyrite mineralization very similar to Wasamac. This shear can be traced a further six kilometres from the Wasamac-Francoeur property boundary to the west of the historic Francoeur mine.

Wasamac, Francoeur, and Arntfield have recorded past production of over 720,000 ounces of gold (see Section 6.6). Exploration on the past producing Francoeur gold deposit will prioritize confirmation and expansion of the historical mineral resources as well as testing high priority targets along the Arntfield-Francoeur segment of the western Wasa Shear.

In addition, there are several parallel shear zones at Francoeur with known mineralization located south of Francoeur, including Lac Fortune, and an interpreted southern splay of the Wasa Shear in the Arntfield area that are targets for drilling and potential mineral resource expansion.

Figure 9-1: Wasamac Infill and Depth Extent Exploration Potential.

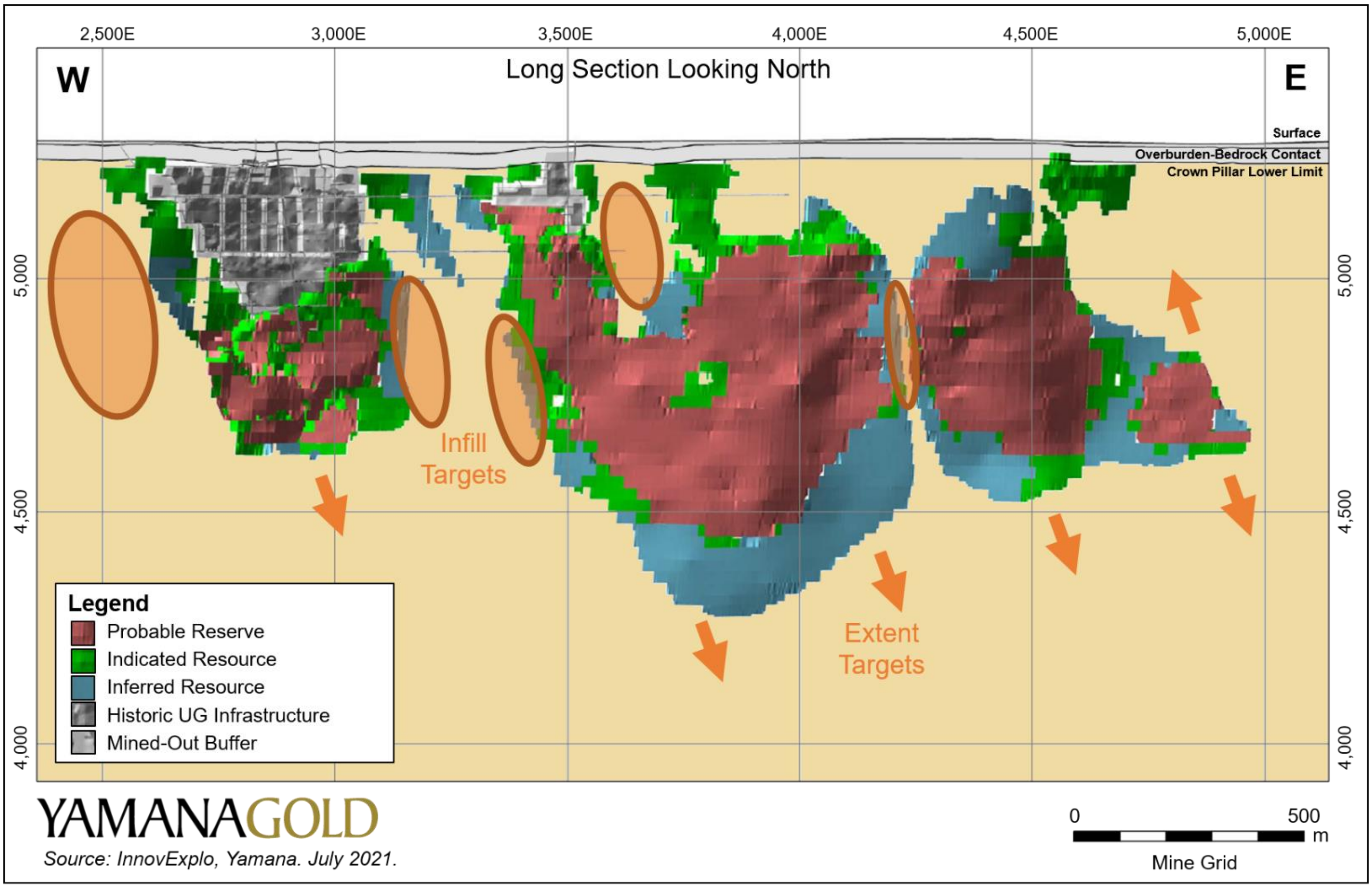
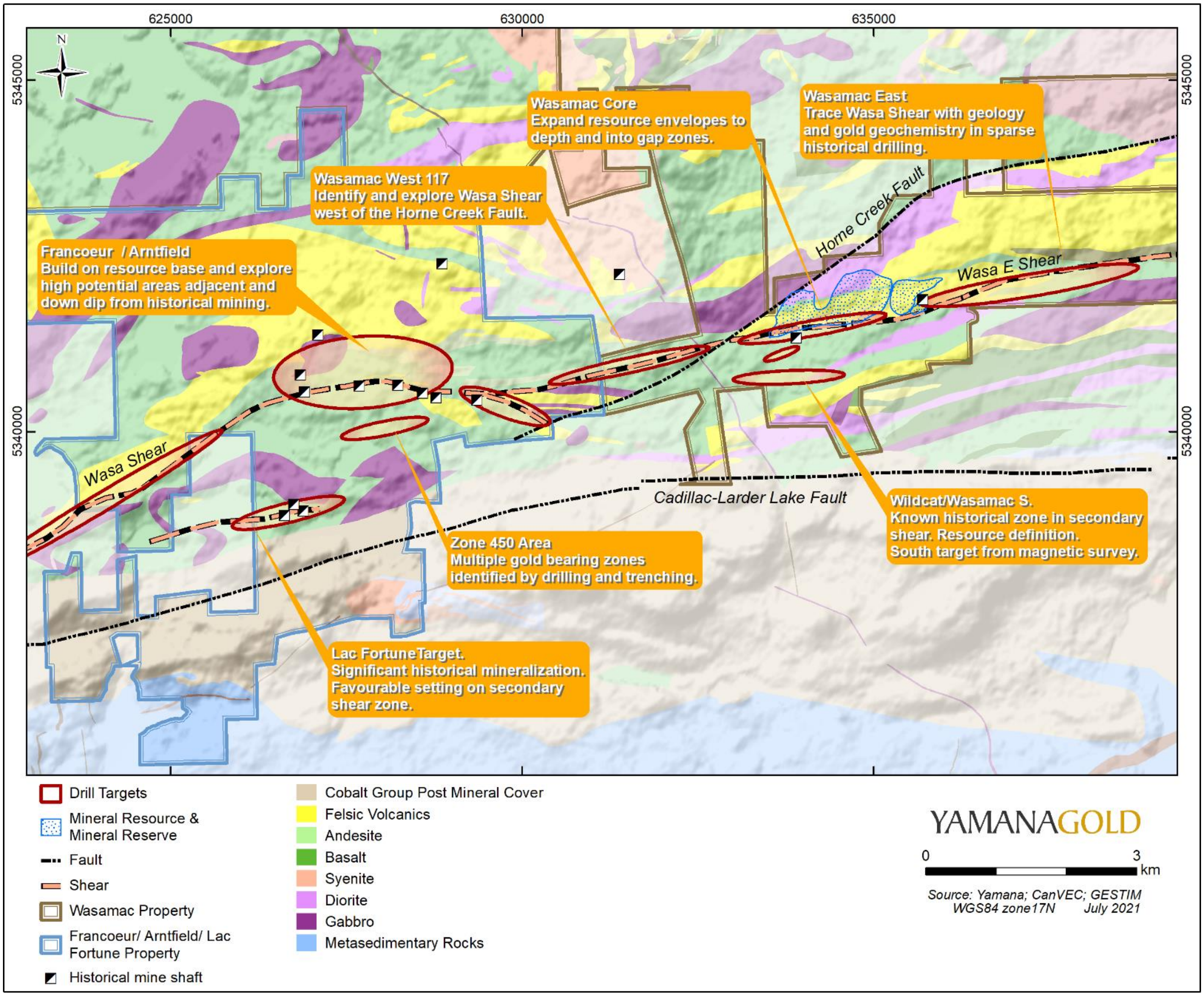


Figure 9-2: Wasamac Property Exploration Potential



10 DRILLING

The issuer has not completed any drilling subsequent to acquiring the property. This section described drilling completed in the vicinity of the Wasamac deposit since 1980. Refer to Item 6 (History) for other historical drilling programs and/or for drilling completed on other claims blocks of the property.

Since the discovery of gold in 1936, at least 14 exploration/mining companies have drilled in the Wasamac mineral resources and mineral reserves area. No drill holes have been drilled since 2016.The drill hole database contains data from 3,317 drill holes in the vicinity of the Wasamac deposit. Of these, 804 drill holes were used for the mineral resource and mineral reserve estimates described in Item 14 on mineral resource estimates.

10.1 Drilling by Previous Operators

Surface drilling completed in 1980 and 1981 by Exploration Long Lac was of BQ core size. All the drill holes were surveyed at that time (Bugnon, 1982).

Richmont drilled eight surface drill holes in 2002 totalling just over 4,500 m. The main geological target was the Wasa shear zone at depth (Zones 1 and 2). Fifteen drill holes were completed in 2003 for 9,475 m of surface drilling targeting the Wasa shear zone at depth (Zones 2 and 3). An additional 3,859 m of drilling was completed in 2004 targeting Zones 2 and 3 of the Wasa shear zone. Richmont recovered BQ size drill core from 2002 to 2005 and generally spaced drill holes at 110 m intervals. Reflex EZ-shot orientation measurements were taken at 30 m intervals.

In 2010, Richmont complete a 10,000 m surface drilling campaign with Bradley Bros. Limited with the goal of reassessing resources using a lower cut-off grade in order to evaluate the potential for an underground bulk mining operation. Twenty-nine drill holes, including two wedges, recovering NQ size drill core were drilled in the Wasa shear zone from May to December 2010, for a total of 19,853 m.

Drilling sites were located mainly on private land with access agreements signed with landowners. Access to drilling sites was via dirt roads. Drilling sites and access were restored after the drilling.

Drilling by Richmont continued in 2011 to verify the extension of the mineralization below the Main Zone, explore between the Main Zone and Zone 1, and delineate and verify the continuity mineralization in Zones 1, 2, and 3. Drilling was planned using a drill hole spacing of approximately 75 m on a vertical longitudinal section. Three to four drill rigs recovering NQ size drill core were used. Seventy-eight drill holes were completed for the 2011 drilling campaign (11 drill holes were stopped and re-drilled due to excessive deviation) totalling approximately 52,000 m.

The 2011 drilling campaign confirmed the mineralization at the bottom of the Main Zone with a large thickness in the western portion, where gold mineralization is also found in the footwall of the shear zone. It also widened Zone 2 at depth to the west and demonstrated its junction with Zone 1, and better delineated Zone 3.

In 2012, Richmont further completed 87 NQ size core drill holes. Most of these drill holes were drilled on private land, and agreements were reached with the respective owners. The 2012 drilling campaign continued the definition of the Main Zone and better defined Zone 3.

More extensive metallurgical tests were initiated on composite samples of drill core for the four main mineralized zones of the property. Over twenty geotechnical holes were also drilled to verify the quality of the rock in the crown pillar and the hanging wall of the Main Zone and Zone 1. A large diameter borehole was also drilled to intersect existing development of the former Wasamac Mine and serve as a dewatering well.

In 2012, 16 drill holes for a total 11,803 m were drilled by Richmont to test the Francoeur-Wasa shear zone between the vertical depths of 200 m and 1,000 m across claims optioned from Globex and now part of Yamana property. All drill holes intersected the Francoeur Wasa shear zone; however, where intersected, the structure appeared to be less highly deformed and not as highly altered as elsewhere.

Drilling between 2011 and 2012 was contracted to local drilling companies Forage Hébert Inc., Foramex (4294858 CANADA INC.), and Forages M. Rouillier Inc.

Richmont drilled two drill holes totalling 600 m targeting the east and west extensions of the Wildcat Zone in 2015-2016.

All drill holes drilled by Richmont were surveyed at surface using various differential GPS or hand-held GPS systems by Richmont personnel or surveying contractors such as Corriveau J.L. & Associés Inc. Down-hole deviation was measured using a Flexit SmartTool or a Reflex instrument at 30 m intervals, corrected to the magnetic deviation. Magnetism was measured at the same time and some readings were discarded due to high values. For some drill holes, a multi-shot reading was taken at the end of the drill hole.

The core recovery for drilling conducted by Richmont was measured to be 95% to 100%. Some fault material was intersected in the most brecciated parts of the Wasa shear zone diminishing core recovery.

The QPs have not identified drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results.

10.2 Planned Drilling by Yamana

An ongoing drilling program by Yamana is in progress at the time of writing. No final results are available for the mineral resources update presented in this report. The issuer has not completed any other drilling since acquiring the property.

Drilling is currently being conducted for further rock mechanic studies and exploration purposes, and other drilling campaigns are at the planning and/or permitting stage. The results from the ongoing and proposed campaigns will dictate future approaches. Ongoing and preliminary plans are in line with recommendations of Chapter 26 and include 120,000 metres of drilling in 2021 and 2022. The main objective of future drilling is infill drilling to better delineate areas expected to be developed in the first three years of production is expected to include 30,000 metres in 2021, with a further 38,000 metres in 2022 to provide further delineation of the remaining mineral resources. This work is expected to increase confidence in grade, improve mine planning, and provide further geotechnical and metallurgical data.

A concurrent exploration effort will focus on expanding the current mineral resource envelopes to depths below the established mineral resource and with testing for mineralization targeting the poorly explored gaps between zones. Exploration on the broader property is expected to include 10,000 metres in 2021 with an effort to delineate secondary zones, such as Wildcat, and test high-priority extensions of the Francoeur-Wasa shear zone.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Core Logging and Sampling Procedures

Limited data is available from work prior to 2002 regarding sampling methods used underground, either for core drilling or wall sampling, or applied to surface drilling.

From 2002 to 2016, core logging was performed by Richmont geological staff using industry-standard procedures. And between 2010 and 2016, core logging was completed on Richmont's Francoeur mine site, where access was controlled by guards at the entrance.

The following data was described and entered into the logging software by Richmont geological staff:

- log header, drill hole location, parameters, and surveys
- descriptions of the main and sub-geological units
- mineralized zones with their mineralogy, attitude, thickness
- structures, alterations, and RQD

Selected core intervals were sawn in half with one half being kept as a reference in core boxes, the other half bagged, labelled, and transported to Laboratoire Expert Inc. in Rouyn-Noranda by Richmont for assay. Assay results and core descriptions were collected and plotted onto vertical sections for interpretation and drill hole planning.

The core boxes were marked with aluminum tags and moved to permanent storage in steel racks on the Francoeur mine site. Since 2003, most of the split core has been stored at the Francoeur mine and remains available. Since 2009, sample rejects and pulps remain stored at the Francoeur mine site.

11.2 Density Determinations

The historic tonnage factor used for the Wasamac mine ore was 12 ft^3/ton, which corresponds to a density of 2.80 g/cm^3.

In May 2010, Richmont requested that Unité de Recherche et de Service en Technologie Minérale (URSTM) make density measurements on samples from Zone 2. The collected samples (from drill holes WS-10-31 and WS-10-36) were sent for metallurgical testing (Lelièvre, 2011). The average density of the 21 samples was 2.823 g/cm^3.

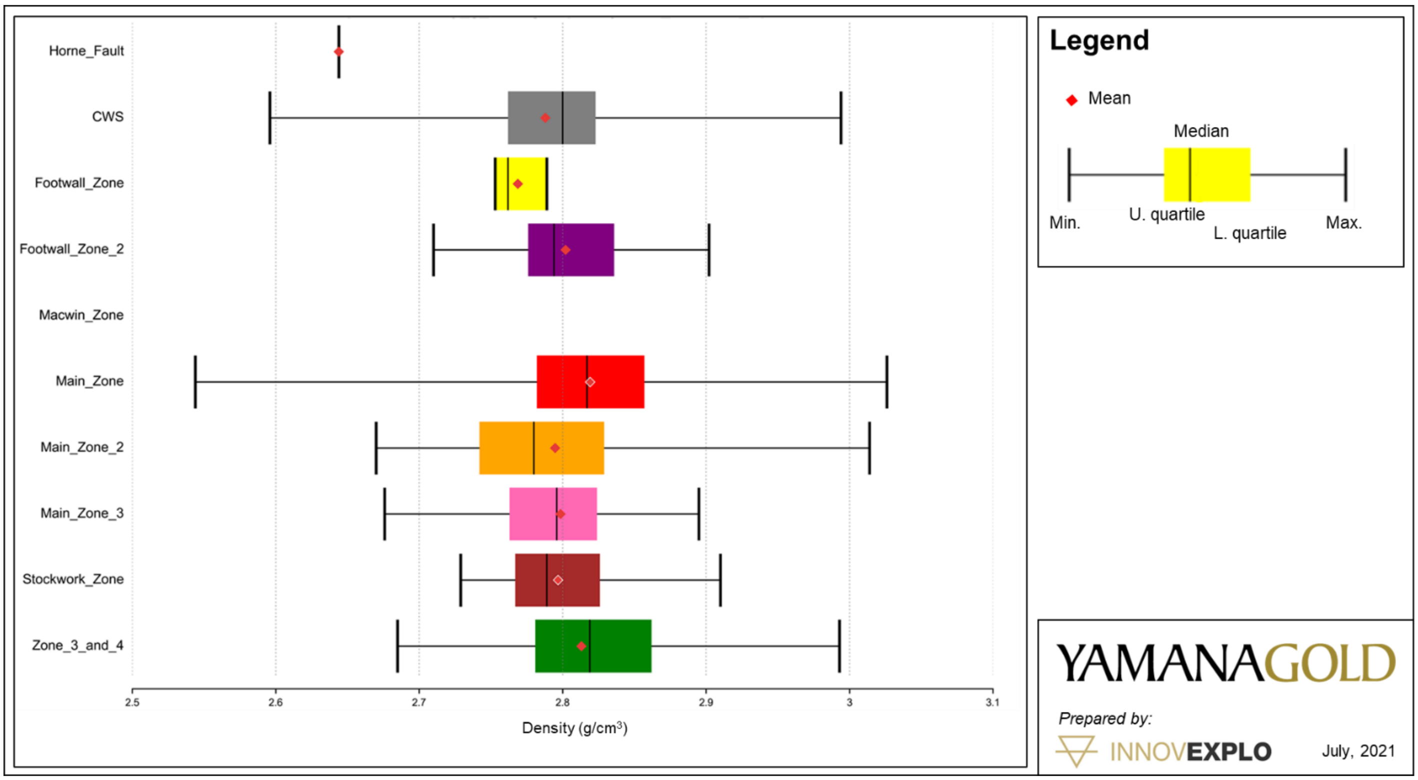
In 2011, approximately 40 new density measurements were made by URSTM. Results averaged 2.80 g/cm^3 for the Main Zone; 2.83 g/cm^3 for Zone 1, and 2.84 g/cm^3 for Zone 2.

In May 2021, densities, determined by standard water immersion methods, were calculated on 586 samples from previously drilled core. During the measurement of the density of those samples, a protocol where every five samples, a standard (either 50 g weight, 100 g weight or a barren rock) was inserted into the sample stream to monitor possible instrument drift.

These 586 samples were collected from the vicinity of the gold domains. The results are presented in tabular form in Table 14-7 and as a box plot in Figure 11-1. The mean of these measurements is 2.80 g/cm^3.

The density value of 2.80 g/cm^3 is considered representative and that value was used to convert volumes to tonnages for the mineral resource and mineral reserve estimates.

Figure 11-1: Box Plot Presenting the Results of the Density Measurements taken in 2021 on 586 Samples



11.3 Sample Preparation and Analysis

Details regarding the analytical procedures used by laboratories prior to 2002 could not be recovered by the authors.

11.3.1 Techni-Lab

During the 2002-2004 surface drilling programs, diamond drill core was logged and split at Richmont's core logging facility. Samples from one half of the core were tagged and bagged and delivered directly to Techni-Lab in Sainte-Germaine-Boulé, Québec. At Techni-Lab, samples were counted and classified. A project list was created, and the sample identification numbers were compared with the order form provided by Richmont. Each sample was allocated two identification tags, one for the pulp and the other for the reject. Wet samples were dried in an oven at 60°C for one hour. Samples were crushed to -2 mm, homogenized, and split with a Jones riffle splitter to retain a 250 g sample, which was then pulverized to 80% passing -200 mesh for three minutes in a ring pulverizer. Pulps were analyzed using standard fire assay methods with an atomic absorption spectrometry finish (FA AAS). If the results were greater than 10 g/t Au, a second 30 g pulp sample was fire assayed with a gravimetric finish (FA Grav).

Techni-Lab was a commercial and independant laboratory (since 2010 is part of the Actlabs group). Techni-Lab employed industry standard quality assurance / quality control (QA/QC) procedures, including daily checks of equipment and the insertion and review of blanks, duplicates, and certified reference materials. For the period from 2002 to 2004, the QPs were not able to verify if Techni-Lab had an any accreditation or certification for testing and calibration laboratories.

11.3.2 Laboratoire Expert Inc.

For the period from 2010 to 2012, 2015 and 2016, samples were sent to Laboratoire Expert Inc. in Rouyn-Noranda. Around 20% of pulps and rejects from the mineralized zones were sent to Techni-Lab in Sainte-Germaine-Boulé, Québec, for the 2010 and 2011 campaigns, and to Accurassay in Thunder Bay, Ontario for the 2012, 2015 and 2016 drilling campaigns, and were analyzed by both FA AAS and FA Grav (30 g).

At Laboratoire Expert Inc., samples were dried if necessary and then reduced to -1/4 inch with a jaw crusher, which was cleaned with compressed air and barren material between samples. The sample was then reduced to 90% -10 mesh with a roll crusher. The roll crusher was cleaned between samples with a wire brush and compressed air and barren material between sample batches. The sample was then split using a Jones type riffle to approximately 300 g, which was then pulverized to 90% -200 mesh in a ring and puck type pulverizer. The pulverizer was cleaned with compressed air between samples and with silica sand between batches.

The samples were analyzed by FA AAS (30 g). Samples greater than 1 g/t Au were systematically re-analyzed in the same laboratory by FA Grav (30 g). The samples with a gold content greater than 10 g/t were analyzed in duplicate using the same method. Both results exist in the database, but the gravimetric result is used in the final gold value column (AUFIN_PPM). When two gravimetric results are available, the average is used.

Samples were assayed in batches of 28, which included a reagent blank and a gold certified reference material.

Laboratoire Expert Inc. was a commercial and independant laboratory. For the period from 2010 to 2012, 2015 and 2016, the QPs were not able to verify if Laboratoire Expert had an any accreditation or certification for testing and calibration laboratories.

Accurassay. was a commercial and independant laboratory. For the campaigns of 2012, 2015 and 2016, Accurassay's Laboratory was ISO 17025 certified.

11.4 Quality Assurance and Quality Control Procedures

There are no records of a quality control program for the period when the mine was in operation from 1965 to 1971.

Exploration completed by Exploration Long Lac in the 1980s included checks for assay results from the 1980-1981 drilling campaign. Two laboratories were used for this campaign: Laboratoire d'analyse Bourlamaque Ltée of Val-d'Or, QC (Bourlamaque), and Assayers Limited of Rouyn-Noranda, QC (Assayers). Some of the samples sent to one laboratory were re-sent to the other. Only the pulp was used, and samples were separated into two groups: one for the WSZ, the other for the Wildcat Zone. Pulps prepared by Bourlamaque and assayed by the two laboratories showed a very good correlation. Pulps prepared by Assayers and assayed by the two laboratories showed a lower correlation (Bugnon, 1982).

For the 2002 to 2004 drill campaigns, the QA/QC program for drill core samples included the re-assay of all samples with good initial results by a second laboratory facility. ALS Chemex of Val-d'Or, QC, an independent and ISO/IEC accredited analytical laboratory, re-assayed those pulps. ALS Chemex conducted the second FA AAS on a 30 g sample. If the second assay returned a value greater than 7.0 g/t Au, an FA Grav was performed on a second 30 g sample obtained from the same pulp.

Assay values obtained from the two distinct laboratory facilities were compared. Differences exist when comparing individual sample results; however, overall, results were similar. It must be noted, however, that Techni-Lab yielded slightly lower assay results than ALS Chemex.

For the drill campaigns from 2010 to 2012, 2015 and 2016, the QA/QC program consisted of inserting certified reference materials and blank samples into the sample stream every 20 samples, and performing check assays at a secondary umpire laboratory to analyze pulps and/or coarse rejects. The QA/QC results of these drilling campaigns are summarized in Section 11.5.

11.5 Results of the QA/QC Programs from 2010 to 2012, 2015 and 2016

During the period considered, 25,373 samples from core drill holes were sent to Laboratoire Expert Inc. in Rouyn-Noranda, Québec. The samples were from drill holes over the entire deposit.

11.5.1 Certified Reference Materials

To control accuracy in the laboratory, certified reference materials purchased from ROCKLABS® (Rocklabs) was inserted into the sample streams along with original samples. Richmont's accuracy control methodology (Rocklabs' recommended approach) used the reference value of the standard and the standard deviation value of the data obtained by the laboratory to set the limits of ±2 standard deviation (SD) and ±3SD. It is important to note that this differs from Yamana's methodology, which uses the expected value and standard deviation values indicated in the certified reference materials certificate (see Table 11-1).

For all certified reference materials, the results by FA Grav are slightly higher compared to those obtained by FA AAS; however, the FA Grav results are more in-line with the expected results as defined by Rocklabs.

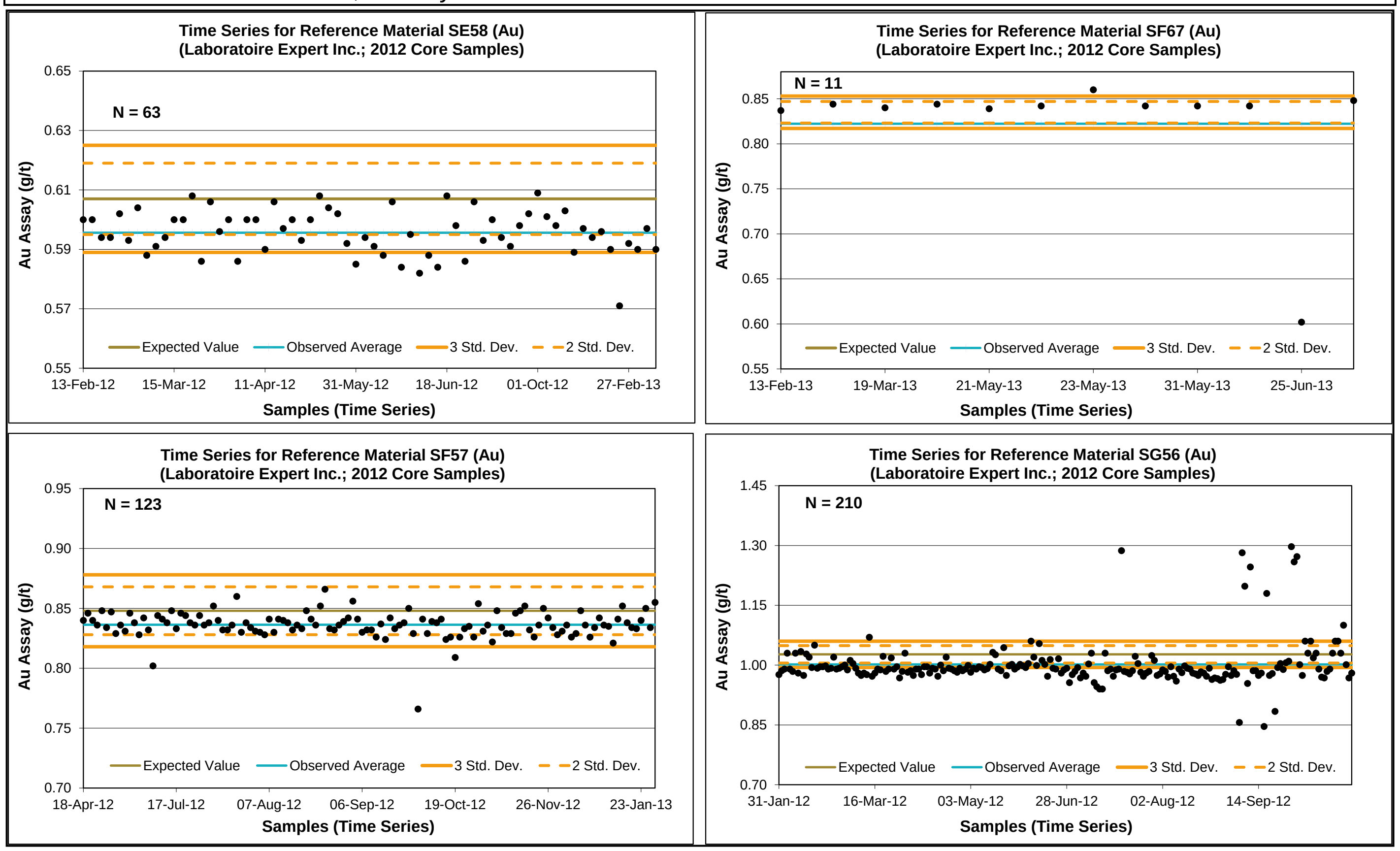
Table 11-1: Certified Reference Materials and their Expected Value and Standard Deviation Used on the Wasamac Project by Year

CRM	Source	Expected Value (Au g/t)	Std. Dev. From Source	Year			
				2010	2011	2012	2015-2016
SI42	Rocklabs	1.761	0.021	x	x		
SL46	Rocklabs	5.867	0.066	x	x		
SP37	Rocklabs	18.14	0.15	x	x		
SI54	Rocklabs	1.78	0.011		x		
SE58	Rocklabs	0.607	0.006		x	x	
SG56	Rocklabs	1.027	0.011		x	x	
SH55	Rocklabs	1.375	0.014		x	x	
SL61	Rocklabs	5.931	0.057		x	x	
SF57	Rocklabs	0.848	0.01			x	
SF67	Rocklabs	0.853	0.006			x	
SH65	Rocklabs	1.348	0.009			x	
OREAS 60c	Ore Research	2.47	0.08				x

As an example, Figure 11-2 shows the control charts and the results of certified reference materials SE58, SF67, SF57 and SG56 used in 2012.

Figure 11-2: Results of Certified Reference Materials SE58, SF67, SF57 and SG56 used in 2012

Project	Wasamac	Statistics	SE58	SF67	SF57	SG56
Data Series	2012 Standards	Sample Count	63	11	123	210
Data Type	Core Samples	Expected Value	0.607	0.835	0.848	1.027
Commodity	Au (g/t)	Standard Deviation	0.006	0.006	0.010	0.011
Laboratory	Laboratoire Expert Inc.	Observed Average	0.596	0.822	0.836	1.002
Analytical Method	FA. AAS (<1 g/t) Grav (>1g/t)	Mean Bias %	-1.9%	-1.5%	-1.4%	-2.5%
Detection Limit	0.005 g/t Au	Failures (> 3 Std. Dev.)	17.5%	18.2%	2.4%	66.2%
Source	Yamana, January 2021.					



11.5.1.1 2010

In 2010, three certified reference materials (from Rocklabs) were used for control of accuracy at Laboratoire Expert Inc. for a total of 149 standards analysed. All three standards showed acceptable performance for gold gravimetric results.

The standard SI42 failed five times out of 68 (above 3SD), which corresponds to a 7.4% failure rate. SL46 had no failures and SP37 had 14 failures (below 3SD), which corresponds to a 7.1% failure rate. The mean bias percentage is considered low at ±0.5%, which is considered a good result.

The percentage of samples between inside 2SD was around 66.2% for certified reference materials SI42, 92.5% for SL46 and 92.9% for SP37. Despite sub-optimal performance of SI42, the 2010 certified reference materials results were acceptable.

11.5.1.2 2011

Eight certified reference materials were used during the 2011 campaign; 378 standards were sent to Laboratoire Expert Inc. The performance of certified reference materials SH55 was poor. Use of this certified reference material was subsequently discontinued by Richmont due to the anomalous values not consistent with expected ranges early in the drilling program. Dispersion of the results, however, was observed at the beginning of sampling but was followed by better results later, suggesting a potential laboratory bias. For certified reference material SE58, 66.7% of the results are within ±2SD, 75% for SG56, 21.7% for SH55, 74.2% for Si42, 23.3% for SI54, 56.5% for SL 46, 93.9% for SL61 and 100% for SP37. These results are below expected ranges; however, except for certified reference material SH55, the mean bias was below ±1.5%, which is considered acceptable.

11.5.1.3 2012

Seven different certified reference materials were used during the 2012 drilling campaign. Three were used to assess the FA AAS method and four to assess the FA Grav method. Some 734 standards were sent to Laboratoire Expert Inc.

The certified reference materials SF67 and SH65 showed a positive bias (1.1% and 0.6% respectively), whereas the other five certified reference materials showed a negative bias, (between 0 and -2.5%), which is considered acceptable as it cannot be associated with the analytical method.

The performance of the certified reference materials during 2012 is below expected limits due to the large number of failures (outside 3SD); however, the bias is within the acceptable range (overall below 2%).

11.5.1.4 2015 and 2016

Only one drill hole was completed during each of the 2015 and 2016 programs. A certified reference materials standard was inserted every 20 samples. Ten samples of certified reference materials OREAS 60C were incorporated into the 2015-2016 sample streams sent to Laboratoire Expert Inc. and analyzed for gold and silver.

Despite a low number of results to the limited drilling program, the results were good for monitored precious metals, gold and silver. No failures occurred for either analyte. The overall mean bias was between ±1.5%, which is considered acceptable.

11.5.2 Blanks

According to the information recovered by Yamana, core taken from previous drilling campaigns and expected to be barren in gold were used as blanks; these were inserted every 20 samples. The source of the barren core is uncertain, and it is unclear what type of testing, if any, was done to verify its gold content.

For the information between 2010 and 2012, it is assumed that the detection limit for the FA AAS method is 0.005 g/t Au. The 2015 and 2016 laboratory certificates specified that the detection limit using FA AAS is 0.005 g/t for gold and 0.02 g/t for silver.

In 2010, 40 blanks out of 151 returned values above the threshold of 10 times the detection limit which corresponds to a 26.5% failure rate. In 2011, 39 blanks out of 379 sent to the laboratory returned values above 10 times the detection limit corresponding to a 10.3% failure rate. In 2012, 26 of 857 blanks returned values above 10 times the detection limit corresponding to a 3% failure rate. An improvement over the years was observed but overall performance remained poor during the 2010, 2011 and 2012 campaign. The cause of the frequent failures may be contamination during the sample preparation process but also the potential existence of gold in the blank material.

In the limited drilling between 2015 and 2016, no blank sample returned values above the detection limit for gold and silver.

As an example, Figure 11-3 on the following page shows the control charts for contamination (the results of blank samples) between 2010 and 2012.

11.5.3 Check Assays (Umpire Lab)

11.5.3.1 Pulp

For the 2010 drilling campaign, 200 pulp samples from Laboratoire Expert were sent to Techni-Lab for umpire check assaying. The average for check samples was 3.49 g/t Au at Techni-Lab, whereas the average of the original assays at Laboratoire Expert was 3.41 g/t Au. For values below 1 g/t Au, the relative percent difference (RPD%) values are observed with a negative trend; the check values are a little higher than the originals. This correlates well with the lower values observed at Laboratoire Expert when using the FA AAS method, which they did for gold values below 1 g/t Au. For values greater than 1 g/t Au, it is observed that the distribution of the pairs is asymmetric with respect to the 0 RPD% axis with some isolated outliers. No bias is observed for samples with gold contents greater than 1 g/t. For the check assay pairs above 0.05 g/t Au, 83.2% had an absolute relative difference within ±20%.

For the 2011 drilling campaign, 221 pulp samples were sent to Techni-Lab laboratory, which acted as the umpire laboratory. In the range of gold values less than 1 g/t, the 2011 samples analyzed at Laboratoire Expert, similarly to what was observed in 2010, showed a negative bias. No bias is observed for check assay pairs above 1 g/t Au. The average for original samples was 3.43 g/t and 3.55 g/t for the check assay pair. Of the check assay pairs greater than 0.05 g/t Au, 85.2% of them had an absolute relative difference within ±20%.

In 2012, 51 pulps were sent to Accurassay for check assaying. Samples with a gold grade above 1 g/t were selected for verification. The mean of the original samples was 3.97 g/t Au and 3.65 g/t Au for the check assay pulps. Of the check assay pairs above 0.05 g/t Au, 51.0% of them had an absolute relative difference within ±20%.

For the campaign of 2015 and 2016, pulps were not sent to a secondary laboratory for umpire check assaying.

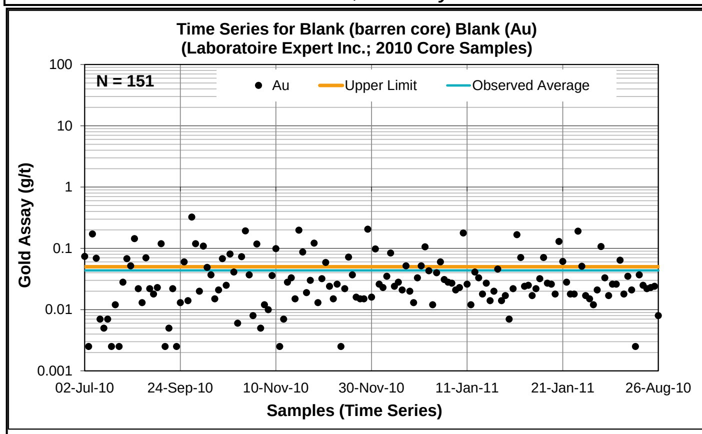
As an example, Figure 11-4 shows the control charts and results of umpire check assaying, for pulp, completed 2012.

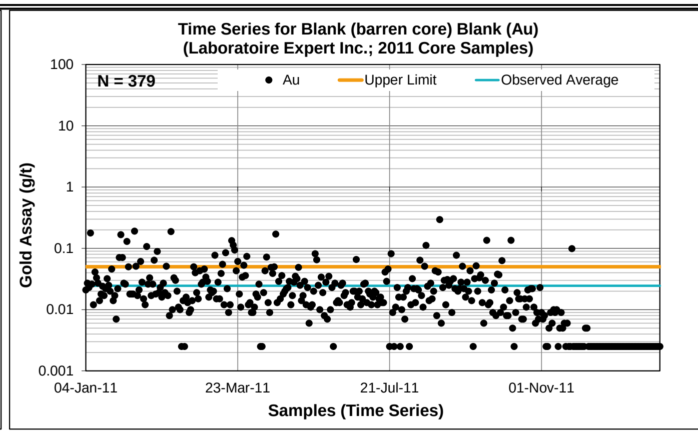
Figure 11-3: Control Charts for Contamination between 2010 and 2012

Project	Wasamac
Data Series	2010-2012 Blanks
Data Type	Core Samples
Commodity	Au
Laboratory	FA. AAS (<1 g/t) Grav (>1g/t)
Analytical Method	Fire Assay - AAS/Grav.
Detection Limit	0.005 g/t Au
Source	Yamana, January 2021.

Control Samples	Blank (barren core)		
Statistics	2010	2011*	2012
Sample Count	151	379	857
Expected Value	0.005	0.005	0.005
Standard Deviation	0.049	0.032	0.106
Observed Average	0.044	0.025	0.014
Upper Limit (10xDL)	26.5%	10.3%	3.0%

** 13 data were removed from the graph and statistics due to the mislabeling of blanks with CRMs.*





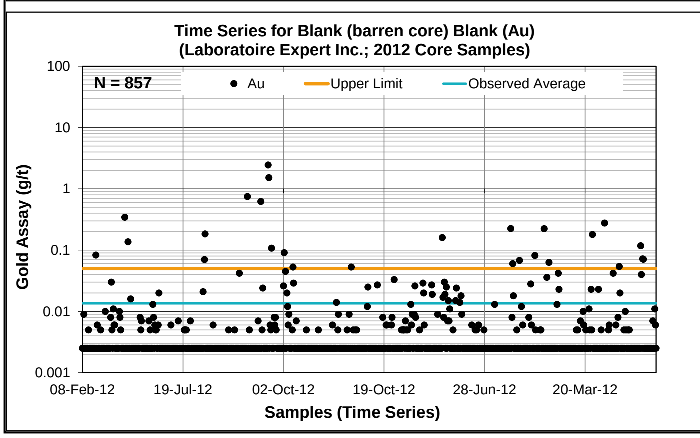
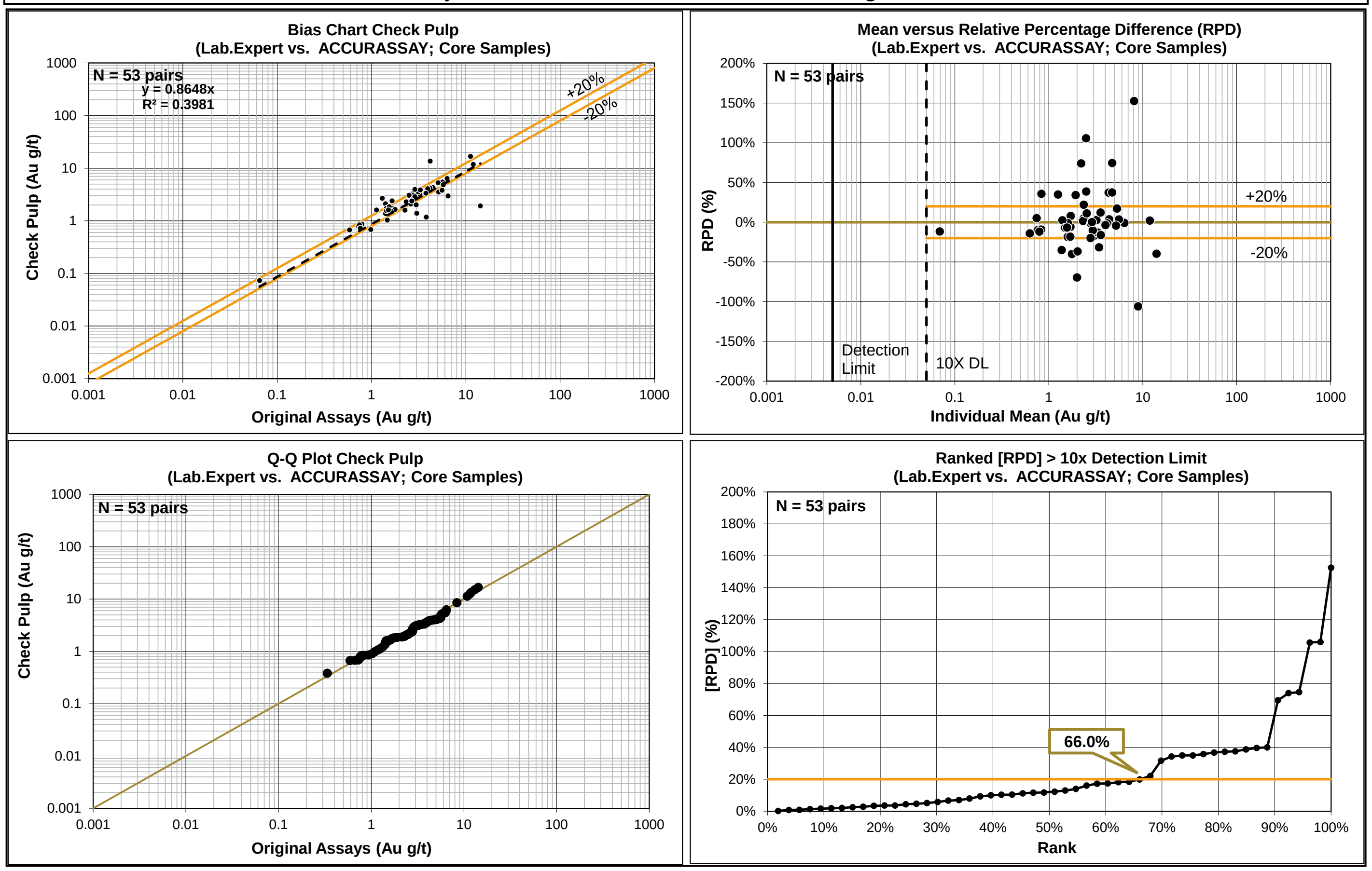


Figure 11-4: Results of Umpire Check Assaying for Pulp in 2012

Project	Wasamac	Statistics	Original	Check
Data Series	2012 Check pulp	Sample Count	53	53
Data Type	Core Samples	Average	3.27	3.17
Commodity	Au in g/t	Minimum Value	0.065	0.073
Laboratory	Expert vs. Accurassay	Maximum Value	14.26	16.75
Analytical Method	Fire Assay - AAS/Grav.	Samples > 10DL	53	
Detection Limit	0.005 g/t Au	Pairs +/- 20% RPD if > 10DL	66.0%	
Original Dataset	Original Assays	Precision = 2xCV		
Paired Dataset	Check Pulp	Precision 10DL to 1 g/t Au	23.7%	
		Precision 1 to 5 g/t Au	49.9%	
Source	Yamana, January 2021.	Precision > 5 g/t Au	47.8%	



11.5.3.2 Coarse Rejects

During the 2010 drilling campaign no coarse reject material was sent for check assaying. In 2011, 227 coarse reject samples were sent to Techni-Lab for preparation and check assay analysis. The average was 3.52 g/t Au for the Laboratoire Expert original samples and 3.47 for the check assay rejects. Of the check assay pairs greater than 0.05 g/t Au, 68.9% of them had an absolute relative difference within ±20%. These results are within expected limits for coarse reject check assays.

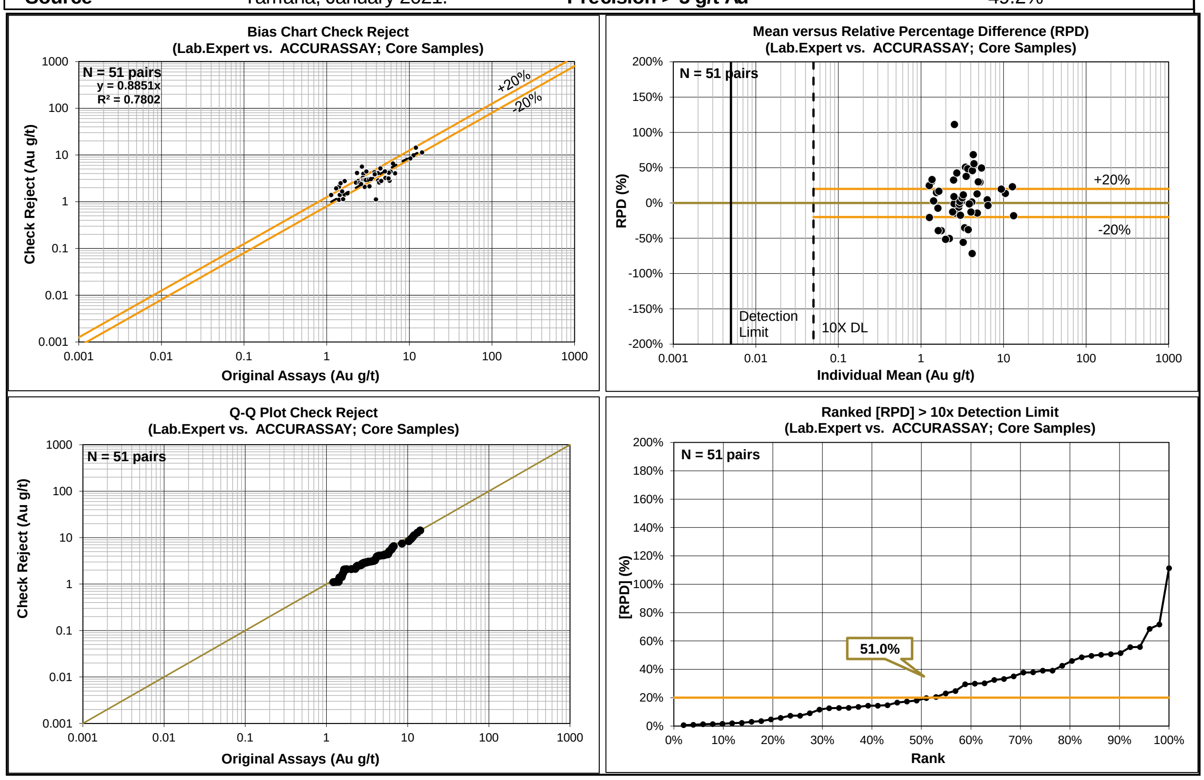
In 2012, 53 coarse rejects were sent to Accurassay for check assay testing. Samples with gold values above 1 g/t were selected. The means of the original samples and the coarse reject check assays are very close, 3.23 and 3.19 g/t Au respectively. Of the check assay pairs greater than 0.05 g/t Au, 66.0% of them had an absolute relative difference within ±20%.

For the campaign of 2015 and 2016, coarse rejects were not sent to a secondary laboratory for umpire check assaying.

As an example, Figure 11-5 shows the control charts and results of umpire check assaying, for coarse rejects, completed 2012.

Figure 11-5: Results of Umpire Check Assaying for Coarse Rejects in 2012

Project	Wasamac	Statistics	Original	Check
Data Series	2012 Check Reject	Sample Count	51	51
Data Type	Core Samples	Average	3.97	3.65
Commodity	Au in g/t	Minimum Value	1.13	1.10
Laboratory	Expert vs. Accurassay	Maximum Value	14.26	14.38
Analytical Method	Fire Assay - AAS/Grav.	Samples > 10DL	51	
Detection Limit	0.005 g/t Au	Pairs +/- 20% RPD if > 10DL	51.0%	
Original Dataset	Original Assays	Precision = 2xCV		
Paired Dataset	Check Reject	Precision 10DL to 2 g/t Au	44.4%	
		Precision 2 to 5 g/t Au	51.0%	
Source	Yamana, January 2021.	Precision > 5 g/t Au	49.2%	



11.6 Comments on Sample Preparation, Analyses and Security

The QA/QC program implemented for the Wasamac Project was adequate. The insertion frequency for the control samples meets industry norms.

Overall, the accuracy of the primary laboratory for the sampling between 2010 and 2016 is considered below expectations with many failures occurring outside three standard deviations; however, the bias was kept under control (i.e., below 2%), indicating that the impact of these failures is minimal.

Blank samples made of barren core (source and methodology for selection is undocumented) were inserted into the sample stream to be assayed along with authentic core samples in order to monitor laboratory contamination. A high incidence of failures due to contamination was observed in 2010 (around 26%); the failure rate decreased over the 2011 and 2012 campaigns. The cause of the errors remains unclear; either core used for blanks was not completely barren, cleaning during sample preparation was poor, or errors in insertion occurred. During 2015 and 2016 no contamination failures were recorded.

For control of precision and detection of bias at the primary laboratory, pulps and coarse rejects were sent for umpire check assaying at secondary laboratories. During 2010 and 2011, pulps and reject were sent to Techni-Lab and in 2012, and pulps and coarse rejects were sent to Accurassay. No significant assay bias was observed.

It is the opinion of the QPs that the procedures followed conform to the industry practices and that the quality of the assay data is adequate and acceptable to support a mineral resource and mineral reserve estimation.

12 DATA VERIFICATION

This chapter covers the data verification of the Wasamac Project's DDH database, used for the in the estimation of mineral resources and mineral reserves (effective as at June 30, 2021). Data verification, by InnovExplo's QPs, included a visit to the project site, as well as an independent review of the data for selected drill holes (assays, QA/QC program, downhole surveys and collar surveys).

InnovExplo's QPs also reviewed and validated the mineral resource and mineral reserve estimation process adhered to by Yamana, including all parameters, block model construction, the script that runs the model, volumetric report, and the validation process.

12.1 Wasamac Project Database

The database contains data from 3,317 DDH drilled for various purposes on the property. Of these, 804 drill holes were drilled in the block model area and therefore used for the mineral resource and mineral reserve estimates. It is important to note that since the discovery of gold in 1936, at least 14 exploration/mining companies have drilled on the property. No drill holes have been drilled since 2015.

12.1.1 Assays

The QPs had access to the assay certificates for historical drill holes in the database dating from 2002 to 2015 that were drilled by Richmont. Certificates from drill holes completed before 2002 and after 1980 were either provided by Yamana from files in their archives (i.e., not from the laboratory), or from hardcopy logs. All assays were verified for selected drill holes (10% of the drill holes from the drilling programs completed between 1980 and 2015). The assays recorded in the database were compared to the original certificates (from Yamana's archives or from Laboratoire Expert). No major errors or discrepancies were found.

Assay data from drill holes drilled from underground could not be verified with respective hard copy logs or laboratory certificates. These drill holes were compared on plans and sections with the closest surface drill holes. Most of the underground drill holes are from inside the "mined-out buffer volumes" (see Section 14.3); and were only used to help the modelling of the research ellipse (variogram models); these volumes were not interpolated and are not part of the mineral resource or mineral reserve estimates.

The samples collected from "bar and arm drilling" completed during the development of the Wasamac mine were not used for the mineral resource and mineral reserve estimates.

The QA/QC database has been validated and is described in Section 11.2.

12.1.2 Drill Hole Collar and Downhole Surveys

Downhole surveys (mainly Multishot surveys) were conducted on the majority of Richmont's surface drill holes. These deviation test results were archived using Microsoft Excel (*.xls) files. Acid tests were completed on drill holes completed between 1980 and 2002 and results archived on hard copy logs. Deviation tests from 10% of these drill holes were compared with the downhole data recorded in the database. No major discrepancies were found.

Hole collars from 2010 to 2015 drilling were surveyed using DGPS and the surveyor's certificates were provided by Yamana. Collar positions of older drill holes were either archived on hardcopy logs or positioned by Richmont's staff using sections and plans. Some of these older drill holes were surveyed when the surface collars were discovered. Archived collar positions of 10% of the drill holes were compared with the collars coordinates found in the database. No major discrepancies were found.

12.2 Mineral Resource and Mineral Reserve Estimation Process

The mineral resource and mineral reserve model for the Wasamac Project was prepared by Yamana, as were the geological interpretation and 3D geological model. The QPs reviewed and validated the geological and estimation models, including the interpretations, selected parameters, scripts, and results of the volumetric model (refer to Chapter 14).

12.3 Site Visit

The QPs, Alain Carrier, P.Geo., Denis Gourde, P.Eng. and Vincent Nadeau-Benoit, P.Geo., along with the Yamana team, visited the property and the logging facility on May 13, 2021. The visit included a tour of the outdoor core storage facility; a review of core intersections of the main geological units and typical mineralization throughout the deposit; a visit of various mechanically stripped outcrops; and a viewing of the proposed location of the planned surface mining infrastructure and surroundings. The photos in Figure 12-1 on the following page document the site visit.

12.4 Comments

There were no limitations in the ability of the QPs to verify the data. The QPs are of the opinion that the data and estimation process for the project are acceptable. The QPs consider Yamana's Wasamac Project database to be valid and of sufficient quality to be used for the mineral resource and mineral reserve estimates in this report.

Figure 12-1: Site Visit of May 13, 2021

Notes: FWSZ= Francoeur-Wasa shear zone; RQD = rock quality designation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Summary

The objective of the metallurgical study was to quantify the metallurgical response of ores from deposits at the Wasamac project site. The program was designed to confirm the parameters for process design criteria for leaching and flotation. The metallurgical program was conducted at Base Metallurgical Laboratories Ltd. (BaseMet Labs) in Kamloops, BC as project BL0729 in February and March 2021, and was performed on composites from Main Zone West, Main Zone Central and Main Zone East (formerly Z1, Z2 and ZP ore zones).

All testwork conducted as a part of this 2021 feasibility study update was intended to validate and optimize the process design parameters as well as operational consumable rates from previous testwork programs and phases of the project.

The samples and test results used in the basis of the process design criteria have been validated to be within the current mine plan and are representative of the variability of the deposit.

13.2 Historical Testwork

Testwork reports previously completed for other studies on the Wasamac Project include the following, as seen in Table 13-1.

Table 13-1: Historical Testwork Programs

Document Name	Issuer	Date	Description
BL0348	BaseMet Labs	November 5, 2018	Metallurgical testing of the Wasamac Project, including comminution testing, chemical composition, mineralogical assessment, leach performance, tailings diagnostic leach, cyanide detox testing, and oxygen uptake testing
Project 13433-001 Report #1	SGS Minerals Services	April 5, 2012	The recovery of gold and silver from the Wasamac deposit
Project 13433-001 Report #2	SGS Minerals Services	January 12, 2012	Gold deportment study for two samples from the Wasamac Gold Project
Project 13433-001 Report #3	SGS Minerals Services	April 17, 2012	A gold deportment study on one sample from the Wasamac Gold Project (Z2)
Project 13433-001 Report #4	SGS Minerals Services	August 19, 2013	The variability of gold and silver recovery from the Wasamac deposit
Project 13433-001 Report #5	SGS Minerals Services	January 2, 2013	A gold deportment study on one sample from the Wasamac Gold Project (Z3)
Solids-Liquid Separation Testing Report	Pocock Industrial Inc.	September 2018	Solids-liquid separation testing report

Summaries of the historic testwork listed above can be found in the following technical reports on the Wasamac project:

- N.I. 43-101 Technical Report on the Feasibiity Study of the Wasamac Project for Monarch Gold Corp. by BBA Inc., December 1, 2018.
- Preliminary Economic Assessment for the Wasamac Project by BBA Inc. for Richmont Mines Inc., May 4, 2012.

Information used from these reports is summarized in the following subsections where applicable.

13.3 Updated Feasibility Study Testwork

13.3.1 Sample Description

Based on a preliminary review of historical results, Ausenco designed the latest testwork program with the following objectives:

- based on optimal conditions from the 2018 program, evaluate testing conditions for typical telluride treatment (higher pH conditions, higher dissolved oxygen, and longer cyanidation time), after removal of free gold prior to leaching with gravity concentration
- conduct a series of grind-recovery tests to confirm 2018 grind selection
- re-evaluate flotation with concentrate (after re-grind) and tailings leaching flowsheet

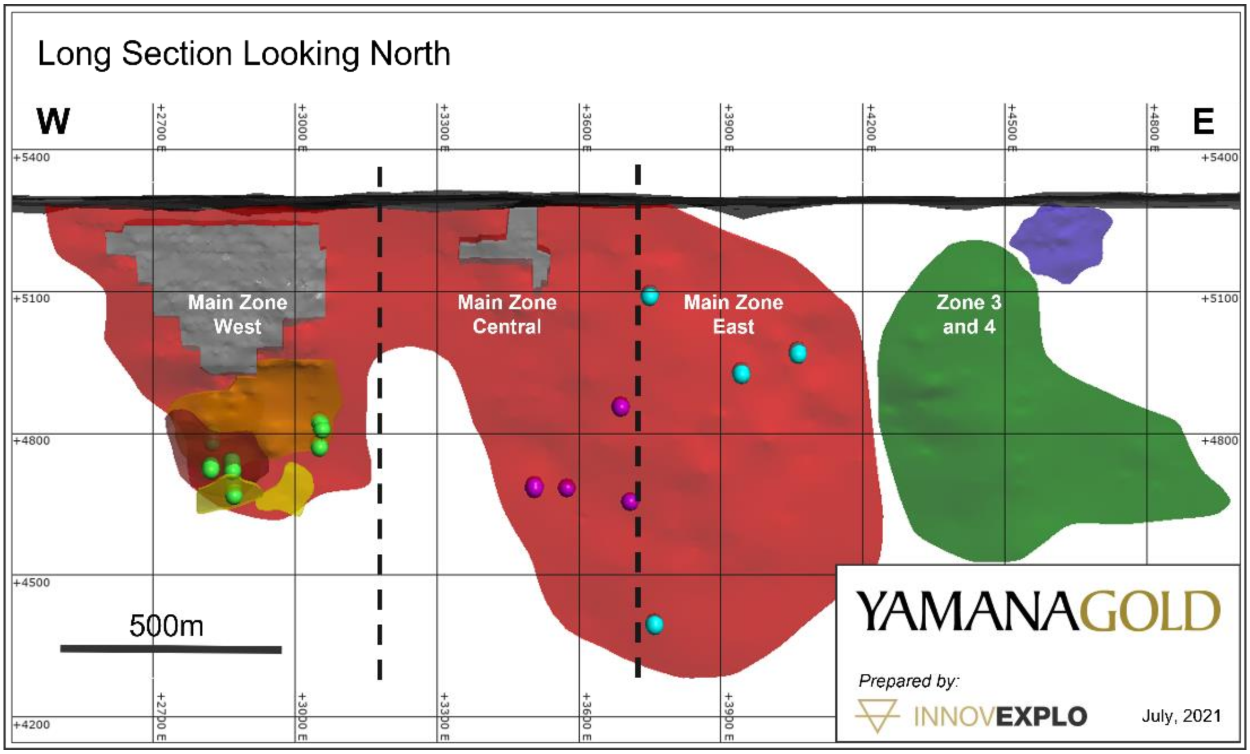
Samples tested aligned with historical zones within the Wasamac deposit. The corresponding nomenclature between the current and historical zones includes:

- Main Zone West was “Zone Principal” or “ZP”
- Main Zone Central was "Zone 1" or "Z1"
- Main Zone East was “Zone 2” or “Z2”

Figure 13-1 includes a longitudinal section with green, fuchsia, and turquoise spheres to show the location of the drill holes where samples for the 2021 testing program were taken.

The samples were limited to Z1, Z2 and ZP; Z1 and Z2 had shown low recoveries. Sample inventory from the 2018 BaseMet program was utilized. Only composites that fell within the new mineralization wireframe zones interpretation were included for Z1 and Z2; ZP sample contains the new stockwork zone as limited sample masses were available for the feasibility study update program. Table 13-2 shows the sample composition breakdown from the 2018 inventory. Table 13-3 shows which of the three composite samples were selected and used for the testwork, indicated by an “X”.

Figure 13-1: Location of Drill Holes used to Generate Samples for 2021 Testing Program



Source: InnovExplo, 2021

Table 13-2: Sample Compositions

Zone	2018 Composite Sample ID	Inventory Available (kg)	Grade Au (g/t)	Sample Required (kg)
Z1	Z1-VAR-1	6.16	2.84	6.16
Z1	Z1-VAR-5	8.24	2.49	8.24
	Total	14.40	2.64	14.40
Z2	Z2-VAR-1	11.60	1.82	11.60
Z2	Z2-VAR-2	7.64	1.99	7.64
Z2	Z2-VAR-4	15.02	4.60	5.50
Z2	Z2-VAR-5	8.18	1.98	8.18
	Total	42.44	2.36	32.92
ZP	ZP-GRIND-4	13.28	2.44	2.50
ZP	ZP-GRIND-5	6.10	3.33	2.50
ZP	ZP-GRIND-6	12.18	2.73	2.50
ZP	ZP-VAR-2	7.78	2.29	2.50
ZP	ZP-VAR-3	16.68	2.24	2.50
ZP	ZP-VAR-5	16.06	2.97	2.50
	Total	72.08	2.67	15.00

Source: Ausenco, 2021

Table 13-3: Testwork completed by composite

Composite	Leach Testing	Rougher Flotation Testing	Rougher Flotation Testing + WOL	Oxygen Uptake Testing
Z1	X		X	X
Z2	X	X	X	X
ZP	X		X	X

Source: Ausenco, 2021

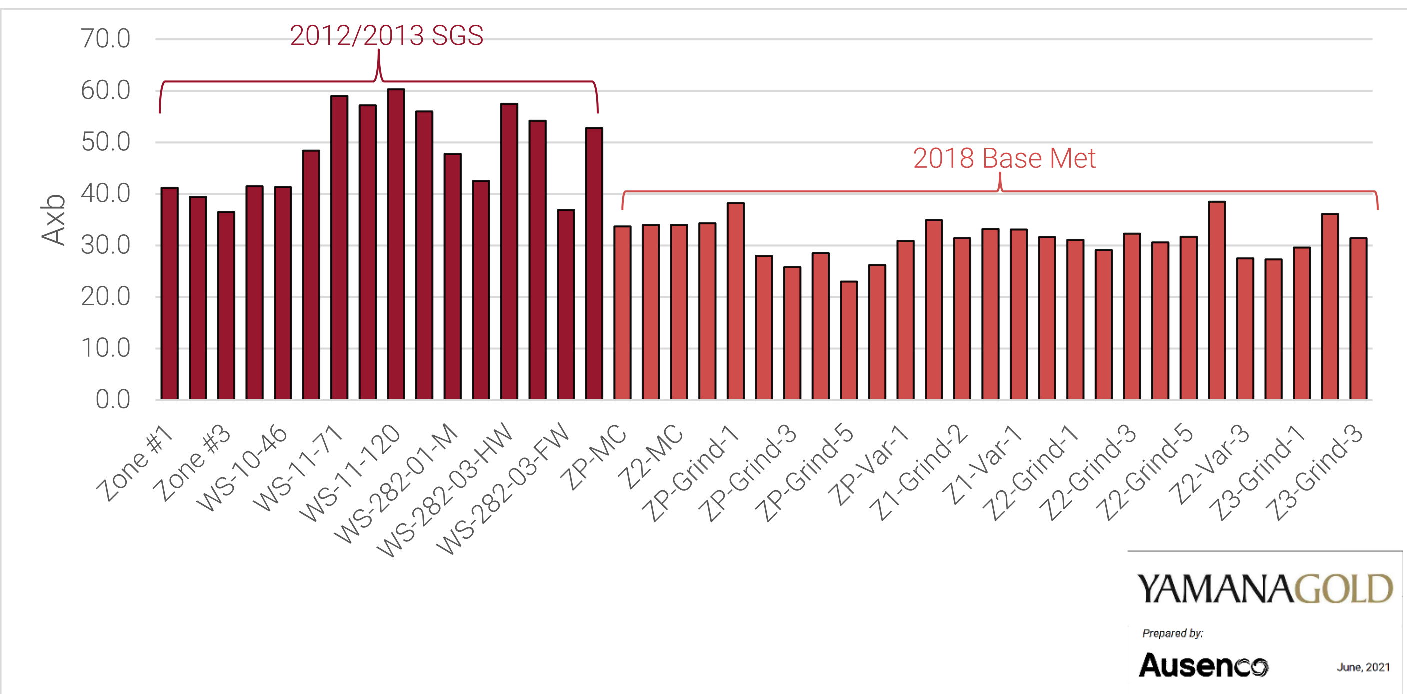
13.3.2 SMC Testing Feed Size Analysis

During the review of previous testwork programs, Ausenco found that the JKTech SMC tests were conducted at two different feed sizes in the 2012/2013 and 2018 test programs. The 2012/2013 SMC testwork performed at SGS was conducted on feed size material of -31/+26.5 mm while the 2013 SMC testwork performed at BaseMet was performed on -22.4/+19 mm material. The SMC test can be performed on either size fraction; however, it was originally developed for a standard feed size of -31/+26.5 mm.

Ausenco contacted JKTech to calculate a correction factor to adjust the results of the -22.4/+19 mm SMC tests.

Figure 13-2 shows the different Axb factor, a measure of ore hardness, for the two datasets conducted at different feed sizes while Figure 13-3 shows the 2018 BaseMet results after the size correction factor is applied by JKTech.

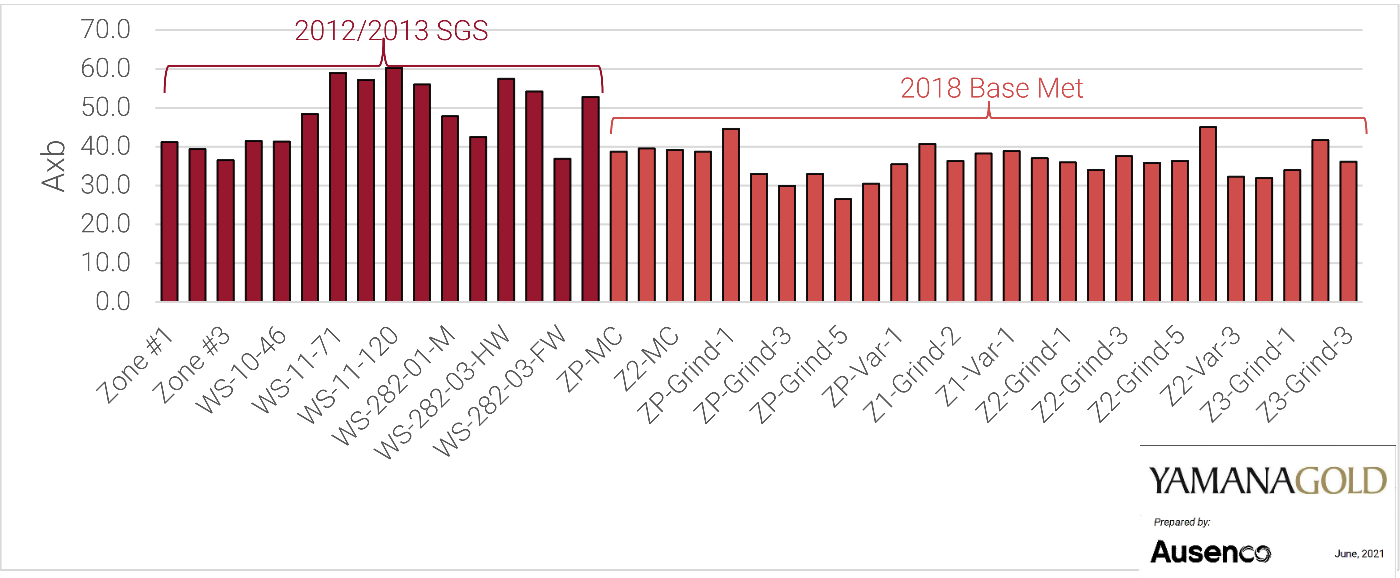
Figure 13-2: SMC Test Results for Two Different Feed Sizes, before Correction



Source: Ausenco, 2021

The corrected SMC test results in Figure 13-3 were then used in the determination of the design point for the Axb factor, as summarized in Section 13.5. It is noted that this process design criterion is now based on more competent ore (lower Axb) than previous studies.

Figure 13-3: SMC Test Results for Two Different Feed Sizes, after Correction



Source: Ausenco, 2021

13.3.3 Leach Testing

Optimized leaching conditions were developed for the three composites (Z1, Z2 and ZP). One sample from each composite underwent gravity recoverable gold (GRG) testing with leaching completed on gravity concentrate and tails. Baseline leach testwork was completed using 1 kg of material on bottle rolls measuring leach kinetics after 48 hours, at which point the leach was terminated. Cyanide leach testwork was evaluated at varying primary grind sizes to confirm the grinding mill equipment sizing and to review the impact on extraction kinetics and recovery at varying pH levels.

The following leach conditions were maintained throughout all baseline tests:

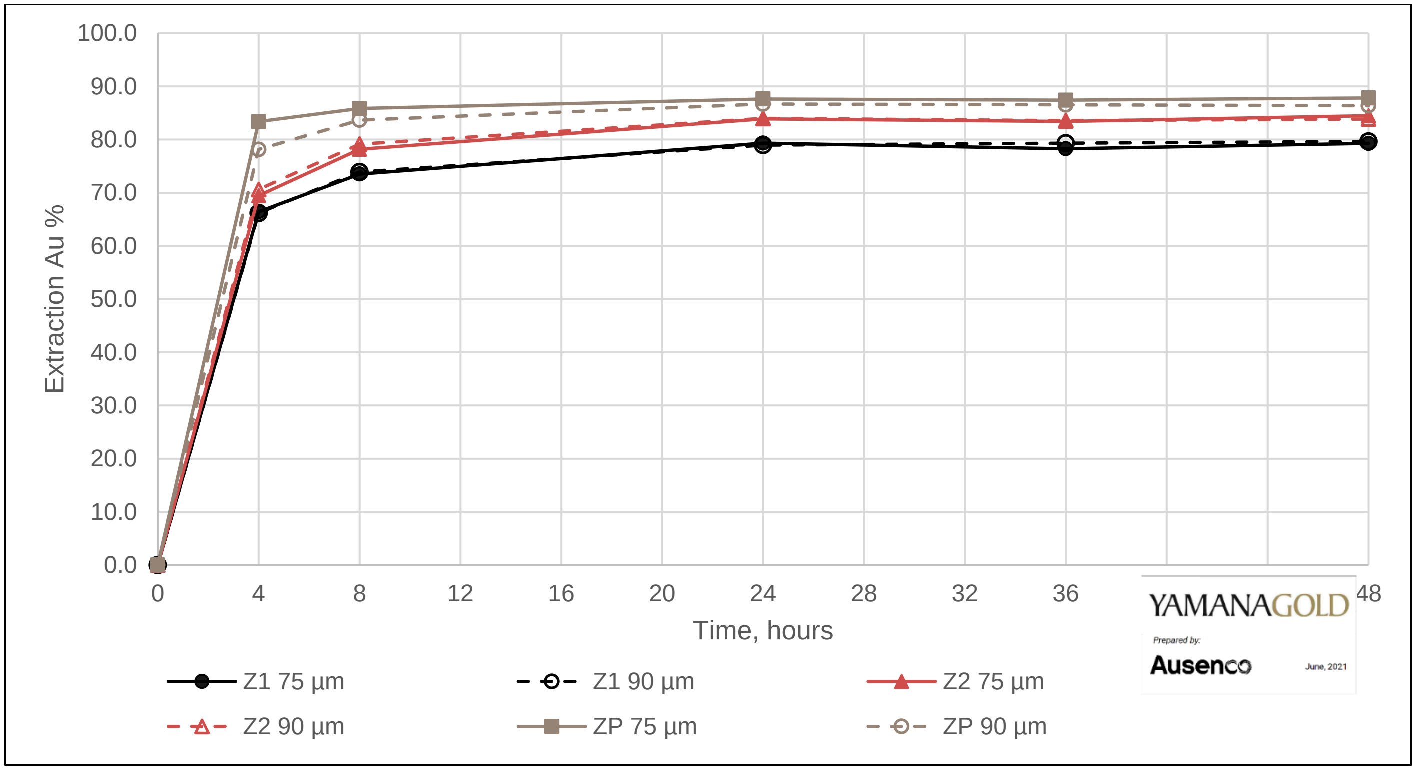
- pulp density = 40%wt solids
- NaCN concentration = 1.0 g/L (maintained)
- dissolved oxygen concentration >20 mg/L
- retention time = 48 hours
- grind size K_{80} = 60 µm

Kinetic leach testing was completed on two additional samples from each composite at the same conditions as stated above, but with leach kinetic measurements at 4, 8, 24, 36, and 48 hours. These samples were assigned Test ID CN22 through CN27. Test IDs CN22 through CN24 were leached at a grind of 80% passing (K_{80}) 75 µm grind size while CN25 trough CN27 were leached at a K_{80} of 90 µm grind size.

The results of all leach testing are shown in Table 13-4 on the following page.

Grind sizes finer than a nominal K_{80} of 75 µm, as shown in Table 13-4 below, did not show significant increases in gold extraction when comparing final residue assays.

Figure 13-4: Leach Test Kinetics by Grind Size



Source: Ausenco, 2021

Table 13-4: Leach Testing Results

Test ID	Sample ID	Grind, K_{80} (µm)	Leach Time (h)	NaCN	Addition (kg/t)		Consumption (kg/t)		Au Grade			Recovery (%)		
									Hd (Calc)	Hd (Dir)	Residue	Grav.	CN	Comb
				g/L	NaCN	CaO	NaCN	CaO	g/t	g/t	g/t	Au	48	Au
CN1	Z1 Comp	60	48	1.0	1.94	0.34	0.53	0.34	2.65	2.26	0.40	4.3	84.4	85.1
CN4	Z1 Comp	60	48	1.0	1.95	0.43	0.63	0.43	2.37	2.26	0.43		82.1	82.1
CN7	Z1 Comp	60	48	1.0	1.87	1.14	0.43	1.14	2.33	2.26	0.43		81.8	81.8
CN10	Z1 Comp	60	48	1.0	2.04	2.32	0.63	2.32	2.34	2.26	0.43		81.6	81.6
CN22	Z1 Comp	75	48	1.0	1.88	0.45	0.46	0.45	2.20	2.26	0.46		79.3	79.3
CN25	Z1 Comp	90	48	1.0	1.76	0.31	0.31	0.31	2.21	2.26	0.45		79.6	79.6
CN2	Z2 Comp	60	48	1.0	2.02	0.41	0.68	0.41	2.09	2.18	0.28	2.6	86.5	86.8
CN5	Z2 Comp	60	48	1.0	2.01	0.40	0.69	0.40	2.14	2.18	0.28		87.1	87.1
CN8	Z2 Comp	60	48	1.0	1.97	1.25	0.57	1.25	2.16	2.18	0.29		86.6	86.6
CN11	Z2 Comp	60	48	1.0	1.85	2.33	0.41	2.33	2.23	2.18	0.30		86.8	86.8
CN23	Z2 Comp	75	48	1.0	1.75	0.27	0.27	0.27	1.93	2.18	0.30		84.5	84.5
CN26	Z2 Comp	90	48	1.0	2.33	0.91	0.91	0.91	1.98	2.18	0.32		83.9	83.9
CN3	ZP Comp	60	48	1.0	1.82	0.46	0.46	0.46	2.13	2.00	0.27	4.8	86.7	87.3
CN6	ZP Comp	60	48	1.0	2.01	0.45	0.64	0.45	2.52	2.00	0.24		90.7	90.7
CN9	ZP Comp	60	48	1.0	1.97	1.24	0.56	1.24	2.47	2.00	0.27		89.1	89.1
CN12	ZP Comp	60	48	1.0	1.97	1.95	0.58	1.95	2.61	2.00	0.27		89.6	89.6
CN24	ZP Comp	75	48	1.0	1.81	0.34	0.34	0.34	2.26	2.00	0.28		87.8	87.8
CN27	ZP Comp	90	48	1.0	1.77	0.31	0.31	0.31	2.34	2.00	0.32		86.4	86.4

Source: BaseMet, 2021

Tests to evaluate common telluride leach conditions did not improve recoveries from the 2018 testing. Gravity concentration removed only 2.6% to 4.8% of free gold in the samples tested and made no impact in overall recoveries. Telluride leach conditions with high pH (free lime concentrations of 0.5 and >1.1 g/L CaO) and maximum dissolved oxygen concentrations made no improvement in recoveries or reagent consumptions compared to standard leach conditions. Despite the presence of gold telluride minerals, telluride leach conditions were not found to be beneficial. Average recoveries from the 2021 tests include:

- Z1 = 82.5% Au
- Z2 = 86.7% Au
- ZP = 89.0% Au

Previous testwork results were analyzed along with the 2021 results summarized here to determine the optimum leach conditions:

- 35 hours of overall residence time
- grind size K_{80} of 60 µm
- sodium cyanide (NaCN) addition rate of 0.6 kg/t of leach feed (design)
- CaO addition rate of 1.0 kg/t of leach feed (design) added to SAG mill feed and maintained at pH 10.5 to 11.0.

The leach circuit feed pulp density selected was 50%wt solids.

The test results showed that 24 hours retention time may be sufficient. Additional testing is planned in the next phase of the project to validate this.

13.3.4 Oxygen Uptake Testing

Oxygen uptake testing was completed on one kg samples at the K_{80} grind size of 60 µm from each of the three composites. The following conditions were maintained during testwork:

- pulp density = 40%wt solids
- NaCN concentration = 1.0 g/L (maintained)
- pH = 11.5 (maintained with lime)
- dissolved oxygen target = 20 mg/L with oxygen addition

The intention of the testwork was to determine oxygen demand in leaching. The testwork indicated moderate to high oxygen demand, in the range of 0.012 to 0.030 mg/L/min after 24 hours of sparging. The test results also provide oxygen consumption data for selecting the required oxygen plant capacity or liquid oxygen supply.

13.3.5 Flotation Testing

13.3.5.1 Rougher Flotation

Rougher flotation testing was completed on one-kilogram samples for three K_{80} grind sizes for the Z2 composite only. The following conditions were maintained or achieved during testwork:

- NaCN concentration = 1.0 g/L (maintained)
- grind sizes, K_{80} = 60, 90, 120 µm
- pH = natural conditions

Table 13-5 shows the ultimate gold recovery and the flotation tails assay for each of the three tested grind sizes (shaded grey) These results indicated that grind size had little to no effect on flotation efficiency.

Table 13-5: Flotation Test Results

Product	Weight		Assay			Distribution		
	%	grams	Au (g/t)	Ag (g/t)	S (%)	Au (%)	Ag (%)	S (%)
Z2 Composite – 60 µm								
Ro Con 1	5.1	51.0	26.6	44	37.9	66.0	46.5	77.6
Ro Con 2	7.2	72.1	22.1	38	32.8	77.3	56.9	95.0
Ro Con 3	8.6	86.1	19.3	34	28.1	80.9	60.7	97.1
Ro Con 4	10.0	99.6	17.1	30	24.5	82.9	62.7	97.9
Rougher Tail	90.0	901.3	0.39	2	0.06	17.1	37.3	2.1
Z2 Composite – 90 µm								
Ro Con 1	5.1	50.5	26.9	42	35.5	63.3	54.2	72.9
Ro Con 2	7.6	75.4	21.6	36	30.2	75.9	68.8	92.6
Ro Con 3	9.6	96.1	18.0	30	24.4	80.6	74.6	95.3
Ro Con 4	11.1	111.2	16.0	27	21.2	82.6	77.3	96.1
Rougher Tail	88.9	887.0	0.42	1.0	0.11	17.4	22.7	3.9
Z2 Composite – 120 µm								
Ro Con 1	4.2	42.3	28.5	44	38.7	54.7	38.4	63.4
Ro Con 2	7.8	77.8	20.5	34	30.0	72.6	54.5	90.4
Ro Con 3	11.3	112.4	15.7	27	21.8	80.2	61.7	94.7
Ro Con 4	12.9	129.0	14.1	24	19.2	82.6	64.1	95.7
Rougher Tail	87.1	870.1	0.44	2	0.13	17.4	35.9	4.3

Source: Ausenco, 2021

13.3.5.2 Rougher Flotation-Concentrate Leach

Rougher flotation testwork was conducted on one kilogram of each of the three project composites. The concentrate from flotation testwork was subsequently reground and leached, and tailings were also leached. The following conditions were maintained or achieved during testwork:

- NaCN concentration = 1.0 g/L (maintained)
- primary grind size K_{80} = 120 µm
- concentrate secondary grind size K_{80} = 20 µm
- pH during rougher flotation = 8.0-8.5 (natural conditions)
- pH during leaching = 11
- leach time = 48 hours

Table 13-6 shows the results of this testwork. The flotation flowsheet produces comparable or better recoveries than whole ore leach recoveries. Testing has consistently shown higher recovery with this flowsheet for Z1 and Z2 samples (2.5% for ZP to 6.6% higher for Z1). These two zones constitute over 60% of the mineral reserves. Recoveries in these zones significantly impact the overall recoveries and project economics, therefore improved recoveries in Z1 and Z2 using flotation should be explored in the future. The whole ore leach flowsheet was carried forward in the project due to its operational simplicity and lower capital cost.

Table 13-6: Rougher Flotation-Leach Test Results

Product	wt. (%)	Au (g/t)	Au Recovery (%)	
			Stage	Overall
Z1 Composite				
Concentrate Leach Solution			87.8	72.2
Concentrate Leach Residue	15.8	1.29	12.2	10.0
Flotation Tailings Leach Solution			74.4	13.3
Flotation Tailings Leach Residue	84.2	0.11	25.6	4.6
Combined Gold Dissolution				85.4
Z2 Composite				
Concentrate Leach Solution			89.8	76.1
Concentrate Leach Residue	19.4	0.88	10.2	8.6
Flotation Tailings Leach Solution			76.1	11.6
Flotation Tailings Leach Residue	80.6	0.09	23.9	3.7
Combined Gold Dissolution				87.7
ZP Composite				
Concentrate Leach Solution			93.4	81.8
Concentrate Leach Residue	16.2	0.81	6.6	5.8
Flotation Tailings Leach Solution			80.4	10.0
Flotation Tailings Leach Residue	83.8	0.07	19.6	2.4
Combined Gold Dissolution				91.8

Source: Ausenco, 2021

13.4 Recoveries

Recent testwork results from BaseMet labs confirm and support recoveries from the 2018 Feasibility Study. The recoveries from the samples excluded and the samples included in the revised mineralized zones are very similar, providing continuity between the 2018 Feasibility Study and the current study. In depth analysis of leach test results did not establish relationships between head and tails grades or spatial recovery relationships. Scatter in the recoveries align with the conclusion to continue to assign single point recoveries to each zone, as seen in Table 13-7.

Table 13-7: Recoveries by Zone

<table>
<tr><th>Zone</th><th>Composite</th><th>Feed
Au (g/t)</th><th>Recovery
(%)</th><th>Yamana 2021
Model Recoveries</th><th>2018 FS Recoveries</th></tr>
<tr><td rowspan="4">Z1</td><td>Z1-VAR-1</td><td>2.38</td><td>79.2</td><td rowspan="4">81.1%</td><td rowspan="4">81.6%</td></tr>
<tr><td>Z1-VAR-3</td><td>3.97</td><td>79.6</td></tr>
<tr><td>Z1-VAR-5</td><td>1.99</td><td>83.4</td></tr>
<tr><td>Z1-AUS</td><td>2.37</td><td>82.1</td></tr>
<tr><td rowspan="6">Z2</td><td>Z2-GRIND-1</td><td>3.04</td><td>85.4</td><td rowspan="6">86.4%</td><td rowspan="6">86.2%</td></tr>
<tr><td>Z2-VAR-1</td><td>1.56</td><td>87.2</td></tr>
<tr><td>Z2-VAR-2</td><td>1.71</td><td>84.4</td></tr>
<tr><td>Z2-VAR-4</td><td>4.26</td><td>90.5</td></tr>
<tr><td>Z2-VAR-5</td><td>1.6</td><td>83.5</td></tr>
<tr><td>Z2-AUS</td><td>2.14</td><td>87.1</td></tr>
<tr><td rowspan="4">ZP</td><td>Z3-GRIND-2</td><td>3.48</td><td>93.5</td><td rowspan="4">93.4%</td><td rowspan="4">92.7%</td></tr>
<tr><td>Z3-VAR-2</td><td>3.00</td><td>93.1</td></tr>
<tr><td>Z3-VAR-3</td><td>3.36</td><td>93.9</td></tr>
<tr><td>Z3-VAR-4</td><td>6.03</td><td>93.1</td></tr>
</table>

Source: Ausenco, 2021

13.5 Conditions from Previous Testwork

Testwork evaluated from previous project phases were used in the selection of design criteria for the following plant conditions. Detailed design criteria can be found in Section 17.2.1 and is summarized in Section 13.5.

- Comminution
  - Axb (design, 25th percentile) = 35.9
  - bond rod mill work index (design, 75th percentile) = 16.9 kWh/t
  - bond ball mill work index (design, 75th percentile) = 15.0 kWh/t
  - abrasion index (average) = 0.243 g
- Gravity gold recovery
  - Wasamac ore is not amenable to gravity separation

- Leach reagent addition rates
  - sodium cyanide addition rate = 600 g/t of leach feed
  - quick lime addition rate to SAG mill feed (design) = 3000 g/t of leach feed
  - quick lime addition rate to SAG mill feed (for operating cost estimation) = 1000 g/t of leach feed
  - hydrated lime addition rate (design) = 1000 g/t of leach feed
  - hydrated lime addition rate (for operating cost estimation) = 500 g/t of leach feed
- CIP carbon concentration = 25 g/L
- Cyanide destruction
  - cyanide detox process selected = O_2/SO_2
  - circuit feed concentration (pre-dilution) = 100 mg/L CN_{WAD} (nominal), 150 mg/L CN_{WAD} (design)
  - residence time = 70 minutes
  - SO_2 addition rate (design) $SO_2:CN_{WAD}$ ratio = 5:1
  - copper addition rate (design) = 50 mg/L Cu^{2+}
  - hydrated lime addition rate (design) = 5.0 g $Ca(OH)_2$/g CN_{WAD}
- Dewatering testwork
  - pre-leach and tails thickener:
    - solids loading at or below 24% thickener feed density = 0.9 (t/h)/m2
    - flocculant addition rate (design) = 45 g/t
  - filtration plant clarifier:
    - solids loading at or below 3% clarifier feed density = 1.8 $(m^3/h)/m^2$
    - flocculant addition rate (design) = 55 g/t

14 MINERAL RESOURCE ESTIMATES

The mineral resource estimate for the Wasamac Project has been completed following standards set out in CIM's "Mineral Resource and Mineral Reserves Estimation Best Practices Guidelines" (November 2019) and were classified according to CIM's "Definition Standards for Mineral Resources and Mineral Reserves" (May 2014) guidelines. The mineral resource estimate presented in this Report is based upon Yamana's internal independently validated mineral resource estimates and is effective as of June 30, 2021. Vincent Nadeau-Benoit, P.Geo. and Alain Carrier, M.Sc., P.Geo., both from InnovExplo and who qualify as independent and qualified persons (QPs) under N.I. 43-101, have audited and validated Yamana's mineral resource estimate, including all key assumptions and interpolation results, using all available information. Final mineral resource classification (according to CIM 2014 Standards), constraining volumes (optimized underground mineable shapes) and cut-off grades were established by InnovExplo. Mineral resources are reported exclusive of mineral reserves and satisfy the reasonable prospects for eventual economic extraction.

The purpose of the updated mineral resource estimate is to incorporate a new 3D geological model (modified mineralized zones), supported by new geostatistical analysis, new interpolation strategy, new assumptions for classification and the creation of potentially mineable shape to constrain the mineral resource estimate.

The mineral resource area for the Wasamac gold deposit covers a strike of 2.7 km and a width of approximately 600 m to a depth of 1,300 m below surface. Leapfrog Geo v.5.1.4 was used for the 3D mineralization models of the deposit prepared by Yamana that included the construction of 11 gold domains, which are supported by the alteration assemblages and the interpreted units and structures of the lithological model and the assay results. Datamine Studio RM v.1.7 was used for the estimation, which consisted of 3D block modelling and the ordinary kriging (OK) interpolation method. Statistical studies (including capping and variography) were completed using GSLIB v.2.907.

The validations were carried out by the QPs using Microsoft Excel, Leapfrog Geo, Leapfrog Edge and Snowden Supervisor.

The main steps in the methodology were as follows:

- compile and validate the databases used for the 3D modelling and for the mineral resource estimation
- ensure availability and accuracy of the void model (inclusions of historical underground openings), assess the use of historical information versus more recent drilling information
- validate the 3D lithological model and interpretation of the mineralized zones based on lithological units, alteration zones, mineralization, structural information, gold assay values and the general geometry of historical stopes
- validate and review the drill hole intercepts database, conduct independent sensitivities on composite length and approaches, conduct independent sensitivities on capping values and the resulting estimation databases for the purposes of geostatistical analysis and variography including independent tests in Snowden Supervisor
- review and validate all key assumptions, including the requirement for additional in-situ density measurements for supporting tonnage estimates
- validate the block model key parameters, interpolation methods, strategies and parameters, conduct independent sensitivities with other interpolation methods in Leapfrog Edge and check Datamine final grade interpolation results

- establish classification criteria, produce clipping areas for mineral resource classification and apply them to the block model
- assess the mineral resources with "reasonable prospects for potential economic extraction" by selecting the appropriate cut-off grades and produce "resources-level" optimized underground mineable shapes
- ensure adequate subtraction of the historical voids and exclusion of the mineral reserves (from this study) from the mineral resource estimate
- generate the final mineral resource estimate

Mineral resources are reported exclusive of mineral reserves. Mineral resources are not mineral reserves and have not demonstrated economic viability. The mineral resource statement of the deposit, as of June 30, 2021, is presented in Table 14-13.

14.1 Drill Hole Database

The database was provided by Yamana as a Microsoft Access file (*.mdb) using a format compatible with Geovia GEMS 6.8.2. The deposit was defined by surface and underground core drilling inside the gold domains of the deposit. All the drill holes have been established on a local grid system; however, UTM drill hole coordinates are also available.

The database contains information including collar information, deviation survey, gold assays, lithological description, alteration, structural measurements from core, mineralization, and major textures.

As shown in Table 14-1, the database includes data from 3,317 drill holes that were drilled for various purposes in the vicinity of the Wasamac deposit from the 1940s to 2012. Of these, 804 drill holes were used for the mineral resource estimate: 355 were drilled from the surface, with an aggregate length of 146,967 m and 28,667 assays (28,843 m assayed); and 449 were drilled from underground, with a total length of 11,024 m and 7,434 assays (10,173 m sampled).

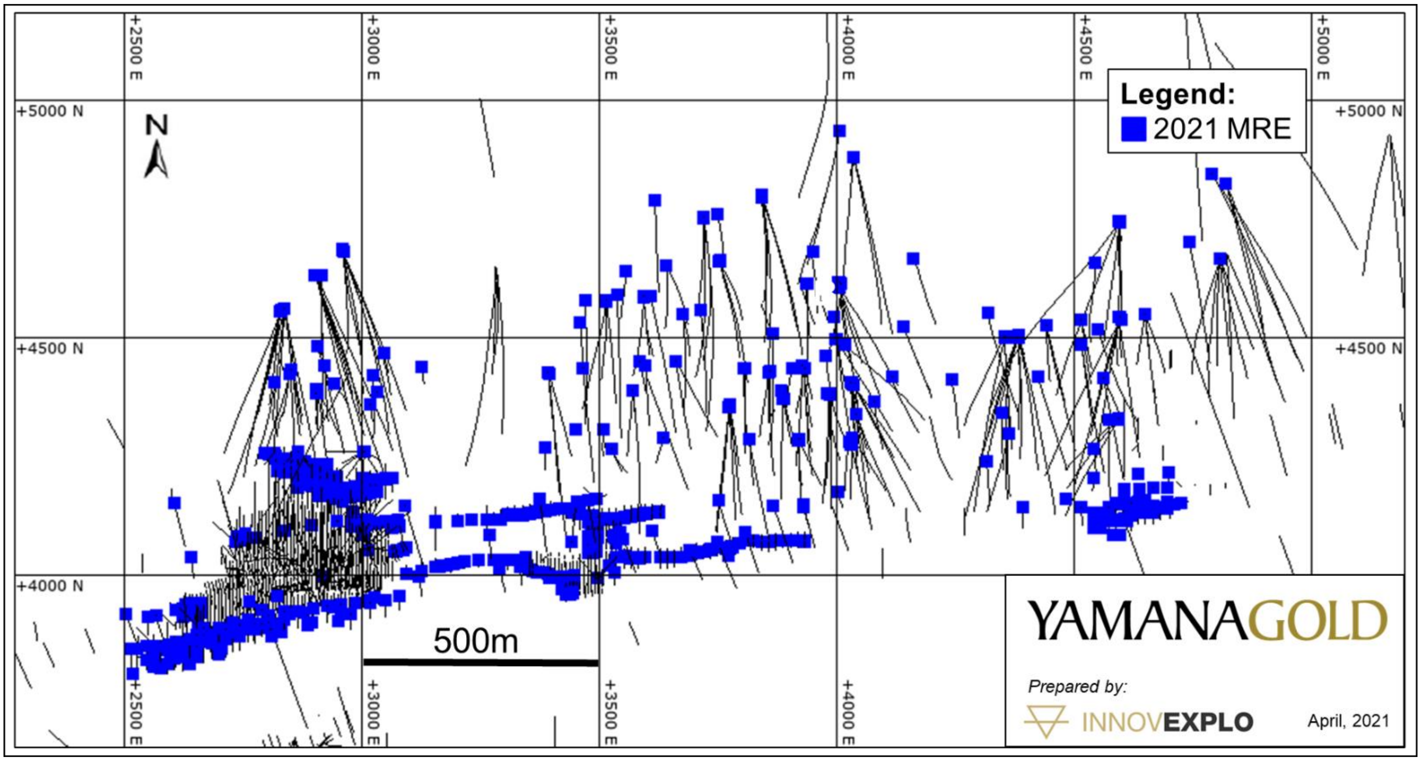
It is important to note that for drill hole intervals transecting the "mined out" volumes, gold values were used to model the variograms but were discarded and excluded for the interpolation; these drill holes are, therefore, not counted in the total amount of the drill holes used in the mineral resource estimate database.

Table 14-1: Number of Drill holes (DDH) in the Database for the Wasamac Gold Deposit

DDH Position	Validated DDH (Total Length)	DDH used for Mineral Resource Estimation (Total Length)	Samples from DDH used for Mineral Resource Estimation (Total Length)
Surface	1,141 (246,107 m)	355 (146,967 m)	28,667 (28,843 m)
Underground	2,176 (46,278 m)	449 (11,024 m)	7,434 (10,173 m)
Total	**3,317 (292,385 m)**	**804 (157,991 m)**	**36,101 (39,016 m)**

A surface plan view of the validated diamond drill holes used in the mineral resource estimate is shown in Figure 14-1.

Figure 14-1: Surface Plan View of the Validated Diamond Drill Holes used in the mineral resource estimate



Note: Local mine grid coordinate system

14.2 Mineralization Model (Definition and Interpretation of Estimation Domains)

An internal mineralization model completed by Yamana and its resultant 3D wireframes define the mineral resource estimation domains (see Figure 14-2). The mineralization model was independently reviewed and validated by the QPs.

Compared to the previous model, which was used in the 2018 Feasibility Study (Caumartin et al., 2018), and based on the database, the updated mineralization model reflects field observations, the complexity of the mineralization and is more representative of the geology. The extents of the domains were modified.

The mineralization model 3D domains are mostly supported by alteration assemblages (defined within an alteration model) and finely disseminated sulphides distributed along and parallel to the interpreted Francoeur-Wasa shear zone (defined within a lithological model) (see Section 7.3). As with the alteration and lithological models, the assay results were used to complete the interpretation when other drill hole data (usually qualitative data) were lacking. No cut-off grades were used to define and/or restrain the estimation domains. This precluded artificially high-grading the mineral resource estimates and explains the disappearance of the "donut holes" from the previous model.

In total, 11 gold domains and sub-domains were created for the deposit, as listed below.

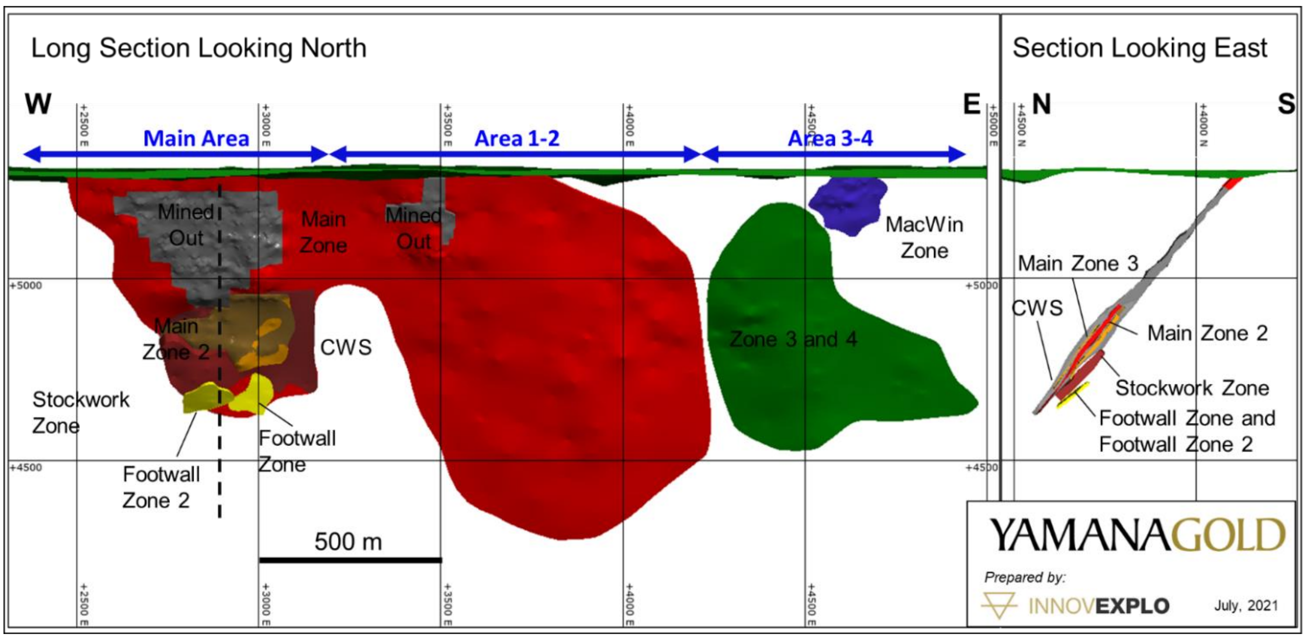
- The new "Main Zone" domain merges what were previously known as the Main Zone, Zone 1 and Zone 2, and includes discrete sub-domains (and discrete wireframes) to adequately separate the various styles of mineralization: CWS, Main Zone, Main Zone 2, Main Zone 3, Stockwork_Zone, Footwall Zone and Footwall Zone 2.

- The new "Zone 3 and 4" domain merges what were previously known as "Zone 3" and "Zone 4".
- The new "MacWin Zone".

The domains extend a maximum of 60 m along dip and strike from the final assayed sample point. At depth, to the west, the extent of the new Main Zone was limited, given the occurrence of the nearby Horne Creek fault and the risk of over-interpreting the down-dip continuity of the zone. The domains' wireframes are snapped to the assay intervals of the drill holes; a maximum snap distance of 2.0 m was used in order to limit local distortions in the wireframes. The solids were designed with no minimum thickness.

Any unestimated material within the block model limits was assigned a value of 0.00 g/t Au and a density of 2.80 g/cm^3.

Figure 14-2: Mineralization Model



Note: Local mine grid coordinate system

14.3 Other 3D Surfaces and Volumes, Topography, Bedrock and Voids Model

The topography, bedrock and voids models were provided by Yamana. Individual 3D surfaces were created to define the surface topography and overburden/bedrock contact. The topography surface was created using data from different surveys completed by Richmont (prior to 2013); its resolution is around 25 m in the vicinity of the deposit. The overburden-bedrock contact surface was modelled using logged overburden intervals and used to clip the mineralization domains' 3D wireframes. The voids represent historical underground workings from the Wasamac mine (combined stopes, drifts and shafts).

The underground development models were generated using hardcopy sections and plan maps from the Wasamac mine's archives. Electronic capture of drift-walls was done using a digitizer or georeferenced scanned maps. For level plans, surveying station coordinates were captured, and wall lines were put at the station elevations using Promine software.

Subsequently, an extrusion was applied to the wall lines to create the models of the drifts in 3D using Geovia GEMS. The heights of drifts were adjusted depending on the level (see Table 14-2).

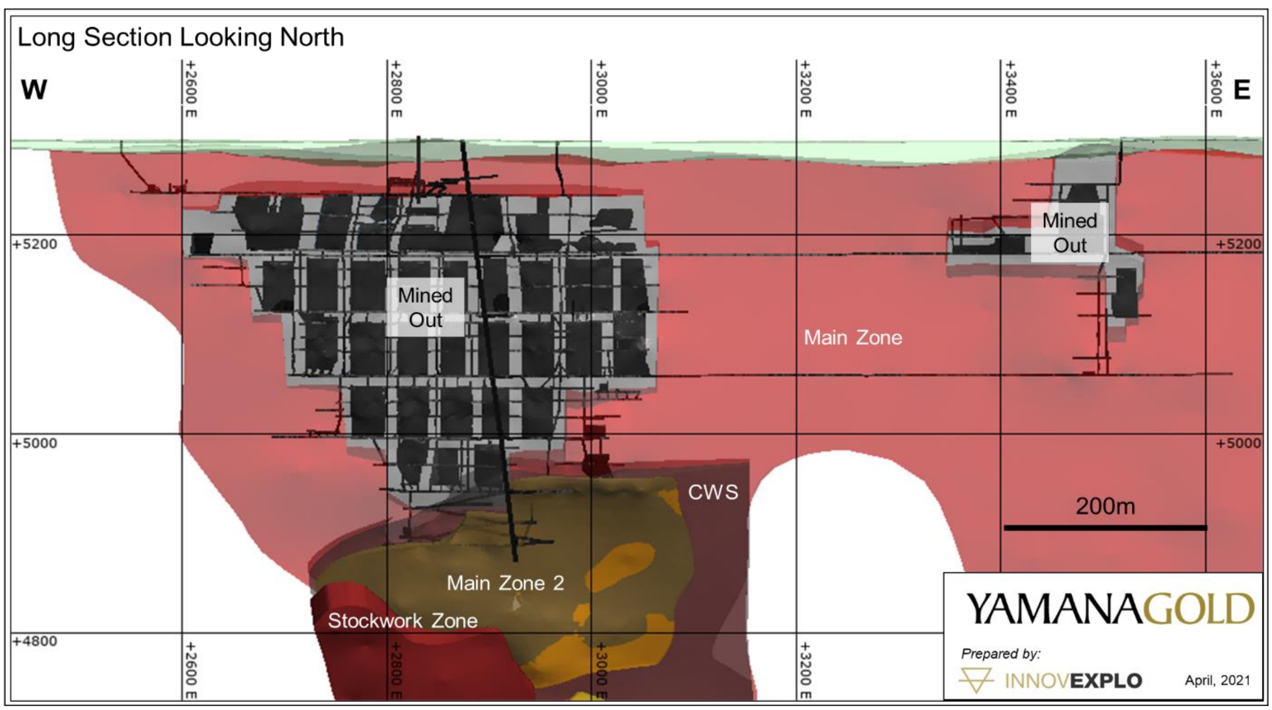
Table 14-2: Historical Drift Heights (from the 2018 Feasibility Study)

Drift Height	Level (Feet Below Surface)
2.43 m (8 ft)	From surface to 1,175
2.74 m (9 ft)	From 1,175 to 1,350
3.05 m (10 ft)	Ramp from 1,175 to 1,350

The volume of all the stopes and the total tonnage mined was calculated using a density of 2.80 g/cm3. The tonnage of the modelled stopes (1,764,879 tonnes) compared well with the historical production records (1,892,448 tonnes), considering that a certain tonnage from development was also likely sent to the Wasamac mill (Gauthier et al., 2012). The location of the stopes was corroborated using drill hole information.

Two 5 to 20 m mined-out buffer volumes were built in the western (historically known at the Main Zone) and eastern part (historically known as Zone 1) of the new Main Zone using two copies of the modelled Main Zone domain clipped with a boundary polyline drawn around the bulk of the historic stopes and underground infrastructures in those areas (see Figure 14-3). These volumes were removed from the gold domains prior to the interpolation. No historical underground workings are present in Zones 3 and 4.

Figure 14-3: Voids Model Mined-Out (Black and Dark Grey) compared to the Buffer Volumes (Light Grey)



Note: Local mine grid coordinate system

Based on the available data, the voids are considered accurate, and the addition of buffer zones add some security margins to their fringes.

14.4 Compositing Methods

Codes corresponding to each gold domains (VCODE) were automatically attributed to the drill hole assay intervals that intersect them (Drill hole Evaluation in Leapfrog Geo). The coded intercepts were used to analyze assay lengths and generate statistics for raw assays and composites. Table 14-3 summarizes the statistical analysis of the original (raw) assays for each domain. The raw assay statistics were used for composite length.

Table 14-3: Summary Statistics for the Drill Hole Raw Assays

Gold Domain(s)	VCODE	Assay Count	Maximum (Au g/t)	Mean (Au g/t)	Standard Deviation	COV
Main_Zone, Main_Zone_2, Main_Zone_3	1100 1200 1300	6385	176.57	2.51	4.66	1.86
Footwall_Zone, Footwall_Zone_2	2100 2200	85	43..06	3.91	5.74	1.47
Zone_3_and_4	3100	492	32.59	2.76	3.34	1.21
Stockwork_Zone	4100	382	99.53	3.64	8.37	2.30
CWS	5100	1818	59.15	0.93	2.41	2.59
MacWin_Zone	7100	350	60.69	3.74	6.03	1.61

A variable composite length was selected that corresponds to the total length of the drill hole intercept interval of each gold domain. The domain thickness (and the variability of it), the nature of the mineralization (especially its homogeneity and the low COVs), the proposed block size, and the original sample length were taken into consideration when choosing this scenario. The summary statistics of the composited length are examined by mean, minimum, and maximum values and are shown in Table 14-4.

Table 14-4: Summary Statistics for the Drill Hole Composites

Gold Domain(s)	VCODE	Assay Count	Maximum (Au g/t)	Mean (Au g/t)	Standard Deviation	COV
Main_Zone, Main_Zone_2, Main_Zone_3	1100 1200 1300	746	41.69	2.42	2.41	0.99
Footwall_Zone, Footwall_Zone_2	2100 2200	11	7.70	3.59	2.05	0.57
Zone_3_and_4	3100	57	7.09	2.70	1.49	0.55
Stockwork_Zone	4100	15	11.27	3.76	3.00	0.80
CWS	5100	255	29.55	1.04	2.21	2.12
MacWin_Zone	7100	48	22.48	3.80	4.42	1.16

A few intersects of the gold domains were only partially sampled; a value of 0.00 g/t Au was assigned to these missing intervals.

The coordinates of the centre of each composite were assigned.

During the validation by the QPs, a scenario was completed using different approaches and parameters. Among others, sensitivity was completed using 3.0 m equal-length composites. In conclusion, no material differences (i.e., greater than 1%) were found, and a final scenario using variable composite length was retained.

14.5 High-Grade Capping

Capping values were applied to the composites on a domain-per-domain basis. Composites were statistically examined for the presence of grade-outliers by using a combination of methodologies, such as the inspection of probability plots and histogram analysis. The value of the coefficient of variation (and how it is affected by capping) and the Parrish method were also considered, for each domain. To limit the influence of these identified outliers, all composites above the defined capping value (established for each domain) were limited to this defined value. Each gold domain was examined separately. Figure 14-4 and Table 14-5 below show the capping analysis plots and statistics of the Main Zone (VCODE 1100, 1200 and 1300).

Figure 14-4: Capping Analysis (Plots) for the Composites of the Main Zone (VCODE 1100, 1200 and 1300)

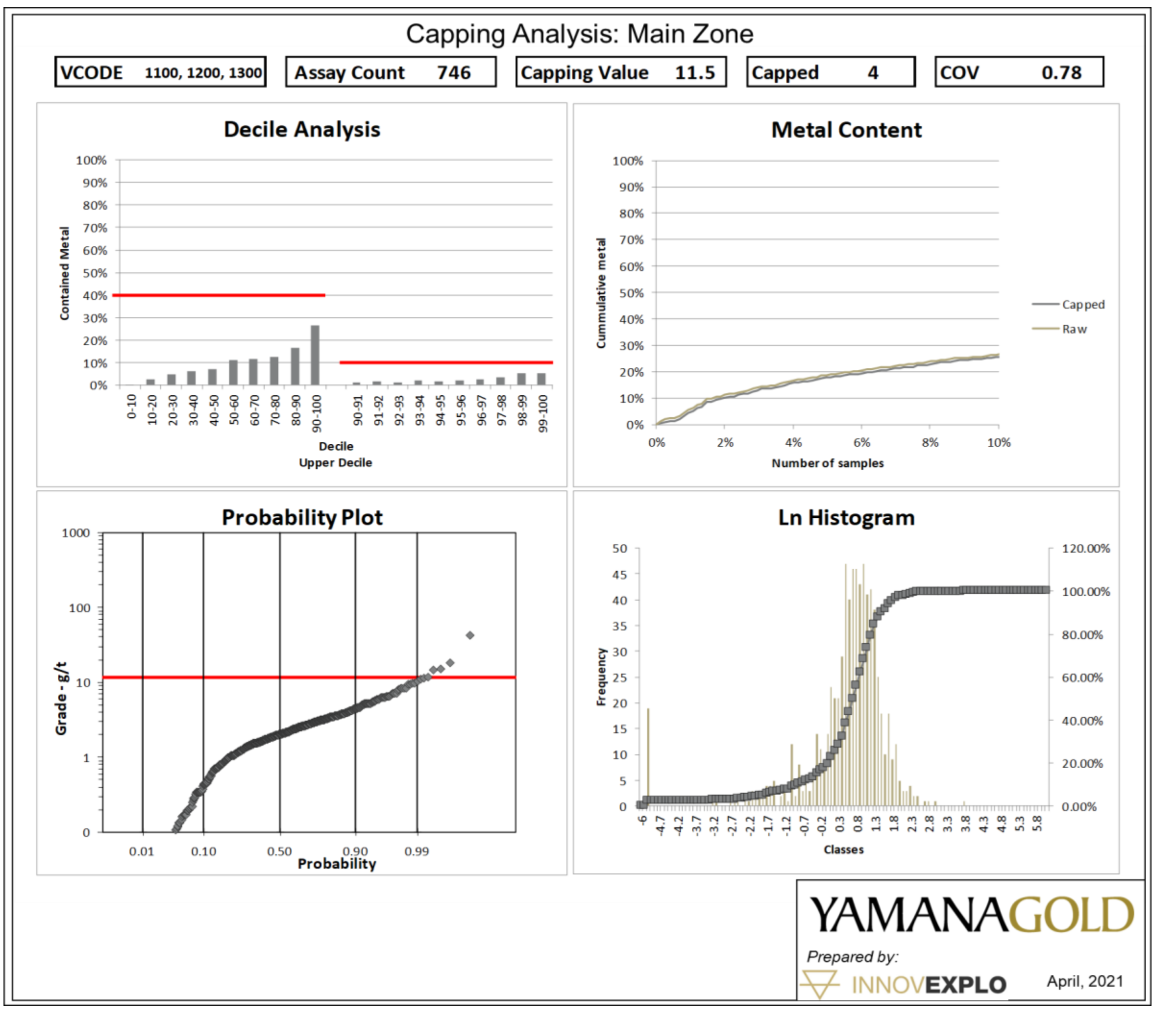


Table 14-5: Capping Analysis (Statistics) for Composites of the Main Zone (VCODE 1100, 1200 and 1300)

Parameter	Composited Au	Capped Composite Au	Composite Length
Mean	2.42 g/t	2.36 g/t	10.15 m
Standard Deviation	2.41	1.85	8.20
C.O.V.	0.99	0.78	0.81
Skewness	7.14	1.96	2.52
Maximum	41.69 g/t	11.50 g/t	86.87 m
Minimum	0.00 g/t	0.00 g/t	0.05 m
Median	2.01 g/t	2.01 g/t	8.60 m
N Sample	746	746	746

The defined capping values are listed in Table 14-6. During the validation by the QPs, scenarios were completed using different approach and parameters. Among others, sensitivity was completed on capping the raw assays before compositing (using different and adapted thresholds obtained from raw assays statistics review). In conclusion, no material differences (i.e., greater than 1%) were found from the sensitivity models and a final scenario using capping applied on the composites was retained.

Table 14-6: Composite Capping Values by Gold Domains

Gold Domain(s)	VCODE	Capping Value (Au g/t)	Composites Capped
Main_Zone, Main_Zone_2, Main_Zone_3	1100 1200 1300	11.5	4
Footwall_Zone, Footwall_Zone_2	2100 2200	-	-
Zone_3_and_4	3100	6.3	1
Stockwork_Zone	4100	7.5	1
CWS	5100	4.3	7
MacWin_Zone	7100	11.7	2

14.6 Density

Historically, a tonnage factor of 12 ft^3/ton of ore was used at the Wasamac mine, corresponding to a density of 2.8 g/cm^3. No additional measurements were found for subsequent historic drilling campaigns. In May 2010, the URSTM performed density measurements on samples from Zone 2 that were sent for metallurgical testing (drill holes WS-10-31 and WS-10-36). The average of the 21 samples gave a specific gravity of 2.82 g/cm^3. In 2011, approximately 40 additional density measurements were performed by URSTM, returning average density results of 2.800 g/cm^3 for the Main Zone, 2.827 g/cm^3 for Zone 1, and 2.843 g/cm^3 for Zone 2.

For the updated mineral resource estimate, additional “in-situ density” measurements were added to support tonnage estimates. In May 2021, densities determined by standard water immersion methods were calculated on 586 samples from previously drilled core. These 586 samples were obtained from within or in the vicinity of the gold domains, and their results are presented in Table 14-7. In conclusion, an average density value of 2.80 g/cm^3 is considered appropriate for all mineralized domains and adjacent waste rocks and was used for the mineral resource estimate.

Table 14-7: Density by Gold Domains (2021 Measurements Campaign)

Domain	Sample Count	Mean	Median	Minimum	Maximum	Standard Deviation	COV
CWS	136	2.79	2.80	2.60	2.99	0.07	0.02
Footwall_Zone	3	2.77	2.76	2.75	2.79	0.02	0.01
Footwall_Zone_2	13	2.80	2.79	2.71	2.90	0.05	0.02
Horne_Fault	1	2.64	2.64	2.64	2.64	-	-
Main_Zone	173	2.82	2.82	2.54	3.03	0.07	0.02
Main_Zone_2	24	2.79	2.78	2.67	3.01	0.08	0.03
Main_Zone_3	16	2.80	2.80	2.68	2.90	0.06	0.02
Stockwork_Zone	33	2.80	2.79	2.73	2.91	0.04	0.02
Buffer Zone	112	2.79	2.79	2.61	3.01	0.07	0.03
Zone_3_and_4	75	2.82	2.82	2.69	2.99	0.07	0.02
Total	586	2.80	2.80	2.54	3.03	0.07	0.02

14.7 Block Model

A rotated block model (blocks and sub-blocks) was constructed in Datamine. The 3D wireframes of the gold domains act as sub-block triggers for their coding. Independent scenarios were completed using different block sizes and no material differences (i.e., none >1%) were found from the sensitivity models.

The origin of each block model is the lower-left corner. Block dimensions reflect the sizes of mineralized zones, variability of the mineralization and plausible mining methods.

Table 14-8 shows the properties of the block model.

Table 14-8: Block Model Properties for the Wasamac Deposit

Description	X	Y	Z
Block Model Origin (Mine Grid Coordinates)	2368.00	3392.00	4241.00
Rotation Angle	None	None	86.00
Parent Block Dimension	15.00 m	15.00 m	15.00 m
Number of Parent Blocks	169	85	68
Minimum Sub-block Dimension	0.25 m	0.25 m	0.25 m

14.8 Variography

Variography used for the Wasamac deposit is based on the capped composites with the attributed VCODES corresponding to the Main Zone (1100, 1200 and 1300). The composites corresponding to the mined-out volumes were added to the dataset and were used mainly to improve the fit of the exponential correlograms model at a short distance.

Experimental correlograms were calculated along the strike, dip and pole directions.

The correlogram models for the Main Zone gold domains (capped composites with VCODES 1100, 1200, 1300 or associated with the mined-out volumes) are shown in Figure 14-5; their parameters are shown in Table 14-9.

Figure 14-5: The Correlogram Models for the Main Zone Gold Domains

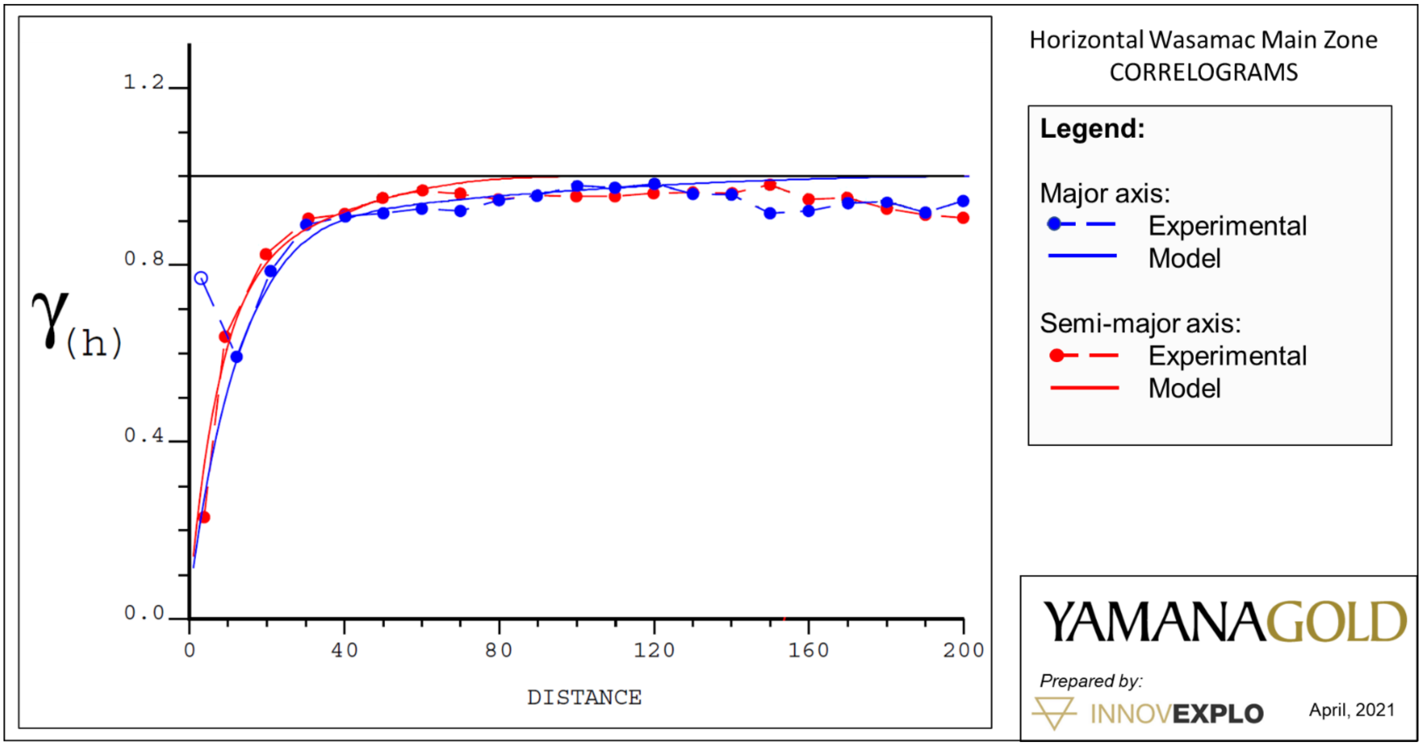


Table 14-9: Correlogram Model Parameters for the Main Zone Gold Domains

VCODE	Structure	Contribution	Model	R1x (m)	R1y (m)	R1z (m)	Z (°)	X (°)	Z (°)
1100,	C0	0.05	Nugget	-	-	-	-	-	-
1200,	C1	0.75	Exp	36	24	93	176	-50	-70
1300	C2	0.10	Sph	38	80	20	176	-50	-70
Mined-Out	C3	0.10	Sph	200	100	20	176	-50	-70

14.9 Grade Interpolation

Each gold domains were estimated independently (as hard boundaries) with ordinary kriging (OK) algorithm using the capped composite grades.

An approach with three estimations-passes was used to estimate all blocks within the domains. Search ellipses were rotated to orient the first axis along the dip direction, the second axis along the strike direction of the vein, and the third axis along the pole direction.

Anisotropic search ranges were derived from the variography for the major and semi-major axis of the ellipses: The first-pass ranges correspond to the distance at a sill of 95 and the second- and third-pass ranges correspond to the distance at a sill of 100 (full distance range). For the minor axis, and because composites correspond to the intersect intervals of each gold domains (so therefore there is only one composite and no pair in the direction of the minor axis) the same ranges of the major axis were used.

The grade estimation parameters are summarized in Table 14-10.

Table 14-10: Estimation Search Parameters

Parameters	1st Pass	2nd Pass	3rd Pass
Interpolation Method	OK	OK	OK
Search Range - Major (m)	70	200	200
Search Range - Semi-major (m)	50	100	100
Search Range - Minor (m)	70	200	200
Minimum Number of Composites	3	3	1
Maximum Number of Composites	10	10	10
Maximum Number of Composites per DDH	1	1	1

14.10 Block Model Validation

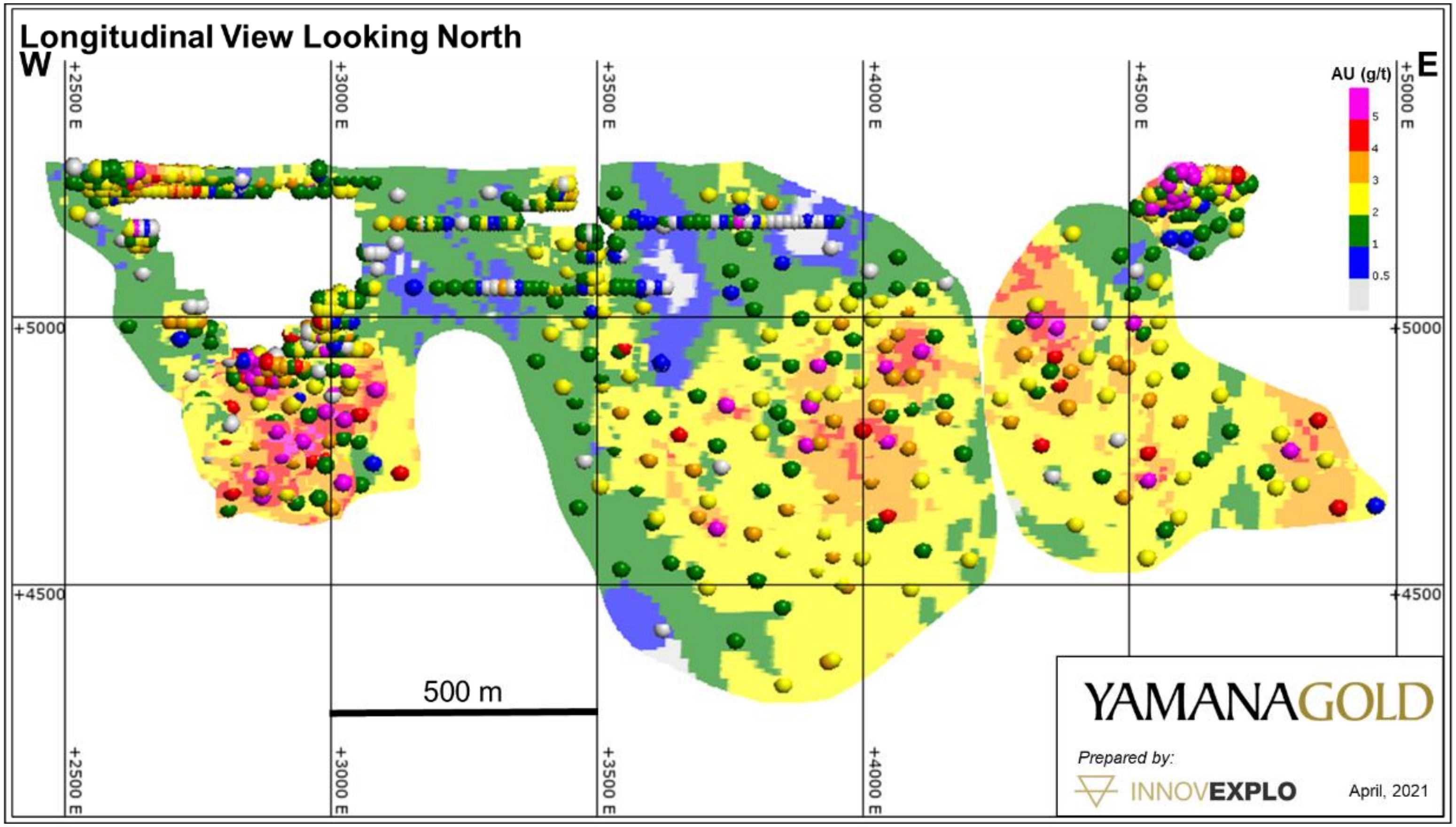
The block models were validated visually and statistically. The visual validation confirmed that the block models honour the drill hole composite data (see Figure 14-6).

A nearest-neighbor (NN) model was produced to check for local bias in the model using the same ellipses (with anisotropy) and estimation domains as the OK model.

The OK model was compared with the NN model using swath plots. The NN and OK models show similar trends although the OK model is smoother, as expected, when compared to the NN model. Figure 14-7 shows the swath plot in the principal direction of the Wasamac deposit as an example. A statistical comparison was also completed between the composites, the results of the NN and OK models.

Comparative results from Main Zone are presented, as an example, in Table 14-11.

Figure 14-6: Longitudinal View Comparing the Block Grades to the Composites Au Values (Points)



Note: Local mine grid coordinate system

Figure 14-7: North Direction Swath Plot (Y) of the Wasamac Deposit

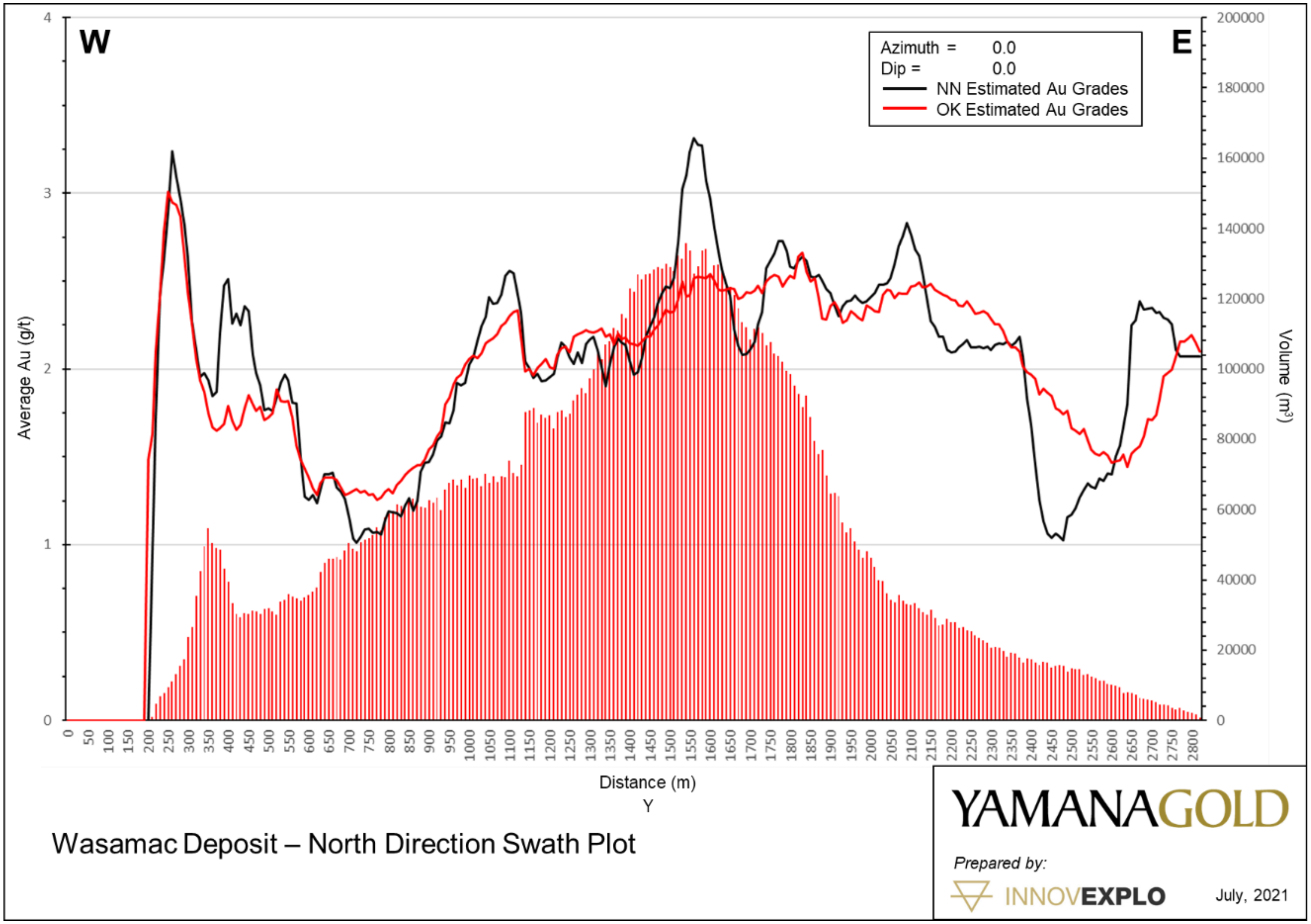


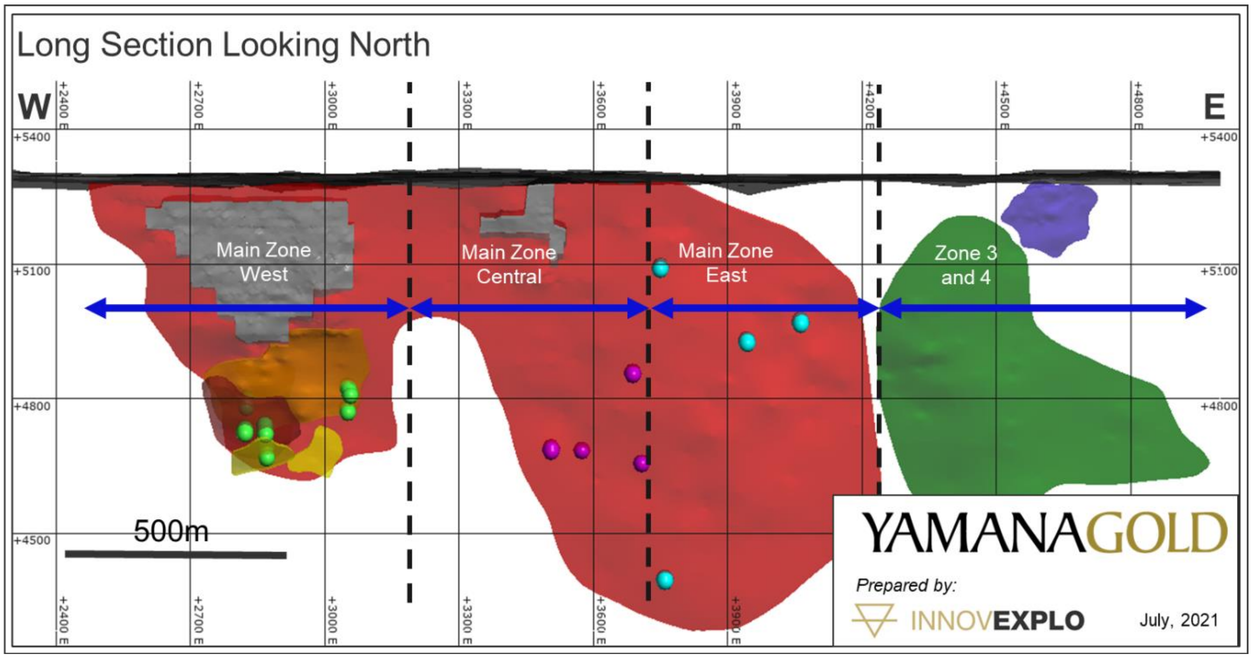
Table 14-11: Statistical Validation of the Estimated Block – Main Zone

Description	Composites	OK	NN
Number of Blocks		3,737,264	3,737,264
Number of Samples	746		
Gold Statistics (g/t Au)			
Minimum	0.00	0.14	0.00
Q1	1.20	2.09	1.62
Median	2.01	2.46	2.13
Q3	3.06	2.59	2.74
Maximum	11.50	9.10	11.5
Mean	2.36	2.34	2.33
Standard Deviation	1.85	0.78	1.42
COV	0.78	0.33	0.61

14.11 Economic Parameters and Cut-Off Grade

Different cut-off grade values were calculated for each metallurgical domains (see Figure 14-8) using the parameters presented in Table 14-12.

Figure 14-8: Metallurgical Domains



Note: Local mine grid coordinate system

Table 14-12: Input Parameters Used to Calculate the Different Underground Cut-off Grades

Input Parameter	Metallurgical Domains			
	Main Zone West	Main Zone Central	Main Zone East	Zone 3 and 4
	Value	Value	Value	Value
Gold Price (US$/oz)	1250	1250	1250	1250
Exchange Rate (USD/CAD)	1.32	1.32	1.32	1.32
Gold Price (C$/oz)	1650	1650	1650	1650
Royalty (%)	1.5	1.5	1.5	1.5
Recovery (%)	92.0	81.6	86.2	92.7
Global Mining Costs (C$/t)	40.00	40.00	40.00	40.00
Processing Costs (C$/t)	14.34	14.34	14.34	14.34
Tailing Management Costs (C$/t)	2.20	2.20	2.20	2.20
G&A Costs (C$/t)	2.67	2.67	2.67	2.67
Total Cost (C$/t)	59.21	59.21	59.21	59.21
Calculated COG (g/t Au)	1.2	1.4	1.3	1.2
75% of Reserves COG (g/t Au)	1.1	1.3	1.2	1.2

Yamana uses cut-off grades for their mineral resource calculations that corresponds to approximately 75% of the break-even cut-off grade used to estimate the mineral reserves. The QPs consider these cut-off grades to be adequate based on the current knowledge of the project. Compared to the 18-months rolling average (around US$1,725/oz Au as of June 30, 2021), the gold price value used (US$1,250/oz) is considered conservative.

The QPs consider the selected cut-off grades to be instrumental in outlining mineral resources with reasonable prospects for eventual economic extraction for an underground mining scenario of the deposit.

A constraining volume was produced with the DSO using a minimum mining shape of 10 m along the strike of the deposit, a height of 5 m and a width of 2 m. The maximum and typical shapes measure 15 m x 25 m x [lens width]. The typical shape size was optimized first. If it was not economical, a smaller stope shape was optimized. Stope optimization used cut-off grades of 1.1 g/t Au for the Main Zone West metallurgical domain, 1.3 g/t Au for the Main Zone Central metallurgical domain and 1.2 g/t Au for both the Main Zone East and Zone 3 and 4 metallurgical domains. These cut-off grades were used for indicated and inferred mineral resources. Blocks lying outside the interpreted mineralized zones were considered to have zero gold grade for stope optimization.

The DSO results were used to constrain the mineral resource estimate.

14.12 Classification of Mineral Resources

Globally, the deposit shows good geological continuity of alteration, structure and mineralization (from drill hole intercepts to drill hole intercepts) and shows relatively low variability of gold grade (low coefficient of variation of the raw assays for all the gold domains). The mineral resource classification was completed using the following main confidence criteria: distance from the closest composite; the interpolation pass; and the requirement for reaching a minimum number of drill holes used. In addition to the employed confidence criteria, mineral resources are now constrained within potentially mineable shapes to demonstrate reasonable prospects for eventual economic extraction, and to align with the reporting standard at other Yamana operations.

No measured mineral resources were defined. Previous measured mineral resources were reclassified as indicated mineral resources in this study to align with Yamana's and InnovExplo's prerequisite that measured mineral resources must be supported by underground development sampling, with the required quality assurance and quality control.

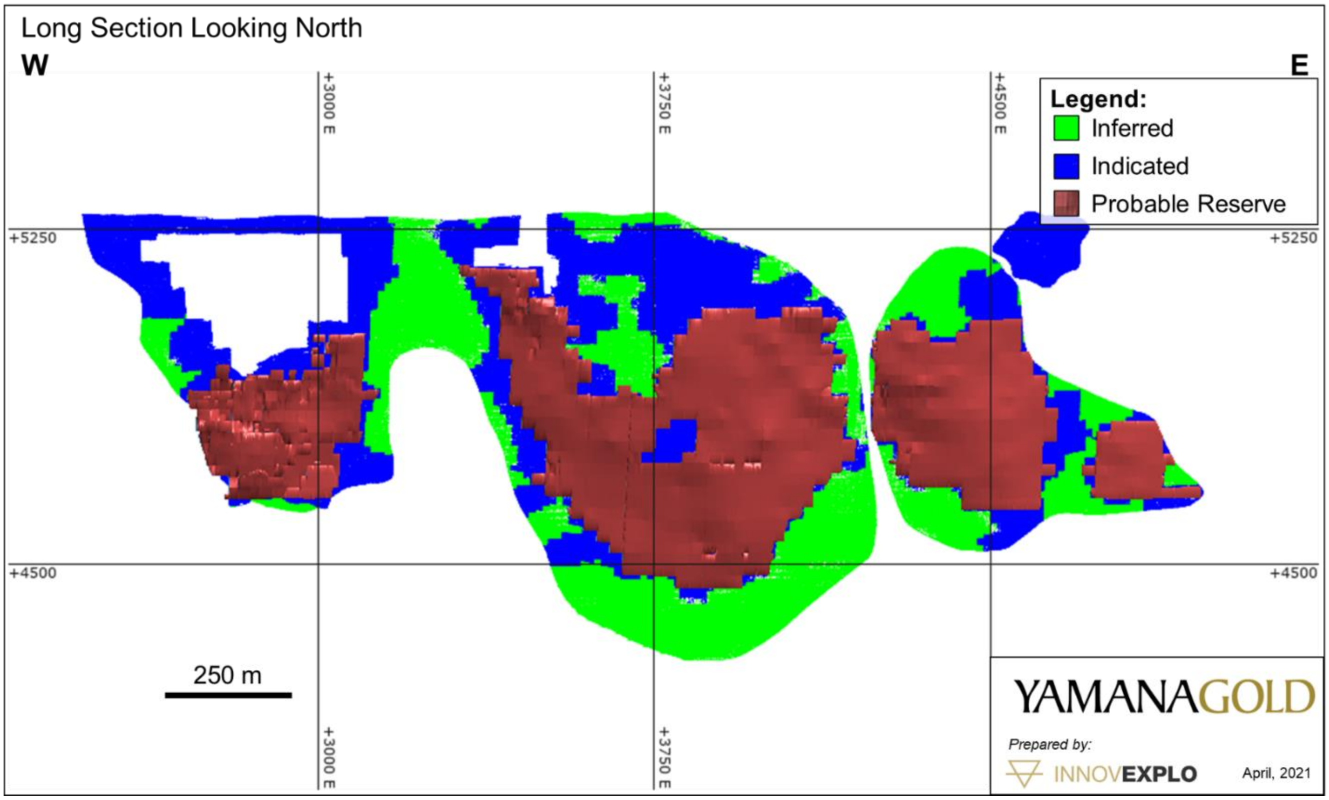
Indicated mineral resources were defined for blocks showing geological and grade continuity interpolated with a minimum of three drill holes during Pass 1 and Pass 2, and within a closest distance of less than 40 m.

Inferred mineral resources were defined by the remaining blocks interpolated from Pass 1, Pass 2 and Pass 3, and included within the interpreted wireframes.

Based on the criteria described above, the final classification was obtained after applying, a series of outline rings (clipping boundaries) created in longitudinal views, keeping in mind that a significant cluster of blocks would be necessary to obtain an indicated mineral resource. Within the indicated category outlines, some inferred blocks were upgraded into indicated, whereas some indicated blocks outside of these outlines were downgraded to inferred category. The QPs consider this a necessary step to homogenize (smooth out) the mineral resource volumes in each category and to avoid the inclusion of isolated blocks in the indicated category.

Figure 14-9 shows the mineral resource classification with the probable mineral reserves for the Wasamac deposit before the resource DSO optimization.

Figure 14-9: Mineral Resource Classification for the Resource with the Probable Mineral Reserves DSO



Note: Classification without cut-off grade and before DSO optimization for the mineral resource. Local mine grid coordinate system

14.13 Mineral Resource Estimate

The mineral resources reported for the deposit are exclusive of mineral reserves and were prepared using conceptual mining shapes from optimized underground mineable shapes that are based on cut-off values and their calculation parameters (see Table 14-12), which correspond to 75% of the cut-off values used to estimate mineral reserves.

Mined-out, sterilized (non- mineable blocks), and current mineral reserves were subtracted from the block models.

The use of constraining conceptual mining shapes to report underground mineral resources and cut-offs to report mineral resources demonstrate that the “reasonable prospects for eventual economic extraction” criteria, as defined in the CIM (2014) Standards, is met.

The mineral resource estimate of the deposit as at June 30, 2021, exclusive of mineral reserves, is presented in Table 14-13.

A long section looking north of the mineral reserves and mineral resources is presented in Figure 14-10.

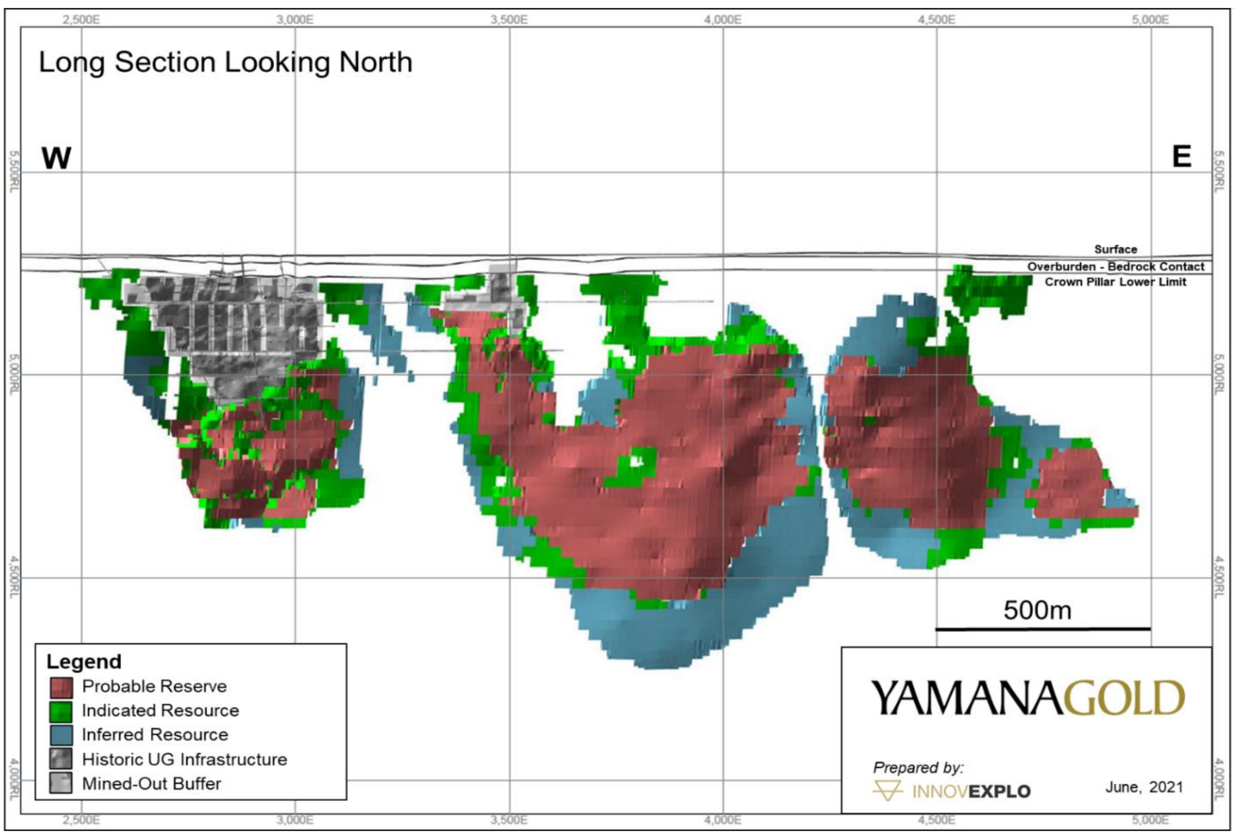
The qualified persons responsible for this section of the technical report are not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that could materially affect the mineral resource estimate.

Table 14-13: Wasamac Estimate of Mineral Resources as of June 30, 2021 (Exclusive of Mineral Reserves)

Category	Tonnage (kt)	Grade Au (g/t)	Contained Gold (koz)
Indicated	5,769	1.76	326
Inferred	3,984	2.01	258

Notes: 1. The qualified persons for the current mineral resource estimates are Mr. Vincent Nadeau-Benoit, P.Geo. and Alain Carrier, M.Sc., P.Geo. (InnovExplo). Mineral resources have been estimated by Yamana and independently audited and validated by InnovExplo. The mineral resource estimate conforms to the 2014 CIM Definition Standards on Mineral Resources and Reserves and follows 2019 CIM definitions and guidelines for mineral resources and are reported exclusive of mineral reserves. 2. The mineral resource estimate has an effective date of June 30, 2021. 3. Mineral resources were evaluated using the ordinary kriging weighting algorithm informed by capped composites and constrained by three-dimensional mineralization wireframes. Mineral resource categories were assigned using clipping boundaries. Indicated category resources were established for blocks interpolated during the first two passes within 40 m closest distance from a drill hole composite within the same mineralized zone. Inferred category resources were established for the remaining interpolated blocks inside the mineralization wireframes. A bulk density of 2.80 g/cm3 was used to convert volume to tonnage. 4. Cut-off grades, which corresponds to 75% of the cut-off grades used to estimate the mineral reserves, are variable based on the metallurgical recoveries and range from 1.10 to 1.30 g/t Au. 5. Mineral resources are below a 32 m surface crown pillar and outside a 5 m minimum buffer around historical underground infrastructures and constrained by potentially mineable shapes based on a minimum mining width of 2 m and considering internal waste and dilution. 6. All figures are rounded to reflect the relative accuracy of the estimate. Sum totals may not add up due to rounding.

Figure 14-10: Long Section looking North of the Mineral Reserves and Mineral Resources

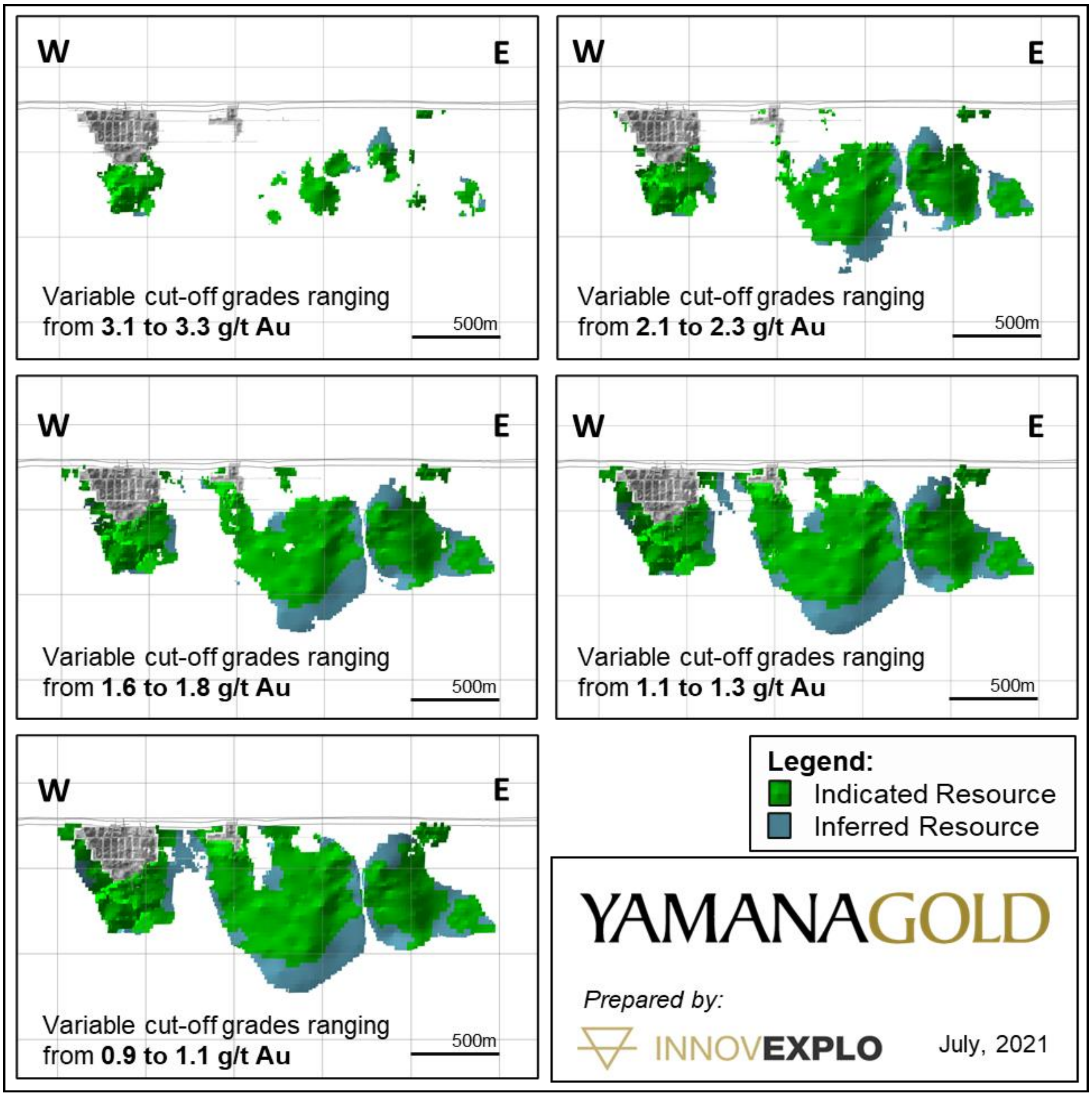


Note: Indicated and inferred mineral resources are exclusives of the mineral reserves. Local mine grid coordinate system

14.14 Sensitivity to Cut-off Grade

Figure 14-11 and Table 14-14 show the cut-off grade sensitivity analysis of the mineral resource estimate. The reader should be cautioned that the numbers provided should not be interpreted as a mineral resource estimate. The reported quantities and grade at different cut-off grades are presented in situ (inclusive of mineral reserves) and for the sole purpose of demonstrating the sensitivity of the mineral resource at different cut-off grades. The Wasamac Project shows clusters of mineral resources at higher cut-off and rapidly get more continuity at intermediate to lower cut-off grades.

Figure 14-11: Long Section looking North showing Sensitivity of the Wasamac Mineral Resources to Cut-off Grades



Note: Constrained by DSO mineable shapes.

Table 14-14: Sensitivity to Cut-off Grade (Constrained by DSO Mineable Shapes)

Category	Cut-off Grade Au (g/t)				Tonnage (Mt)	Grade Au (g/t)	Ounces (koz)
	MZ	Z1	Z2	Z3			
Indicated	3.1	3.3	3.2	3.2	6.515	3.90	817
	2.1	2.3	2.2	2.2	18.330	3.08	1 813
	1.6	1.8	1.7	1.7	23.788	2.81	2 147
	1.1	1.3	1.2	1.2	28.231	2.59	2 349
	0.9	1.1	1.0	1.0	31.131	2.51	2 517
	1.0	1.0	1.0	1.0	29.658	2.51	2 397
Inferred	3.1	3.3	3.2	3.2	0.194	3.45	21
	2.1	2.3	2.2	2.2	1.786	2.61	150
	1.6	1.8	1.7	1.7	2.918	2.33	219
	1.1	1.3	1.2	1.2	4.113	2.05	271
	0.9	1.1	1.0	1.0	4.694	1.93	291
	1.0	1.0	1.0	1.0	4.820	1.90	295

Note: The highlighted grey lines in Table 14-14 represent the results of mineral resources (inclusive of mineral reserves) before the final DSO optimization for obtaining the mineral reserves.

The highlighted grey lines in Table 14-4 represent the results of mineral resources (inclusive of mineral reserves) before the final DSO optimization for obtaining the mineral reserves.

15 MINERAL RESERVE ESTIMATES

15.1 Introduction

Mineral reserves were classified in compliance with the CIM "Definition Standards for Mineral Resources and Mineral Reserves". As such, the mineral reserves are based on measured and indicated mineral resources and do not include any inferred mineral resources. Measured and indicated mineral resources are inclusive of proven and probable mineral reserves. Mineral reserves are the estimated tonnage and grade of ore that is considered economically viable for extraction.

Mineral reserves for the Wasamac deposit incorporate dilution and mining recovery factors based on the selected mining method and design. Also, economic analyses were completed to validate the profitability of particular areas of the mineral reserves.

The following sources of information were instrumental in the mineral reserve estimation process:

- mineral resource block model (last updated May 21, 2021)
- current 3D model of existing underground workings and historical stope outlines
- current 3D litho-structural model of the site

15.2 Estimation Procedure

The mineral resource block model from the last update of May 21, 2021 was used as the basis for estimating the mineable tonnage considered in the mine plan. Cut-off grades for the different mining areas were first estimated, then the stope shapes were optimized according to various parameters, such as geometry and dilution. The final mineral reserve estimate was obtained after completing the stope and underground mine designs, including the economic validation and considering additional factors, such as mine recovery.

15.2.1 Cut-Off Grades Calculations

The cut-off grade calculations are based on parameters from benchmarks as well as Yamana and InnovExplo estimates. Due to the variation in metallurgical recoveries along the deposit, four cut-off grades were used for the stope optimization. The parameters used in these calculations are summarized in Table 15-1.

Table 15-1: Cut-Off Grade Calculation Parameters

Input Parameters	Unit	Metallurgical Domain			
		Eastern Part of Main Zone	Central Part of Main Zone	Western Part of Main Zone	Zone 3 and 4
Gold Price	US$/oz	1250	1250	1250	1250
Exchange Rate	CAD:USD	1.32	1.32	1.32	1.32
Royalty	%	1.5%	1.5%	1.5%	1.5%
Refining Cost	$/oz	4.00	4.00	4.00	4.00
Processing Cost	$/t treated	14.34	14.34	14.34	14.34
Metallurgical Recovery	%	92.0%	81.6%	86.2%	92.7%
Mining Recovery	%	95.0%	95.0%	95.0%	95.0%
Mining Dilution	%	11.9%	15.1%	17.4%	25.9%
Mining Cost	$/t treated	40.00	40.00	40.00	40.00
Tailing Management	$/t treated	2.20	2.20	2.20	2.20
General and Administration	$/t treated	2.67	2.67	2.67	2.67
Cut-Off Grade	g/t	1.45	1.68	1.63	1.62

15.2.2 Dilution Factor Calculation

Internal dilution was considered when optimizing stope shapes and converting them into planned mineable stope shapes. External dilution was also considered during stope optimization by using appropriate ELOS values (see Chapter 16) based on stope size and location and rock mechanics properties. Backfill dilution was added afterwards, based on the location of each stope and the mining sequence.

The following parameters were used to estimate stope dilution:

- Dilution is expected to come from the hanging wall (0.5 m) and footwall (0.25 m) for most stopes.
- In the chlorite schist unit, the dilution is estimated to be 1.0 m in the hanging wall and 0.5 m in the footwall.
- A backfill dilution of 0.5 m is estimated for both walls and the back of any stopes in contact with backfill.
- A lower recovery is assumed for sill pillar stopes to support a lower dilution from the paste.
- Secondary ground support is planned to reduce dilution for specific stopes.
- No dilution is assumed for the stope floors.
- For internal, external, and backfill dilution, the calculations assumed a dilution grade of 0 g/t.

In summary, the external waste dilution is estimated to be 11%. When considering the backfill dilution, the average dilution of the project is estimated to be 13%.

15.2.3 Mining Losses

Mining loss (or mining recovery) is based on the material in the model that is left behind; for example, to provide structural support when facing blasting or operational challenges and rock mechanics issues.

The estimated mining recoveries for the site range from 86% to 95%. Recovery varies mainly according to blasting method and the associated challenges, as well as rock mechanics conditions such as sill pillar recovery. Recovery values are based on past experience and estimates for typical stopes, and the factors seem reasonable given the selected mining method and the ground conditions.

The average mining recovery for the project is 93.6%.

15.2.4 Stope Shape Optimization

The geological block model was the primary input in the Deswik Shape Optimizer (DSO) version 2020.3, a Deswik software application used to optimize individual stope shapes from the block model using Stope Shape Optimizer algorithms from Alford Mining Systems and the parameters listed in Table 15-2.

Table 15-2: Stope Shape Optimization Parameters

Parameters	Value
Cut-off Grade Value	Presented in Table 15-1
Waste Density	2.8 g/t
Typical Sublevel Interval	25 m
Typical Stopes Length - Longitudinal Long Hole Stoping - Primary Traverse Long Hole Stoping - Secondary Traverse Long Hole Stoping	 30 m 20 m 30 m
Minimal Stope Width	3 m
Minimum Horizontal Pillar between Parallel Lenses	10 m
External Dilution Included	Presented in Section 15.2.2
Minimum Slope Wall Angle	43°

15.3 Mineral Reserve Estimate

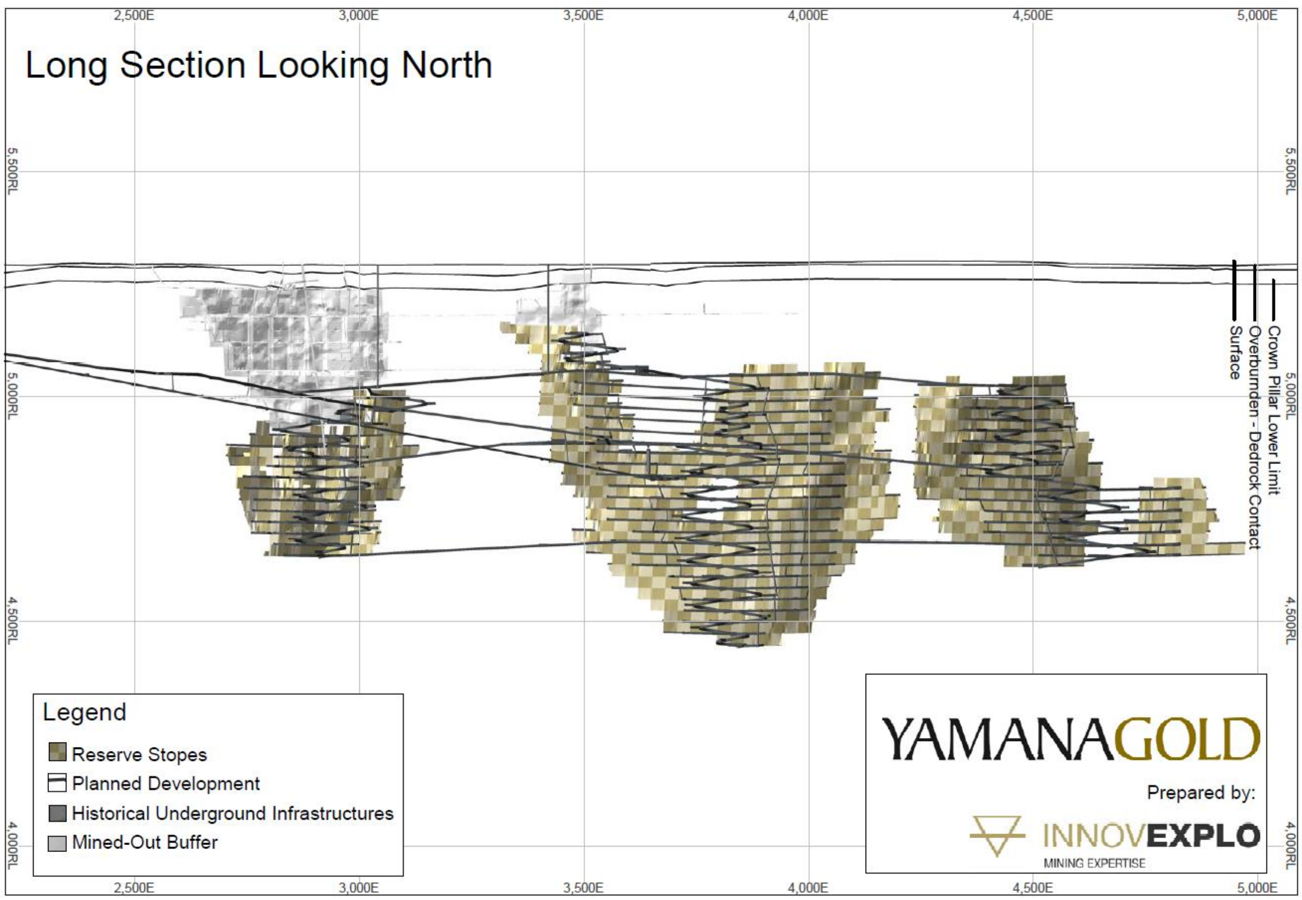
An estimate of mineral reserves for the Wasamac Project as of June 30, 2021 is provided in Table 15-3. An illustration of mineral reserves with planned development and stopes is shown in Figure 15-1.

Table 15-3: Wasamac Estimate of Mineral Reserves as of June 30, 2021[1]

Category	Tonnage (kt)	Grade Au (g/t)	Contained Gold (koz)
Proven	---	---	---
Probable	23,168	2.56	1,910
Total P & P	23,168	2.56	1,910

Notes: 1. The QPs for the mineral reserve estimate are Mr. Denis Gourde, P.Eng. and Sébastien Tanguay, P.Eng. (InnovExplo). The mineral reserve estimate conforms to the 2014 CIM Definition Standards on Mineral Resources and Reserves and follows 2019 CIM definitions and guidelines. 2. Mineral reserve estimate has an effective date of June 30, 2021. 3. The metallurgical recoveries varies with the metallurgical domain: 92.0% for the eastern part of the main zone; 81.6% for the central part of the main zone; 86.2% for the western part of the main zone; 92.7% for zones 3 and 4. 4. Estimated at US$1,250/oz Au using an exchange rate of US$1.32:C$1.00, variable cut-off value related to the metallurgical domain: 1.45 g/t for the eastern part of the main zone; 1.68 g/t Au for the central part of the main zone; 1.63 g/t for the western part of the main zone; 1.62 g/t for zones 3 and 4. 5. Mineral reserve tonnage and mined metal have been rounded to reflect the accuracy of the estimate and numbers may not add due to rounding. 6. The total refining, processing, mining, tailing management and general and administration cost is estimated at 63.21$/t. 7. Mineral reserves include both internal and external dilution. The internal mining dilution varies with the metallurgical domain: 11.9% for the eastern part of the main zone; 15.1% for the central part of the main zone; 17.4% for the western part of the main zone; 25.9% for zones 3 and 4. The external dilution is estimated to be 11%. the average dilution of the project is estimated to be 13%. 8. The estimated mining recoveries for the site range from 86% for the sills to 95% for the stopes. The average mining recovery factor was set at 93.6% to account for mineralized material left in each block in the margins of the deposit. For the purpose of the COG calculation, mine recovery was set at 95%. 9. The qualified person responsible for this section of the technical report is not aware of any mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the mineral reserve estimate.

Figure 15-1: Illustration of Mineral Reserves with Planned Development and Stopes



15.4 Factors that May Affect the Mineral Reserves

Areas of uncertainty that may materially impact the mineral reserve estimate include the following:

- commodity prices, market conditions and foreign exchange rate assumptions
- cut-off grade estimates
- capital and operating cost assumptions
- geological complexity and mineral resource block modelling
- stope stability, dilution and mining recovery factors
- metallurgical recoveries and contaminants
- rock mechanics (geotechnical) constraints and the ability to maintain constant underground access to all working areas

The qualified person responsible for this section of the technical report is not aware of any mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the mineral reserve estimate.

16 MINING METHODS

16.1 Introduction

The Wasamac project will involve an underground operation with a proposed mining method optimized to the deposit geometry and employing a variety of long-hole (longitudinal and transverse) stoping. To access the deposit, two 3-km-long ramps will be developed: one for material handling and the second for personnel and equipment. The total projected underground development was optimized and has decreased compared to the 2018 Feasibility Study. The objective was to decrease both development costs and waste production by implementing three design improvements: increasing level spacing; reducing the requirement for footwall drifts in waste; and optimizing the material handling system to minimize ramp development requirements. The materials handling systems were improved by optimizing various haulage scenarios, available technologies, and combinations thereof.

Mining voids will be filled using a combination of paste fill (delivered from an underground paste fill plant), cemented rockfill (CRF), and rockfill, with the intention of increasing mining recovery, providing stable rock conditions, and minimizing the mine surface footprint and closure requirements. The surface impact of the underground mine will be minimized because of the mine's proximity to the Rouyn-Noranda municipality. Minimal surface break-through of raises, minimal surface infrastructure, and low blasting vibration through the utilization of best blasting practices are considered important objectives.

The increase in mineral reserves, reduction in development metres, and optimization of the material handling systems will allow the project to sustain a production of 7,000 t/d. The Wasamac Project's wide stopes (typically 10 to 15 m), relatively shallow depth, underground conveyor system, and adoption of modern technology are expected to establish it as a low-cost underground mining operation.

Another objective of the project is the reduction of CO2 emissions by utilizing best technologies and appropriate mining strategies and practices. The Wasamac underground mine is designed to create a safe working environment and reduce consumption of non-renewable energy through the use of electric and high-efficiency equipment. Yamana will take advantage of additional technological advances, including ventilation-on-demand and high-efficiency fans, to reduce power requirements, as they become available.

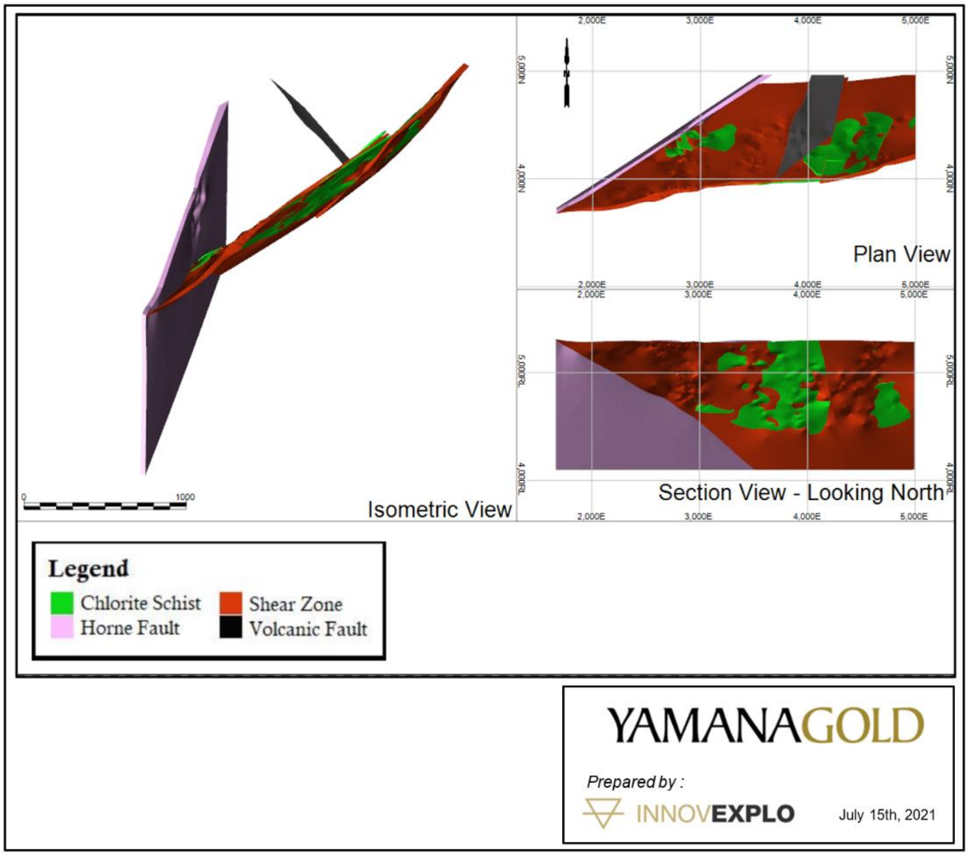
The underground mine study used a local coordinate system to fit with the geology model.

16.2 Rock Engineering

16.2.1 Historic Practices

Historic stoping at the Wasamac mine (see Figure 16-1) used a long-hole stoping method (longitudinal retreating) with 30 m wide x 60 m high stope panels and 12 m wide pillars (Cullen, 1965). The stope development included an access raise, driven along the footwall, with sublevels at 18 m vertical spacing. Sublevels were slashed to ore-width to permit drilling of parallel and angled long-hole blast holes. Mined stopes were not backfilled.

Figure 16-1: Geomechanical Domains – Structures



The former Wasamac Mine ceased operation in April 1971. A review by Little Long Lac Gold Mines Ltd. (1981) identified the main reasons for the shutdown, such as low metal prices, excessive hanging wall dilution, and poor ground conditions in the stoping areas at depth.

16.2.2 Rock Mechanics Domains

A litho-structural domain model was created by Yamana and validated by InnovExplo in March 2021. The model used available rock mechanics and geology data to interpret the domains required for the rock mechanics aspect. There are twelve lithological domains interpreted for the deposit and immediate surroundings. The shear zone "lithology" domain corresponds, in general, to the mineralized material.

The properties of the various structures and lithologies are discussed in detail in the following sections of the report. Data for some units, as for the faults, is limited. The observations made through the litho-structural model are useful to guide the next steps of the rock mechanical field campaign.

The interpreted domains are used to better estimate the dilution of the stopes, the optimal size of the underground openings, rock support requirements and other design consideration.

16.2.3 Rock Strength

The intact rock strength was estimated from laboratory tests carried out at École Polytechnique de Montréal in 2011. A total of 40 uniaxial compressive strength (UCS), 41 Brazilian (indirect tensile strength) and 40 triaxial strength tests were done. InnovExplo reviewed the results of these tests with the geomechanical domains. Using the drill hole identification numbers, along with the respective depth of each test, the Authors were able to identify the main lithology of each sample and its location. Moreover, the Authors were able to use the photographs of the tested core, combined with their experience and École Polytechnique's observations, to validate or reject the results depending on whether they showed intact or structural failure. The result was 24 validated UCS tests (see Table 16-1), 35 validated Brazilian tests (see Table 16-2) and 20 validated triaxial tests (see Table 16-3). The UCS done tests considered the deformation of the intact rock to evaluate the Young Modulus (E) and Poisson coefficient (ν).

Table 16-1: Results of Uniaxial Compressive Strength Tests

Lithologies	Tests Done	Valid Tests	UCS (MPa)		E (GPa)		v	
			Average	Standard Deviation	Average	Standard Deviation	Average	Standard Deviation
Intermediate Tuff with Lapilli	9	7	111.9	20.5	75.6	7.6	0.26	0.08
Andesite	17	9	136.4	41.6	76.5	13.7	0.27	0.03
Gabbro	2	0	N/A	N/A	N/A	N/A	N/A	N/A
Lower Porphyritic Rhyolite	1	1	135.6	0.0	75.1	0.0	0.20	0.00
Flow Breccia	4	4	134.3	41.0	84.9	8.4	0.25	0.03
Shear Zone	7	3	132.8	30.6	78.8	3.9	0.29	0.07

Table 16-2: Results of Brazilian Tests

Lithologies	Tests Done	Valid Tests	T0 (MPa)	
			Average	Standard Deviation
Intermediate Tuff with Lapilli	10	10	14.0	4.8
Andesite	16	14	11.3	2.6
Gabbro	2	1	17.7	0.0
Lower Porphyritic Rhyolite	2	1	11.3	0.0
Flow Breccia	3	2	12.5	1.9
Shear Zone	8	7	11.1	3.4

Table 16-3: Results of Triaxial Strength Tests

Lithologies	Tests Done	Valid Tests	Confining Pressure Tested (MPa)	
			σ3 Minimal	σ3 Maximal
Intermediate Tuff with Lapilli	9	5	5	15
Andesite	16	5	5	15
Gabbro	2	1	5	5
Lower Porphyritic Rhyolite	2	2	5	10
Flow Breccia	4	2	5	10
Shear Zone	7	5	5	15

16.2.4 Rock Mass Jointing

The rock mass jointing data was obtained from the acoustic and optical televiewer surveys carried out by Qualitas in 2013. The digital model created by InnovExplo shows that all the televiewer surveys were concentrated on the hanging wall of the deposit and in proximity of areas that are now mined out. The resulting 10,657 discontinuity measurements provide geomechanical information on the main joint sets, such as orientation, aperture and spacing. The rock mass jointing database was divided into three zones: "first 20 to 35 metres"; "middle"; and "20 metres before mine openings". The middle zone was the most noteworthy from the point of view of pre-mining rock mass conditions. A total of 3,169 discontinuities were utilized in the joint-sets analysis.

The location of the recorded discontinuities does not provide enough coverage to allow for an adequate interpretation and analysis of the joint sets for the entire deposit. Also, most of the boreholes have the same preferential orientation (i.e., perpendicular to the targeted mineralized zones), so any joint sets parallel to the axis of the drill holes would not have been properly considered in the analysis. Nevertheless, the available data were reinterpreted from the point of view of geomechanical domain for the Middle zone to guide the decision about stope sizing and ground support. The assumption that these data were representative of the various geomechanical domains was taken. The resulting stereonets are presented in Figure 16-2. A summary of joint sets by geomechanical domain is provided in Table 16-4.

Figure 16-2: Stereonet Plots of Main Structural Joint Sets

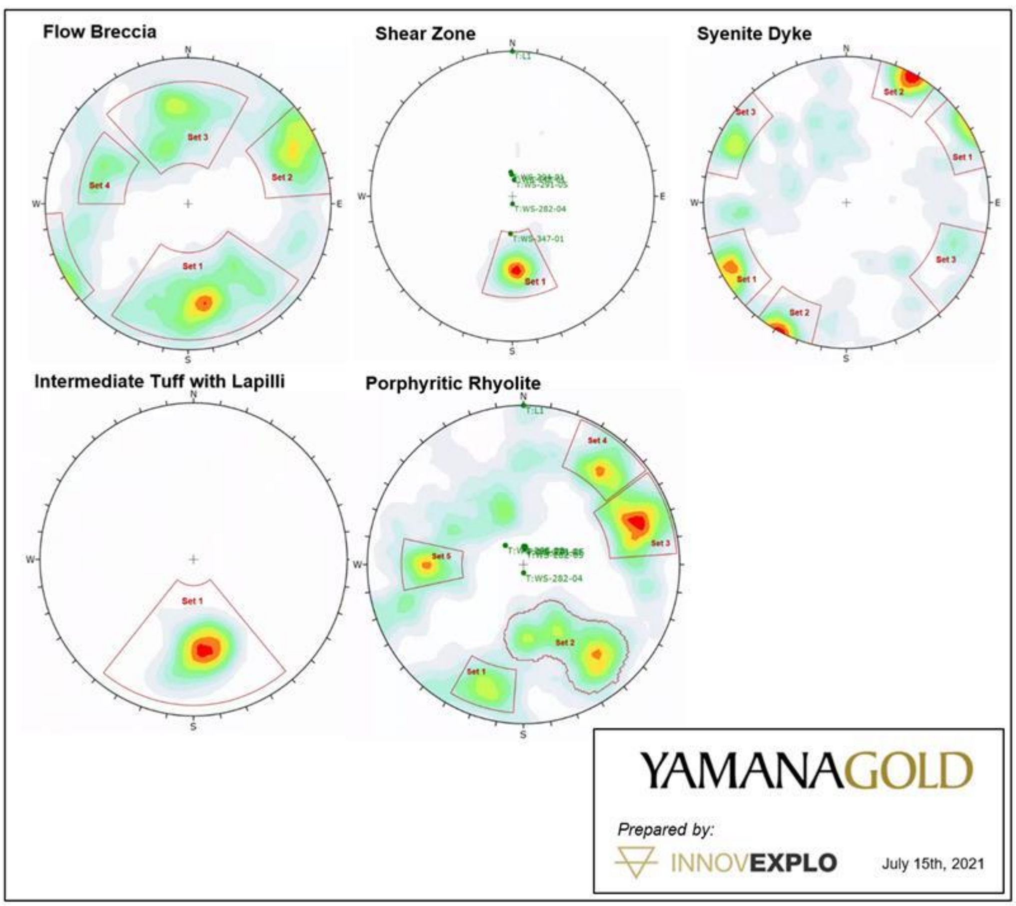


Table 16-4: Summary of Joints Sets by Geomechanical Domain

Lithologies	Number of Joints	Average Spacing (m)	Jn	
			Median	Standard deviation
Intermediate Tuff with Lapilli	474	0.8	3.00	0.94
Flow Breccia	1823	0.6	3.00	2.87
Shear Zone	221	1.0	3.00	0.97
Syenite Dyke	145	0.3	2.00	0.72
Porphyritic Rhyolite	506	0.6	2.00	1.86

16.2.5 Rock Mass Characterization

The rock mass characterization was updated for the current study based on the new litho-structural model. No geomechanical drilling campaigns or logging were carried out at Wasamac, so some parameters, such as Jn, Jr and Ja of the previous rock mass characterization, were based on the field visit review of available drill core, and photographs of drill core (2018 Feasibility Study; Itasca technical report). The rock-quality designation (RQD) data were obtained from a Yamana database of numerous drill holes.

Based on available data, (sometimes minor) and the current understanding of the rock mass, the rock mass was separated into four main geomechanical domains (Table 16-5). Lithologies with similar properties were merged to create these geomechanical domains. Others rock mass characterizations were estimated for minor geomechanical domains, such as the Horne fault, mafic dyke and flow breccia, but are not shown in Table 16-5, as they are used only for particular cases.

Table 16-5: Rock Mass Classification

Description	Footwall	Shear zone	Chlorite Schist	Hanging wall - General
Lithologies	Andesite and Gabbro	Shear zone	Chlorite schist	Tuff, Syenite and Rhyolite
RQD	82 (Good)	74 (Fair)	48 (Poor)	76 (Good)
Fracture Spacing	Unknown	1.0 m	Unknown	0.7 m
Joint Conditions	Jr = 1.5 Ja = 1.0	Jr = 1.5 Ja = 1.0	Jr = 1.0 Ja = 4.0	Jr = 1.5 Ja = 1.0
Number of Joint Sets (Jn)	3.00 (One joint set plus random joints)	2.24 (One joint set plus random joints)	2.24 (One joint set plus random joints)	2.39 (One joint set plus random joints)
Groundwater Conditions	Dry			
RMR_{1989}	77 (Good rock)	73 (Good rock)	60 (Fair rock)	77 (Good rock)
Q’	40.9	49.6	5.3	47.7
Q’ lower bound	3.6	3.4	1.3	3.6
GSI	72	68	34	69

16.2.6 In-Situ Stress

As no in-situ stress measurements have been measured at the Wasamac site, the information on in-situ stresses was estimated based on models available in the literature (e.g., Young and Maloney, 2015) and on measurements collected at comparable geological environments from the surrounding Wasamac project. As the in-situ stresses for the Rouyn-Noranda region are generally well understood, the assumptions shown in Table 16-6 were judged to be sufficiently accurate for the current feasibility study. The assumed stresses parameters are summarized in Table 16-6..

Table 16-6: In-Situ Stress Assumption

Stress	Orientation
σ_1 (MPa) = 2.4 σ_3	Horizontal trend on Azimuth = 356°
σ_2 (MPa) = 1.6 σ_3	Horizontal trend, normal to σ_1
σ_3 (MPa) = σv (MPa) = 0,03 z	Vertical
z = depth below surface (m)	

16.2.7 Anticipated Rock Mass Behaviour

The occurrence, persistence and characteristics of the rock mass fabric and the major geological structures (e.g., fault and shear zones), as well as their intersections, will control the rock mass behaviour around underground openings.

With depth or increased extraction of the mineralized zones, mining induced stresses are expected to concentrate around excavations and are anticipated to be generally observable as localized fracturing and light spalling in the footwall. Where mine excavations are oriented subparallel to the regional foliation, minor to moderate wall deformation may develop due to its anisotropy. In the chlorite schist domain, more deformation and failure of the intact rock is probable.

16.2.8 Stope Dimensioning and Dilution

An analysis of preliminary stope optimization shows the dominant geomechanical domain that should influence stope sizing. An example of that preliminary analysis is illustrated in Table 16-7 and indicates the rock unit influencing the dilution on the footwall and hanging wall of the stopes. The "Competent Rock" entry refers to the units on the footwall and hanging wall of the shear zone. A detailed analysis shows that the ''Competent Rock'' comprises mainly the andesite unit in mining area E29.

Table 16-7: Dilution Summary by Geomechanical Domains

Description	Hanging Wall	Footwall
Geomechanical Domain	%	%
Shear Zone	85%	72%
Chlorite Schist	8%	21%
Competent Rock	7%	6%
Total	100%	100%

After identifying the rock unit that most influences stope sizing and dilution, stope-sizing analyses were carried out for many scenarios including various depth, stope sizing and the dominant rock units.

Stope dimensioning was determined using a combination of the Stability Graph method (Potvin, 1988) for long-hole stoping; a review of the stability of historic Wasamac mine stopes, and the experience of the Authors with similar deposits. Analysis was constrained such that a minimum of 75% of the stopes had stable walls without additional ground support. A transition zone, without support, was accepted for the back of the stopes, as ground support will be present in the drifts. For the other stopes, secondary ground support is planned in order to maintain stability and to minimize dilution. The 75% factor is important in consideration of the variations of the rock mass conditions in the deposit. Table 16-8 presents the typical parameters used in the calculation of the stability number that was used to estimate dilution and stope dimensions based on the stability graph and ELOS method.

To make the mine design and planning easier and uniform, the stope dimensions shown in Table 16-9 were uniformly used in the study. A constant sub-level spacing of 25.0 m was selected to allow the flexibility to reduce stope length when the rock condition and stresses constraints are unfavourable (e.g., in the chlorite schist units). The proposed stope-sizing respected the stability criteria selected for the study for the deeper stopes planned. In other words, the main factors influencing the stability and the dilution of the stopes at the Wasamac Project should be the joint sets and the geomechanical units, not the induced stresses.

Table 16-8: Parameters Used to Determine Stability Number Used in Stability Graph and ELOS Analysis

Parameter	Q'	Factor A	Factor B	Factor C
Hanging Wall	Depending on rock mass and location. (Refer to **Table 16-5**)	1.0	0.3	4.5
Footwall		1.0	0.3	6.8
Sidewall		0.9-1.0	1.0	6.8
Back		0.1-0.5	1.0	2.0

Table 16-9: Stopes Dimensions Summary

Description		Transverse - Primary Stope	Transverse - Secondary Stope	Longitudinal
Sub-level spacing (m)		25	25	25
Andesite	Maximum Stope Width (m)	20	20	20
	Stope Length (m)	20	30	30
Shear Zone	Maximum Stope Width (m)	20	20	20
	Stope Length (m)	20	30	30
Chlorite Schist	Maximum Stope Width (m)	10	10	10
	Stope Length (m)	10	15	15

The maximum hydraulic radius of the hanging wall of the stopes is limited to 8.3 m. For comparison, the stopes of the historic Wasamac mine workings were mined with an average hydraulic radius of 10.7, which had led to stability and dilution challenges. The dilution estimates were based on the experience of the Authors with similar deposits, mining methods, and blasting methods. The dilution was also estimated by the ELOS method described by Clark (1997). The proposed value assumed good blasting practices and a good understanding of the rock mass to reduce the stope length or add secondary ground support where required. The estimated dilutions for the stopes are shown in Table 16-10.

Table 16-10: Dilution Estimation

Description	Footwall (ELOS value) (m)	Hanging Wall (ELOS value) (m)
Andesite	0.25	0.50
Shear Zone	0.25	0.50
Chlorite Schist	0.50	1.00

16.2.9 Sill Pillars and Mining Sequence

Sill pillars will be temporarily left in place and will serve to increase the number of production centres in concurrent locations. The sill pillars are designed to 20 m vertical height, similar to the regular long-hole stope height, and will be placed typically at 125 m or 150 m vertical intervals, the equivalent of a five or six vertical stope progression. The vertical interval will vary depending on the mine planning and the eventual centre of production. The sill pillars will be recovered as mining progresses upward.

Since, in many cases, the mining sequence will follow a longitudinal retreat toward the centre of the mineralized zone, the sill pillar dimensions will be reduced, causing an increase of the induced stresses in its centre. Additional ground support and potential drift rehabilitation could be required in the ore drift at sill level. Considering these aspects, the recovery for stopes mined in the sill pillar was reduced to 86% with 25 m floor to floor uppers.

To reduce the induced stress between two centres of production, the centres will be mined in a chevron shape to avoid any stresses concentration where possible. For example, in the case of two longitudinal stoping centres of production side by side, the induced stresses will increase when mining became closer to the drawpoints and closer to the sill pillar. For the transverse stoping, mining will start at the centre of the mining horizon and thereby distribute the induced stresses to the border areas. These typical mining sequence strategies have the benefit of minimizing induced stress challenges and being able to more easily anticipate and plan possible challenging sectors of the mine. The location of the sill pillar was optimized to ensure that minimal induced stresses will affect main infrastructures and levels.

16.2.10 Stand-Off Distance

In some parts of the mine, mineralized material remains close to the old Wasamac stopes. As explained in Chapter 14, "Two 5 to 20 m “mined-out” buffer volumes were built in the western (historically known as the Main Zone) and eastern part (historically known as Zone 1) of the new Main Zone". In addition to that buffer, an additional pillar of 10 m was added between the historical stopes and the new ones. So, a total pillar of 15 to 30 m was used between most of the historical stopes and the current ones. At these locations, the historical stopes have a typical width between 5 m and 15 m. In summary, the pillar between the historical stopes and the new ones would correspond to a ratio (pillar height/stope width) around 2:1 and should be stable based on empirical data for these types of pillars.

It is planned to mine the stopes close to the old workings towards the end of the mine life. That delay allows time to improve the understanding of the rock mass, drill and blast approaches and other production activities. At that point, it is assumed that it will be possible to mine these stopes without important challenges.

16.2.11 Crown Pillar Stability

The crown pillars criteria were estimated using the empirical scaled span method (Carter, 2000). Based on the current mine plan, the closest stopes to surface have a crown pillar around 115 m. Based on the analysis done, the crown pillar stability should not be an issue.

16.2.12 Other Pillars and Criteria for Design

Numerical analysis was carried out to confirm design criteria with RS2 from Rocscience. Based on these simulations, a minimum pillar of 40 m between the haulage drift and the stopes is recommended. That pillar should minimize the disturbance of the haulage drifts and closest infrastructures, while maintaining a relatively short drawpoint. Other design criteria were provided for the design based on empirical data and common rules of thumb used in similar operations.

16.2.13 Ground Support

16.2.13.1 Ground Support Required for Development

Anticipated ground support requirements for development are summarized in Table 16-11. The preliminary ground-support templates were developed from common empirical support practice, taking excavation dimension, and the anticipated stress and structural influences into account. Other drift sizes are planned for the project but are not summarized in Table 16-11 as they are not major ground support patterns. These other drifts size (e.g., 5.75 x 7.3 and 7.0 x 5.0) used similar patterns to the ones explained in Table 16-11 for standard design. The recommended ground support is also the same for the ramps and haulage drifts.

Table 16-11: Typical Ground Support for Development

Openings	Dimensions	Planned Ground Support
Start-up (Permanent)	5.75 mH x 5.75 mW (Ramps and access drifts) And 5.3 mH x 5.0 mW (Haulage drift)	Back: 2.4 m fully resin grouted rebar (1.5 m x 1.5 m) with 1.8 m galvanized split set in a dice pattern, Wall: 1.8 m galvanized split set (1.5 m x 1.5 m), Screen: #6 gauge, galvanized welded wire mesh to within 1.8 m of floor Face: 1.8 m split set (1.5 m x 1.5 m)
	4.5 mH x 4.5 mW (Conveyor drifts)	Back: 1.8 m fully resin grouted rebar (1.5 m x 1.5 m) with 1.5 m galvanized split set in a dice pattern, Wall: 1.5 m galvanized split set (1.5 m x 1.5 m), Screen: #6 gauge, galvanized welded wire mesh to within 1.5 m of floor
Standard design (permanent)	5.75 mH x 5.75 mW (Ramps and access drifts) And 5.3 mH x 5.0 mW (Haulage drift)	Same as start-up but pattern converted to 1.2 m x 1.2 m
	4.5 mH x 4.5 mW (Conveyor drifts)	Same as start-up but pattern converted to 1.2 m x 1.2 m
Standard design (temporary)	5.3 mH x 5.0 mW (Ore drifts and drawpoints)	Back: 2.4 m fully resin grouted rebar (1.2 m x 1.2 m) with 1.8 m split set in a dice pattern, Wall: 1.8 m split set (1.2 m x 1.2 m), Screen: #6 gauge, galvanized welded wire mesh to within 1.8 m of floor Face: 1.8 m split set (1.2 m x 1.2 m)
Intersection	Span < 7.0 m	Primary support: As per development support outlined above
	7.0 m < Span < 9.0 m	Primary support: As per development support outlined above Secondary Support: 3.8 m SuperSwellex (2.0 m x 2.0 m)
	9.0 m < Span	Primary support: As per development support outlined above Secondary Support: 5.0 m cables (2.0 m x 2.0 m)
	Note	Replace SuperSwellex by cables in permanent haulage drifts due to proximity to production activities. Replace cables by SuperSwellex in temporary intersection.

Different ground support patterns are planned between the start-up of the project (mainly the development of the two access ramps and some infrastructures), and the remainder of the project. A lighter ground support is planned to accelerate the development cycle time at the beginning of the project. This ground support during the start-up seems reasonable considering the good ground conditions estimated for the footwall of the deposit. Also, the drifts developed during the start-up will be further from the production activities, so would be slightly less affected by mining-induced stresses and rock mass deterioration. If additional ground support is required in some part of the development to ensure long-term stable excavation, it will be installed after the critical development cycle has been completed.

Note that the ground support requirements should be reassessed once the development layout is finalized, and when the local rock mass conditions and behaviour are confirmed from underground lateral development. For example, if the rock condition is better than estimated, split sets at the back could be replaced by 0.5 m split set, to pin screen to the rock.

In addition to Table 16-11, 0.5 m split sets are planned when required to affix screening to rock faces, unless other criteria are mentioned. In the presence of wide fault zones, fibre-reinforced shotcrete (minimum 50 mm) is recommended to be added to the primary support.

For excavations larger than 7.0 m (e.g., the paste plant and garage), 5.0 m cables on a 2.0 m x 2.0 m pattern will be applied at the back as secondary support. For permanent infrastructures, e.g., where employees will work on a daily basis, wire mesh will be extended close to floor level and 50 mm shotcrete added to the back and the walls.

16.2.13.2 Secondary Ground Support Required for Stopes

Considering variation in the rock mass properties and mining sequence that may damage some sectors due to mining-induced stresses, it is estimated that 25% of the stopes will need secondary ground support to stay stable and keep a low level of dilution. This ratio is based on stope dimensioning analysis, preliminary results of numerical analysis of the sequence, and experience with similar operations. The secondary ground support templates were developed from common empirical support practices, considering stope dip, stope dimension and the anticipated stresses and structural influences. These empirical criteria provided an estimation of the density and length of the cables required for the hanging wall and back of stopes. These were converted to an "average tonne/cable metres required" ratio, as shown in Table 16-12.

Table 16-12: Typical Cable Quantity Necessary as Secondary Ground Support for Stopes When Required

Geomechanical Domain	Stope Type		Tonne / Cable Metre Ratio	
			Less than 8 m Wide	More than 8 m Wide
Andesite	Longitudinal stoping		17	34
	Transverse stoping	Primary	NA	26
		Secondary	NA	29
Shear Zone	Longitudinal stoping		17	34
	Transverse stoping	Primary	NA	26
		Secondary	NA	29
Chlorite schist	Longitudinal stoping		20	24
	Transverse stoping	Primary	NA	28
		Secondary	NA	24

The secondary support is mostly planned for stopes in chlorite schist, secondary transverse stoping, and stopes in the sill pillars. Some random stopes were also planned to be cabled in consideration of the lower rock mass properties in some parts of the mine.

16.2.14 Estimation of Backfill Strength Requirements

The backfill strength requirements were assessed using empirical criteria and numerical analysis made in RS2 finite elements code software. Table 16-13 shows the strength required for vertical exposure for various stope conditions. Table 16-14 illustrates the results of the analysis for underhand mining or, in other words, when it is planned to mine a stope under a backfilled one. For that situation, to minimize the backfill cost, it is recommended to create a pillar (sill mat) of the required strength at the bottom of the stope to utilize the strength required for the vertical exposure for the upper part of the stopes. For underhand mining, a lower recovery at the top of these stopes is planned to provide an additional support to the backfill.

Backfill barricade will be made with a combination of rockfill and shotcrete. They will provide a sufficient confining capacity to being able to fill the mining voids with paste. Barricades and more are discussed in Section 16.5.4.1.2..

Table 16-13: Paste Fill Strength Requirements for Vertical Exposure

Mining Method	UCS (MPa)
Longitudinal stoping - Less than 8 m wide	0.20
Longitudinal stoping – 8 m to 15 m wide	0.30
Longitudinal stoping – 15 m to 20 m wide	0.40
Transverse stoping - Primary	0.40
Transverse stoping - Secondary	0.50

Table 16-14: Paste Fill Strength Requirements for Underhand Mining

Mining Method	UCS (MPa)	Sillmat Height (m)
Longitudinal stoping - Less than 8 m wide	0.5	4
Longitudinal stoping – 8 m to 15 m wide	1.2	8
Longitudinal stoping – 15 m to 20 m wide	1.5	10
Transverse stoping	1.5	10

When CRF is to be used to fill the mining voids, similar strength and backfill requirements are planned but detailed analysis will be required to validate these requirements. In the case of vertical exposure, only part of the stope is required to be filled with CRF. The other part of it could be filled with rockfill. That strategy minimizes the cement cost, while maintaining backfill stability. The rockfill proportion to be used in combination with CRF to backfill the mining voids varies between 32% to 53%, depending on the stope geometry and sequence.

16.3 Hydrogeology

The hydrogeological aspect of the current study has been conducted by Hydro-Ressource Inc. In 2014, a hydrogeological study was carried out to better understand the flow at site (Richelieu Hydrogéologie Inc., 2014). The following work was completed:

- gathering existing information, including SIH governmental system
- drilling of shallow drill holes in the overburden and the upper bedrock
- testing the new holes drilled
- testing some existing exploration drill holes with packers
- pumping the existing workings through a connecting well
- carrying flow simulations using for inflow prediction

Among the work that was done, the new drill holes were shallow and average 23 m in length. Their utility is limited and mostly allowed for water table elevation measurements and groundwater sampling.

Among the nine exploration drill holes that were tested, most of the packer tests were carried out in the shallow part of the rock formation. Based on current results, the first few metres of the rock formation appear to have higher hydraulic conductivities, which is a common in the Canadian Shield due to the Ice Age glacial effects, freeze-thaw cycles and surface alteration. Therefore, there is no clear trend suggesting higher permeability at shallow depth in the remaining part of the rock formation.

The mineralized zones are associated with the Francoeur-Wasa shear zone. This shear zone and other potential transversal faults have not been characterized through the packer campaign. The packer tests have all been carried out in the hanging wall part of the main shear zones. Hydraulic conductivity values range around 10-8 to 10-9 m/s.

A pump test was carried out twice in an existing shaft. In December 2013, the pump test showed a flow rate of around 515 USgpm. The drawdown observed reached about 20 m after ±5 days of pumping.

A flow simulation model was prepared using Visual Modflow, a finite difference software developed in 1988 by the USGS. The version of Visual Modflow that has been used does not allow for the incorporation of discrete faults, except by multiple simplifications: the shear zone has been modelled with large cells. Three distinct zones have been considered in the model: the Francoeur-Wasa shear zone, the hanging wall, and the footwall. The values of hydraulic conductivities in the model range from 10-8 to 10-11 m/s, which are lower than the observed values. The values applied to the shear zones are closer to the one measured, even though they were not measured in the shear zone. The lower K values suggest an under-estimation of the inflow for prediction. The model has been calibrated in steady state and has not been calibrated in the transient state using the pump test results.

Simulation results obtained from the Modflow model suggest an inflow of 209 USgpm. This number seems underestimated considering the low K values used in the model and the absence of discrete faults. Also, this model has been prepared with a smaller footprint of the future mine (as compared to the updated mine plan), which can induce a bias in the prediction of the inflow. To overcome this situation, an inflow of 420 USgpm was recommended to be used for dewatering engineering (pumps and infrastructures).

Additional investigations are required for operational purposes. Among other things, the main shear zone must be characterized more precisely, and other potential transversal faults have been identified with the updated geological model, which could induce higher flow in some areas of the future mine. Once additional data has been gathered, the model should be updated accordingly, and new predictive calculation should be made.

16.4 Mine Design

The Wasamac project will utilize optimized mining methods and mining sequences, utilizing a combination of longitudinal and transverse stoping with backfill. The project is designed as a modern underground operation with a small footprint and minimal surface infrastructure to the south of Highway 117. Tailings will be deposited underground as paste fill and in a filtered dry stack tailings storage facility, approximately 6 km northwest of the processing plant. Waste material will also be used as rockfill.

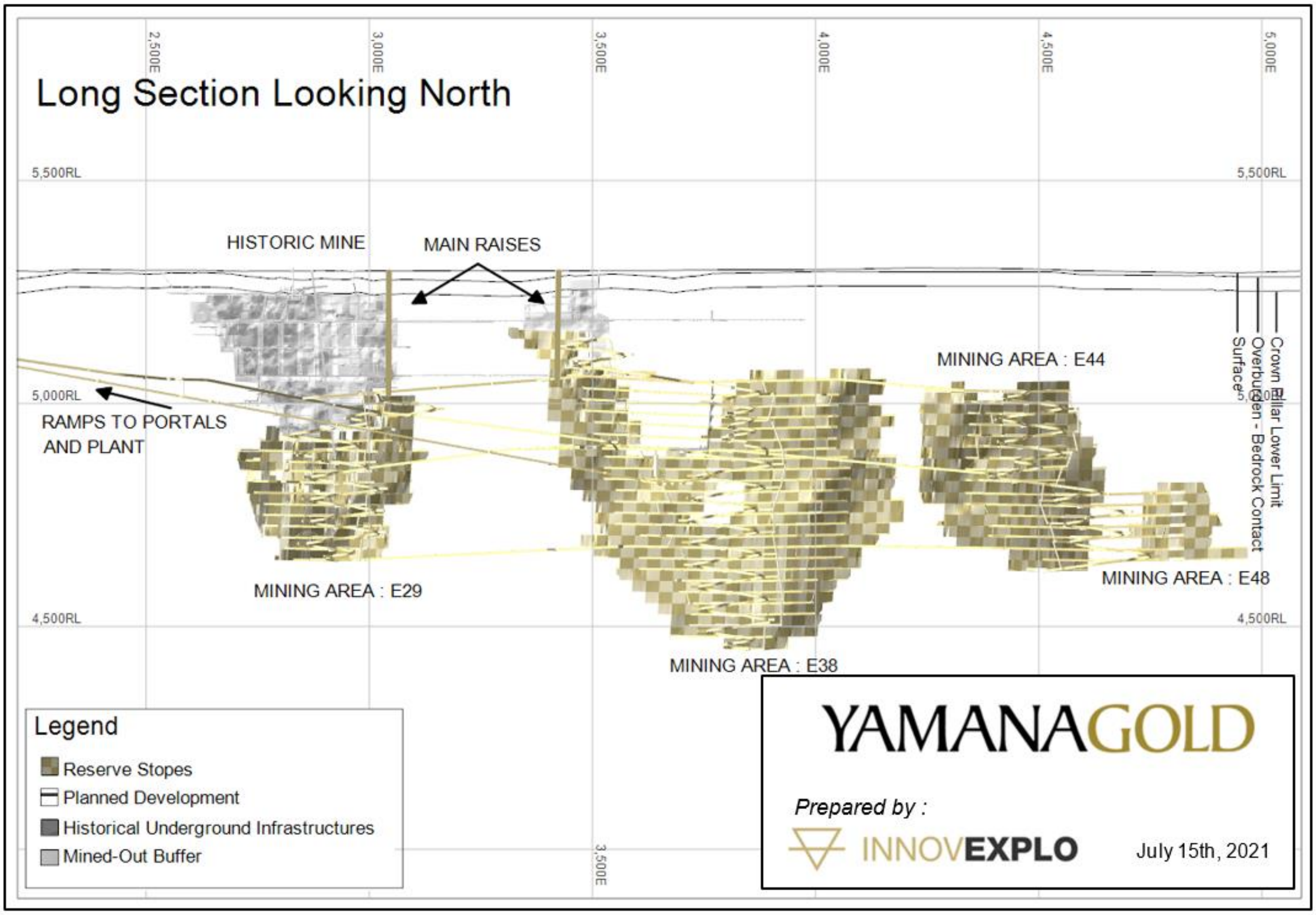
The new project is built around and below the historic Wasamac mine. Mine dewatering, waste management and pillar evaluation are all aspects that needed to be considered while designing the new mine around the old workings. To reduce the project's surface footprint, most infrastructures will be installed underground. A main hub containing the main crusher and an underground conveyor to surface will ensure ore output from the mine, while minimizing equipment activities on surface. A full paste fill plant is also planned for underground, using mill tailings and cement. These underground installations minimize surface structures, while maximizing waste material and tailings returned underground.

Levels are spaced every 25 m and connected by decline ramps; levels include all the necessary infrastructures required for large-scale mechanical long-hole stoping. Two main levels will be used as centres of operation for major infrastructures like the service bay and main hub; they also provide connecting access ramps between all zones.

Fresh air will be supplied to the mine by two ventilation raises, with high-efficiency fans installed underground. The ventilation shaft housings and the cement discharge at surface are few of the only infrastructures south of Route 117.

Figure 16-3 presents an overview of the project at completion.

Figure 16-3: Mine Overview at Completion



16.4.1 Main Infrastructure

Most major infrastructures will be located underground and centralized in mining area E38. This includes the service bay, the main crusher, the 7,500-tonne ore bin, the main conveyor to surface, the main dewatering system and the paste fill plant.

Most major infrastructures will be located on the two main transfer levels (L425 & L650) between mining areas E29, E38 and E44. The main ventilation fans and paste fill plant are designed to be located on L275, near the top of mining areas E29 and E38.

The first main level to be developed is L425, which includes the service bay, parking area (with access to 600 V for electric equipment), powder and cap magazine, and the grizzly and rockbreaker room.

16.4.1.1 Service Bay

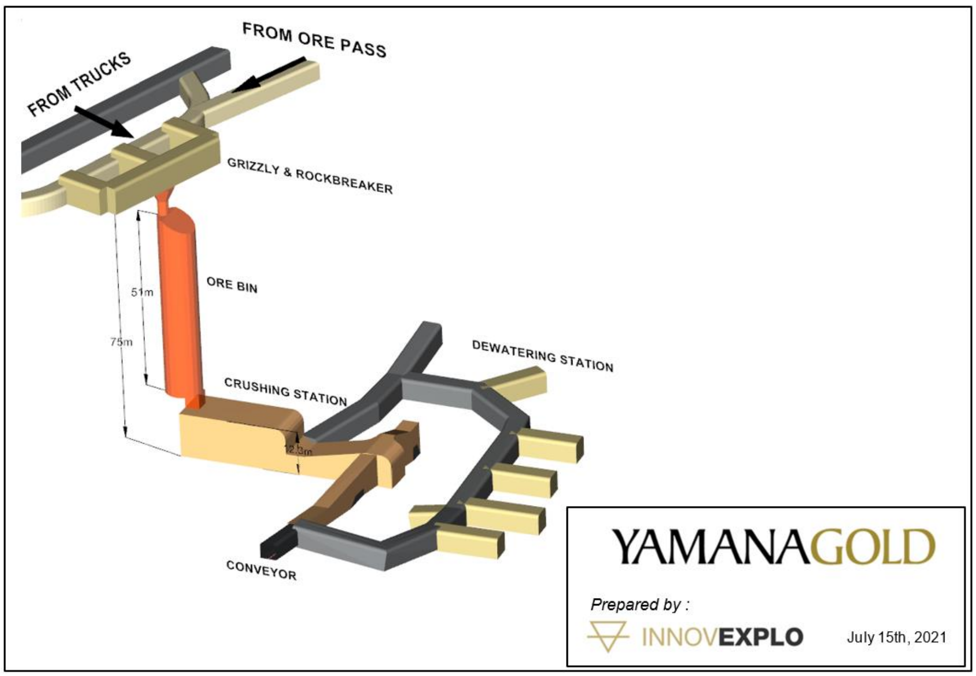
The service bay will have an access area, welding bay, garage, tires storage, washing bay, warehouse, greasing bay, fuel bay and parking. The service bay will be located on L425 between mining area E29 and E38. The garage will be able to accommodate up to three large pieces of equipment at the same time as trucks, and the parking area will have room for at least 10 vehicles.

The overall service bay area will have a total volume of 25,000 m^3 for a total of 750 equivalent metres.

16.4.1.2 Crushing Station and Main Dewatering Station

An overview of the crushing station is shown in Figure 16-4.

Figure 16-4: Crushing Station Overview



The main hub is located between L425 and L500. The full rockwork includes a rockbreaker at the grizzly level (including two dumping points), a 7,500-tonne circular ore bin (10 m diameter), a crushing station, and conveyor transfers. Considering the main hub is the primary ore output for the mine, it is logically situated near the centre of mass of the deposit. The grade of the conveyor ramp is 17% and will allow fast and direct access to the crusher station rockworks. The objective of having the main hub infrastructure underground is to limit surface impact by minimizing unnecessary equipment and excessive noise and dust emission.

Alongside the crushing station, on the same level, is the main dewatering station. Details of the dewatering system are outlined in Section 16.7.6.

16.4.1.3 Paste Fill Plant Location

The paste fill plant, located at L275, and paste fill operation are detailed in Section 16.5.4. The paste fill plant location has been reviewed and optimized since the 2018 Feasibility Study; the underground location near the top of area E38 is the best option for reducing surface infrastructure, and providing quick access and overall paste fill distribution efficiency. The paste fill plant will use tailings directly from the mill through a system of pipes in the main service ramp (reducing tailings output at surface). A ~6 hours of storage tank is planned at the paste plant for tailings. Cement will be dumped by truck directly into a borehole at surface, ensuring constant component availability.

16.4.1.4 Main Fans Location

The main fans are located at L275 between mining areas E29 and E38. The ventilation network is detailed in Section 16.8. These locations ensure rapid access to the main ventilation network and the resulting surface breakthrough for both main raises on Yamana properties. The location is optimal to easily ventilate the main mining areas E29 and E38, while the connecting drift will allow ventilation of the farthest zones E44 and E48. Also, the raises are positioned in order to avoid any unnecessary development in the Francoeur-Wasa shear zone. This will allow for long-term stability of the raises, with minimal ground support. The location is also beneficial in the event that potential mineral reserves are converted between mining areas E29 and E38.

The main fans set-up includes excavations for the fans and electrical and maintenance bay. The excavations hosting main fan designed to be approximately 10.6 wide x 5.0 m high x 60.0 long. The bottom of each shaft is also designed to minimize pressure loss by eliminating abrupt changes in direction (blasting brow and backfilling where needed).

16.4.1.5 Additional Infrastructure

Additional infrastructure includes emergency underground refuge stations, powder and cap magazines, ore passes, and internal ventilation raises. There are two instances of powder and cap magazines, both situated on the two main levels L425 and L650, which can easily accommodate the explosive requirements of the project. The powder and cap magazines have been designed with an area of 96 m2 and 40 m2, respectively, with room spare for material manipulation and to comply with all federal and provincial requirements.

The main ore pass is in the mining area E38 upper-eastern zone, between levels L275 and L425. Fingers are connected at each intermediate levels and include cone plug systems to control ore and air flows. The ore pass design is discussed in Section 16.4.2.

Each underground refuge station is designed and located to accommodate the necessary number of workers at any given time. The refuges are located closer than the required 1,000 m to ensure no delays in the development sequence. Depending on the volume of activities in any mining area, one of the three refuge station types is designed to accommodate the correct number of workers (12, 18 or 24 workers). All amenities will be found in the refuges to serve as a lunchroom: tables, chairs, washing station, lunch supplies, long-term evolution (LTE) connection, etc.

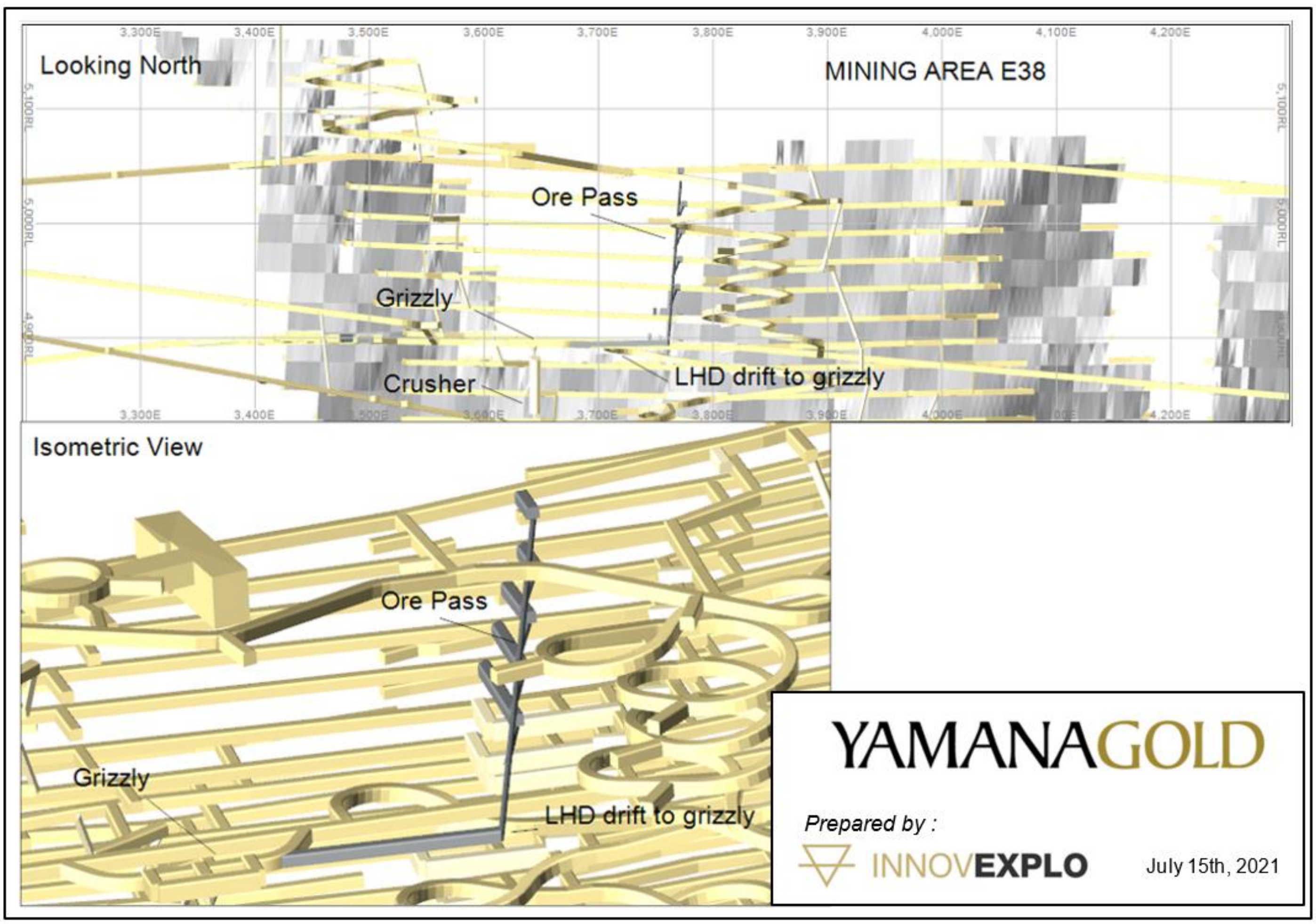
16.4.2 Ore Handling System

Ore and waste are hauled by LHDs from the production area to either a remuck or a loading bay near the access drift and then loaded into trucks to be hauled to the main grizzly and rockbreaker station (L425). The production is based on the automation potential of the equipment between shifts to ensure maximal ore output throughout life of mine; guaranteeing a continuous mineral reserve in the ore bin located under the grizzly.

Material from the ore bin is crushed and transported by conveyors to the surface stockpile. Including the crusher's output conveyor, there is a series of three conveyors hauling ore to surface. The system also includes two underground transfer towers (L500 and L75). The waste is hauled by trucks to surface or to empty stopes for backfill requirements as needed. The overall hauling waste objectives are to minimize the waste moved to surface and optimize trucks cycles times and efficiency.

Possible locations for ore passes have been evaluated for all mining areas above main level L425, but the only mining area with enough tonnage to be more profitable than the automated fleet is E38 upper levels. Development of one main ore pass (2.4 m diameter) will facilitate the ore handling from L275 to Main Level L425. Ore will be fed through four finger raises (1.8 m x 1.8 m), which will be capped by a cone plug system to minimize ventilation recirculation. Ore will be re-hauled by LHD from a chain control system on L425. LHD will tram 130 m directly to dump in the main grizzly. Figure 16-5 shows the ore pass configuration selected for the project.

Figure 16-5: Ore Pass Overview



The main ore handling system is an automated diesel fleet (trucks and LHD). The equipment pairing retained is the TH663i (63 tonnes) with the LH621i (21 tonnes), supplied by Sandvik. The chosen models are ready for optimal automation and provide the best cost per tonne compared to equivalent equipment.

16.4.3 Mine Design Criteria

Permanents drifts (ramps and access drifts) are generally 5.75 m wide by 5.75 m high, whereas the ore and waste drifts not used by trucks are 5.3 m wide by 5.0 m high. The following subsections describe the different types of rockworks and heading (e.g., main ramps, typical level, loading bay, and emergency egress). Various development parameters are summarized in Table 16-15, whereas the general pillars set by rock mechanics are listed in Table 16-16.

Remucks are generally spaced every 150 m for development efficiency.

Table 16-15: Mine Design Parameters

Development Heading	Width (m)	Height (m)	Gradient
Main Ramp	5.75	5.75	15% maximum
Conveyor Ramp	4.50	4.50	17% maximum
Access	5.75	5.75	2.0%
Drift (Ore & Waste)	5.30	5.00	2.0%
Loading/Unloading Bay	5.75	7.30	2.0%

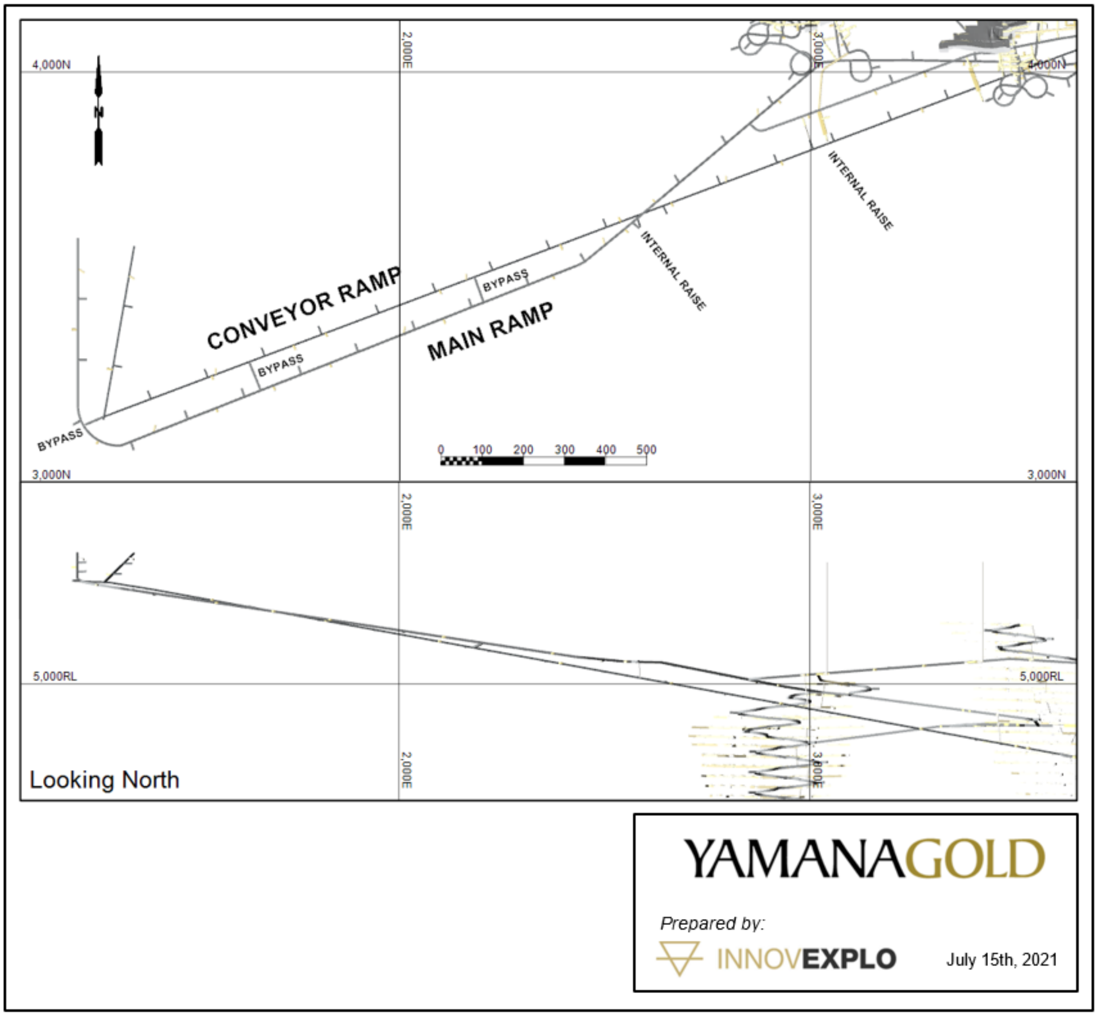
Table 16-16: Mine Design Pillars

Pillar Type	Minimal Distance (m)
Ramp / Stope	50
Haulage Drift / Stope	40
Raise / Stope	40
Ramp / Drift	15
Drift / Drift	10
Raise / Ramp	30
Raise / Drift (Ore Pass, Ventilation Raise)	30
Raise / Drift (Drop Raise)	20

16.4.3.1 Main Ramps

The main service ramp (see Figure 16-6) width and height is 5.75 m x 5.75 m at a maximum gradient of 15%. Secondary electrical stations are excavated every 300 m, which is the maximum conservative distance without loss of charge. Sumps are positioned every 500 to 600 m and will only be used during development. The conveyor ramp has the same infrastructure specifications as the main service ramp, except it will be 4.5 m x 4.5 m and have a maximum gradient of 17% to facilitate access to the main crushing station. As they will eventually be used for emergency egress, the minimal distance of 30 m between the ramps is respected in areas that are considered to be exit pathways.

Figure 16-6: Ramps Overview



Three horizontal bypasses will be developed to facilitate proper ventilation of the access ramps and diminish pressure challenges during pre-production. Three additional ventilation raise bypasses are also planned along the conveyor ramp that will also be used to facilitate ventilation of the two main ramps.

Depending on the phase of the project, both access ramps will have their own sets of associated services.

During pre-production (before commissioning of main fans), the main service ramp will have 6″ pipes for water and compressed air, and a 4″ pipe for pumping water to surface. A rigid ventilation duct will only be required when the ventilation will not flow naturally to parts of the drifts during their development. During pre-production, the conveyor ramp will have the same layout as the main ramp, except with two parallel 48″ ventilation ducts for temporary ventilation.

During production, the rigid ventilation ducts will be removed and reused elsewhere. A 10″ pipe will be installed in the main ramp to feed the paste fill plant with mill slurry from the surface. The conveyor ramp will be equipped with a 6″ pipe to pump clean water from the main dewatering station to surface. For details process water, refer to Section 16.7.8.

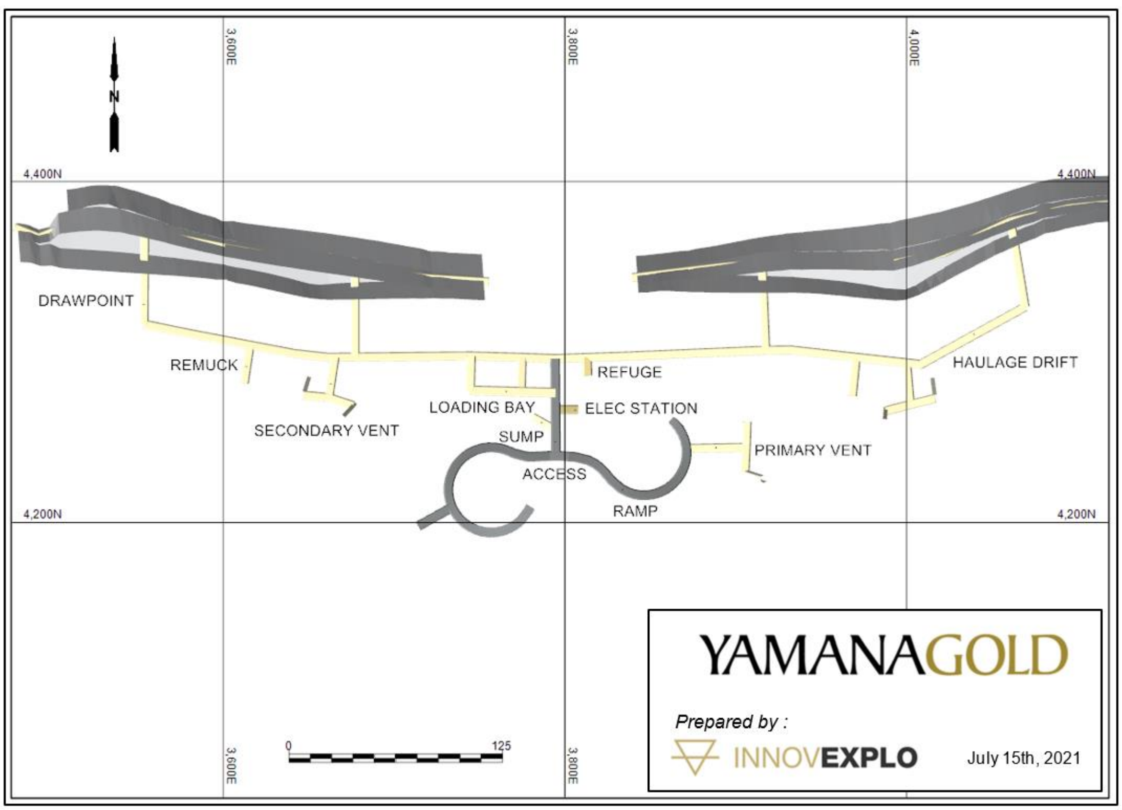
To maximize production through automation, the ramps between levels will have a minimal turning radius of 25 m, where possible (in some instances, due to deposit shape or design, the minimum turning radius will be 20 m). To minimize maintenance and operator fatigue, ramps are designed in an "8" shape as much as possible. A remuck bay is also planned between every level for development efficiency.

There are three sets of connecting ramps between mining areas E29, E38 and E44. Connecting ramps in the upper parts of the mine at L275 are primarily used for ventilation and paste purposes, with the two main fans located between E29 and E38. The remaining connecting ramps join the three main mining areas through the main levels L425 and L650. The grades of the connecting ramps have been designed to optimize trucks haulage and to facilitate mine dewatering and pumping. Major infrastructures are planned on the two main connecting ramps (L425 and L650).

16.4.3.2 Typical Level

A typical production level includes an access drift, a sump, a primary/secondary electrical station, a loading bay, a ventilation access (connected to the level or the ramp), and haulage drifts and ore drifts (see Figure 16-7). Depending on location, it can also include a refuge, an ore pass access (mining area E38), and other major infrastructure.

Figure 16-7: Typical Level (Mining Area E38)



A sump will be excavated roughly at 60° and -15% from the access to facilitate mucking of mud excess. Each level has at least one secondary electrical station; main electrical stations are positioned every five or six levels.

The access drifts and loading bay (5.75 m x 5.75 m) are used by trucks and LHDs, whereas haulage drifts will only accommodate LHD (5.3 m x 5.0 m). Waste haulage drifts are typically offset by at least 40 m from the ore body to respect pillars set by rock mechanics analysis. Drawpoints are generally spaced every seven stopes (210 m) for longitudinal mining, whereas transverse mining requires a drawpoint for every stope panel. Developments are designed to respect the 2% minimal gradient to facilitate water runoff to level sump.

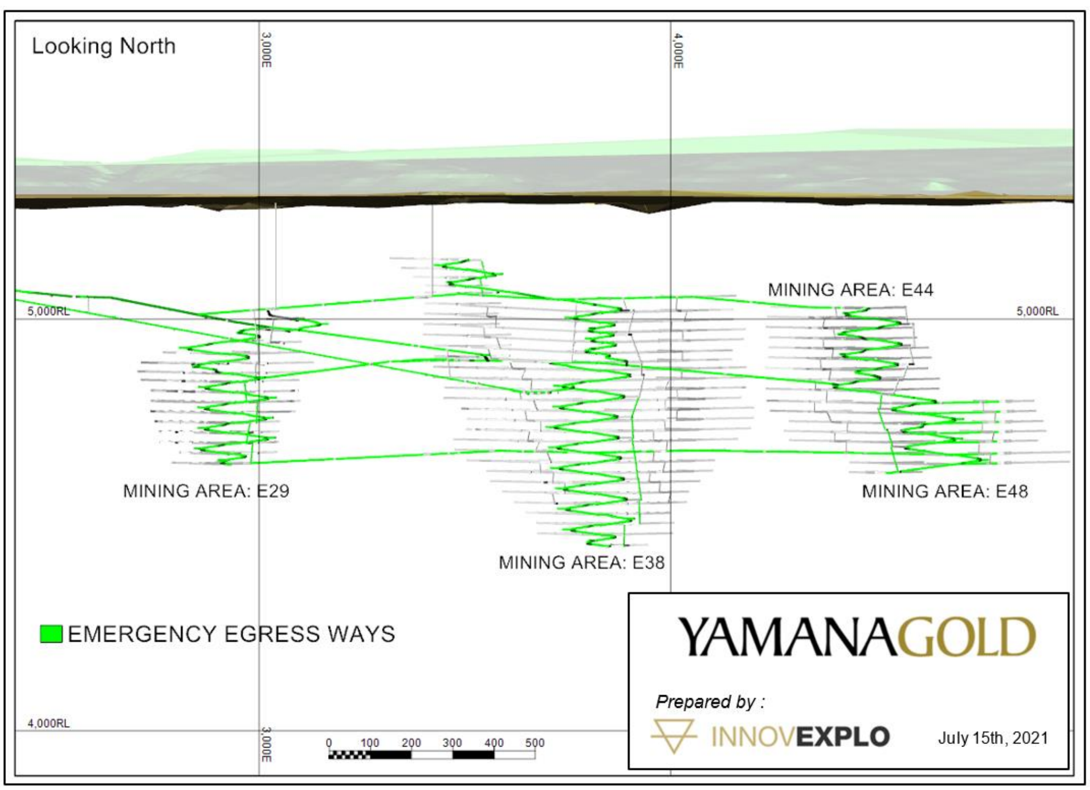
16.4.3.3 Loading Bay

Loading bays are designed to optimize automation of the truck and LHD fleets. A split-level loading concept has been used for the design, which entails excavating a higher drift for LHDs dumping, maximizing visibility and fill. The double-access shape ensures flexibility by having the option to dump backfill or ore in the second access. Supplier input and past experiences have proven that this type of layout is optimal to maximize productivity.

16.4.3.4 Emergency Egress

By the end of the mine life, as all mining areas will be connected by multiple ramps, so different exits for each mining area will be developed (See Figure 16-8). To start production as early as possible in some mining area, some of the internal ventilation raises will need to be outfitted with manways. This will add flexibility to the production sequence, while also multiplying egress routes for additional safety. The two main access ramps will be considered as the two emergency egress routes out of the mine.

Figure 16-8: Emergency Egress Routes – Overview



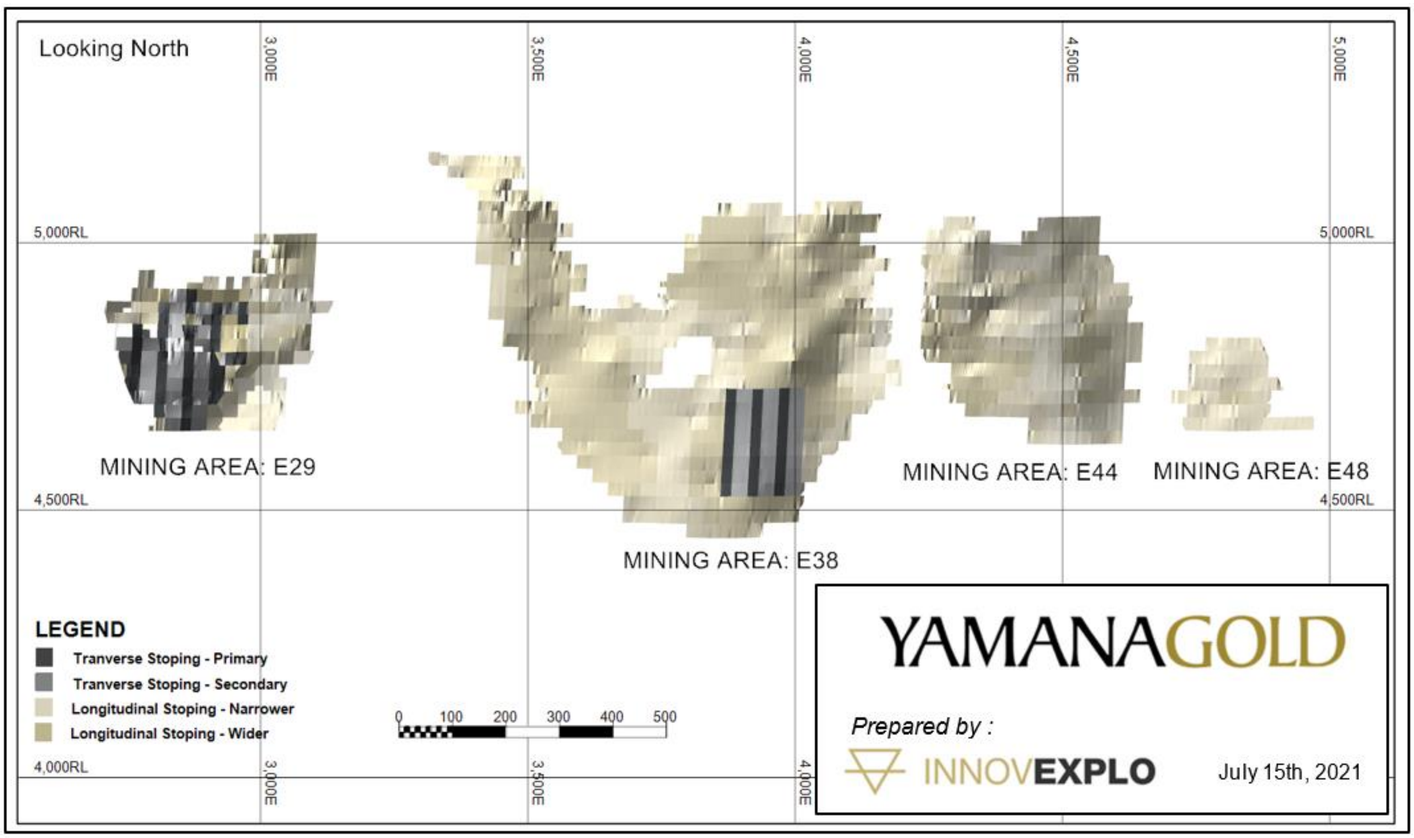
Manways are installed in 4.0 m diameter raises, which are also used as primary ventilation. Manways will be needed as follow:

- mining Area E29: from the lowest level (L675) to the L475
- mining Area E38: from the lowest level (L875) to the L525, and between L275 and L175 (no secondary access)
- mining Area E44 & E48: from the lowest level (L700) to the L525

16.5 Mining Methods

Mine development at the Wasamac Project will employ numerous production fronts to maximize productivity and flexibility to reach the targeted 7,000 t/d rate. Two main long-hole mining methods will be employed: longitudinal and transverse (see Figure 16-9). The transverse stoping is mostly concentrated in mining area E29, the widest zone of the mine. Mining areas have individual production centres based on the main mining method of each sector. Mining of each production centre will ascend from the lowest to the highest level.

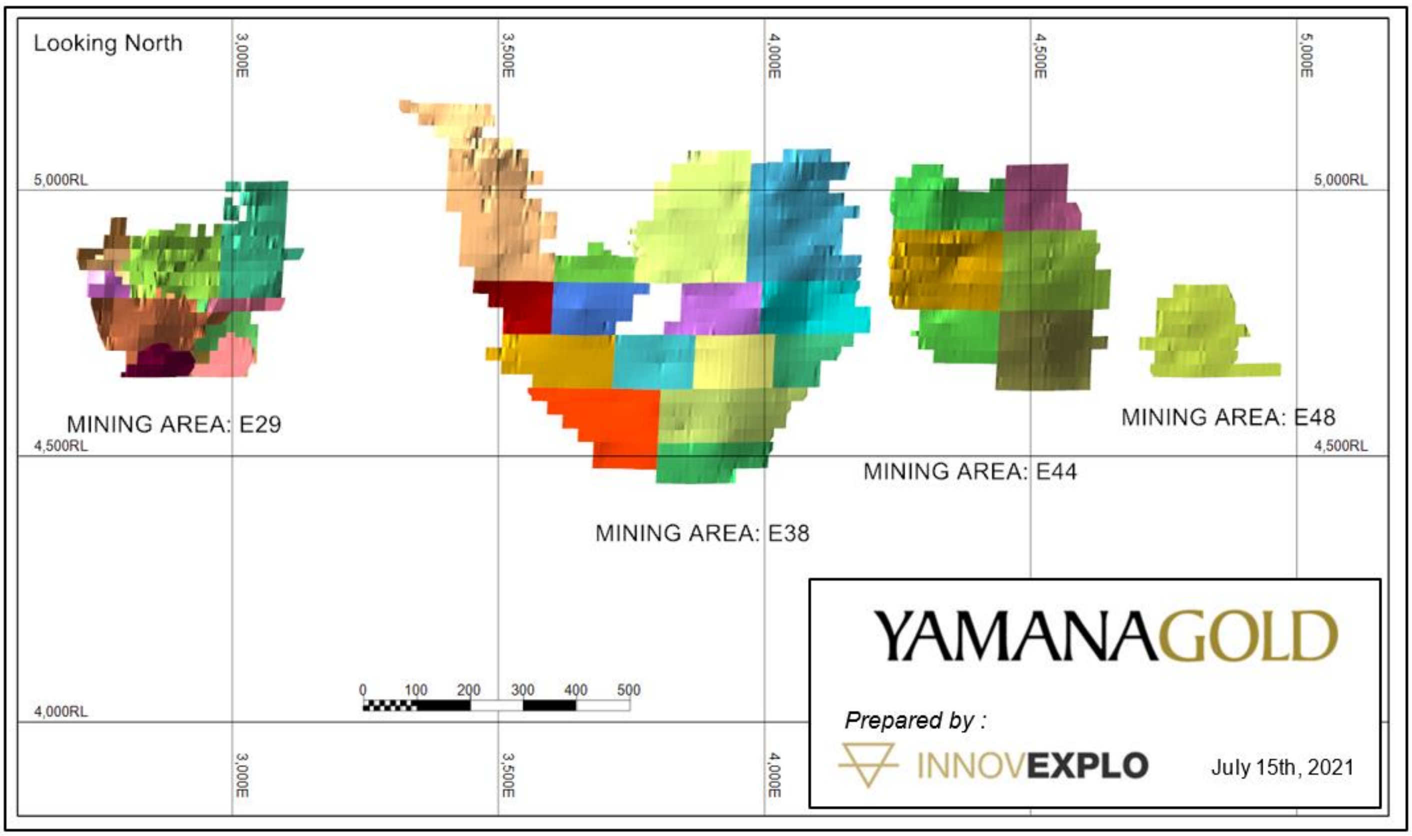
Figure 16-9: Mining Methods



The last level in a sequence, the sill pillar, will be recuperated by uppers and backfilled by paste. The lowest (first level) of a production centre will be backfilled with stronger paste fill to ensure a full and safe recovery of the sill pillars. Some stopes will be drilled using upper drilling (e.g., stopes at the apex of zones and sill pillars). Production centres are normally five to six levels high and seven stopes wide. At the highest part of the mine, mining horizons are generally higher due to the current mine design that will start the production around level L450 to minimize backfill costs and optimize rock mechanics by diminishing the number of sill pillars.

Figure 16-10 presents an overview of the different production centres.

Figure 16-10: Planned Production Centres



16.5.1 Stope Design – Longitudinal Long Hole

Longitudinal long-hole methods will be used for stopes less than 20 m wide (see Figure 16-11 for an example). These stopes are classified based on their rock mechanics and width. Stopes having a hanging wall affected by the chlorite schist formation are classified separately as longitudinal chlorite schist and are smaller to reduce dilution. The number of stopes and resulting tonnages are summarized in Table 16-17. Dimensions and parameters are discussed in Section 16.2.7.

Figure 16-11: Longitudinal Example – Mining Horizon 27 & 30

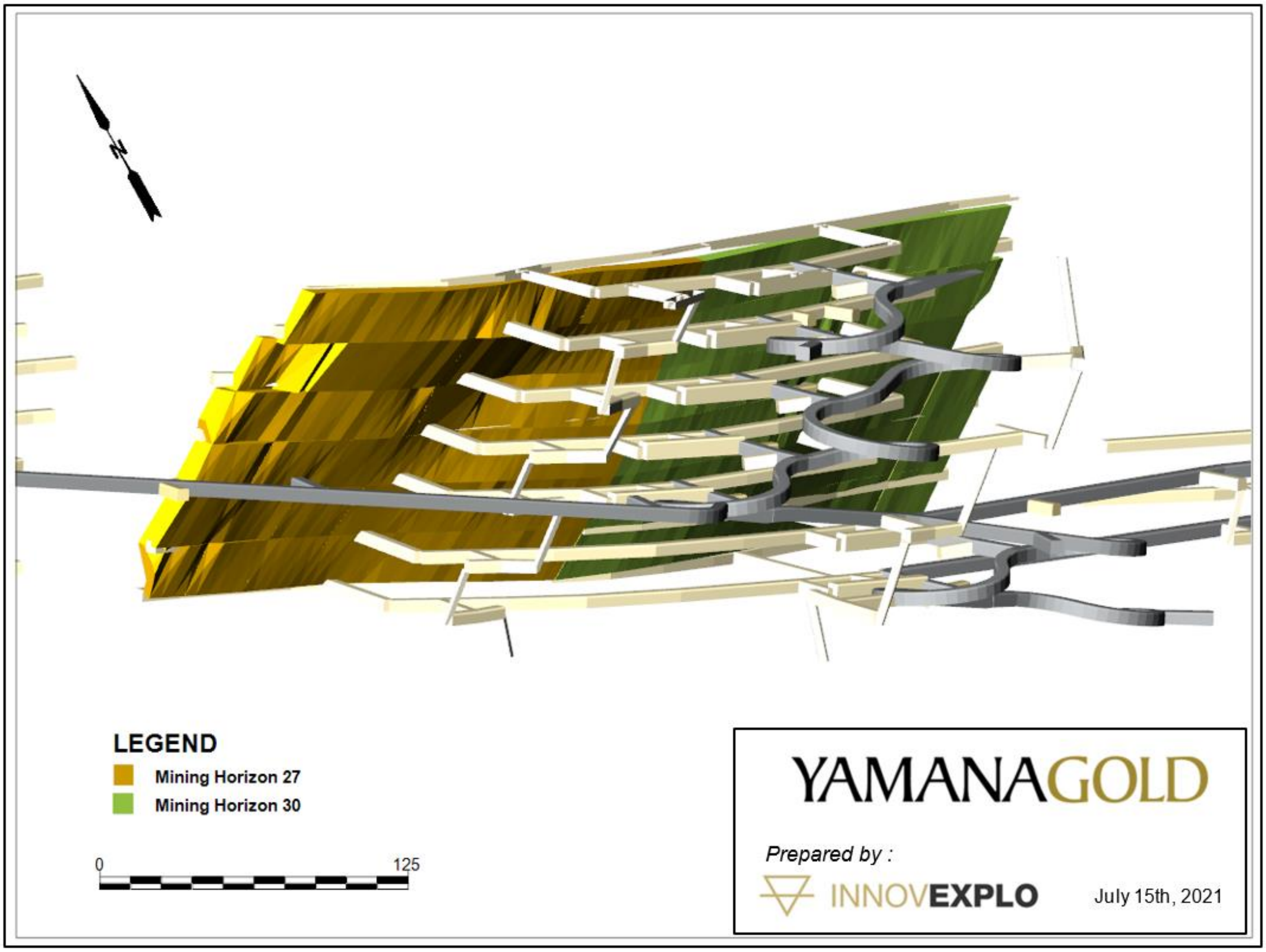
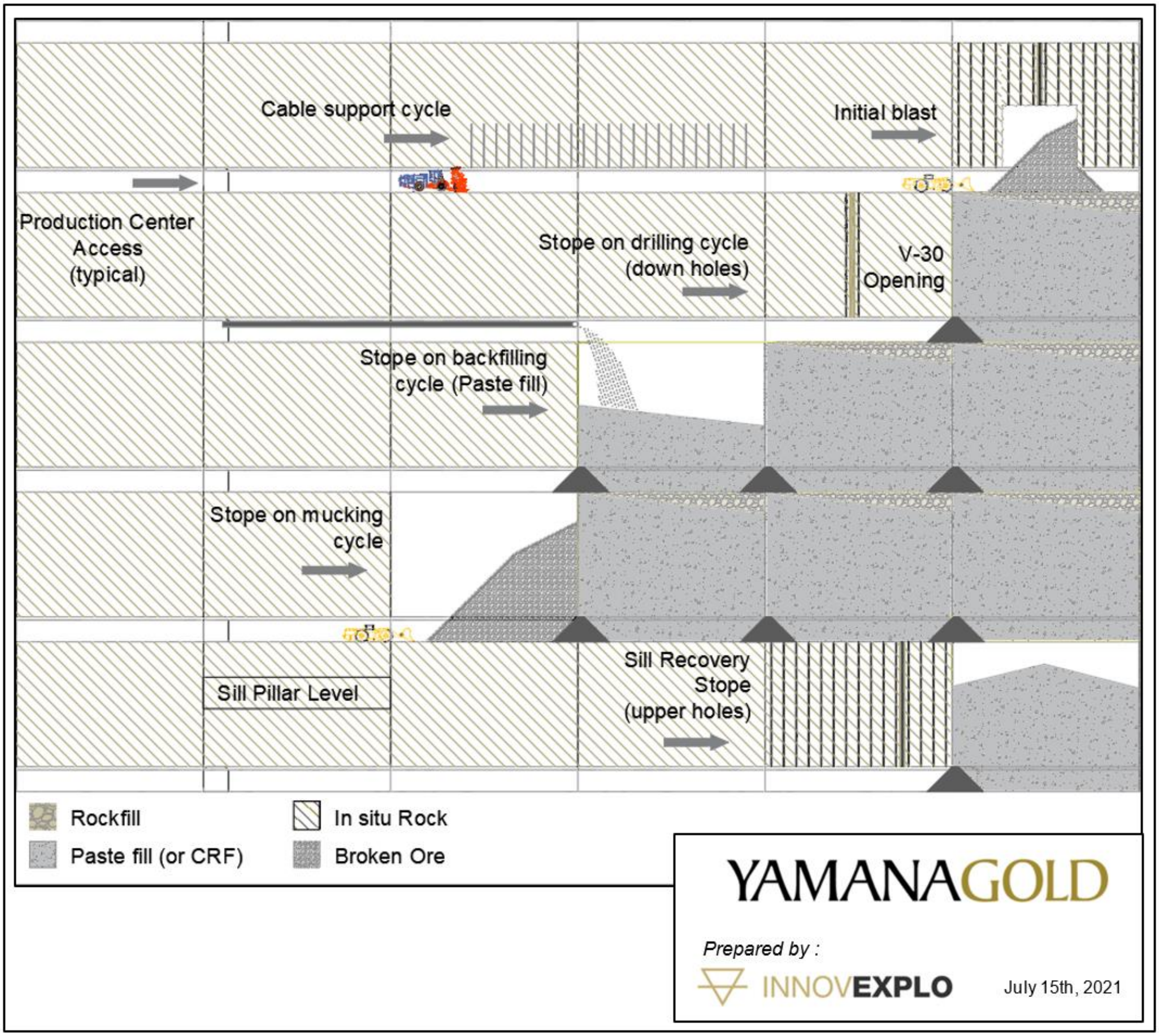


Table 16-17: Mining Methods – Longitudinal Stoping Summary

Methods - Longitudinal	Number of Stopes	Tonnage
Longitudinal Chlorite Schist	169	1,694,480
Longitudinal 3.0 m to 8.0 m	311	4,768,681
Longitudinal 8.0 m to 20.0 m	375	10,736,148

A typical mining cycle includes secondary ground support where required. V-30 slot-drilling is made in advance of the production drill mobilization, followed by the complete production drilling of the stope. Longitudinal stopes are blasted in two phases: a primary blast for the void and the secondary blast after the first blast is mucked out. The second blast may be loaded during mucking to maximize efficiency. Once the stope is blasted and mucked out, a barricade is built in the undercut and the stope is backfilled with paste fill or CRF. Rockfill is used as backfill when possible or to finalize and level the drift floor. The longitudinal retreat method is used to create pyramidal shapes as mining progresses in a production centre (see Figure 16-12). This is designed to maximize the stability of the mining area by diverting the induced stresses outside of the mining area.

Figure 16-12: Typical Mining Cycle – Longitudinal



16.5.2 Stope Design – Transverse Long Hole

A transverse long-hole method will be used with the remaining zones (i.e., width > 20 m). These stopes are differentiated into primary and secondary categories depending on the sequence; they are also classified depending on the chlorite schist formation influence on the hanging wall and end wall (see Figure 16-13). This method entails having a drawpoint for each stope panel. When parallel drifts are impossible due to sizing of the smaller schist stopes, access with "Y" shape are used. The number of stopes and resulting tonnages are summarized in Table 16-18.

Figure 16-13: Transverse Example – Mining Horizon 6

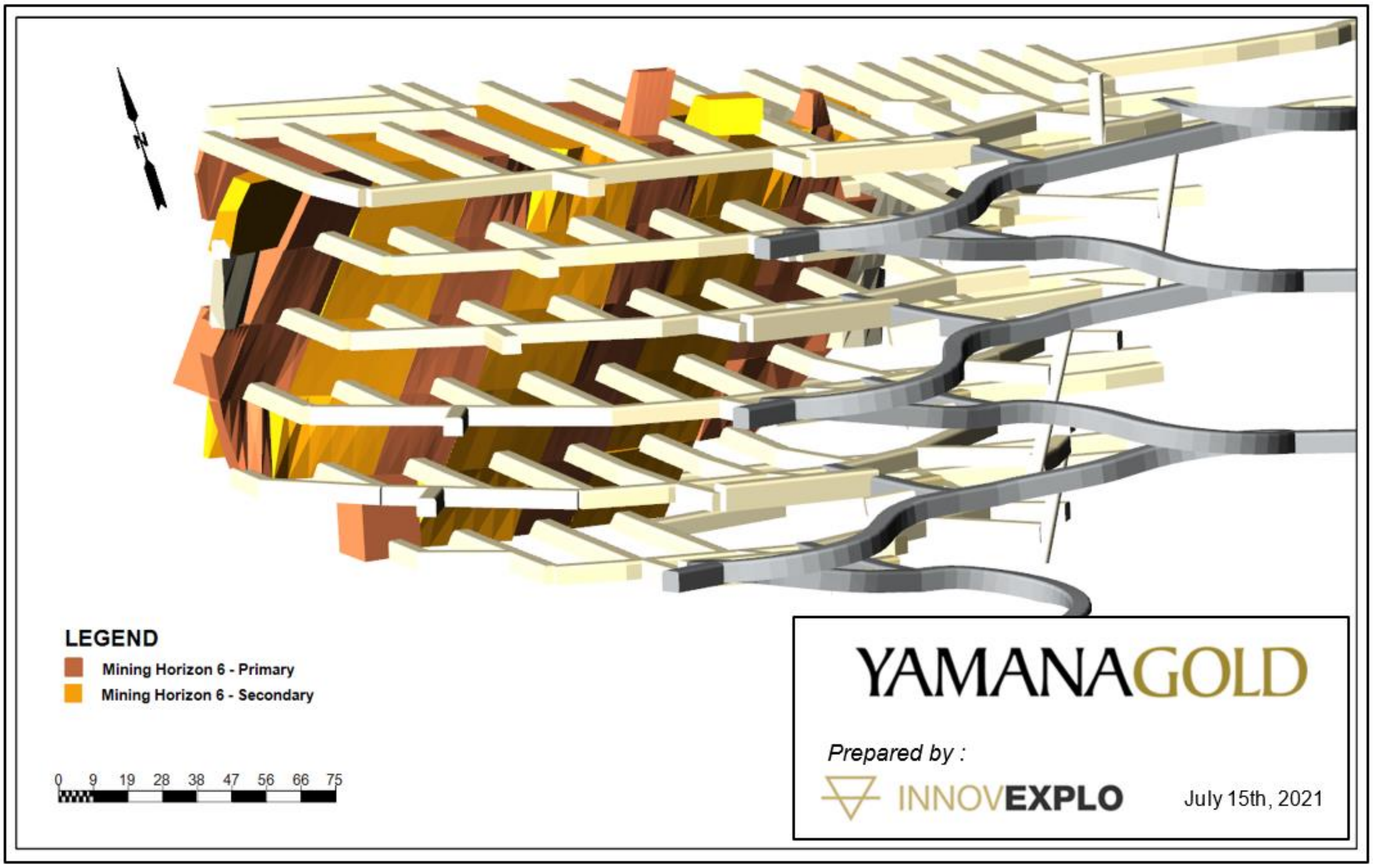


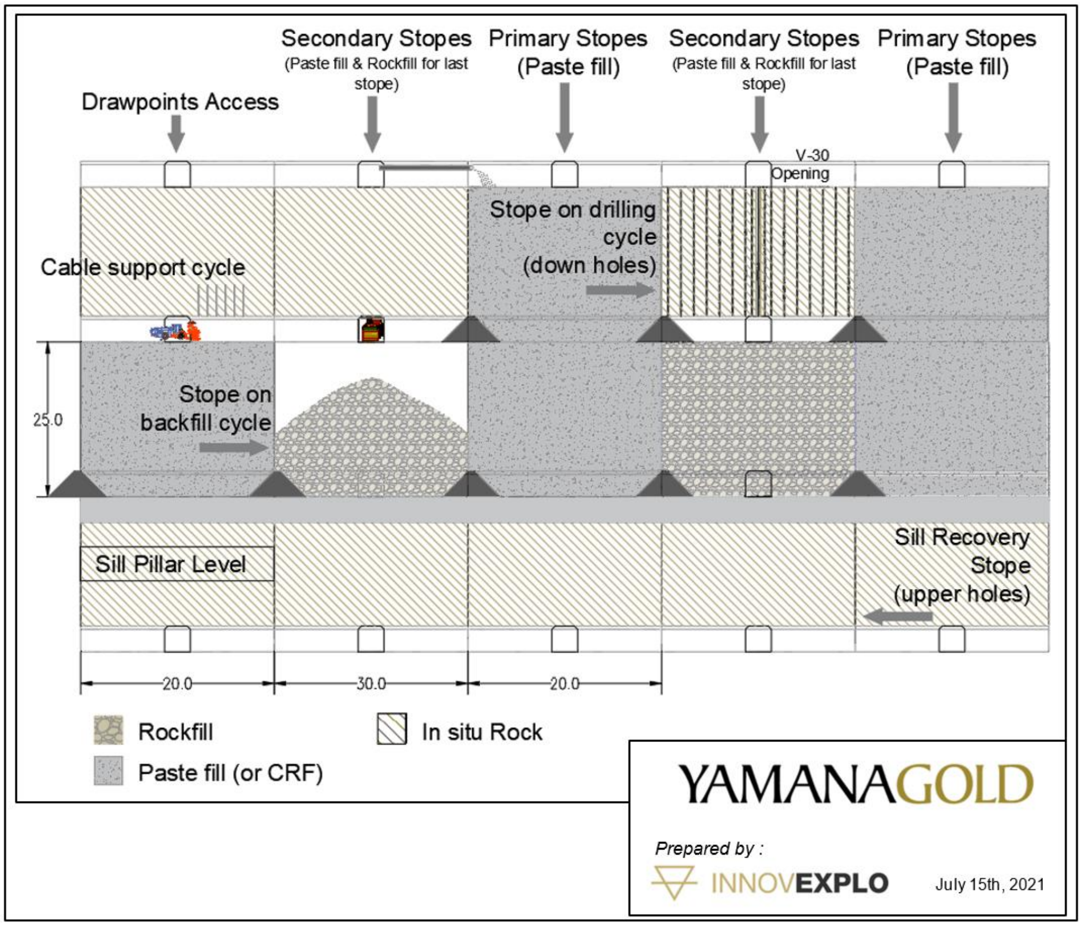
Table 16-18: Mining Methods – Transverse Stoping Summary

Methods - Transverse	Number of stopes	Tonnage
Chlorite Schist Primary	30	339,439
Chlorite Schist Secondary	18	293,552
Transverse Primary	116	2,707,644
Transverse Secondary	120	3,778,578

Like longitudinal stoping, typical mining cycles includes secondary ground support where required, V-30 slot-drilling (see Section 16.5.3), production drilling, mucking and backfilling. The mining sequence starts with the primary stopes from bottom to top, whereas the secondary stopes are blasted when both adjacent primaries are backfilled. For the same drawpoint, the farthest stope is mined first and the sequence retreats towards the hauling drift. This sequence creates a pyramidal shape with the mining voids when the mining progress in a centre of production and is beneficial with respect to the rock mechanics and production aspects. Most transverse stopes need two blasts. Secondary transverse stopes require a third blast because of the important volume.

Figure 16-14 shows an example of the main activities included in the typical mining cycle of transverse long-hole mining.

Figure 16-14: Typical Mining Cycle – Transverse



16.5.3 Drill and Blast Design

The long-hole methods chosen for the Wasamac project make use of fan drilling to maximize recovery from a single overcut or undercut drift. Production drill holes are drilled with a 101.6 mm (4″) diameter using a Sandvik DL432i. Most holes are between 3 and 25 m (level spacing at 25 m floor to floor), but some of the longer holes can reach 29 to 30 m. At this length, diameter, and with the selected equipment, deviation can easily be controlled and avoided. Impact on dilution would be minimized.

Wasamac plans to take full advantage of the DL432i automation capabilities. Intelligent drilling control system and automated drilling through shift changes ensure optimal drilling, while maximizing utilization.

To limit personnel exposition and production cycle time, the main method for cut opening will be a raisebore hole. The most economic, flexible, and risk-free method is achieved using Machines Roger as a contractor to drill the necessary V-30 openings (30‴ holes). Other raiseboring rigs have been evaluated; however, the dip of the ore body does not allow the use of most of the common rigs in the industry. For safety purposes, the excavated hole will be mechanically capped or not fully raisebored to the overcut, eliminating open hole exposition.

16.5.3.1 Drill and Blast – Parameters

As stope geometry can vary greatly between each method, some adjustments are to be expected between each type. Seven typical stopes configurations were used for the planning and cost estimates. Little variations to these configurations are made for upper drilling. The burden for each method is 2.5 m and the spacing vary from 2.5 to 3.0 m depending on the drilling pattern method. The method to assess the drilling and the resulting load on planning is to evaluate the drilling ratio (t/m) and re-drilling factor (%). The calculated results are summarized in Table 16-19. Minor modifications to these ratios and factors are applied when the stope is to be drilled as upper.

Table 16-19: Drilling Ratio and Re-drilling Factor

Methods	Drilling Down		Upper Drilling	
	Drilling Ratio (t/m)	Re-drilling factor (%)	Drilling Ratio (t/m)	Re-drilling Factor (%)
Longitudinal Chlorite Schist	10.8	13%	10.4	13%
Longitudinal 3.0 to 8.0 m	9.7	3%	8.9	8%
Longitudinal 8.0 to 20.0 m	12.7	3%	12.5	8%
Chlorite Schist Primary	7.4	12%	9.1	12%
Chlorite Schist Secondary	9.2	15%	10.4	5%
Transverse Primary	8.6	2%	10.7	7%
Transverse Secondary	9.2	5%	9.3	5%

Regarding blasting, the targeted void is 20%. All methods can achieve this easily with two mass blasts. With experience acquired during production, parameters like length and height of the first blast can be optimized to improve safety, production rate and rock stability.

Generally, collars will be 2.0 m with 1.5 m of stemming. Electronic detonators will be used to optimize flexibility and fragmentation. Additional parameters include two boosters and detonators and the use of bulk emulsion to minimize risks and costs. Another design addition is the use of inflatable plugs for break-through holes, which accelerate stope preparation and improve cycle time.

Figure 16-15 and Figure 16-16 present typical drilling and blasting patterns for downhole longitudinal and transversal stoping.

Figure 16-15: Schematic Sections of Longitudinal Long Hole Stoping

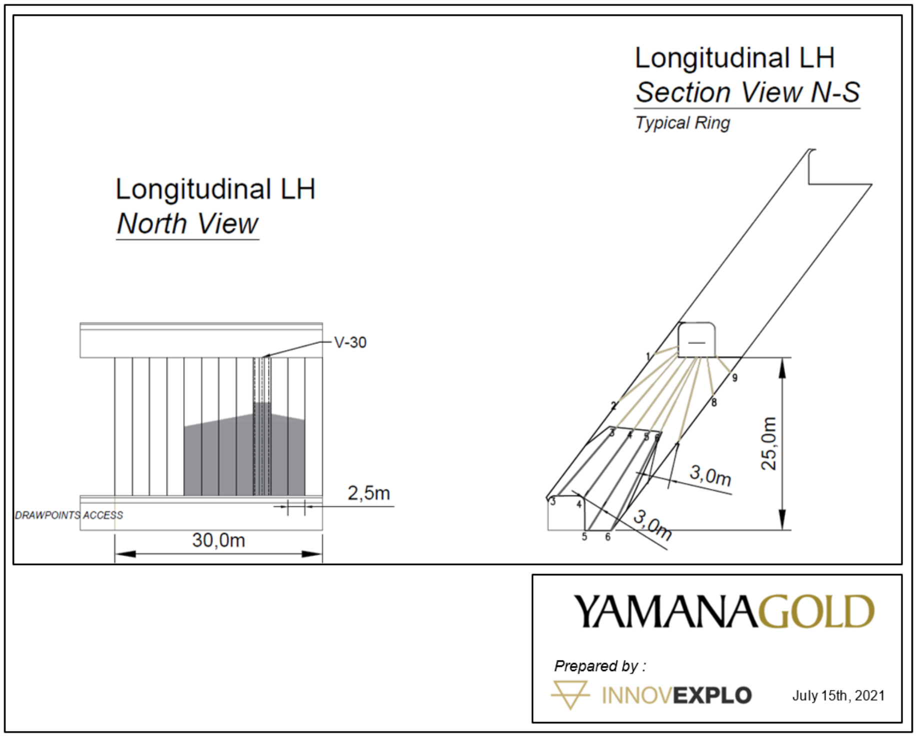
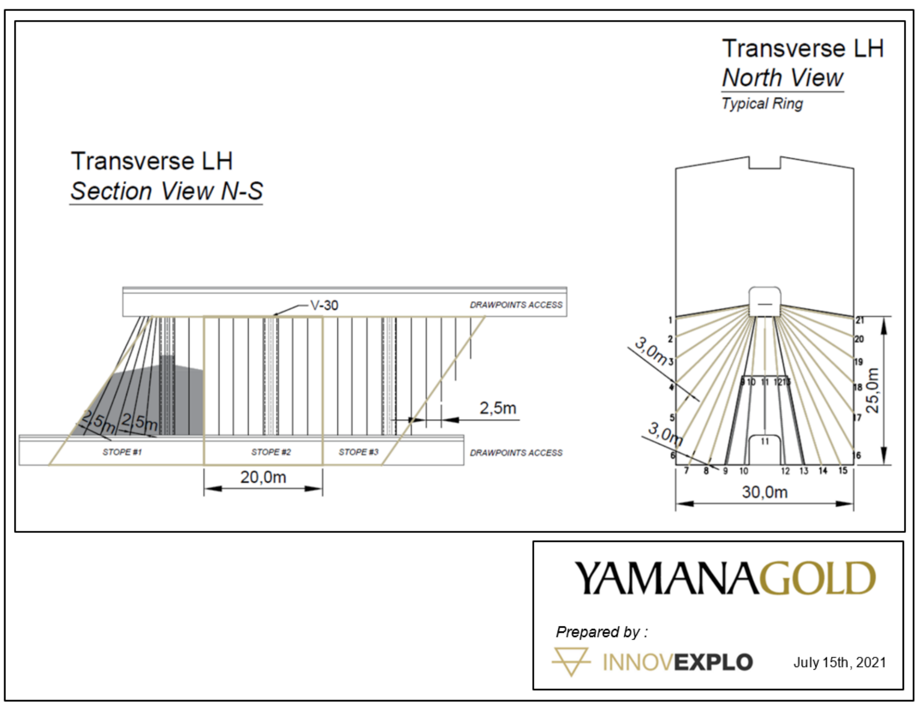


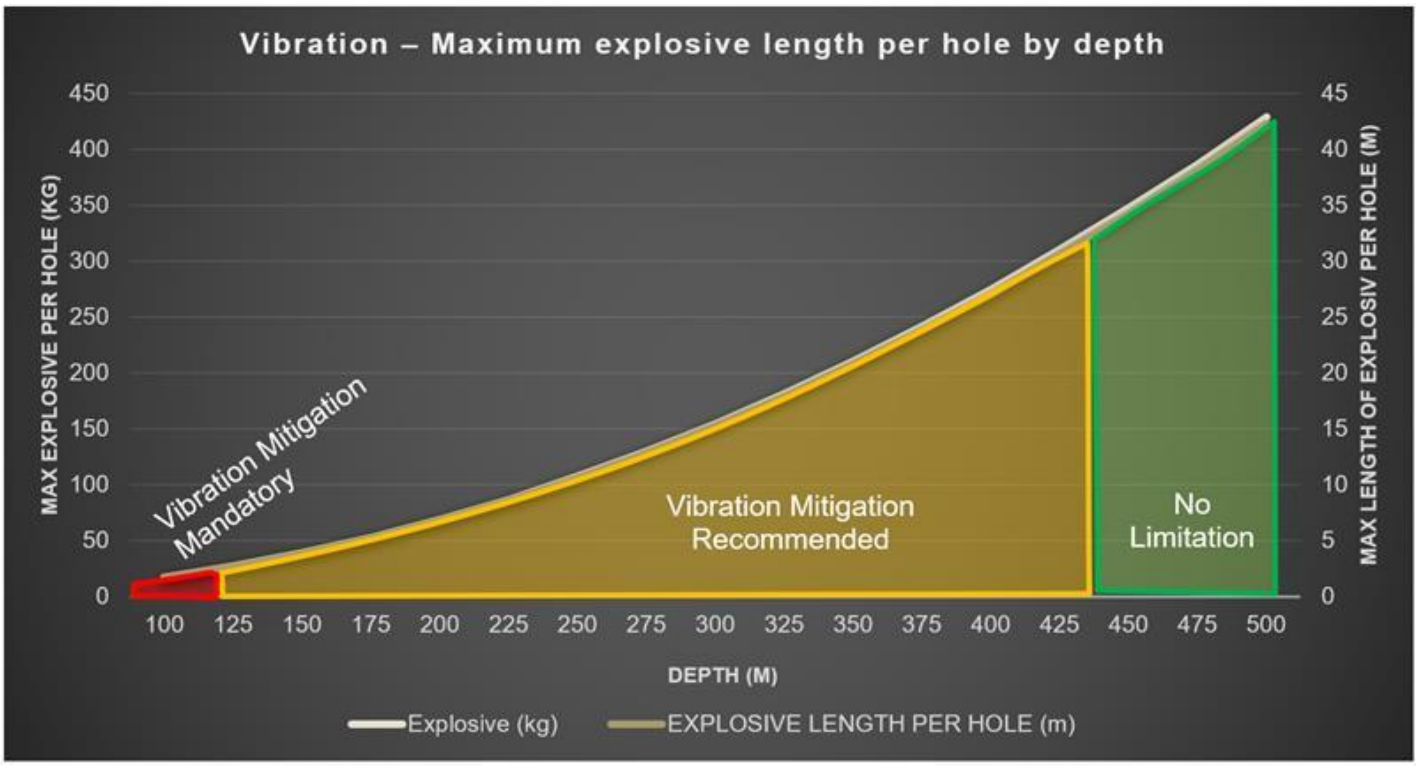
Figure 16-16: Schematic Sections of Transversal Long Hole Stoping



16.5.3.2 Vibration Control

Vibration control at surface is paramount to acceptability by the neighbouring community. Blast-induced vibration has been evaluated theoretically using the USBM formula (Duval, 1962). The evaluation has been done for depths between 100 and 500 m, since below this depth no production charge will be longer than the admissible charge length. Directive 019 dictates that for an underground exploitation, the maximal admissible vibration at surface is 12.7 mm/sec. In consideration of the neighbouring community, the maximal vibration has been set at 7.0 mm/sec, which also provides an additional safety factor. The Figure 16-17 summarizes the results for maximum explosive length by depth.

Figure 16-17: Vibration – Maximum Explosive Length per Hole by Depth



Most downhole stopes, both longitudinal and transverse, affected by the limitation will be able to mitigate charge length by designing the primary blast sequentially. Whenever separating longer charges between two blasts is not possible, the decking technique will be used to ensure that no charge is longer than the admissible length per depth.

In the few instances that decking is not possible (notably upper stopes), some alternative methods may be used. Webgen™ wireless initiating system combined with upper hole decking (both technologies from Orica) is the most efficient way the targeted ore can be recuperated without sacrificing safety or vibration control.

Additional expertise can be used to ensure optimal monitoring of sound and vibration at surface. Soft dB, engineering experts in acoustic and vibration, have proposed an option for a system monitoring and mitigating the blast surface repercussions. The first proposition includes a surveillance system including six stations monitoring sound and vibration. Additionally, artificial intelligence can be used to identify any sound sources not coming from the mine, minimizing workhours required for in-depth monitoring. The system will ensure respect of the mine's decree, if applicable, or the minimal requirements for sounds and vibration (Note 98-01 and Directive 019). Conformity reports will be sent to the Ministry, as needed. Costing for this or a similar system has been included in the budget.

16.5.4 Backfill

Various types and strengths of backfill are planned to be use at the Wasamac Project to backfill mining voids (Table 16-20). The primary type of backfill will be paste. It will represent 80% of the backfill used at the mine. Rockfill will be used when possible (18% of the time). For example, it will be used for the last secondary transverse stope on a same drawpoint or the stope at the drawpoint location for the longitudinal retreat to the centre of production. CRF will be used at the beginning of the life of mine when the backfill plant will not be available. CRF will also be used later in the mine plan as part of a strategy to minimize the waste material being hauled at surface and the overall surface footprint.

The strength of the backfill required will depend on the size of the stope and the mining sequence. For example, a stope above sill pillar will require a higher strength, as stopes will be mined beneath them.

Table 16-20: Tonnage of the Different Types of Backfill Planned

Backfill Required	Rockfill (tonnes)	CRF (Tonnes)	Paste (Tonnes)
UCS = 0.0 MPa	2,810,154	-	-
UCS = 0.2 MPa	-	61,502	3,487,424
UCS = 0.3 MPa	-	117,938	3,882,030
UCS = 0.4 MPa	-	81,282	3,145,024
UCS = 0.5 MPa	-	17,292	1,401,023
UCS = 1.2 MPa	-	51,635	217,034
UCS = 1.5 MPa	-	-	345,553
Total	2,810,154	329,649	12,478,088

16.5.4.1 Paste Backfill

The paste backfill is produced from mill tailings that are thickened at the mill and then pumped either to the filter plant for disposal, or underground to the paste backfill plant. The backfill plant is designed to match the throughput of the mill, so that the tailings stream is not split, but is pumped to one of the two plants following thickening.

16.5.4.1.1 Paste Testing Campaign

A backfill testing campaign was conducted in support of the feasibility study update. All sample material remaining from the metallurgical testwork was homogenized and tested by RMS Mining Solutions Inc. Testing included material characterization, rheology, dewatering (both thickening and filtration), and strength testing.

Characterization and rheological results were used as inputs into the backfill plant design, as well as helping to validate the mine tailings balance.

Thickening test results validated the thickener design within the mill, while also giving insight into what range of densities should reasonably be expected from the thickener during operation. The filtration testing then used a range of feed densities to ensure that the selection and sizing of filters for the backfill plant can accommodate a range of thickener underflow densities.

Strength testing utilized primarily a 90:10 blend of ground granulated iron blast furnace slag and general use cement. A single batch was cast with 100% cement. The strength with the slag cement blend was significantly higher than the 100% cement test.

The results from the strength test campaign were used along with strength targets to estimate binder requirements for the operating cost estimate. The results of strength testing to date are summarized in Table 16-21, with "TBD" referring to cylinders that have yet to be broken.

Table 16-21: Backfill Strength Testing Results

Slump (mm)	Binder Type	Binder Content (% w/w)	UCS Strength (kPa)					
			7 Day	10 Day	28 Day	90 Day	180 Day	365 Day
175 mm	GGIBFS/GU (90/10)	2%	153		494			
		3%	233		787		TBD	
		5%	534	696	1600	TBD	TBD	TBD
		7%	791		2072			
		10%	1765		2992		TBD	
	GU	5%	273		495			
250 mm	GGIBFS/GU (90/10)	5%	440		1428	TBD	TBD	
		7%	891		2010			

16.5.4.1.2 Paste Fill Barricades

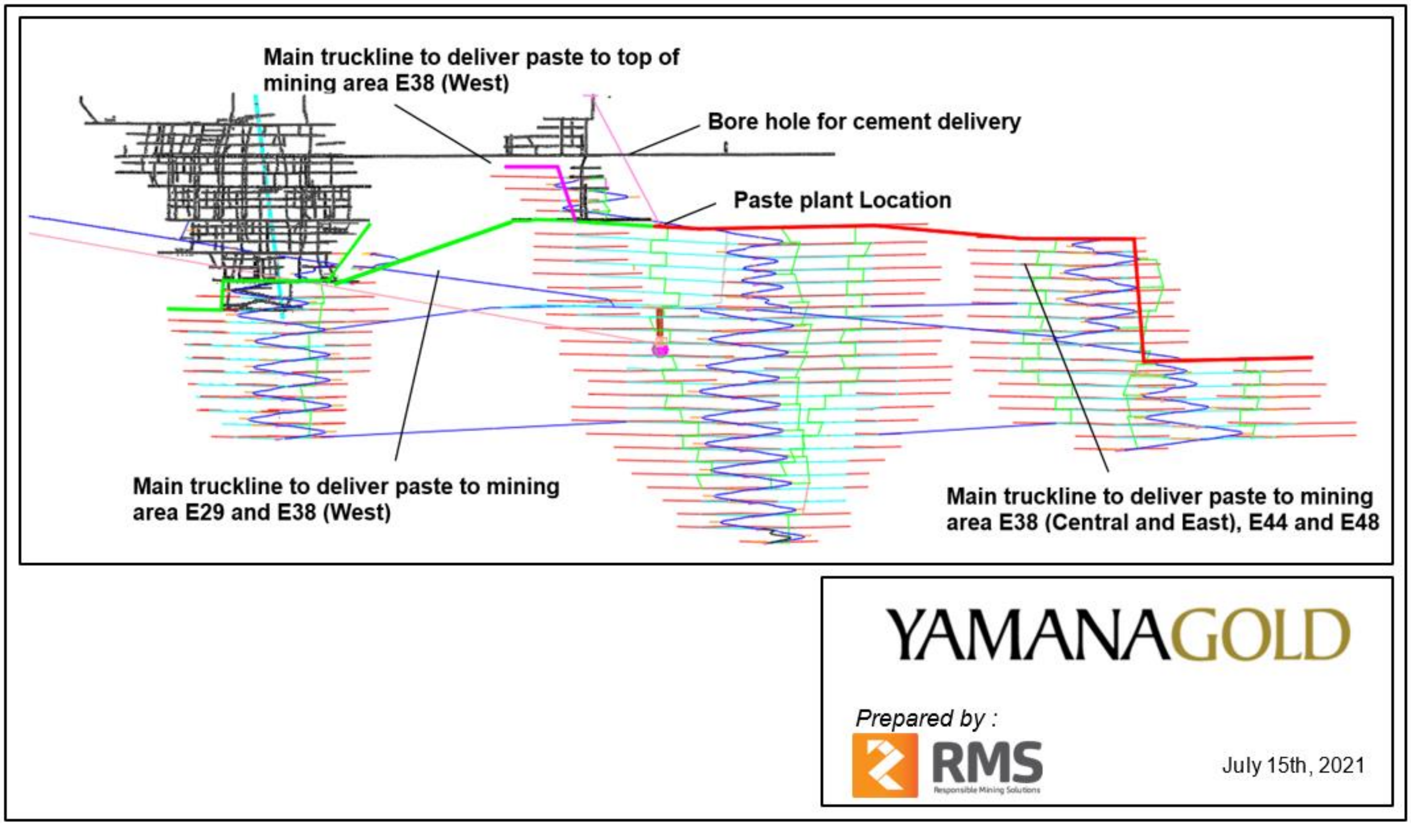
It is planned to create engineered barricades for the paste fill with rockfill and shotcrete. First, the rockfill will be dropped from the top of the stope. Then, rockfill will be pushed to the barricade on the lower drift by an LHD to fill the void and add additional rockfill. A minimum of 3.0 m will be filled in the drift. Finally, 3″ thick shotcrete will be added to prevent any seepage. A total of 332 m³ of rockfill and 3.5 m³ of shotcrete is planned for every barricade. These types of barricades are common in operating Canadian mines. Based on the Authors' experience they are an efficient, safe and rapid construction approach while being cheaper than other type of barricade.

16.5.4.1.3 Backfill Plant Location and Distribution System

The ideal location for a paste fill plant is on surface over the centre of the orebody. As this is not a viable option due to environmental and social considerations, the plant will be located underground, as high as possible within the orebody considering the design of the mine, minimization of additional development costs and delay for construction. The cement will be delivered from surface to the paste plant by a borehole. The paste plant is to be located above the centre of gravity of the planned stopes of the complete project to optimize the paste distribution.

Figure 16-18 shows the approximate location of the plant (where the borehole from surface intersects the development), as well as the major distribution routes that were used to validate the plant location with flow modelling. Preliminary routes for these lines to both extents along the top of the orebody were modelled (bold green and red lines), as well as to the upper extents of the planned stopes (bold magenta line). Flow modelling confirmed that the entirety of the orebody can be reached with reasonable pressure losses using a single paste pump sized for 80 bar of pressure nominally (100 bar peak). From these upper locations, the paste will flow down to lower stopes with gravity even though the orebody is inclined rather than vertical. The distribution system is a 10″ pipe diameter system in order to maintain reasonable pipeline velocities and avoid excessive system wear.

Figure 16-18: Backfill Trunkline Used for Flow Modelling



16.5.4.1.4 Paste Backfill Plant Layout

The layout of the backfill plant allows for access from both the haulage drift on Level L275 and the ramp to Level L250. The layout was used in support of the capital cost estimation and for the planning and is shown in Figure 16-19 on the following page.

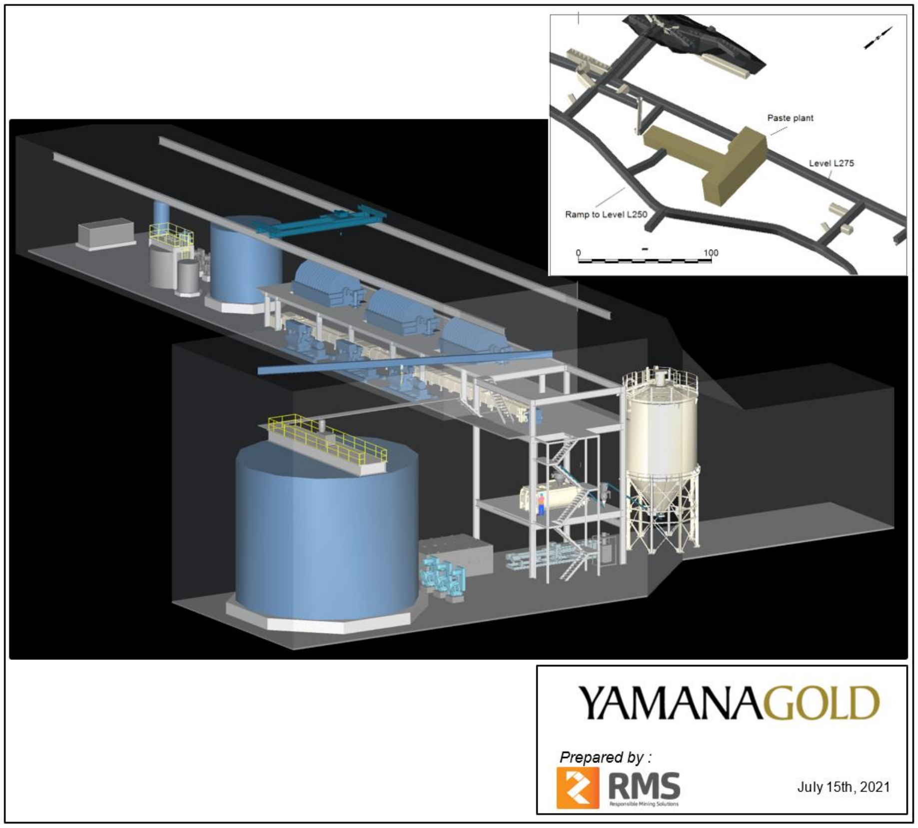
16.5.4.2 Cemented Rockfill and Rockfill

The rockfill will be provided from development waste rock. The waste material will be moved to the hauling bay of the required level by the trucks. Then an LHD will haul the rockfill material to fill the appropriate mining voids. No rock material should be rehandling back from the surface to the underground mine.

The cemented rockfill is planned to be sub-contracted to Swatcrete. They will provide a CRF-12- type mobile plant and operators. This strategy has the benefit of adding no additional infrastructure and providing greater flexibility to meet the requirements of the project.

The mobile plant will go to the site and levels where CRF backfill activities are required. It will provide the amount of cement to be mixed properly to the rockfill to meet the backfill strength requirements of the stopes. This manipulation will be added to the one made by the LHDs to fill the stopes with typical rockfill.

Figure 16-19: Paste Backfill Plant Model



16.6 Mine Schedule

Mining areas and underground infrastructures are accessed through the service and conveyor ramps, starting near the main surface laydown, north of Highway 117. Both ramps will be developed simultaneously using a rapid-development method to minimize pre-production duration. The pre-production objective is to reach all major infrastructure as soon as possible and to prepare mining areas E29 and E38 for the production phase. The major targets during pre-production are the crusher and ore silo, paste plant, and main ventilation raises.

Once production has started, the objective is to maintain an underground production rate of 7,000 t/d, while postponing unnecessary development and expenses. To maximize the profitability of the project, higher grade stopes will be mined at an early stage.

The mine schedule directly considered critical activities like horizontal and vertical development and main production activities. The typical production stope cycle times included the main following activities (stope preparation, secondary ground support (if required), V-30 drilling, production drilling, loading and blast #1, mucking #1, loading and blast #2, mucking #2, and backfill activities (including barricade, if required).

Delays are added to some activities to consider additional tasks such as barricade creation, validation of the tasks, mobilization, and demobilization of the equipment, cave monitoring system (CMS) survey, curing time of the backfill, etc.

16.6.1 Production Rates and Performance Parameters

To maintain accurate underground mine scheduling, detailed cycle times were calculated for main underground activities. The operational parameters used for the Wasamac project are detailed in Table 16-22. Each day includes two 10-hour shifts and considers all related operational activities (e.g., shift changes, lunch break, refuelling, loss of time and transportation to workplaces). Five lost days per year are estimated due to unforeseen events. To maximize the efficiency of main equipment, automation is planned between shifts. The automation will begin at the end of the pre-production period, around July 2027. A three-month period is planned to get an efficient ramp-up for these new technologies. It is planned that at the beginning of the production period, the equipment will be fully automated and efficient between shifts.

Table 16-22: Operating Parameters

Operating Parameters	Unit	Quantity
Average Production Days per Year	Days	360
Number of Shifts per Day	Shifts	2
Effective Hours per Day (No Automation)	Hours	17
Effective Hours per Day (Automation Ramp-up)	Hours	19
Effective Hours per Day (With Automation)	Hours	21

Production operating hours have been defined depending on each equipment cycle time. An overall efficiency of 85% is assumed for major equipment. Production rate and cycle times have been evaluated by activities and tasks, mining area, and sub area. Rates per zone also vary depending on automation availability. Table 16-23 summarizes the rate used for critical production tasks in the scheduling.

Table 16-23 : Main Production Activities Rates

Equipment	Tasks	Minimal Rate	Maximal Rate	Unit	Automation between shifts
Cable bolting	Stopes Support	3,969	7,344	m/mth	Yes
V-30	Cut Opening	380	380	m/mth	No
Production Drills	Upper Drilling	7,491	10,662	m/mth	Yes
	Down Drilling	7,579	11,280	m/mth	Yes
LHD	Mucking	1,091	1,622	t/day	Yes
	CRF	1,115	1,348	t/day	No
	Rockfill	1,213	1,845	t/day	Yes
Paste plant	Paste fill	6,192	6,192	t/day	No

Regarding horizontal development, the pre-production phase rapid-development is paramount to achieve the targeted commercial production start. Various strategies are planned to accelerate the development rate at that phase such as blast at will, lighter ground support on critical path, modern and efficient equipment. Contractors are used for all development during pre-production and are expected to achieve the required development rate. The rates for the development vary mostly by drift sizes and the development strategy (multiple versus single faces). The development planning is made first by estimating the required numbers of working jumbos. Then, based on the quantity of required jumbos, the number of such other related equipment such as bolters, LHDs (required for development) are estimated based on detailed cycle time of the development path. A summary of the main development rates is described in Table 16-24.

The same cycle time calculation process is used to estimate the vertical development rates. The rates vary based on the selected method used and the size of the excavation. To these rates, additional delays are applied to consider other activities when required such as ground support and manway construction.

Table 16-24: Main Horizontal Development Rates

Heading	Single Face (m/d)	Multiface	
		Per jumbo (m/d)	Per heading (m/d)
5.75 x 5.75 – Rapid Development	7.0	16.3	5.4
4.5 x 4.5 – Rapid Development	7.0	14.9	5.0
5.75 x 5.75	3.9	8.3	2.8
5.3 x 5.0	4.2	9.3	3.1
7.0 x 5.0	3.7	8.0	2.7
5.75 x 7.3	3.3	6.9	3.5

16.6.2 Production Plan

It is planned that the underground development will start on October 1, 2024. The first ore development is planned for the last quarter of 2026, and the first stope would be mined in March 2027. The commercial production period is scheduled to start in October 2027 when the mine will reach 7,000 t/d for the first time after three years of pre-production. During the pre-production period, major infrastructures like the service bay, underground crusher and conveyor, paste plant and main raises will be excavated and all associated equipment installed and commissioned.

During the ramp-up to production, while the crusher and conveyor complete their installation and commissioning, trucks will haul the ore to the surface. The same idea is applied to the paste plant. During its construction and commissioning, so as not to delay the production activities, CRF will be used to backfill the required mining voids.

The life-of-mine plan shows a rapid ramp-up in production in the first year, with production rising to approximately 225,000 oz/a for the subsequent four years. Average gold production is expected to be 190,000 oz/a over a life of mine of 10 years. The ounces and other material reported in chapter 16 refer to diluted reserve that consider mining recovery and other underground mining factors but do not consider mill recovery. Based on current mineral reserves, Wasamac has a mine life to October 2036, but potential conversion of mineral resources and exploration potential could possibly extend the mine life. Mining areas E29 and E38 are mined across the mine life but more intensively at the beginning of the operation. Mining area E44 is projected to start production in 2030, and mining area E48 in 2033.

Contractors will be employed to develop the two access ramps and major infrastructures. From mid-2026, with sufficient working faces available, mine development teams will be hired by the mine. They will work in parallel with the contractors' teams until the second trimester of 2028, when only three mine jumbos will be required. An average of 8,700 m of horizontal development are realized per year with a maximum of 15,800 m in 2027. To do that, a maximum of five jumbos are required at work during the last pre-production year and the first months of the production periods. An average of three working jumbos are required during the complete life of mine.

Overall, around 9% of the required LHDs for production and development will be needed for backfill tasks, 29% for development and 62% for mucking activities (maximum of 10 LHDs).

During the life of mine, a median of 16 levels in operation are required at the same time, including all main activities.

A summary of the underground schedule overall and by mining area is provided in Table 16-25 and Table 16-26.

Table 16-25: Underground Schedule Summary

Years	Unit	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	Total
Horizontal Development	m	1,520	6,918	13,988	15,451	10,692	9,624	9,542	9,758	8,801	6,355	6,453	5,538	56	104,696
Vertical Development (Raisebore)	m	-	236	764	346	149	98	318	131	152	117	77	-	-	2,388
Total Development	m	1,520	7,155	14,752	15,797	10,842	9,722	9,859	9,889	8,953	6,471	6,530	5,538	56	107,084
Stopes	kt	-	-	-	1,172	2,294	2,348	2,344	2,281	2,379	2,375	2,327	2,359	1,386	21,264
Ore Development	kt	-	-	74	381	233	172	176	230	148	145	181	161	3	1,903
Total Ore	kt	-	-	74	1,552	2,527	2,520	2,520	2,512	2,527	2,520	2,508	2,520	1,389	23,168
Ore per day (average)	t/d	-	-	206	4,311	6,999	7,000	7,000	6,977	7,000	7,000	6,965	7,000	4,629	NA
Ounces	koz	-	-	6.2	132.5	227.9	225.8	223.5	224.2	209.0	196.6	187.2	176.5	100.8	1,910.2
Gold Grade	g/t	-	-	2.61	2.65	2.81	2.79	2.76	2.78	2.57	2.43	2.32	2.18	2.26	2.56
Waste Produced	kt	124	581	1,261	909	646	657	671	585	622	412	364	311	1	7,143.8
Rockfill and CRF	kt	-	-	-	182	184	341	433	530	450	348	349	267	56	3,140
Paste Fill	kt	-	-	-	634	1,473	1,381	1,302	1,138	1,348	1,398	1,327	1,483	994	12,478

Table 16-26: Underground Schedule Summary per Mining Area

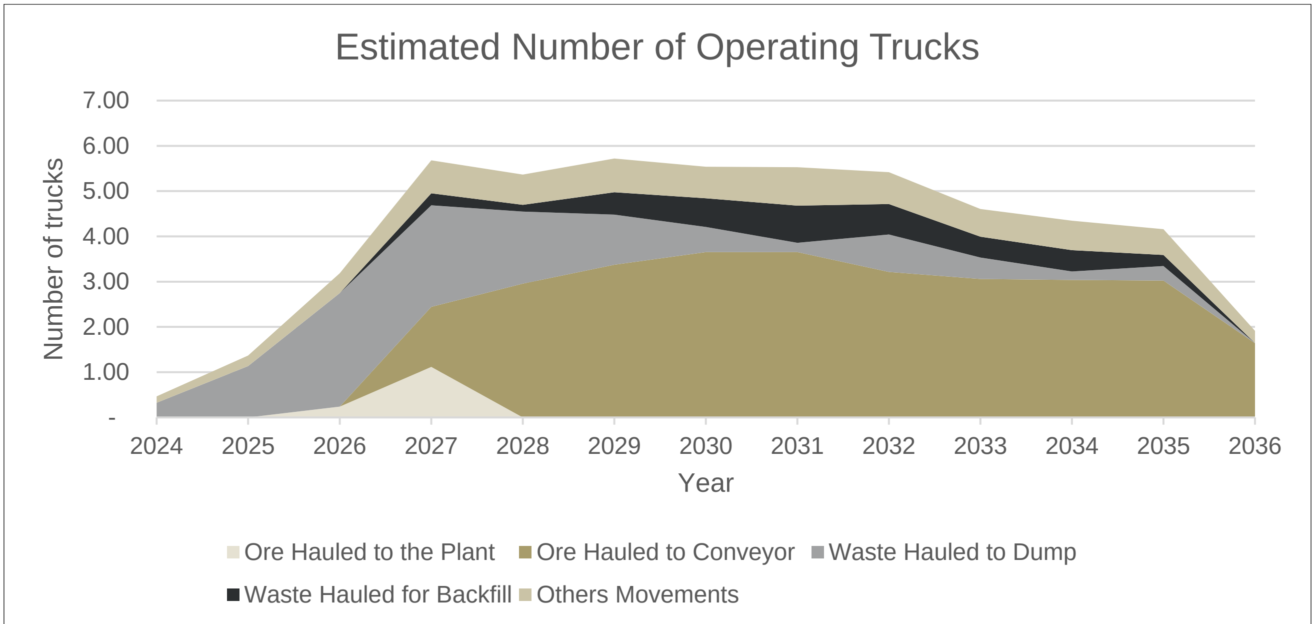
Years	Unit	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	Total
Mining Area E29															
Total Ore	kt	-	-	-	684	1,163	1,140	1,089	1,017	1,010	314	541	84	166	7,208
Ounces	koz	-	-	-	63.0	111.3	112.5	103.4	103.5	90.6	26.6	40.2	5.3	9.6	666.2
Total Development	m	789	3,540	2,382	9,484	5,029	2,472	825	630	687	649	179	493	56	27,217
Mining Area E38															
Total Ore	kt	-	-	74	868	1,364	1,380	1,387	1,162	841	1,803	1,014	1,517	482	11,891
Ounces	koz	-	-	6.2	69.5	116.6	113.2	115.4	93.4	61.3	138.4	72.1	102.4	30.7	919.2
Total Development	m	731	3,614	12,370	6,313	5,812	5,918	4,478	4,003	6,900	657	4,877	399	-	56,073
Mining Area E44															
Total Ore	kt	-	-	-	-	-	-	44	332	676	259	822	804	568	3,504
Ounces	koz	-	-	-	-	-	-	4.6	27.3	57.1	21.0	63.3	58.5	44.1	275.8
Total Development	m	-	-	-	-	-	1,331	4,556	5,255	969	3,293	1,473	2,967	-	19,844
Mining Area E48															
Total Ore	kt	-	-	-	-	-	-	-	-	-	144	131	115	173	564
Ounces	koz	-	-	-	-	-	-	-	-	-	10.7	11.7	10.3	16.4	49.0
Total Development	m	-	-	-	-	-	-	-	-	397	1,873	-	1,680	-	3,950

16.6.3 Trucks Estimation

Truck operating hours were estimated using Deswik.LHS software. A cycle time was estimated for all hauling tasks in the mine planning, based on the truck parameters provided by Sandvik (i.e., cycle time calculated for truck loading, etc.) The truck cycle times took into consideration design constraints, such as grade and optimal dropping points. For example, when the crusher is not available, trucks will haul ore to surface. When the crusher is available, trucks will deliver ore to the grizzly. For the waste hauling strategies, trucks will haul to the closest backfilling activities or otherwise to the waste dump.

Figure 16-20 shows the average estimated number of operating trucks per year for the life of mine.

Figure 16-20: Estimated Number of Operating Trucks



16.7 Mine Services

16.7.1 Electrical Services

Refer to Section 18.13.5, Underground Electrical Distribution, for details.

16.7.2 Communication Network

The underground communications network will consist of fibre-backbone and co-wireless equipment with cyber security recommended by Ambra Solutions based on the size of the mine. This equipment includes non-redundant EPC cores for up to 1,100 users with VoLTE and Push-To-Talk application server with a capacity to support 250 users. Underground personnel will be equipped with smartphones with radio and tracking capability.

The fibre-optic network will be installed throughout the ramps as they develop to facilitate communication with the mine production equipment connected to the underground electrical rooms and mine power centres.

16.7.3 Mine Automation and Monitoring System

The mine automation system was proposed by Sandvik and includes the fleet management for trucks, scoop operators, and drill operators. Operator workstations sited within the control room for the underground will manage the automation and remote operation of the mine equipment from surface in the office complex.

The fleet management system is equipped with a traffic awareness feature that will be implemented throughout the underground mine as the levels advances and based on numbers of vehicles in service.

The underground monitoring system will include a surveillance system. A total of 88 cameras will be installed as the levels develop over the life of mine. In addition, miners' lamps with geolocation systems have been included for underground mining personnel.

16.7.4 Fuel Distribution Network

Fuel supply will be stored at surface in two 90,000 L tanks. There will be two underground fuel bays, positioned strategically between mining areas E29 and E38 (L425 and L650 connecting ramps). The total fuel consumption estimated for the complete life of mine is 40,300,000 L. During a full production year, average fuel consumption is estimated at 4,100,000 L per year.

A 2-inch steel pipe and automated pumping system will ensure constant flow to both fuel stations. The fuel line can be installed during pre-production during the main ramp development. Two 32 m parallel holes (for redundancy) will be drilled between the main ramp and the fuel station (L425) as soon as development permits. Two additional 310 m diamond-drilled holes will be drilled between level L425 and the L650 fuel station. Both pumping systems use automated valves to ensure full and independent automation of each fuel bay. Float level sensors connected directly to the LTE system will determine when refilling is needed.

16.7.5 Compressed Air and Water Supply

Compressed air is provided by two 1,476 cfm electric compressors during pre-production. They will be replaced by two additional units in the newly built compressor room. The two temporary units will be relocated alongside the others in the compressor room.

The compressed air piping network will be installed along the ramp, main drifts, and egress ways throughout the mine. Compressed air will provide power to the pumps for dewatering development work and handheld drills (for specific and limited use in planned development), as well as emergency air to the refuge stations.

The main service ramp will have 6″ pipes for water and compressed air, and a 4″ pipe for water pumped to surface. A 10″ pipe will also be installed in the service ramp to feed the paste fill plant with mill slurry from the surface. The conveyor ramp will have a 6″ pipe to pump clean water from the main dewatering station to surface. For more details on process water management, refer to Section 16.7.8.

Internal ramps are designed with 4″ pipes for water and air, depending on the estimated pressure. Levels and haulage drifts are designed with 2″ pipes for water and air. Water pressure reducing valves are installed every level to ensure proper control of water pressure.

16.7.6 Historical Mine Dewatering

The estimated total volume of water in the decommissioned mine is approximately 1,000,000 m^3 (Caumartin et al., 2018). The preferred option is to dewater the former mine through the 518 m historic shaft, which has a 55° inclination. Based on the current plan, an average pumping flowrate of 200 m^3/h is planned to completely dewater the former mine before development reaches that section. To do so, dewatering can be completed in parallel with the main ramp development. The estimated dewatering time is 7 months and could be completed between October 2024 (pre-production start) and October 2025 (target to reach mining area E29).

Multiple contractors have been consulted to evaluate the required installation to effectively pump 200 m^3/h. The solution selected is to lower four submersible pumps in series using a winch system (around 600 hp requirement). Since the shaft is inclined and the state of the old workings is unknown, multiples cables will be used to stabilize the system. The descent will be monitored every 5 metres to ensure that no problems occur and that the pumps are lowered safely. If ground or infrastructure conditions are too poor for the main option, an alternative solution is to dewater the mine using one or multiple targeted boreholes. Rigid casing and multistage pumps would be needed in this instance.

In terms of water quality, two samples of the flooded void water were taken by WSP in June 2021: one sample during the pumping phase and one at the end of development. The water quality testwork showed that the majority of constituents of concern based on analysis of the two void water samples occurred at concentrations that were lower than compliance levels; therefore, traditional treatment methods (dosing, reverse osmosis, ion exchange) are not needed provided the water chemistry data on which this analysis is based provides an accurate characterization of the void water.

While iron is observed to be in exceedance of the discharge limit, there is reasonable evidence based on known thermodynamic-redox relationships evidenced by a standard Pourbaix (iron) stability diagram that iron will not exceed the discharge limit.

Ausenco notes that the background iron concentration is also very close to the water quality guideline. This may be due to organic matter in the water or that relatively stagnant water is being tested. The following simple solution may provide an immediate fix in the vicinity of the discharge if a measurement of iron is considered a threat to discharge. By aerating (forcibly) the discharge this will accelerate oxygen dissolution from atmospheric gas into the water, which will raise the redox potential, rapidly shifting the environment (to one favourable to Fe^{3+}, rather than Fe^{2+}).

The following solutions may be employed:

- discharging the water from some height above the receiving environment (create splashing at the entry point)
- aerating using churning
- aerating using sparging

While any of these methods would accomplish aeration, it is noted that discharging from above the water surface would likely be associated with no cost.

16.7.7 Permanent Mine Pumping Network

The permanent mine pumping network design was completed in collaboration with Technosub. The design infiltration used is a conservative value of 95 m^3/h, based on an 100% increase value from the 48 m^3/h from the original hydrogeological study (Richelieu Hydrogéologie Inc., 2014).

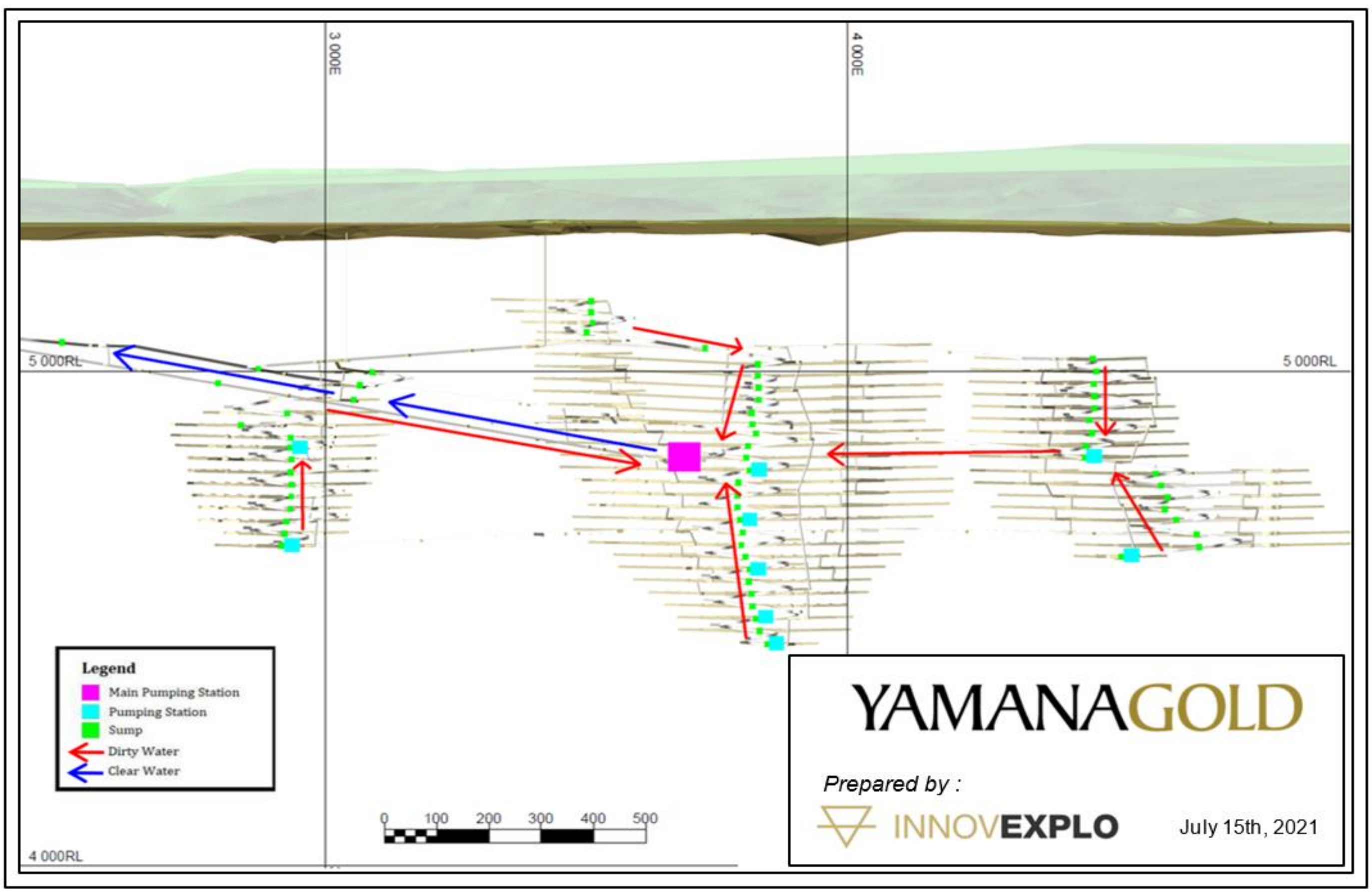
The equipment fleet water consumption is based on assumptions from past experiences. The maximum estimated water consumption is summarized in Table 16-27.

Table 16-27: Equipment Fleet – Water Consumption

Equipment	Quantity	Water Consumption (m^3h)	Total Water Consumption (m^3h)
Jumbo Two Booms	4	16	64
Bolter	6	16	95
Production Drill	3	18	55
Raise Bore	1	20	20
Water Truck	1	11	11
Total			245

Permanent dewatering is to be conducted with a system of drain holes and submersible pumps able to handle high solid content. Muddy water is transported by gravity whenever possible and strategically placed secondary pumping stations allow the water to be transported to the main pumping station (L500, located near main hub). Figure 16-21 illustrates the dewatering strategy of the project. The secondary pumping stations consist in a portable tank and one or two submersible pumps (Technoprocess 60HP or Tsurumi 20HP), depending on the estimated flow.

Figure 16-21: Dewatering Strategy General Overview



Muddy water is treated by a Mudwizard ® system. Two Mudwizard ® systems are installed in parallel; each with a maximum capacity of 182 m^3/h. Resulting mud is managed by a decantation set-up in adjacent drifts meant for this purpose; the dewatered mud is ultimately moved and used as backfill. Clear water is reused as process water throughout the mine (paste fill, equipment, etc.) using two Technojet MH80-125/4 (350 hp). The excess water is pumped to the surface through the conveyor ramp by two Technojet MH80-125/6 (500 hp) running in parallel.

16.7.8 Process Water

At full production, the maximum water consumption is estimated at 245 m^3/h (Table 16-25). Installation at the main pumping station will ensure adequate water flow all over the workplaces.

16.8 Mine Ventilation

The fresh air requirement has been determined to meet the Quebec Regulation Respecting Occupational Health and Safety in Mines (S-2.1, r. 14 - Règlement sur la santé et la sécurité du travail dans les mines). The fresh air rate to dilute emissions must be the one appearing on the certificate of homologation issued by the CANMET Mining and Mineral Sciences Laboratories, CANMET-MMSL.

To help represent a typical situation, a utilization rate has been applied to account for when machines may be mechanically unavailable or not in use. The utilization rates are 80% for production equipment and 50% for most service equipment and machinery that functions primarily with electricity. In addition, ventilation on demand (VOD) is planned for the underground mine to optimize the utilization of the ventilation.

Several pieces of equipment will operate on batteries (refer to section 104 of the RSSTM) to provide the minimum flow at the workplace. This air can theoretically be reused if the network allows it. A minimum flow of 13 kcfm (corresponding to a speed of 0.25 m/s) and a reuse rate of 25% is estimated for battery equipment. Air volumes of 45 kcfm as well as 12 kcfm are also added to dilute dust that might be generated from the conveyor and the crusher.

A contingency of 10% has been applied on the total estimated fresh air requirements to allow for additional equipment and workers that may be added during the life of mine. This includes an allowance for any potential leaks in the system. 850 kcfm is then established for the Wasamac Project and as the design criterion for the underground ventilation infrastructure. This air volume has been established for 2026.

16.8.1 Pre-production Ventilation Plan

Concerning ventilation in pre-production, the objective is to take advantage of the excavation of the two ramps excavated simultaneously. First, as it is presented in Figure 16-22, it is necessary to create a first loop by making two independent auxiliary circuits by the two ramps. Then, when the first branch connecting the two ramps is excavated, a temporary fan is installed in the conveyor ramp (near surface) to generate an internal ventilation circuit. The development of the ramps will thus be able to continue in this way with a flow rate of around 300 kcfm. Stage 7 of the pre-production strategy after the third ventilation loop is illustrated in Figure 16-23.

During development of both access ramps, a 48″ rigid duct will be used to send fresh air to the working faces. In the service ramp, one rigid duct will be used but two in parallel will be required in the conveyor ramp. During production, the rigid ventilation ducts will be removed and reused in other isolated working face before permanent ventilation is installed.

Figure 16-22: First Stage of Pre-production Ventilation Plan of Wasamac

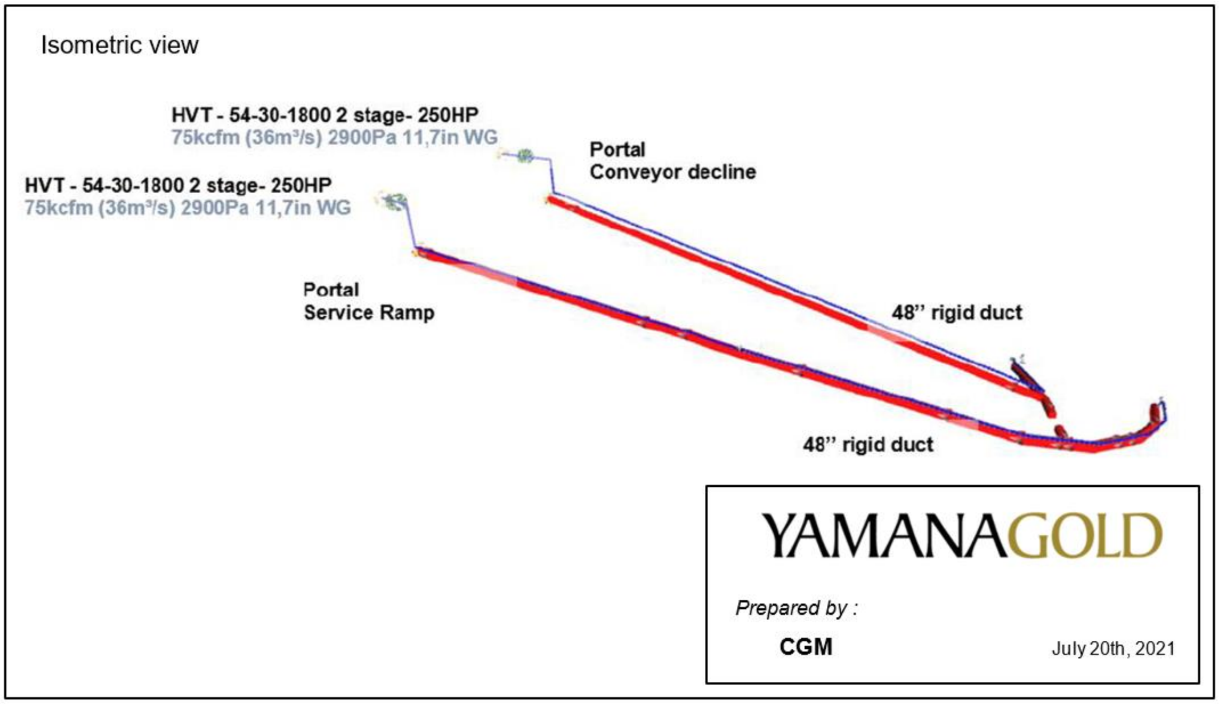
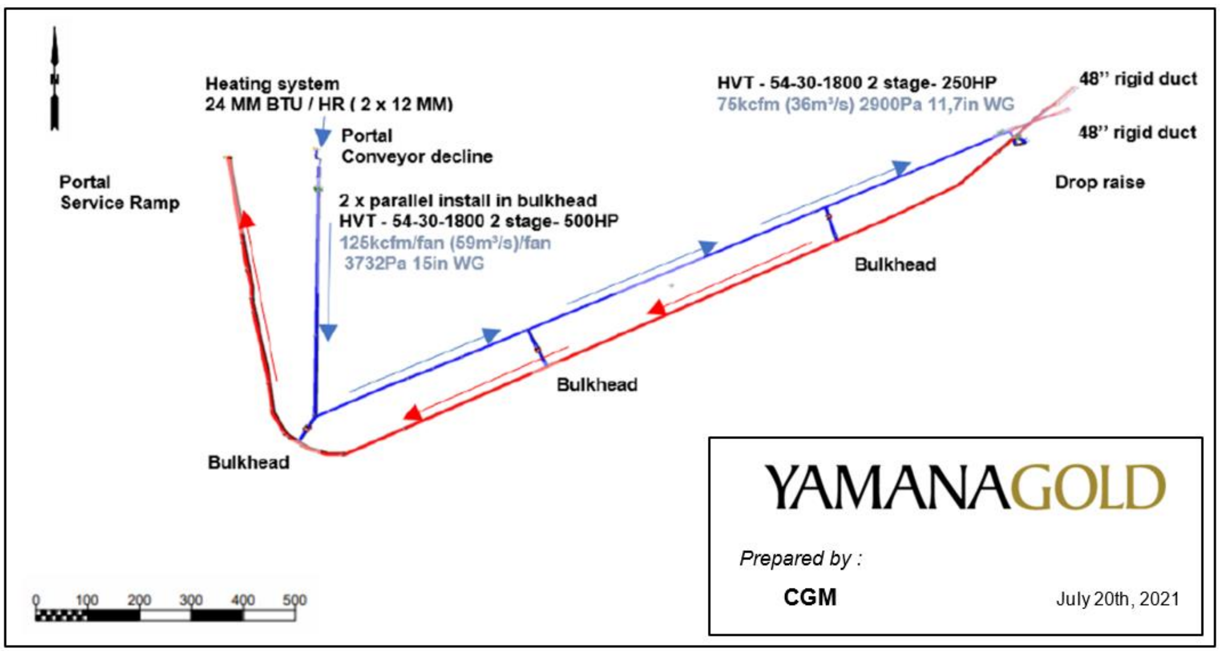


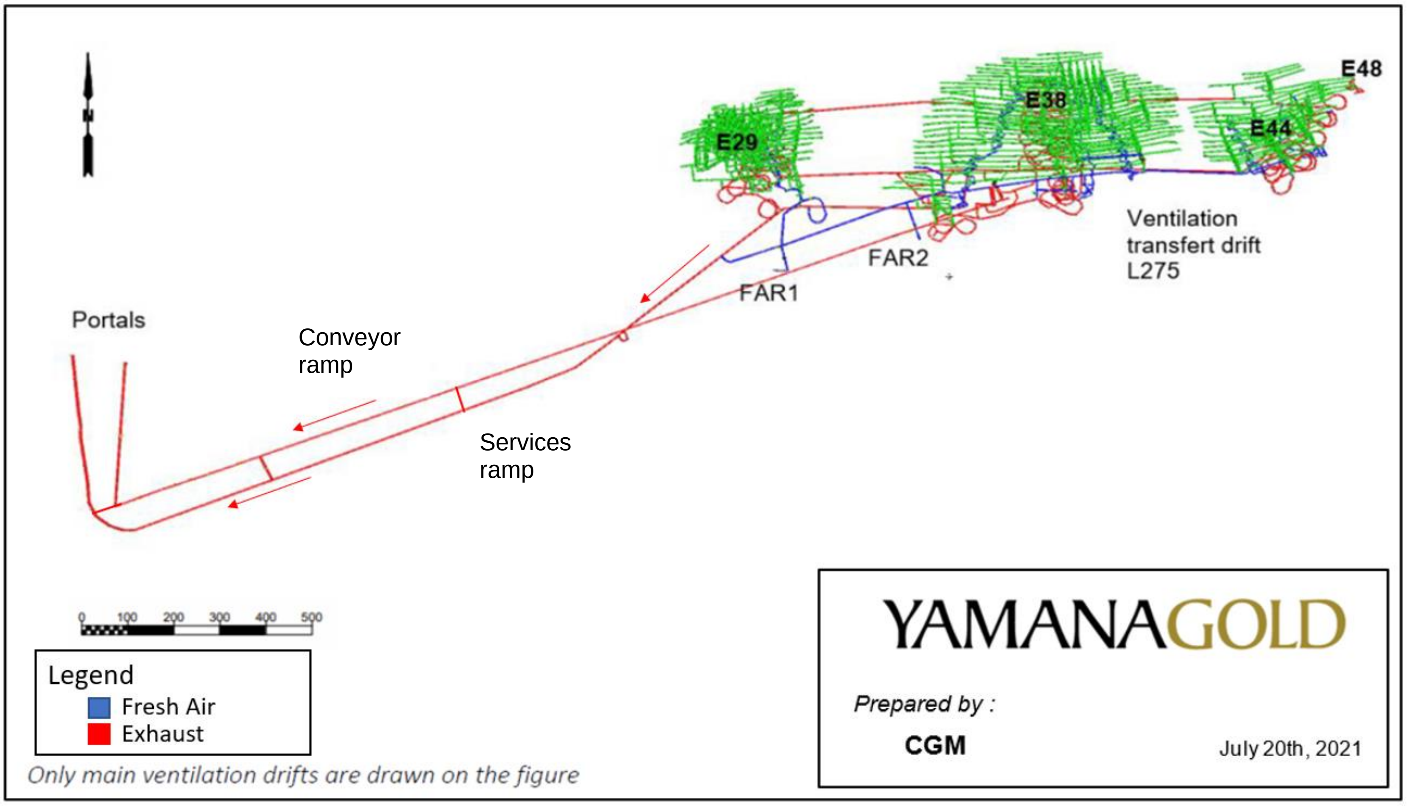
Figure 16-23: Plan View of Stage 7 of the Pre-production Ventilation Plan of Wasamac



16.8.2 Final Ventilation Network

As presented on Figure 16-24, the mining operation is divided into four mining areas: E29, E38, E44 and E48. All zones communicate with each other.

Figure 16-24: Plan View of the Wasamac Ventilation Final Network



As shown in Figure 16-25 and Figure 16-26, the intake system will consist of two intakes, FAR1 (ϕ = 3 m), FAR2 (ϕ = 4.5 m). The FARs are located on the west side of the mine; on level L275 the fresh air must go through a transfer drift to access mining area E38, E44 and E48. A system of doors will be installed where required in the internal ramp to ensure that the air is going into the internal raise system and not into the ramp. The main ramp and the conveyor decline are used as the main exhaust system. Several raises are necessary for the operation of different mining areas. The need will depend on the mining equipment used and the productivity for each zone. Internal drop raises for all zones are 2.4 m x 2.4 m and the raisebores are 4.0 m in diameter. It is planned that some ventilation raises will also serve as escapeways below level 650.

The fans and regulators will be controllable from a control room. Modulating these as needed on a weekly or daily basis will allow the system to obtain proper airflow when required. A dynamic tracking system is planned to maximize the efficiency of the ventilation network.

Figure 16-25: Longitudinal View of the Wasamac Ventilation Final Network

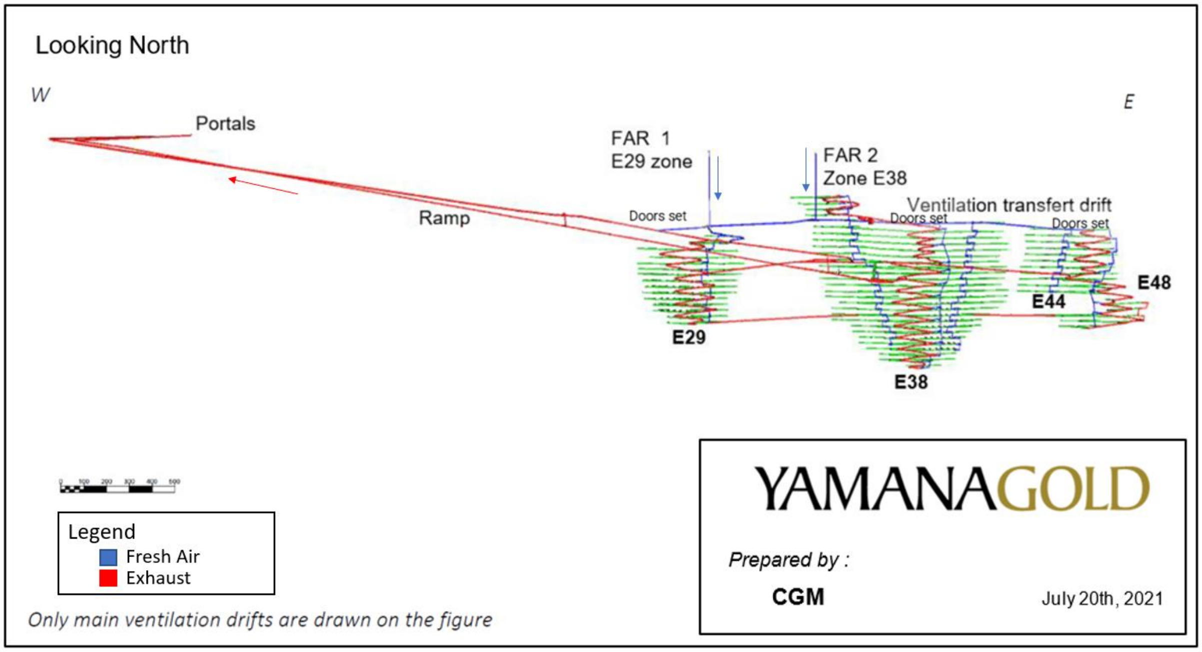
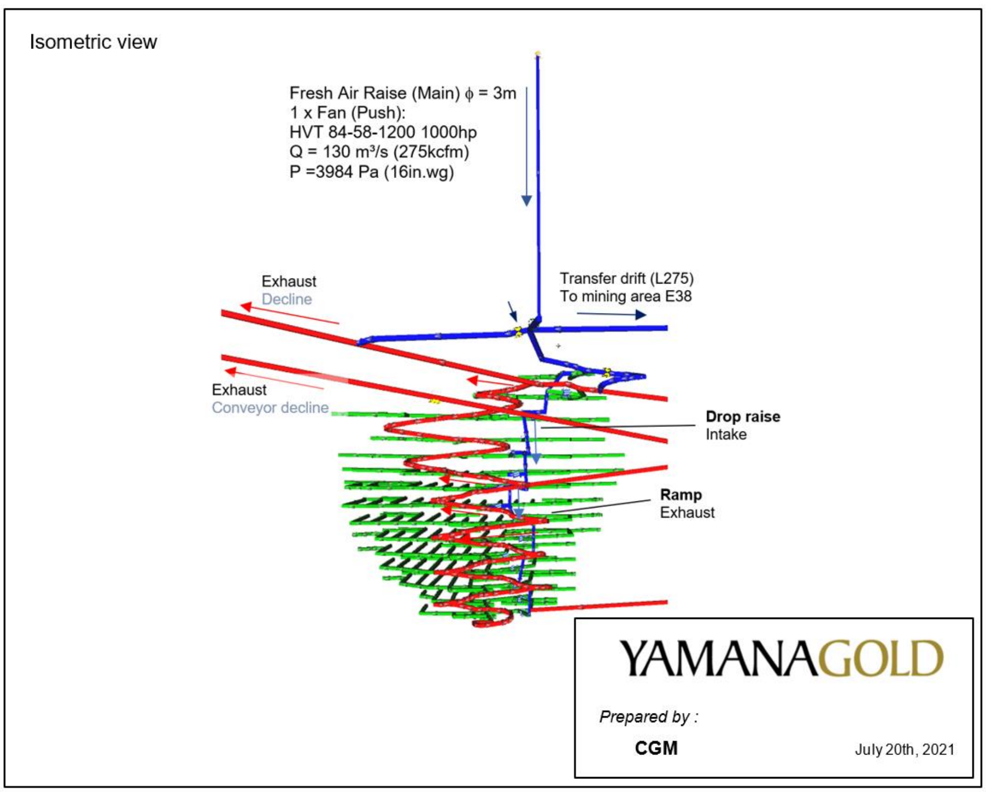


Figure 16-26: Mining Area E29 Ventilation Network



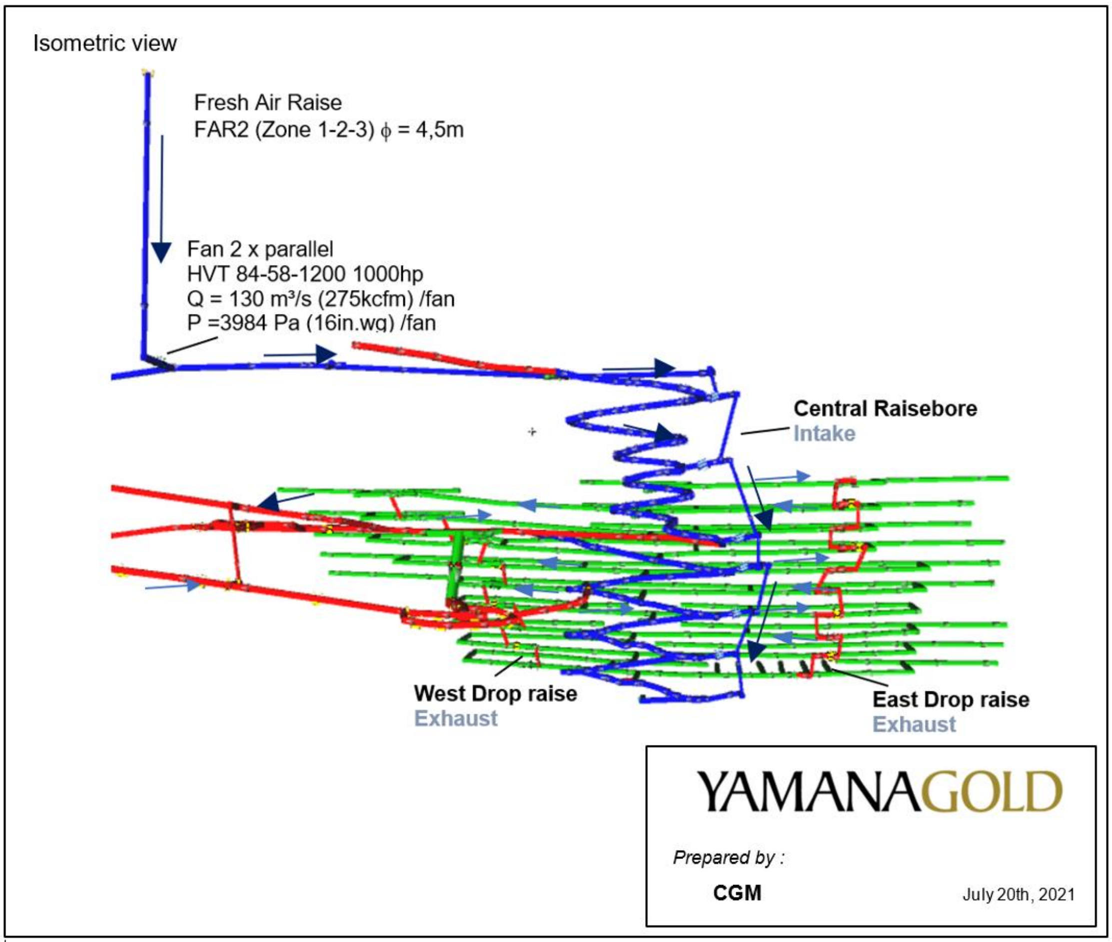
16.8.2.1 Mining Area E29 Ventilation Network

The ventilation network in mining area E29 uses the fresh air from FAR 1 (see Figure 16-26). The fresh air is transferred into a central system of raisebore with a 4 m diameter. From this raisebore system, automated regulators will control the flows on each level. The ramp is used as the exhaust.

16.8.2.2 Mining Area E38 Ventilation Network

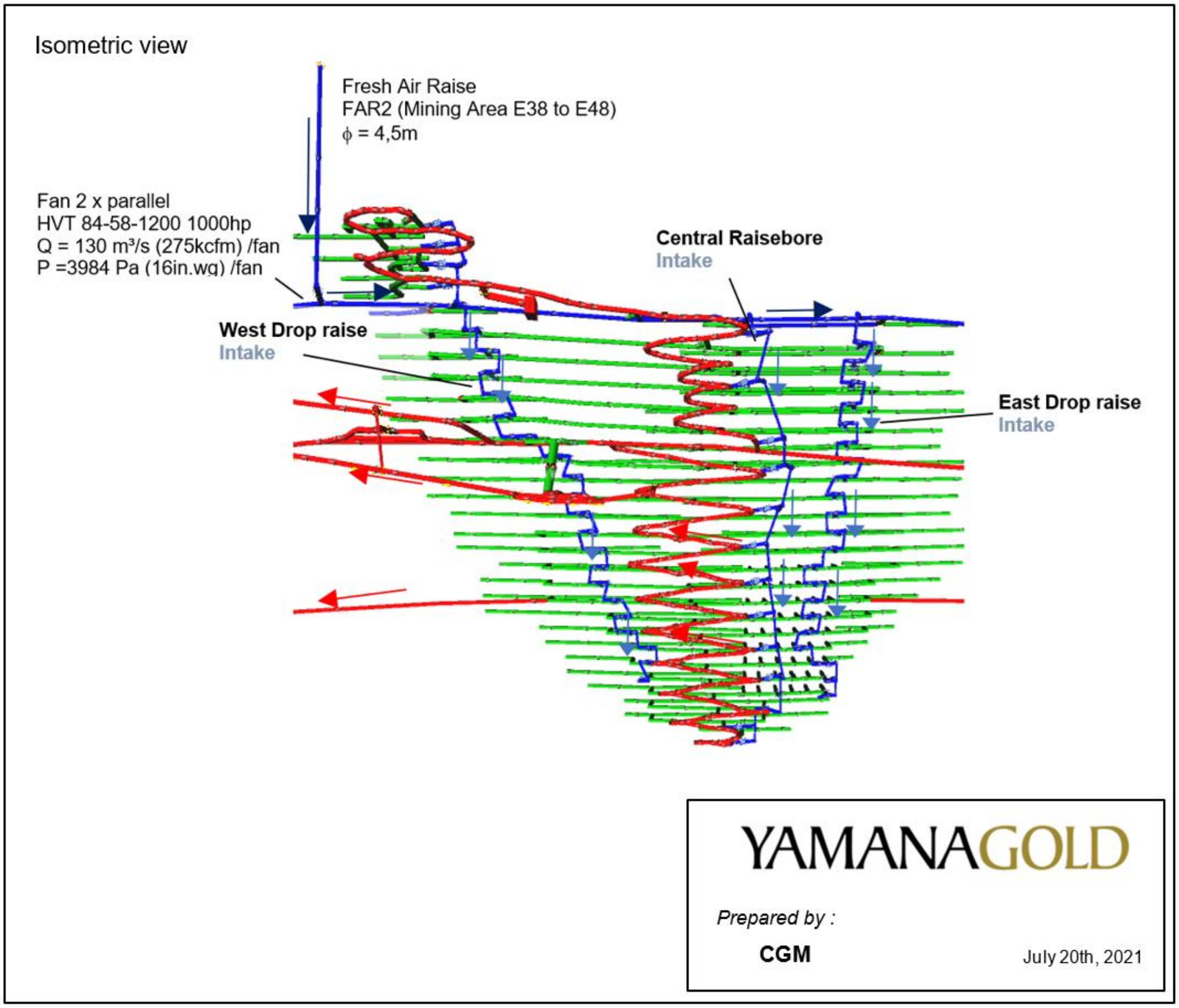
The ventilation of mining area E38 will be done in two stages. As presented on Figure 16-27, the first stage consists of ventilation by dilution from the ramp (intake). For this type of network, doors and controls must be installed in certain places to keep the circuits without recirculation. At that point, the drop raises are used to create a ventilation loop to reduce the secondary ventilation system requirements.

Figure 16-27: Mining Area E38 Ventilation Network, 2026



Once the drop raise system (east and west) network will breakthrough to the transfer drift on level L275, the second stage will be ready to be set up (Figure 16-28). It consists of ventilating the levels of fresh air from east to west from the drop-raise accesses. Automated regulators will control the air on the levels. The ramp is then exhausted.

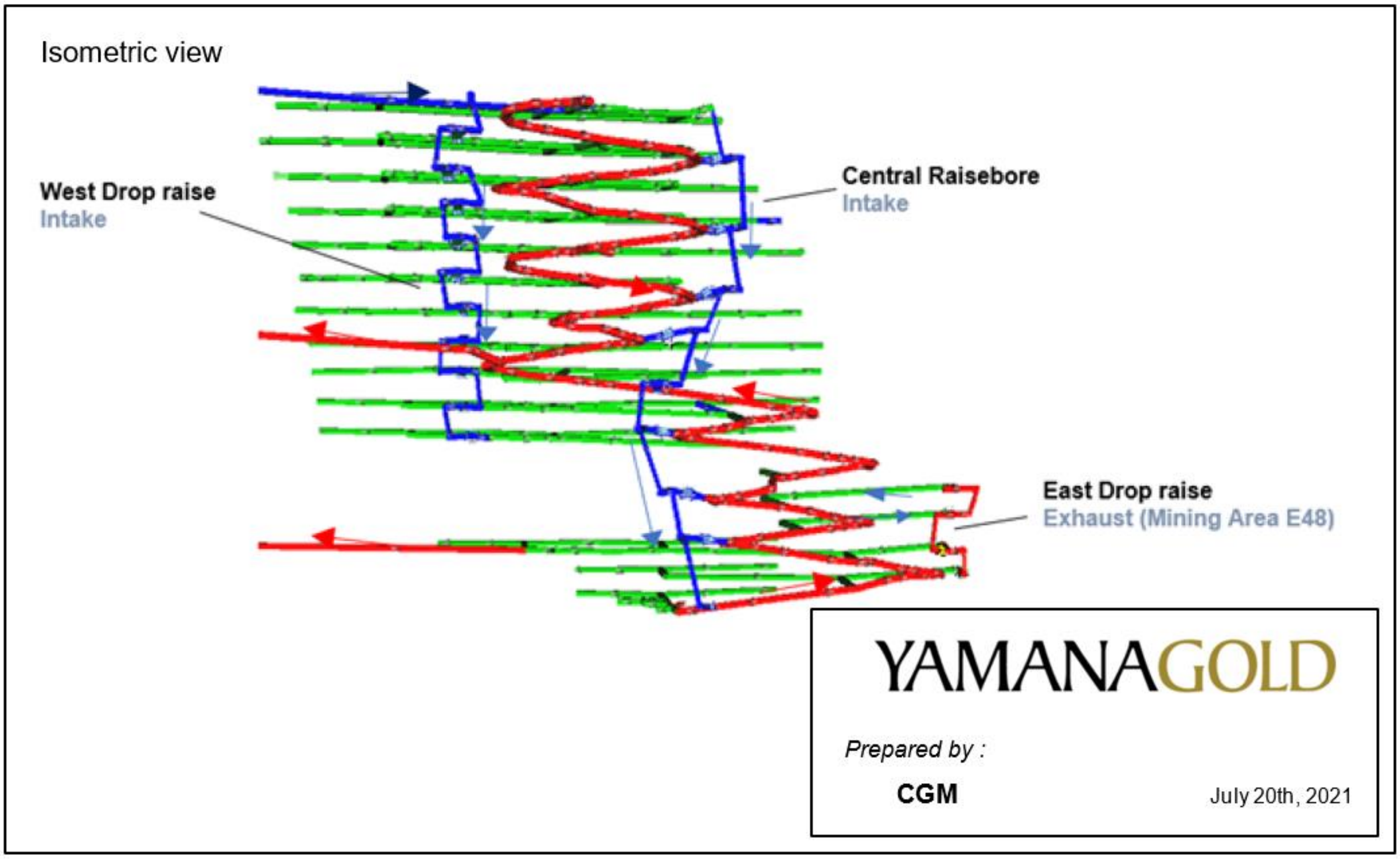
Figure 16-28: Mining Area E38 Ventilation Network, 2030



16.8.2.3 Mining Area E44 and E48 Ventilation Network

The fresh air of the mining area E44 and E48 is sourced from the transfer drift (L275). As per the mining area E29 and as presented on Figure 16-29, the air is transferred in a raise circuit and is distributed over the levels. The flows are controlled by automated regulators. The mining areas E44 and E48 use the same ventilation network. The area E48 requires a drop raise to create a ventilation loop to reduce the auxiliary set up that would be required for that zone otherwise. It is the same strategy as the one used in the longer level of the mining area E38.

Figure 16-29: Mining Area E44 and E48 Ventilation Network – Year 2030



16.8.3 Level Ventilation

Ventilation of the levels will be carried out by an automated regulator control and the air will be directed towards the sites or the working faces with 42″ flexible ducts. In other words, a ventilation duct will only be required when the ventilation will not flow naturally to parts of the drifts during its development and production activities.

Auxiliary fans will be used in combination with ventilation ducts to send fresh air to working areas that need it. Three models of auxiliary fans are required (40 hp, 50 hp and 75 hp). The 75 hp fans are planned for the major infrastructures such as the garage and crusher. For the production levels and working faces, 50 hp fans are planned, and for small infrastructures like the welding bay and powder magazine, 40 hp fans are planned.

16.8.4 Fan and Heating System

A fan and heater system was pre-selected based on the estimated air volumes and flow distribution following the production and development plan. The simulations were carried out using Howden Ventim Design™ software. Table 16-28 shows the selected model of fan and heater for the development period of both access ramps. Stage 1 refers to the period when the two ramps are developed in parallel. From stage 2, connections are developed between the two ramps to share the main ventilation systems and create loops. At that point, the main fans and heaters are installed at the portal of the service ramp.

Table 16-28: Pre-production Fan and Heater Capacity, Operating Points (simulated) from the Portal

Year	Fan Model	Heater Nominal Capacity	Nominal Power (hp/fan)	Total Pressure (Pa)	Airflow/Fan (m^3/s)
2024 Stage 1	2 x HVT - 54-30-1800 2 stage- TYPE 2000 (1 per duct)		250	2900	36
2024 Stage 2	2 x HVT - 54-30-1800 2 stage- TYPE 2000	HITT 24 MM BTU/h (2 x 12 MM)	500	3732	59

Table 16-29 shows the selected model of fan and heater for the FAR 1. The maximal pressure is obtained in 2033. From 2025 to 2036 the average pressure is around 3800 Pa with a constant pressure of 4450 Pa from 2028 and 2033. The main fans of the fresh air raises will be installed underground to minimize noise and footprint at surface.

Table 16-29: FAR 1 Fan and Heater Capacity, Operating Points (simulated)

Year	Fan Model	Heater Nominal Capacity (MM BTU/h)	Nominal Power (hp/fan)	Maximal Pressure (Pa)	Airflow (m^3/s)
2033	1 x HVT 84-58-1200_60Hz	HITT 24 - 26 MM BTU / HR (1X)	1000	4 641	131

Table 16-30 shows the selected model of fan and heater for the FAR 2. Before 2027 only one fan of the selected model would be required for that raise. Later, two fans in parallel will be required. The maximal pressure is obtained in 2029. From 2025 to 2036 the average pressure is around 3800 Pa with a constant pressure of 4,600 Pa from 2029 and 2033.

Table 16-30: FAR2 Fan and Heater Capacity, Operating Points (simulated)

Year	Fan Model(Pa)	Heater Nominal Capacity (MM BTU/h)	Nominal Power (hp/fan)	Maximal Pressure (Pa)	Airflow/Fan (m^3/s)
2026	1 x HVT 84-58-1200_60Hz	HITT 24 - 26 MM BTU/h (2X)	1000	1360	56
2027	2 x HVT 84-58-1200_60Hz	HITT 24 - 26 MM BTU/h (2X)	1000	3492	88
2029	2 x HVT 84-58-1200_60Hz	HITT 24 - 26 MM BTU/h (2X)	1000	4677	131

16.9 Underground Mine Equipment

16.9.1 Maintenance Schedule

The maintenance schedule is based on an overall underground maintenance team and is calculated in personnel workhours. Maintenance for all equipment is done following the supplier recommendation. To maximize equipment life, rebuilds are planned following supplier rebuild cycle recommendation. The maintenance of the equipment will be done at the underground service bay. Maintenance is planned to minimize downtime and maximize work in-between shifts, whenever possible. A full maintenance schedule will be established with the maintenance team at the start of the project in order to ensure efficient rotation of equipment. An overall mechanical availability of 85% or more is expected.

16.9.2 Mining Equipment List

The required operational quantity for all major and critical equipment (jumbo, cable bolter, production drills, LHDs, trucks, etc.) were estimated during the planning process. Yearly operation hours have been estimated for all other secondary services equipment based on typical operation and current mine scheduling requirements. For this secondary equipment, yearly operation hours range between 1,200 and 2,400 (20% to 50% utilization).

All equipment listed in this study is to be acquired by Yamana between 2024 and 2028. Everything costed in the cashflow does not include the equipment required before mine production; it is assumed contractors will provide the production/development fleet required during pre-production.

Battery electric vehicles (BEVs) will be favoured over diesel models whenever possible, most notably for service vehicles. A fleet of alternating batteries is needed throughout life of mine to satisfy the electrical needs of this equipment (provided by the supplier).

The required mobile equipment fleet by year is presented in Table 16-31.

Table 16-31: Mine Equipment List

Equipment Type	Model	Max.	2024	2025	2026	2027	2028 to 2035	2036
Major Equipment								
Jumbo - 2 Booms	Sandvik - DD422iE	4	0	0	2	4	4	2
975 Bolter	MacLean - 975	6	0	0	3	6	6	4
Production Drill	Sandvik - DL432i	3	0	0	1	3	3	2
Cable Bolter	Sandvik - DS422i	1	0	0	0	1	1	1
Production LHD – 21 t capacity	Sandvik - LH621i	10	0	0	1	7	10	10
Mine Truck – 63 t capacity	Sandvik - TH663i	7	0	0	2	5	7	4
Service Equipment								
Explosive Truck (Emulsion)	MacLean - EC3	2	0	0	1	2	2	2
Development Explosive Loader	Orica - Maxiloader 1081	2	0	0	1	2	2	2
Scissor Lift	MacLean - SL3	4	0	0	2	4	4	4
Boom Truck	MacLean - BT3	2	0	0	1	2	2	2
Personnel Carrier	MacLean - PC3	2	0	0	1	2	2	2
Mechanical Truck	MacLean - MT2	1	0	0	1	1	1	1
Shotcrete Sprayer	MacLean - SS5	1	0	0	1	1	1	1
Fuel-Lube Truck	MacLean - FL3	1	0	0	1	1	1	1
Underground Grader	MacLean - GD5	2	0	0	1	2	2	2
Transmixers	MacLean - TM3	1	0	0	0	1	1	1
Block Holer	MacLean - BH3	1	0	0	0	1	1	1
Electric Lift	MacLean - LR3	2	0	0	1	2	2	2
Water Truck	Getman	1	0	0	1	1	1	1
Excavator	Takushi	2	0	0	1	2	2	2
Services LHD - 3.5 t capacity	Sandvik – LH203	2	0	0	1	2	2	2
Light Vehicle - Tractor	Kubota	8	3	4	6	8	8	8
Light Vehicle	Toyota	6	2	3	5	6	6	6
Total Equipment		71	5	7	34	66	71	63

16.10 Mine Personnel

Mine personnel are split between three areas: technical services, maintenance underground (mechanical and electrical), and underground services (supervision, construction, development and production).

Operators and maintenance personnel generally work on a 5-4-4-5 schedule. This results in four crews alternating days and nights, when necessary. The electrical and mechanical supervisors will alternate day and night shifts at times; a supervisor or senior employee will always be present to oversee the shifts. Additional supervisors, technicians and some specific workers will work Monday to Friday on a 5-2 schedule, day shifts only.

16.10.1 Mine Operations, Services and Construction

The operators include those required for the major and secondary equipment, as well as blasters. Underground supervision includes a supervisor, trainer, and some operators related to service tasks. The list of underground operation, services and construction personnel required over the life of the mine is presented in Table 16-32.

Table 16-32: Operation, Services and Construction Personnel List

	MAX	2024	2025	2026	2027	2028	2029	2030 to 2035	2036
Operators									
Jumbo Operator	16	0	0	8	16	16	16	16	8
Bolter Operator	24	0	0	12	24	24	24	24	15
Development Service	20	0	0	9	20	20	20	20	1
Blaster	24	0	0	10	23	24	24	24	15
Production Drill Operator	12	0	0	2	10	12	12	12	8
Cable Drill Operator	5	0	0	2	4	5	4	4	2
Scoop Operator	44	0	0	4	27	40	43	44	41
Truck Operator	28	0	0	6	17	26	28	28	13
Underground Supervision									
Mine Superintendent	1	1	1	1	1	1	1	1	1
Mine Captain	2	2	2	2	2	2	2	2	2
Supervisors	4	2	2	3	4	4	4	4	4
Production Technician	1	0	0	1	1	1	1	1	1
Mine Trainer	1	1	1	1	1	1	1	1	1
Paste Backfill Service	8	0	0	0	4	8	8	8	8
Support Service	6	0	0	3	6	6	6	6	6
Grader Operator	4	0	0	2	4	4	4	4	4
Crusher/Hammer Operator	4	0	0	2	4	4	4	4	4
Construction									
Construction Miner	8	0	0	4	8	8	8	8	8
Shotcrete Construction	4	0	0	2	4	4	4	4	4
Total	215	6	6	74	180	210	214	215	146

16.10.2 Underground Service and Maintenance Personnel

Maintenance staff includes mechanics and electricians for the underground mine; the crew includes a full operational team able to fulfil preventive and unplanned maintenance. A list of underground maintenance personnel required over the life of the mine is presented in Table 16-33.

Table 16-33: Underground Maintenance Personnel List

Maintenance Underground Services	MAX	2024	2025	2026	2027	2028 to 2036
Mechanical Department						
Maintenance Superintendent	1	1	1	1	1	1
Surface Supervisor	1	1	1	1	1	1
Fix Equipment Supervisor	1	1	1	1	1	1
Mechanics Supervisor	1	1	1	1	1	1
Maintenance Planning Supervisor	1	0	0	1	1	1
Maintenance Planner	1	0	0	1	1	1
Reliability Technician	1	0	0	1	1	1
Senior Mechanic	4	2	2	3	4	4
Feed Mechanic	4	0	0	2	4	4
Electro Mechanic	4	0	0	2	4	4
Fuel & Lube Attendant	4	0	0	2	4	4
Junior Mechanic	8	0	0	4	8	8
Welder (Day + Night)	8	0	0	4	8	8
Fixe Mechanic Surface	2	0	0	1	2	2
Fixe Mechanic Underground	2	0	0	1	2	2
Welder (Day)	2	0	0	1	2	2
Loader Operator	2	0	0	1	2	2
Mechanics Construction	2	2	2	2	2	2
Electrical Department						
Maintenance Assistant Superintendent	1	1	1	1	1	1
Electrical Supervisor	1	1	1	1	1	1
Instrumentation Technician	1	0	0	1	1	1
Electrician (Day)	4	4	4	4	4	4
Electrician (Day + Night)	8	4	4	6	8	8
Electrician Construction	2	2	2	2	2	2
Total	66	20	20	45	66	66

16.10.3 Technical Services

Most of the staff in technical services work at the mine site office during the day, with weekends off (5-2 schedule). A list of technical services personnel required over the life of the mine is shown in Table 16-34.

Table 16-34: Technical Services Personnel List

Technical Services	MAX	2024	2025	2026	2027	2028 to 2036
Geology						
Senior Geologist	1	1	1	1	1	1
Geologist	2	2	2	2	2	2
Resource Geologist	2	2	2	2	2	2
Geology Technician	2	2	2	2	2	2
Engineering						
TS Superintendent	1	1	1	1	1	1
Senior Planning Engineer	1	1	1	1	1	1
Planning Engineer	2	2	2	2	2	2
Planning Technician	2	2	2	2	2	2
Drill and Blast Engineer	1	0	0	1	1	1
Drill and Blast technician	2	1	1	2	2	2
Project Engineer	1	0	0	1	1	1
Project Technician	1	0	0	1	1	1
Ventilation Technician	1	1	1	1	1	1
Surveyor	4	4	4	4	4	4
Rock Mechanic Engineer	1	1	1	1	1	1
Ground Control Technician	1	1	1	1	1	1
Total	25	21	21	25	25	25

17 RECOVERY METHODS

17.1 Overall Process Design

The provided testwork was thoroughly analysed and several process flowsheets were considered in the initial stages of the feasibility study. Based on the analysis, the most appropriate process flowsheet was selected for the deposit's metallurgical testwork results and subsequent economic modelling. The unit operations selected are all typical for gold processing. The proposed flowsheet uses conventional processes for:

- crushing/grinding
- leach/carbon adsorption
- carbon desorption/electrowinning/refining
- cyanide destruction/tailings dewatering
- tailings filtration

Key process design criteria are as follows:

- design throughput of 7,500 t/d or 2.7 Mt/a, with a nominal throughput of 7,000 t/d
- crushing plant availability of 70%
- plant availability of 92% for grinding, leach plant, and gold recovery operations
- tailings filtration plant availability of 82.5%

The process plant has been specifically designed to process 7,500 t/d to provide additional capacity to both allow for potential future increases in mining rate and to ease the plant ramp up process.

17.2 Mill Process Plant Description

The process design is comprised of the following circuits:

- primary crushing of run-of-mine (ROM) material
- covered, crushed material stockpile to provide buffer capacity ahead of the grinding circuit
- semi-autogenous grinding (SAG) mill with trommel screen and pebble crusher followed by a ball mill with cyclone classification
- trash screening
- leach + carbon adsorption (L/CIP)

- cyanide destruction of tailings using SO_2/O_2 process
- tailings thickening
- tailings filter plant facility with drystack and collection pond
- acid washing of loaded carbon and pressure Zadra-type elution followed by electrowinning and smelting to produce doré
- carbon regeneration
- effluent water treatment prior to discharging to the James Bay basin

17.2.1 Plant Design Criteria

Key process design criteria for the mill are listed in Table 17-1 on the following page.

17.2.2 Primary Crushing & Stockpiling

The crushing circuit is designed for an annual operating time of 6,132 hours or 70% availability at a design capacity of 7,500 t/d (2.74 Mt/a) to allow for future capacity above the mining rate of 7,000 t/d and an easier ramp-up period for operations.

The primary crushing facility is located underground. Material is hauled from the mine and direct tipped into to the run-of-mine (ROM) bin, which has a static grizzly screen on top to remove oversize ore. A fixed rockbreaker is utilized to break oversize rocks at the feed to the ROM bin. Material from the ROM bin is crushed by a primary jaw crusher. The ROM bin is equipped with a vibrating grizzly feeder at 417 t/h to feed the jaw crusher. The crushed material is conveyed from the underground crushing pocket to a covered stockpile that provides approximately 8,000 tonnes of live storage. The stockpile disconnects crushing from the mill to allow for crusher maintenance.

The mill feed stockpile is equipped with two apron feeders to regulate feed at 317 t/h into the SAG mill via the SAG mill feed conveyor. Pebble lime is added at a constant rate to the SAG mill feed conveyor for pH control in leaching. Pebbles from the SAG mill are fed to a pebble crusher for size reduction, which discharges on the SAG mill feed conveyor to recycle to the SAG mill.

The material handling and crushing circuit includes the following key equipment:

- ROM hopper
- vibrating grizzly
- fixed rockbreaker
- primary jaw crusher
- mill feed apron feeders (equipped with variable speed drives or “VSDs”)
- material handling equipment (including conveyors to move the material from the underground crushing pocket to the stockpile at surface)

Table 17-1: Key Design Criteria

Design Parameter	Units	Value
Plant Throughput, Design	t/d	7,500
Gold Head Grade – Maximum for Design	g/t Au	3.67
Crushing Plant Availability	%	70
Mill Availability	%	92
Bond Crusher Work Index (CWi), 75th percentile	kWh/t	19.5
Bond Rod Mill Work Index (BWi), 75th percentile	kWh/t	16.9
Bond Ball Mill Work Index (BWi), 75th percentile	kWh/t	15.0
SMC Axb, 25th percentile	-	35.9
Bond Abrasion Index (Ai)	g	0.243
Primary Crusher		jaw crusher, 1150 mm x 760 mm
SAG Mill Dimensions		7.6 m dia. X 4.0 m EGL
SAG Mill Installed Power	MW	4.1, with VSD
Ball Mill Dimensions		6.1 m dia. X 8.8 m EGL
Ball Mill Installed Power	MW	6.0
Primary Grind size (P_{80})	µm	60
Leach + CIP Residence Time	h	35
Leach Extraction	% Au	90
Leach-CIP Operating Density	%wt solids	50
Leach Sodium Cyanide Addition	kg/t	0.6
Leach Hydrated Lime Addition	kg/t	1.0
Leach pH target	-	10.5-11
Leach Dissolved Oxygen Target	mg/L	20
Leach & CIP Tanks	#	3 + 7
Tonnes of Carbon per Elution Column	t	7.0
Loaded Carbon Grade, Average	g/t Au	2970
Detoxification Residence Time	min	70
Detoxification Tanks	#	1
Detoxification Feed CNWAD Concentration, Max (Design)	mg/L	150
Detoxification Discharge CN_{WAD} Concentration, Design	mg/L	<10
Detoxification SO_2 addition	SO_2:CN_{WAD} ratio	5.0
Detoxification lime addition	$Ca(OH)_2$:SO_2	1.0
Final Tails Thickener Underflow Density	%wt solids	63
Tailings Filtration – Type		horizontal fast-opening, 62 to 74 chambers, 3.5 m x 2.5 m plates, filter cloth
Filter Feed Pulp Density	%wt solids	63
Filter Cake Moisture	%wt H_2O	10-12

17.2.3 Grinding Circuit

The grinding circuit consists of a SAG mill followed by a ball mill in closed circuit with hydrocyclones. The circuit is sized based on SAG mill feed size of 80% passing 99 mm and a ball mill product of 80% passing 60 µm. The SAG mill slurry discharges through a trommel screen, where oversize pebbles are crushed in a pebble crusher then recycled to the SAG mill via conveyors. Trommel screen undersize discharges into the cyclone feed pumpbox. The SAG mill is powered by a VSD to accommodate changes in ore hardness.

The ball mill is fed by cyclone underflow. The ball mill discharges through a trommel and the oversize is screened out and discharged to a scats bunker. Trommel undersize discharges into the cyclone feed pumpbox.

Water is added to the cyclone feed pumpbox to obtain the appropriate density prior to pumping to the cyclones. Cyclone overflow at 33%wt solids is sent to a trash screen followed by a pre-leach thickener. Flocculant is combined with the feed to the thickener to improve the settling rate of the material. The thickener overflow is recycled as process water in the circuit. The thickener underflow continues to the leach & adsorption circuit.

The grinding circuit includes the following key equipment:

- 25 ft diameter x 13.0 ft effective grinding length (EGL) 4,100 kW SAG mill (equipped with VSD)
- 20 ft diameter x 29.0 ft EGL 6,000 kW ball mill
- 940 mm 132 kW pebble crusher (or equivalent)
- cyclone feed pumpbox
- classification cyclones
- trash screen
- high-rate pre-leach thickener, 22 m diameter

17.2.4 Leach & Adsorption Circuit

The leach-adsorption circuit consists of three leach tanks and seven carbon-in-pulp (CIP) tanks. The circuit is fed by pre-leach thickener underflow. Barren solution from electrowinning cells is periodically transferred to the leach circuit. The leach and CIP tanks have a total circuit residence time of 33 hours at 50%wt solids pulp density.

Oxygen is sparged to each tank to maintain adequate dissolved oxygen levels for leaching at 20 mg/L. Hydrated lime is added to further adjust the operating pH to the desired set point of 10.5 to 11 and cyanide solution is added to the first leach tank. Fresh/regenerated carbon from the carbon regeneration circuit via the carbon sizing screen is returned to the last tank of the CIP circuit and is advanced counter-currently to the slurry flow by pumping slurry and carbon. The carbon sizing screen is used to ensure that carbon fines are not introduced to the CIP circuit. Slurry from the last CIP tank flows to the cyanide detoxification tanks.

The intertank screen in each CIP tank retains the carbon whilst allowing the slurry to flow by gravity to the downstream tank. This counter-current process is repeated until the loaded carbon reaches the first CIP tank. Recessed impeller pumps are used to transfer slurry between the CIP tanks and from the lead tank to the loaded carbon screen mounted above the acid wash column in the elution circuit.

The leach and carbon adsorption circuit includes the following key equipment:

- leach/CIP tanks and agitators
- loaded carbon screen
- intertank carbon screens
- carbon sizing screen

17.2.5 Cyanide Destruction

CIP tailings at 49%wt solids flow by gravity to the carbon safety screen. The screen oversize (recovered carbon) is collected in a bin for potential return to the CIP circuit. The screen undersize is pumped to the cyanide destruction tank. The water used for acid rinse and carbon transfer is also included in the feed to detoxification circuit. The density of the feed to the detoxification circuit is less than the CIP tailings at 48%wt solids.

The tank operates with a total residence time of approximately 70 mins to reduce weak acid dissociable cyanide (CN_{WAD}) concentration from a maximum of 150 mg/L to less than 10 mg/L.

Cyanide destruction is accomplished using the SO_2/O_2 method. The reagents required are oxygen, lime, copper sulphate, and sodium metabisulphite (SMBS). The cyanide destruction tank is equipped with oxygen addition points and an agitator to ensure that the oxygen and reagents are thoroughly mixed with the tailings slurry. The detoxification tank feeds the tailings thickener by gravity flow.

The main equipment in this area includes:

- carbon safety screen
- cyanide destruction tank and agitator
- oxygen supply system

17.2.6 Tailings Thickening

Detoxified tailings are thickened before discharge to the paste plant or tailings filtration plant. The overflow of the thickener is reused as process water in the plant. Flocculant is combined with the feed to the thickener to improve the solids settling rate. The underflow is pumped to either the paste plant for use as backfill in the underground mine or to the tailings filtration plant. Excess water from the process water tank is sent to effluent treatment prior to discharge.

The main equipment in this area includes:

- high-rate thickener, 22 m diameter
- overflow pumpbox for transfer to pre-leach thickener process water tank
- thickener underflow / final tailings pumps (two-stage)

17.2.7 Tailings Filtration

Thickened tailings from the underflow will be pumped to either the tailings filtration plant, located northwest of the main processing plant or the paste plant. The pipeline discharge is collected in a tailings stock tank with a 2.5-hour residence time which then feeds two plate and frame pressure filters. The filters discharge at a rate of 412 t/h and 10%wt to 12%wt moisture to a conveyor which feeds a temporary stockpile. When the filter plant is not operational, the tailings will be sent to the paste plant.

Trucks are loaded from this stockpile and transport the filtered tailings to the drystack. The filtrate and wash water from the filter presses feed a clarifier. The underflow from the clarifier feeds back into the tailings stock tank while the overflow is recycled as process water within the filtration plant. Excess water from the tailings filtration plant is pumped back to the process water tank at the process plant.

The main equipment in this area includes:

- tailings stock tank
- plate and frame pressure filters: horizontal fast-opening, 62 to 74 chambers, 3.5 m x 2.5 m plates, filter cloth (x2)
- filtrate clarifier
- material handling of filtered tailings, including feeders and conveyors

17.2.8 Carbon Acid Wash, Elution & Regeneration Circuit

17.2.8.1 Carbon Acid Wash

Prior to gold elution, loaded carbon is treated with a weak hydrochloric acid solution to remove calcium, magnesium, and other salt deposits that could render the elution less efficient or become baked on in subsequent steps and ultimately foul the carbon.

Loaded carbon from the loaded carbon recovery screen flows by gravity to the acid wash column. Entrained water is drained from the column and the column is refilled from the bottom up with the hydrochloric acid solution. Once the column is filled with the acid, it is left to soak, after which the spent acid is rinsed from the carbon and discarded to the cyanide destruction tank.

The acid-washed carbon is then hydraulically transferred to the elution column for gold stripping.

The main equipment in this area includes:

- acid wash carbon column (7 tonne carbon capacity)
- hydrochloric acid feed pump
- spent solution discharge sump pump

17.2.8.2 Carbon Stripping (Elution) & Electrowinning

The gold stripping (elution) circuit uses the pressure Zadra process.

A high-cyanide, caustic solution is recirculated through a pressure elution column at 140°C to strip the precious metals from the carbon. The precious metal-rich solution from the column exchanges heat with barren solution going to the column. Cooled pregnant solution then flows through electrowinning cells to deposit the gold and silver on the cathodes before the solution is recycled back to the elution column.

The stripping circuit includes the following key equipment:

- carbon elution column (7-tonne carbon capacity)
- strip solution heater (propane-fired) with heat exchangers
- strip eluate, and pregnant solution tanks

17.2.8.3 Gold Room

Gold/silver sludge is recovered from the electrowinning cells and smelted to produce doré bars.

The gold-rich sludge is washed off the steel cathodes in the electrowinning cells using high-pressure spray water and transfers by gravity to the sludge hopper. The sludge is filtered, dried, mixed with fluxes, and smelted in an electric induction furnace to produce gold doré. The electrowinning and smelting process takes place within a secure and supervised gold room equipped with access control, intruder detection, and closed-circuit television equipment.

The electrowinning circuit and gold room include the following key equipment:

- electrowinning cells with rectifiers
- sludge pressure filter
- drying oven
- flux mixer
- induction smelting furnace with bullion moulds and slag handling system
- bullion vault and safe
- dust and fume collection system
- gold room security system

17.2.8.4 Carbon Regeneration

Carbon is regenerated in a propane-fired rotary kiln. Dewatered barren carbon from the stripping circuit is held in a 7-tonne kiln feed hopper. A screw feeder controls the rate of addition of carbon into the regeneration kiln, where it is heated to 650° to 750°C in an atmosphere of superheated steam to restore the activity of the carbon.

Carbon discharges from the kiln and is quenched in water prior to screening on a carbon sizing screen located on top of the CIP tanks to remove undersized carbon fragments. The undersize fine carbon gravitates to the carbon safety screen, whilst carbon screen oversize is directed to the CIP circuit.

As carbon is lost by attrition, new carbon is added to the circuit using the carbon quench tank. The new carbon is then transferred along with the regenerated carbon to feed the carbon sizing screen.

The carbon regeneration circuit includes the following key equipment:

- carbon dewatering screen
- regeneration kiln (propane) including feed hopper and screw feeder
- carbon quench tank

17.2.9 Effluent Treatment Plant

The effluent treatment plant (ETP) was designed with partial data for process water characteristics as testing was not previously carried out for certain analytes. The chemical signature of the mine water, including ammonia, nitrates, and hydrocarbon levels, was estimated based on similarly sized operating mines; the upper bookend was selected as design criteria for the plant. The design remains conservative until further characterization of the water to be treated is carried out.

Excess water from the process plant will be treated in the ETP to meet the emission limit values in the Metal and Diamond Mining Effluent Regulations (MDMER) 2021 discharge regulations as well as the Québec Directive 019 obligations.

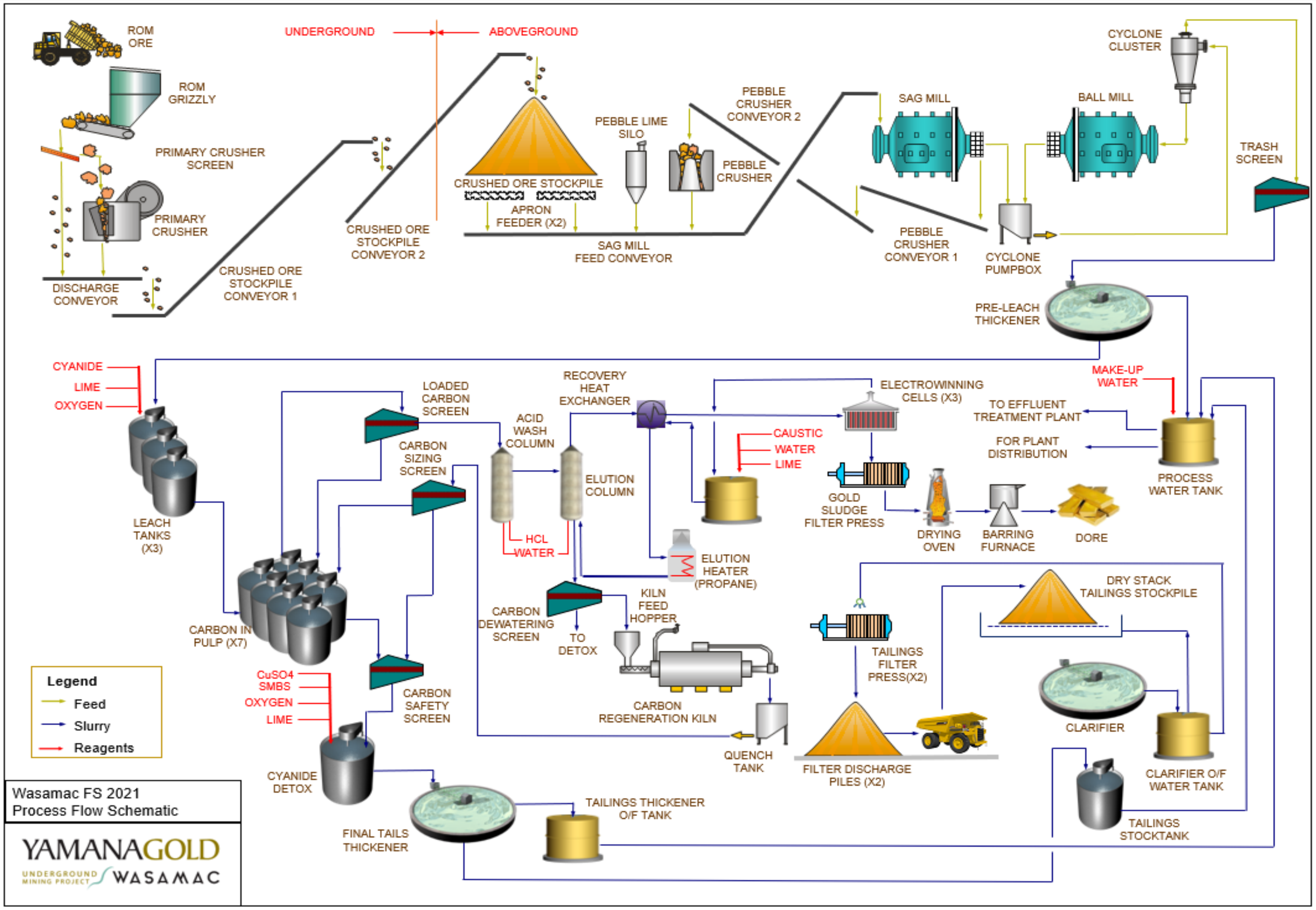
Dissolved metals removal by precipitation with lime will reduce contained metals (primarily copper) in solution to meet regulations. The precipitate sludge will be filtered in Geotubes and report to the TSF. The filtrate will report to a moving bed bofilm reactor for treatment that will subsequently reduce ammonia contained in effluent to meet MDMER 2021 and Québec Directive 019 guidelines for discharge treated water to the environment.

The ETP is sized to process 200 m3/d in order to maintain the nominal site water balance and have additional capacity to reduce holding pond levels after high rain or snow melt events.

17.2.10 Flowsheet & Layout

An overall process flow diagram showing the unit operations in the selected process flowsheet is presented in Figure 17-1. The paste plant is not included in this flowsheet. Views from the 3D model of the proposed plant are provided in Figure 17-2 to Figure 17-7.

Figure 17-1: Overall Process Flow Diagram



Source: Ausenco, 2021

Figure 17-2: Overall Plant Layout

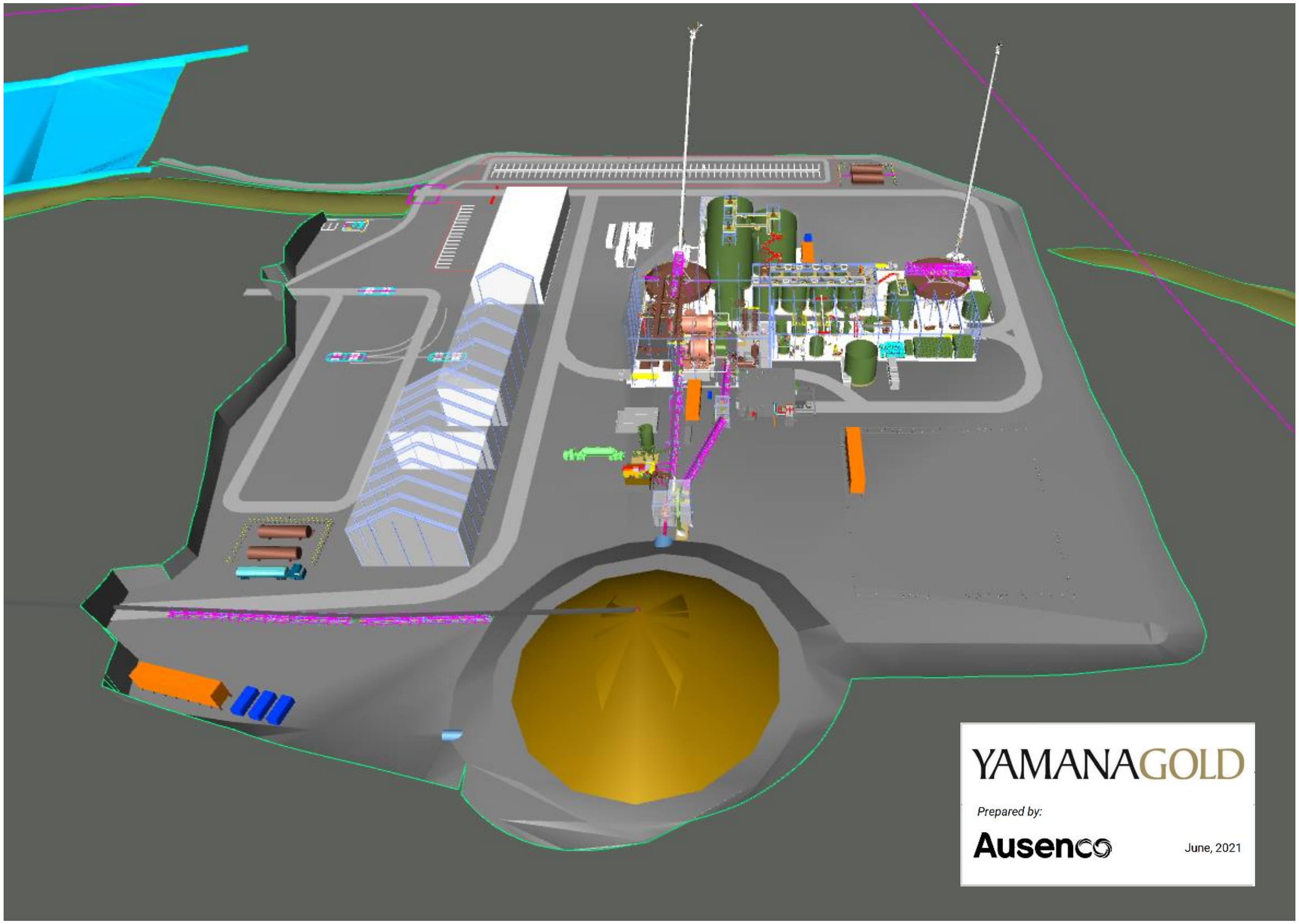


Figure 17-3: Tailings Filtration and Effluent Treatment Plants

Figure 17-4: Stockpile Area Section

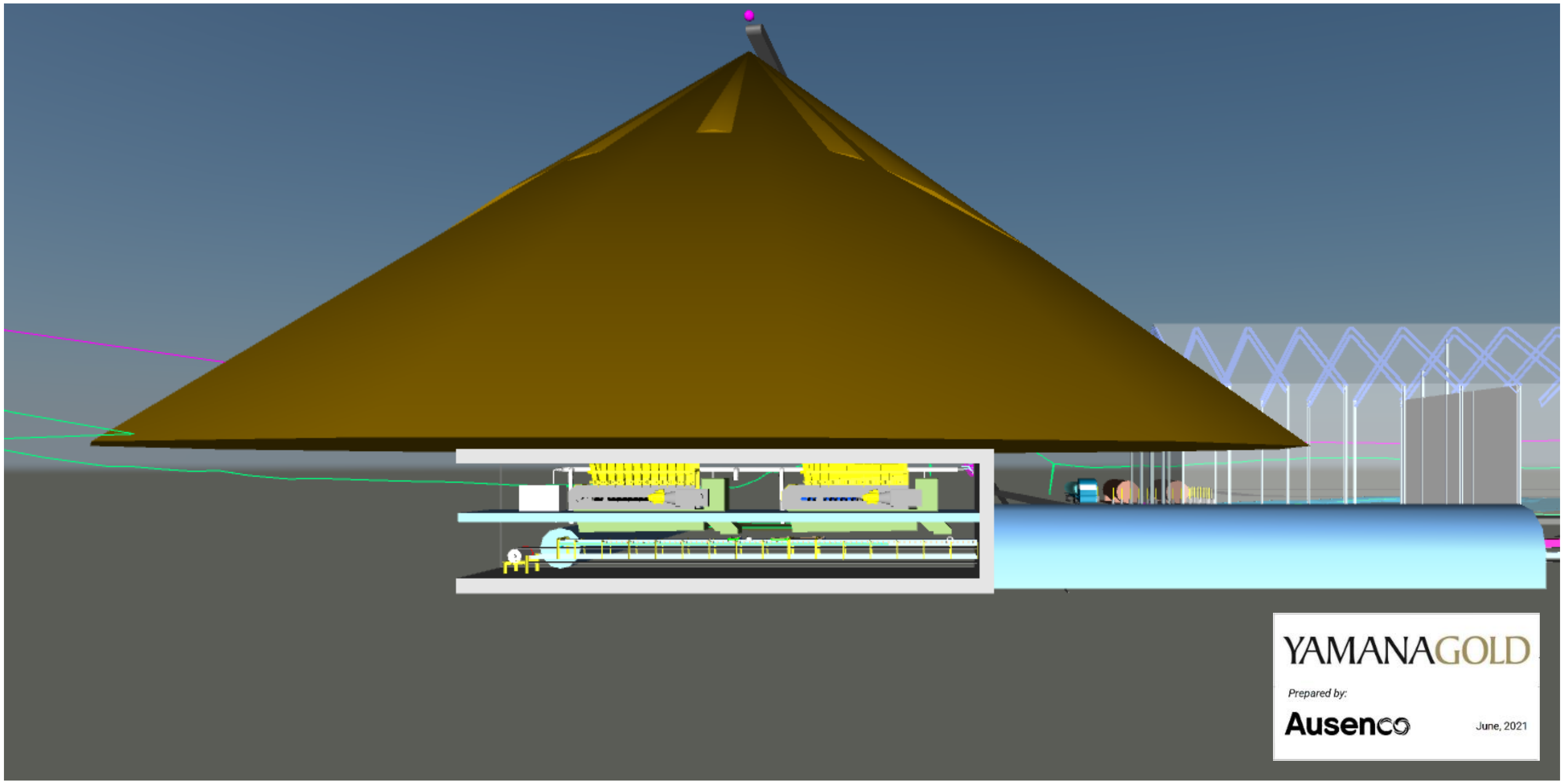


Figure 17-5: Grinding & Tank Area Section

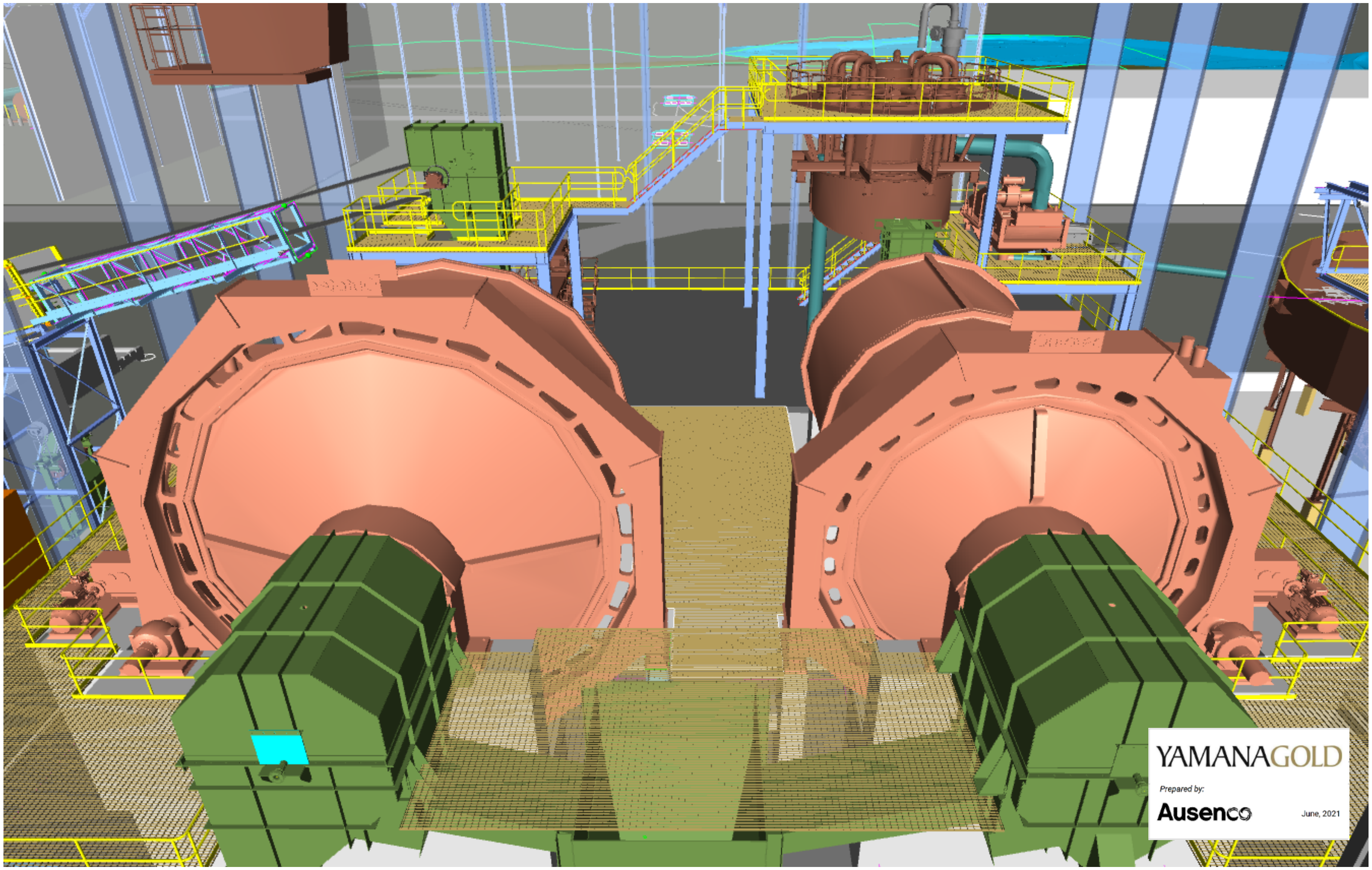


Figure 17-6: Plant Services Area Section

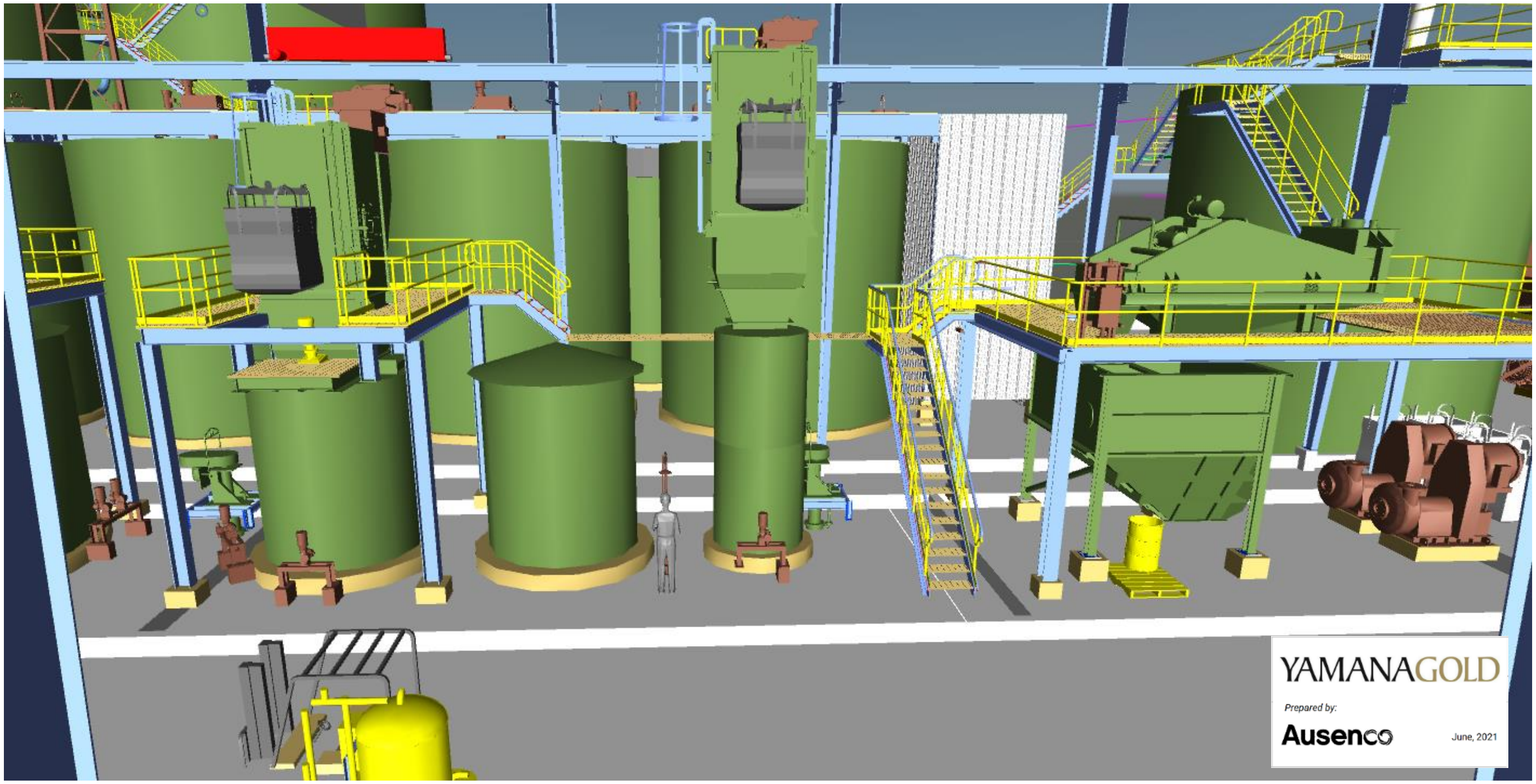
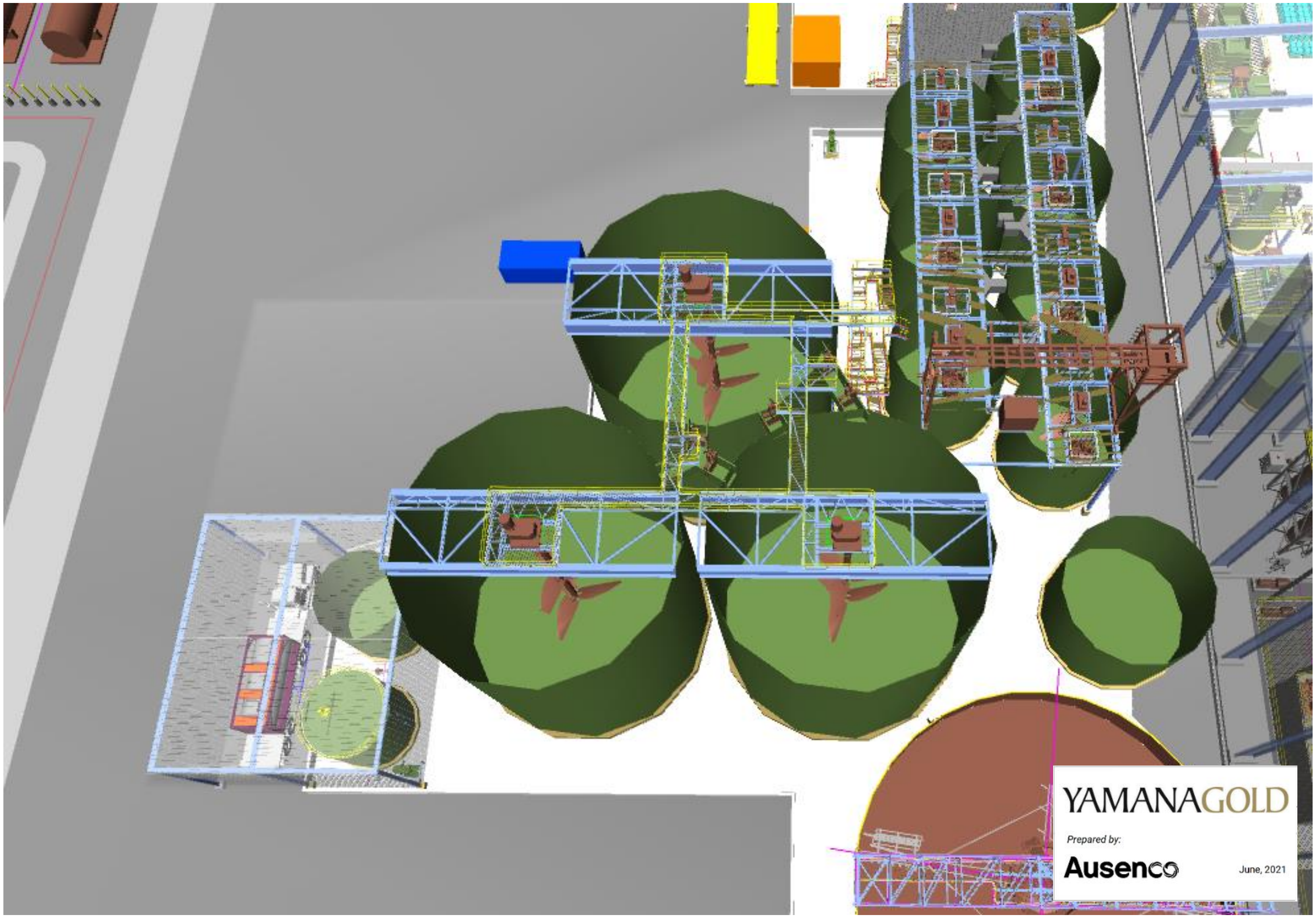


Figure 17-7: Leach/CIP Area Section with Cyanide Off-Loading



17.3 Reagent Handling & Storage

Each set of compatible reagent mixing and storage systems are located within curbed containment areas to prevent incompatible reagents from mixing. Storage tanks are equipped with level indicators, instrumentation, and alarms to ensure spills do not occur during normal operation. Appropriate ventilation, fire and safety protection, eyewash stations, as well as material safety data sheet (MSDS) stations are located throughout the facilities. Sumps and sump pumps are provided for spillage control.

The following reagent systems are required for the process:

- pebble lime
- hydrated lime
- sodium cyanide
- hydrochloric acid
- copper sulphate pentahydrate
- sodium metabisulphite
- sodium hydroxide
- flocculant
- activated carbon
- smelting fluxes
- liquid oxygen

17.3.1 Pebble Lime

Pebble lime is delivered in bulk and is pneumatically conveyed from the tanker to the pebble lime silo located adjacent to the SAG mill feed conveyor. Pebble lime is withdrawn from the lime silo via screw feeder and fed onto the SAG mill feed conveyor in a solid form for pH control in leaching as required. The dosing strength of the pebble lime is 90% CaO.

17.3.2 Hydrated Lime

Hydrated lime is delivered in bulk bags, which are lifted using a frame and hoist into the hydrated lime bag breaker on top of the mixing/storage tank. The solid reagent discharges into the tank and is slurried in process water to achieve the required dosing concentration of 25%wt. The slurried hydrated lime is pumped through a ring main with distribution points at the pre-leach thickener, in leaching, and in cyanide destruction. An extraction fan is provided over the lime bag breaker/mixing tank to remove reagent dust that may be generated during reagent addition/mixing.

17.3.3 Sodium Cyanide (NaCN)

Sodium cyanide is delivered to site in an isotainer containing about 18 tonnes as briquettes. Fresh water is recycled through the isotainer to dissolve the briquettes. Once the dissolution cycle is completed, the contents are transferred to the cyanide storage tank. Compressed air is used to remove any residual solution in the isotainer into the sodium cyanide mixing tank.

Sodium cyanide solution is delivered to the leach circuit and elution circuit at 20%wt strength with dedicated dosing pumps.

17.3.4 Copper Sulphate

Copper sulphate pentahydrate is delivered in solid crystal form in small bags and stored in the warehouse. Process water is added to the agitated copper sulphate mixing tank. A pallet of bags is lifted using a frame and hoist, and periodically a single bag is placed on the copper sulphate bag breaker on top of the tank. The solid reagent falls into the tank and is dissolved in process water to achieve the required dosing concentration.

Copper sulphate solution is transferred by gravity to the copper sulphate storage tank, which has a stacked arrangement with the mixing tank. Copper sulphate is delivered to the cyanide detoxification circuit at 20%wt strength using the copper sulphate dosing pump. An extraction fan is provided over the copper sulphate bag breaker/mixing tank to remove reagent dust that may be generated during reagent addition/mixing.

17.3.5 Sodium Metabisulphite (SMBS)

Sodium metabisulphite (SMBS) is delivered in the form of solid flakes in bulk bags and stored in the warehouse. Process water is added to the agitated SMBS mixing tank. Bags are lifted using a frame and hoist into the SMBS bag breaker on top of the tank. The solid reagent falls into the tank and is dissolved in water to achieve the required concentration. After the mixing period is complete, SMBS solution is transferred to the SMBS storage tank using the SMBS transfer pump. SMBS is delivered to the cyanide detoxification circuit at 20%wt strength using the SMBS dosing pump. An extraction fan is provided over the SMBS mixing tank to remove SO2 gas that may be generated during mixing. Monitoring and alarms will be installed to detect and warn of dangerous SO2 concentrations. The SMBS mixing area is ventilated using the SMBS area roof fan.

17.3.6 Sodium Hydroxide (NaOH)

Sodium hydroxide (caustic soda) is delivered in intermediate bulk containers (IBCs) as a solution and stored adjacent to the elution circuit until required. During winter months, the reagent concentration may be adjusted to prevent it from freezing in the IBCs. Dosing pumps automatically deliver the reagent at 35%wt strength to the required locations—elution circuit, electrowinning and cyanide mixing—to ensure the dosing requirements are met.

17.3.7 Hydrochloric Acid (HCl)

Hydrochloric acid is delivered in IBCs as a solution and stored adjacent to the elution circuit until required. Hydrochloric acid is mixed with raw water (inline) to achieve the required 3% w/v concentration. Hydrochloric acid is delivered to the acid wash circuit using the hydrochloric acid dosing pump.

17.3.8 Flocculant

Powdered flocculant is delivered to site in bulk bags and stored in the warehouse. A self-contained mixing and dosing system is installed, including a flocculant storage hopper, flocculant blower, flocculant wetting head, flocculant mixing tank, and flocculant transfer pump. Powdered flocculant is loaded into the flocculant storage hopper using the flocculant hoist. Dry flocculant is pneumatically transferred into the wetting head, where it is contacted with fresh water.

Flocculant solution, at 0.50% w/v, is agitated in the flocculant mixing tank for a pre-set period. After a pre-set time, the flocculant is transferred to the flocculant storage tank using the flocculant transfer pump. Flocculant is dosed to the pre-leach and tailings high-rate thickeners using variable speed helical rotor style pumps. Flocculant is further diluted to 0.05% w/v concentration just prior to the addition point.

An IBC is periodically filled and transported by pick-up truck to the filtration plant to dose the filtration clarifier. It is diluted just prior to the addition point at the clarifier.

17.3.9 Activated Carbon

Activated carbon from coconut shells, at a size of 1.7 mm x 3.35 mm, is delivered in solid granular form in bulk bags. When required, the fresh carbon is introduced to the carbon quench tank, where it is attritioned and screened prior to use in the CIP circuit.

17.3.10 Anti-Scalant

Anti-scalant is delivered as a solution in IBCs and stored in the warehouse until required. Anti-scalant is dosed without dilution. Positive displacement-style dosing pumps deliver the anti-scalant to the strip solution tank and filtration plant filter presses as needed.

17.3.11 Oxygen

Oxygen is injected into the leach tanks and in the cyanide detoxification tanks. For this purpose, bulk liquid oxygen is supplied by the vendor and is stored in a vendor supplied bulk liquid storage tank. From there, it goes through vaporizers, then feeds the leach and detoxification tanks as required.

17.3.12 Gold Room Smelting Fluxes

Borax, silica sand, sodium nitrate, and soda ash are delivered as solid crystals/pellets in bags or plastic containers and stored in the warehouse until required.

17.4 Services & Utilities

17.4.1 Plant / Instrument Air

High-pressure air at 700 kPa is produced by compressors to meet plant requirements. The high-pressure air supply is dried and used to satisfy both plant air and instrument air demand. Dried air is distributed via the air receivers located throughout the plant.

17.4.2 Plant Air – Filtration Plant

Compressed air at the filtration plant is injected into the plate and frame filter presses to release the dried cake onto the conveyor belt below. It is also used for the squeezing and drying stages of the filter press cycle.

17.5 Water Supply

The water supply for the process plant is described below. A more detailed description of the site water balance is provided in Section 18.

17.5.1 Fresh Water Supply System

Fresh water is supplied to a raw water storage tank. Raw water is used for all purposes requiring clean water with low dissolved solids and low salt content, primarily as follows:

- gland water for pumps
- reagent makeup
- elution circuit makeup
- fresh water is treated and stored in the potable water storage tank for use in safety showers and other similar applications
- fire water for use in the sprinkler and hydrant system
- cooling water for mill motors and mill lubrication systems (closed loop)

17.5.2 Process Water Supply System

Overflow from the pre-leach thickener, final tailings thickener, and filtration plant clarifier meet the main process water requirements. Mine waste water and mill contact water provide any additional makeup water requirements.

17.5.3 Gland Water

One dedicated gland water pump is fed from the fresh water tank to supply gland water to all slurry pumps in the main plant. At the filtration plant, one dedicated pump pulls clarifier overflow water and filters it for use as gland water for all filtration plant slurry pumps.

17.6 Reagent & Consumable Requirements

Reagent consumptions are based on testwork results and standard industry practices. A summary of the estimated reagent and consumables rates is shown in Table 17-2.

Table 17-2: Estimated Reagent Consumptions

Reagent	Form	Unit	Consumption
Activated Carbon	Coconut shell, grade 6 x 12 mesh	g/t feed	40
Copper Sulphate	Blue crystal, pentahydrate, 99.5% minimum purity	kg/t feed	0.22
Flocculant	Powder, 97.5% minimum purity	kg/t feed	0.10
Hydrochloric Acid	Liquid, 33%wt	m^3/strip	0.92
Pebble Lime	Coarse solids, 90% minimum available CaO	kg/t feed	3.49
Hydrated Lime	Powder, 90% minimum available CaO	kg/t feed	1.89
Sodium Cyanide	Briquettes, 98% minimum purity	kg/t feed	0.65
Sodium Hydroxide	Liquid, 50%wt	kg/t feed	0.29
SMBS	Flakes, 97.5% minimum purity	kg/t feed	1.23
Oxygen	Bulk liquid	kg/t feed	1.96
Anti-scalant	Liquid	kg/t feed	0.015
Sulphamic Acid	Liquid	g/t feed	5.0
Borax	Powder	kg/100 kg concentrate	60
Silica	Powder	kg/100 kg concentrate	30
Sodium Nitrate	Powder	kg/100 kg concentrate	5
Sodium Carbonate	Powder	kg/100 kg concentrate	5
SAG Mill Media	125 mm balls	kg/t feed	0.57
Ball Mill Media	50 and 75 mm balls	kg/t feed	0.87

17.7 Power

The process plant is calculated to have an installed total power of 19.0 MW, with a nominal operating demand of 12.2 MW. G&A services are expected to require an additional 1.9 MW, with a nominal demand of 1.3 MW. A breakdown of power consumption by WBS area is shown in Table 17-3.

Table 17-3: Estimated Power Consumption

WBS	Area	Installed (kW)	Nominal Demand (kW)
2800	Ore Receival and Crushing	1,923	1,442
3100	Crushed Ore Storage and Reclaim	169	119
3200	Grinding	11,822	7,676
3300	Leaching	1,727	1,311
3400	Elution and Goldroom	308	180
3500	Tailings Disposal	791	339
3600	Reagents	208	94
3700	Tailings Filtration Plant	896	386
3800	Process and Tailings Air & Water	1,039	545
3900	Process Buildings	112	77
Total		18,995	12,168
4200	HV Power Switchyard and Power Distribution	154	100
4400	Truck Shop and Fuel Storage	391	223
4500	Buildings	1,152	829
4800	Tailings Storage and Management Facilities	396	249
Total		1,940	1,301

18 PROJECT INFRASTRUCTURE

18.1 Introduction

The overall site plan (see Figure 18-1 on the following page) shows the major project facilities, including the underground portal access, mine vent raise collars, the process plant, filtration plant, tailings storage facility (TSF), tailings stockpile, TSF road and pipelines, waste storage areas, tailings pond, existing roads and railways, and the site access road. Access to the facility is from the south side of the property from a secondary local road off of Highway 117. Process plant access will be via the security gatehouse at the property access point.

The site will be fenced around the perimeter of the process plant and mine waste storage areas as well as the TSF. The process plant is located immediately north of Highway 117, behind a natural hillside so that as little of the plant as possible is visible from the highway. This location was selected to reduce the impact of noise and dust on local residents.

Site selection and location took into consideration the following factors:

- natural elevation changes to mitigate sound propagation
- proximity to the proposed mine ramp portals

18.2 Roads and Logistics

The project is accessed from Provincial Highway 117, which links Rouyn-Noranda and the community of Arntfield. The property is located approximately 15 km west-southwest of Rouyn-Noranda, which is serviced by daily flights to Montreal. The property is accessed from rang Jacques-Paquin off Provincial Highway 117 (the Trans-Canada Highway) that links Rouyn-Noranda and the community of Arntfield (see Figure 18-2).

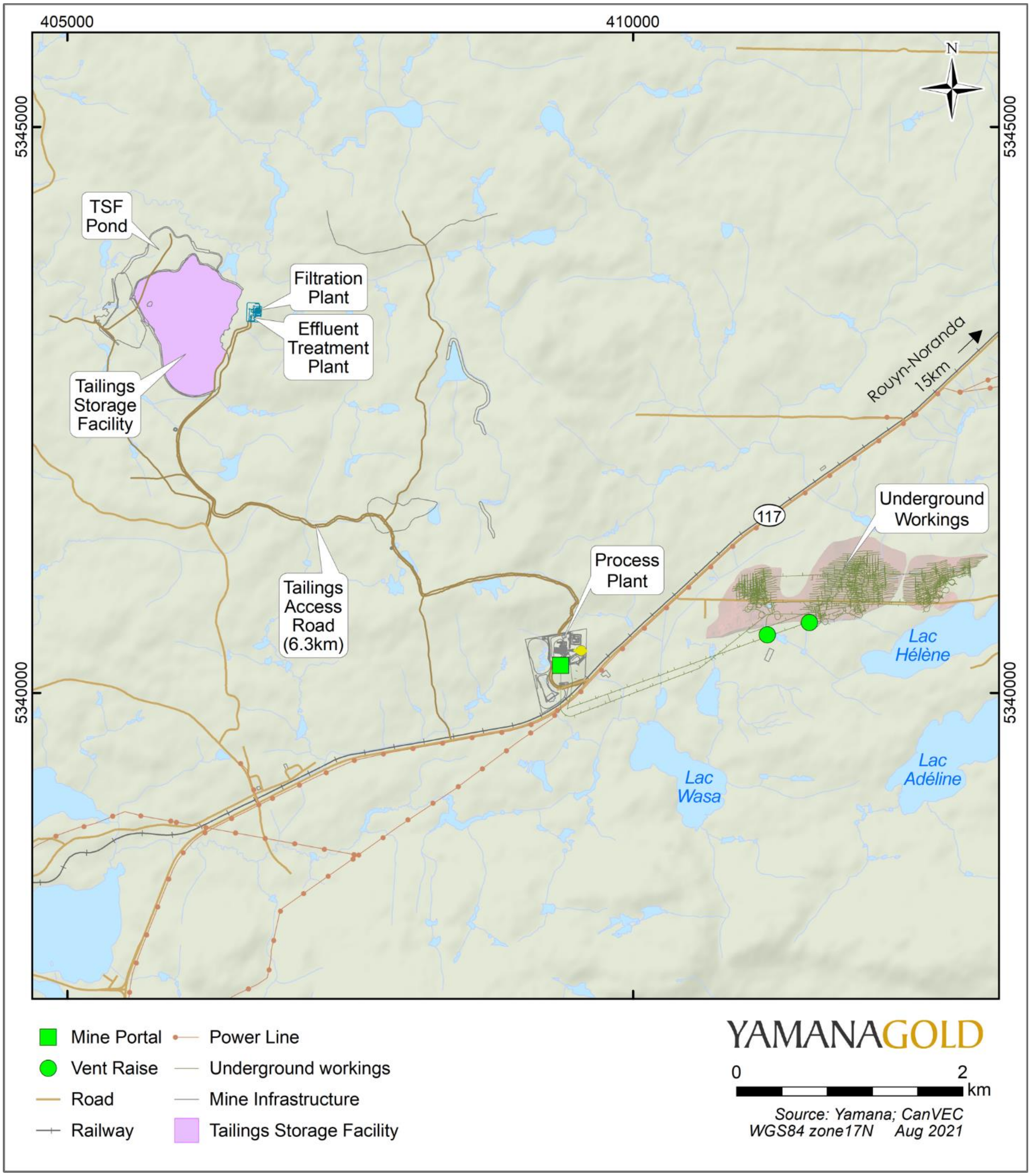
A secondary road (rang des Cavaliers) leads directly to the Wasamac deposit (above the underground mine) approximately 1 km east-northeast of the main project site along Highway 117.

Load and size limits for trucking goods to site are governed by the Quebec Highway Safety Code. The major provisions can be found in Quebec Regulation C-24.2, r. 31 of the Highway Safety Code.

18.2.1 Rail

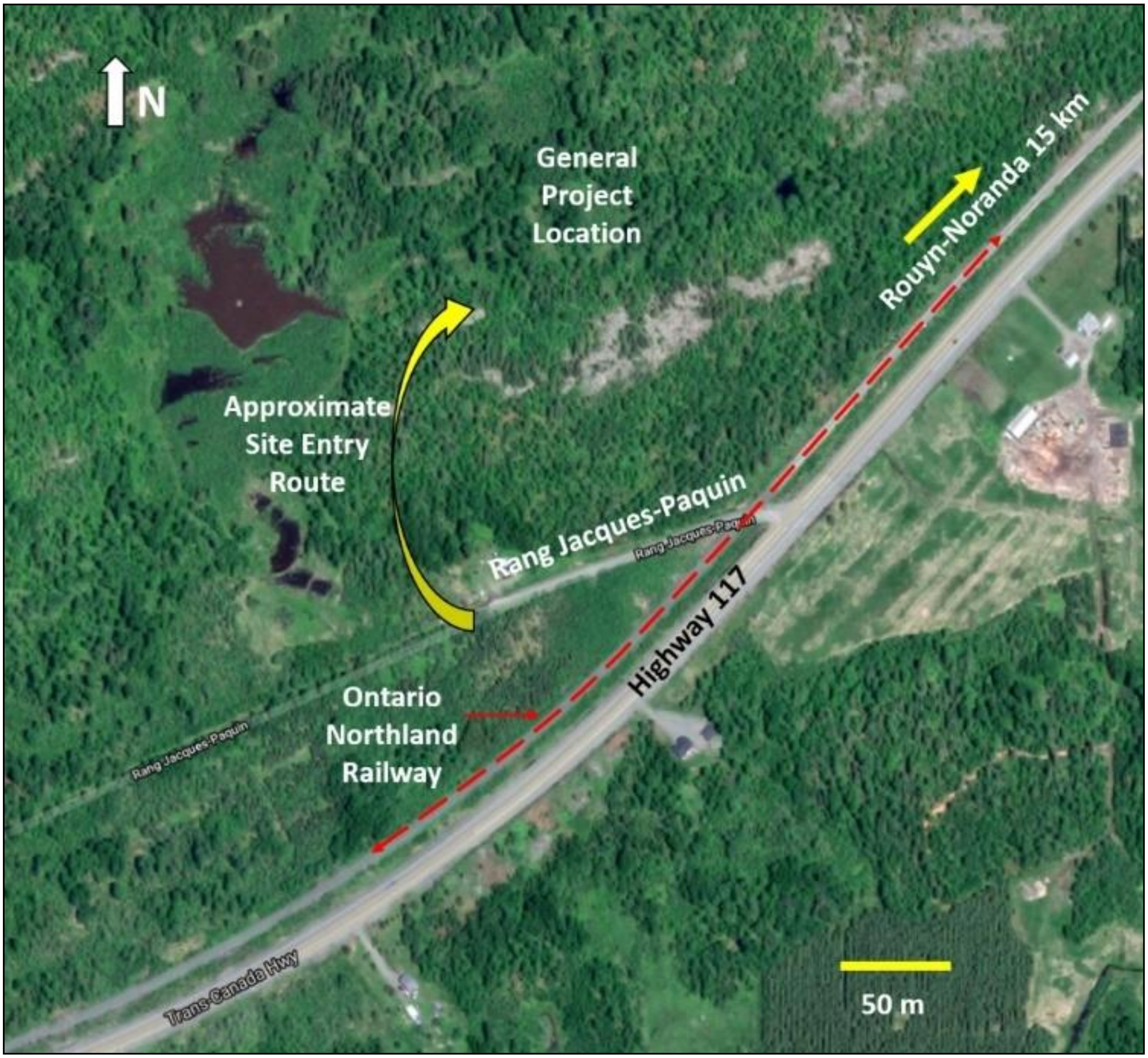
The Ontario Northland Railway runs immediately south of the of the property, parallel to Provincial Highway 117. The railway is situated between the property and Highway 117; a railway crossing is necessary on the main access road to the site entry, as identified in Figure 18-1.

Figure 18-1: Overall Site Plan



Source: Yamana, 2021

Figure 18-2: General Site Entry



Source: Revised from Google Maps, 2021.

18.2.2 Air

There is no airstrip on the project site. Nearby airport facilities are listed in Table 18-1.

Table 18-1: Airport Facilities

Airport	Distance to Site (Road Travel)
Rouyn-Noranda Airport (YUY) (Regional)	38 km
Val D'Or Airport (YVO) (Regional)	133 km
Timmins Airport (YTS) (Regional)	223 km
North Bay/Jack Garland Airport (YYB) (Regional)	270 km
Greater Sudbury Airport (YSB) (Regional)	340 km
Kapuskasing Airport (YYU) (Regional)	341 km
Ottawa International Airport (YOW) (International)	563 km
Toronto Pearson Airport (YYZ) (International)	602 km
Montréal-Pierre Elliot Trudeau Airport (YUL) (International)	650 km
Sault Ste. Marie Airport (YAM) (International)	661 km

18.2.3 Port Facilities

Nearby port facilities are listed in Table 18-2.

Table 18-2: Port Facilities

Port	Distance to Site (Road Travel)
Toronto Port	637 km
Old Port of Montreal	640 km
Port de Trois-Rivieres	740 km
Port Saguenay	847 km
Port of Thunder Bay	954 km
Port of Sept-Iles	1367 km

18.2.4 Access Road and Security

Year-round access to the site will be made available via the main access road off rang Jacques-Paquin. Highway 117 will be expanded to add deceleration/acceleration lanes in both directions with sufficient length for appropriate cueing as personnel turn onto the site access road or merge with Highway 117 traffic.

The section of rang Jacques-Paquin from which the access road will branch consists of aged asphalt requiring stripping, widening, and construction of a new road base. The main access road from Jacques-Paquin to the process plant site has not been developed; typical road building works will be necessary along this 520 m section. The entire access from Highway 117 to the process plant site will be constructed as a dual-lane road for conventional commercial transport traffic.

Immediately north of the entrance of rang Jacques-Paquin from Highway 117 is the Ontario Northland Railway. The railway will require an upgrade due to heavy truck usage during construction. Signal crossings will be installed for increased safety given the higher traffic flow.

The access to the process plant site is controlled by an entrance gate located 150 m from Highway 117. A light/heavy traffic security system will be installed at the access point to the processing plant pad to provide security for the cross traffic between the underground mining trucks and light vehicles (e.g., pick-up trucks, delivery trucks, tankers, etc.). A site perimeter fence will be installed to restrict access to site by unauthorized people and wildlife.

18.2.5 Tailings Management Access and Pipeline Service Road

Except for a small portion at the processing plant and TSF area, the tailings and reclaim water pipes will run above ground on a granular pad (pipe bench) next to a one-lane service road that runs between the process plant and the TSF. On this one-lane service road, four sections will be widened at strategic locations to facilitate circulation. The cost related to these four widened road sections is included in the current project.

The pipe bench is designed with two low points allowing the tailings pipe to be completely drained for maintenance as necessary. A sedimentation pond with sufficient capacity to store the slurry and rainwater volumes is designed at each low point in the case of an emergency spill. From station 5+800 to station 6+300 the service road will be widened to allow trucks to circulate between the TSF site and the toe of the tailings stockpile.

18.3 Electrical Power System

18.3.1 Electrical System Demand

The maximum electrical demand for the Wasamac site is estimated at 28.25 MW. The forecast load for the process plant and building complex is 15.10 MW, whereas 13.15 MW is approximated for the underground mine pre-production and operations.

18.3.2 Facility Power Supply

Primary power will be supplied to the Wasamac site by the Hydro-Quebec utility via a 120 kV overhead transmission line that terminates at the plant’s 120 kV outdoor substation. Emergency power will be generated on site by diesel-powered standby generators that are optimally sized and located near the critical electrical consumers.

18.3.3 Electrical Outdoor Substation

The 120 kV outdoor substation will be a radial power distribution system with N-1 transformer contingency and will be located to the northeast corner of the process plant. Three 18/24 MVA, 120 kV / 13.8 kV oil-filled, substation-type transformers are proposed to deliver the maximum power required by the site. Two of the transformers are dedicated to powering the process plant’s nominal load whereas the third transformer will power the underground mine. Each transformer is sized to offer 20% future spare, and the system is configured to ensure reliability in the event a single transformer is temporarily out of service. This outdoor substation will also include 16 MVAR power factor correction equipment.

18.3.4 Site Power Reticulation

The 120 kV incoming supply by Hydro Quebec will be stepped down via 120 kV / 13.8 kV power transformers and distributed across the site via overhead lines originating from the plant's primary 13.8 kV distribution switchgear housed within the primary electrical room at the outdoor substation.

Overhead distribution lines will be constructed using aluminum conductor steel-reinforced (ACSR) cable and supported by wooden poles. Generally, these will be routed along the plant perimeter and terminate at the electrical rooms (situated at the process plant, tailings management facility and the mine portal areas), the administration and lab buildings.

18.3.5 Standby / Emergency Power Supply

Six standby diesel generators in weatherproof enclosures will be provided to supply emergency power to the process plant, underground mine loads, and life safety systems.

Three low-voltage (600 V) standby diesel generators will provide auxiliary power for process plant emergency loads. Each generator will be sized and located adjacent to the designated electrical room and connected to the respective emergency MCC via an automatic transfer switch.

Three 1 MV, 13.8 kV standby diesel generators connected in parallel and located adjacent to the mine portal electrical room will provide emergency power to the underground mine loads.

18.3.6 Plant Power Distribution

The SAG and ball mills are the largest electrical loads at the process plant. Both motors are squirrel-cage induction motors equipped with a combination variable frequency drive (VFD) and bypass system to minimize voltage drop impact on the utility system during motor start-up. These mill motors will be supplied via buried cable circuits which originate from the plant's primary 13.8 kV switchgear. All other process and non-process plant loads will be powered via 600 V motor control centres (MCCs) located in the electrical rooms in the process plant, tailings management facility, and mine portal areas.

Power to the electrical rooms will be supplied by resistance-grounded, secondary substation type, oil-filled distribution transformers (13.8 kV / 600 V) located adjacent to the respective electrical room. All electrical rooms will be adequately rated for the environment and outfitted with heating and ventilation, lighting and small power transformers, distribution boards, and uninterrupted power supply (UPS) systems. To reduce installation time, the electrical rooms were considered prefabricated modular buildings, installed on structural framework above ground level for bottom entry of cables. Additionally, electrical rooms are strategically located as close as practical to the electrical loads thereby managing voltage drop concerns and reduced cable cost.

Solidly grounded pad-mounted and pole-mounted transformers will be used to step down the voltages at the building complex and laboratory and will terminate at the building's local 600 V distribution boards.

18.3.7 Construction Power

The project's current construction strategy requires that surface subcontractors provide their own diesel power generator sets during plant construction prior to grid connection and energization of the 120 kV outdoor substation. Underground subcontractors will be provided with fuel, water, and electricity by Yamana.

18.4 Fuel Storage

Two 90,000 L tanks of diesel will be used to fuel equipment on site during operations. These tanks will be located on a concrete pad southeast of the warehouse, away from traffic.

Two propane tanks will be stored on site for heating the main buildings. The propane will also be used to fuel the strip solution heater burner, regeneration kiln burners, and for smelting. The propane tanks will be located on a concrete pad north of the parking lot, away from traffic. A propane tank will also be located on a concrete pad at each of the two mine air heat plants to heat the underground mine air.

18.5 Support Buildings

18.5.1 Main Plant Site Area Buildings

The main plant site area consists of either pre-engineered or modular buildings. All pre-engineered buildings have a roof built with cladding 24 Ga galvalume finished standing seam roof with R-28 fiberglass blanket insulation and vapour barrier. Wall cladding is 26 Ga with exterior coating, R-28 fiberglass blanket insulation, and vapor barrier facing. All buildings include canopies over doors and concrete slab-on-grade flooring. The HVAC system consists of propane-fired heating and local ventilation for areas as per code requirements (minimum 5°C operating temperature). The electrical system consists of LED lighting including the emergency lights and small power receptacles as per code. No provision was carried for a washroom unless otherwise indicated.

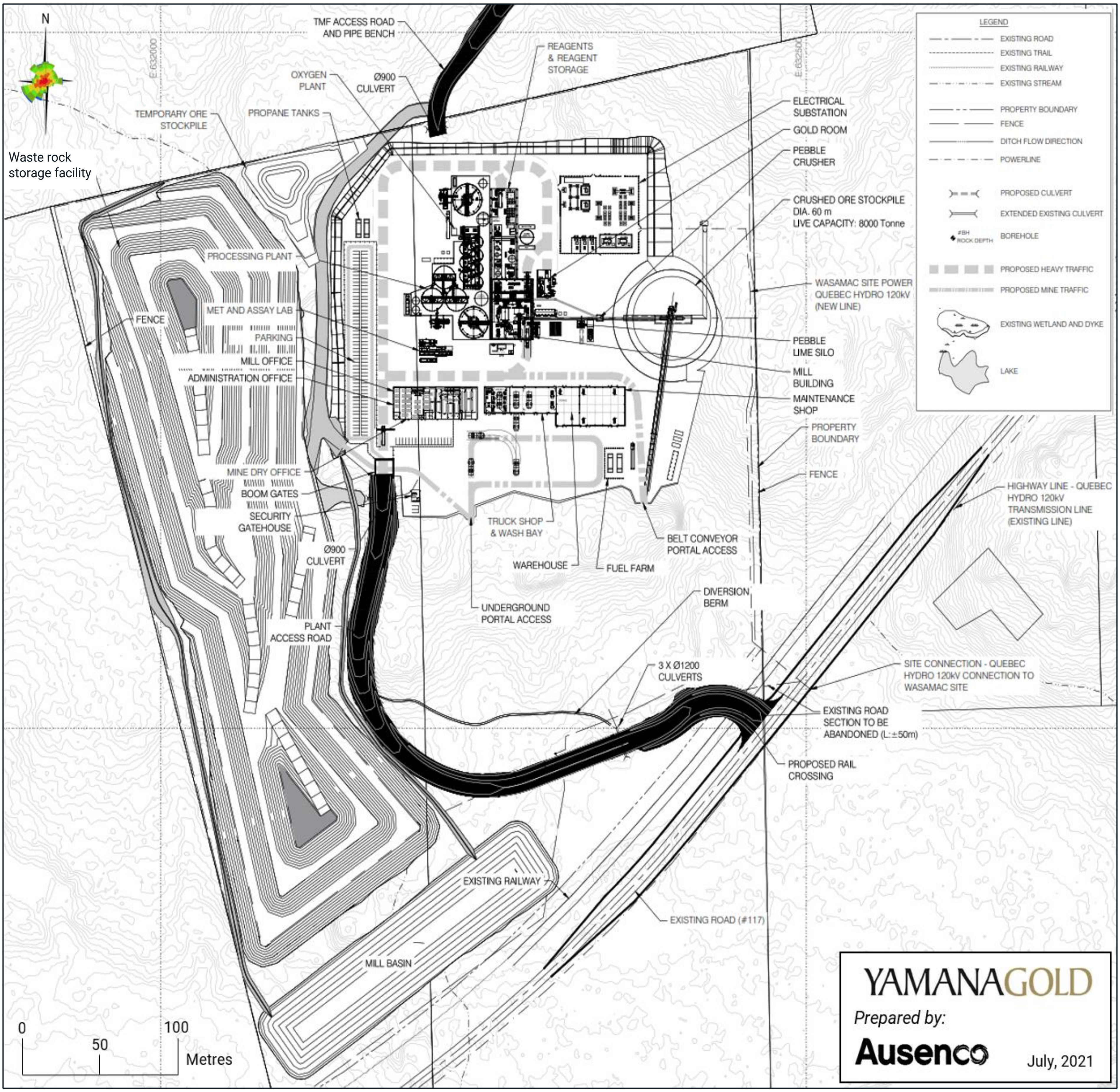
The foundation of modular buildings is to be built by the contactor, as per code requirements. The HVAC system is an electric heating system, as per code requirements. The electrical system includes lighting and small electrical receptacles as per code requirements.

An overall site plan and 3D rendering of the process plant area is shown in Figure 18-3 and Figure 18-4, respectively.

18.5.1.1 Crushed Ore Storage Building

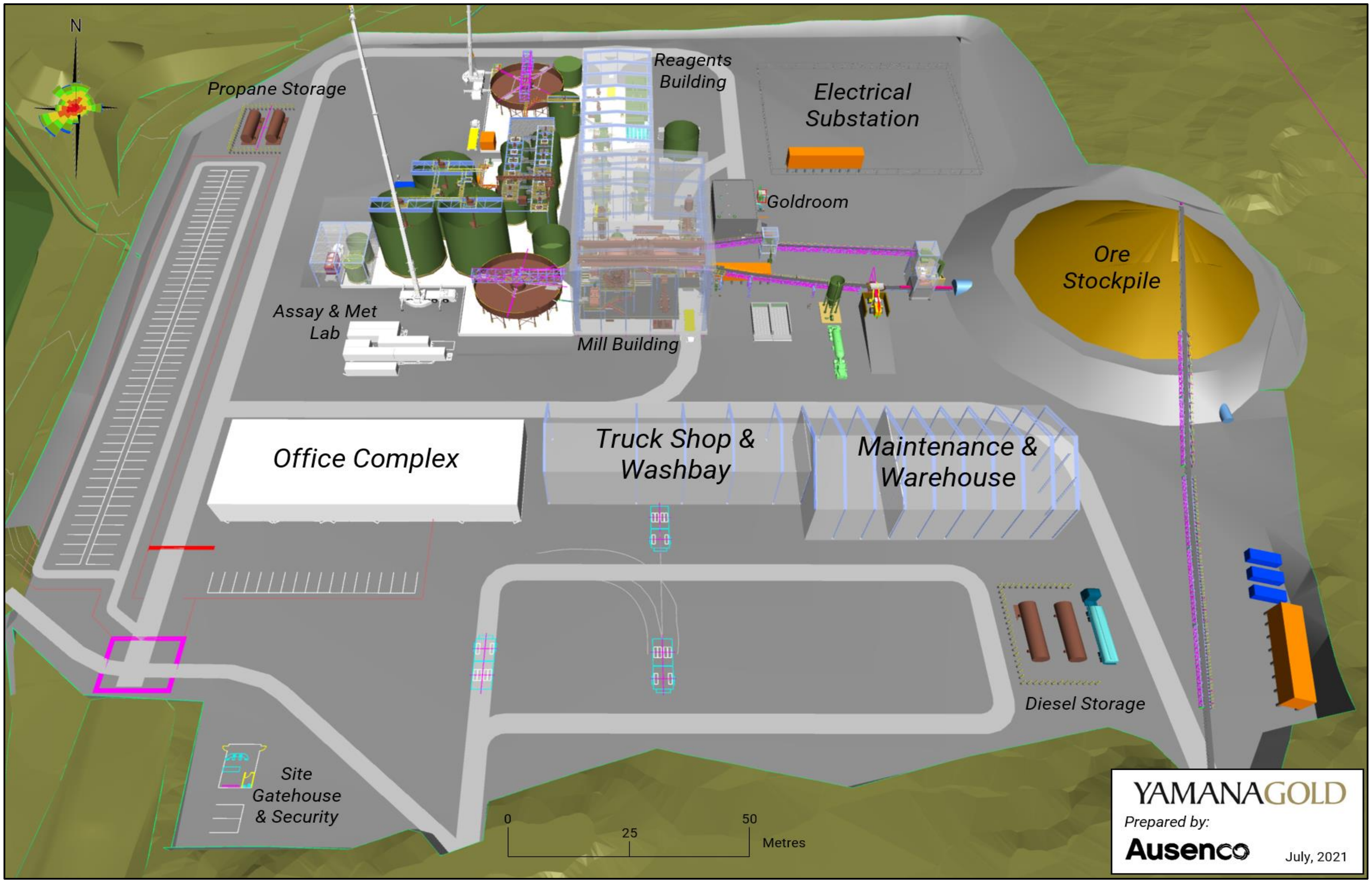
This building is a single-storey inflatable structure measuring 62 m diameter (nominal) x 23 m high for the 8,000 live tonne ore stockpile. The structure material is to be provided by the contractor. No Insulation or crane is required to mount the building. Building doors consist of one 5.0 m wide x 5.0 m high overhead door with curtains and four 0.914 m wide x 2.13 m high metal personnel doors. The structure has a concrete foundation, and no heating is required. Electrical lightings and small electrical receptacles as per code.

Figure 18-3: Overall Site Plan – Process Plant Area



Source: Ausenco, 2021

Figure 18-4: 3D Rendering of the Process Plant Area



Source: Ausenco, 2021

18.5.1.2 Mill Building

This building is a single-storey pre-engineered rigid frame metal building measuring 45 m long x 31 m wide x 23.2 m high (internal clearance envelope) with a gable roof (1:12). Areas included in this building are as follows:

- SAG mill
- ball mill
- hydro cyclones
- acid wash and elution area

The building has a 40-tonne overhead crane with a 5-tonne auxiliary hook that runs along the length of the building. There are eight metal personnel doors measuring 0.914 m wide x 2.13 m high and two 5.5 m wide x 4.5 m high overhead doors complete with curtains.

18.5.1.3 Reagent Building

This building is a single-storey pre-engineered rigid frame metal building measuring 64.0 m long x 20 m wide x 12 m high (internal clearance envelope) with a gable roof (1:12). Storage areas included in this building are as per reagents described in Chapter 17.

The building has a 7.5-tonne overhead crane that runs along the building length. There are twelve 0.914 m wide x 2.13 m high metal personnel doors and two 5.5 m wide x 4.5 m high overhead doors complete with curtains.

18.5.1.4 Gold Room Building

This building is a single-storey pre-engineered rigid frame metal building measuring 16.6 m long x 11 m wide x 8.5 m high (internal clearance envelope) with a gable roof (1:12). Areas included in this building are electrowinning and a gold room, equipped with restricted access key cards and security cameras.

The building has a 2-tonne overhead crane that runs along the building length. The building includes two metal personnel doors measuring 0.914 m wide x 2.13 m high and one 2.4 m wide x 2.4 m high overhead door complete with curtains.

18.5.1.5 Washbay / Truckshop Building

This building is a single-storey pre-engineered rigid frame metal building measuring 54.5 m long x 17.4 m wide x 12.2 m high (internal clearance envelope; 7.5 m crane headroom clearance) with a gable roof (1:12). Areas included in this building are as follows:

- one washbay
- three bays for major equipment
- one bay for minor equipment

The building has a 10-tonne overhead crane that runs along the building length. The building includes five metal personnel doors measuring 0.914 m wide x 2.13 m high, three haul truck doors measuring 6.0 m wide x 4.0 m high, one haul truck washbay door measuring 6.0 m wide x 5.5 m high, one overhead door measuring 4.5 m wide x 3.0 m high complete with curtains and one overhead door measuring 3.0 m wide x 3.0 m high complete with curtains. An interior wall 19 m from the west end of the building will separate the washbay from the truckshop.

18.5.1.6 Truckshop Washroom

This building is a single-storey modular building with dimensions that will be defined by the contractor (minimum ceiling height of 2.7 m). The foundation is to be built by the contactor to code requirements. The module will have an HVAC system and an electric heating system, as per code. The building will include the following:

- changeroom for men with three showers
- changeroom for women with one shower
- small office (10 m^2)
- cubicles
- washrooms

18.5.1.7 Warehouse / Maintenance Shop

This building is a single-storey pre-engineered rigid frame metal building measuring 52.3 m long x 26.4 m wide x 9.1 m high (internal clearance envelope) with a gable roof (1:12). The building has six metal personnel doors measuring 0.914 m wide x 2.13 m high and six overhead doors measuring 3.5 m wide x 3.0 m high complete with curtains.

18.5.1.8 Underground Control Room

This building is a single-storey modular building of 71 m^2, the dimensions of which will be defined by the contractor with a minimum ceiling height of 2.7 m. The building includes one control room with seating for eight people and one computer server room.

18.5.1.9 Assay and Metallurgical Laboratory

This building is a single-storey network of modular buildings totalling 260 m^2 on pre-cast concrete blocks, housing the typical equipment for the mine and plant assays.

18.5.1.10 Mill Office

This building is a two-storey building that houses the main administration office, mill office and mine dry areas. The modular building with dimensions defined by contractor will have a minimum ceiling height of 2.7 m. The rooms required for the mill office area are as follows:

- two offices (10 m^2)
- six cubicles

- lunch for room 12 people
- meeting room for six people

18.5.1.11 Main Administration Building

This two-storey building houses the main administration office, mill office and mine dry areas. The modular building with dimensions defined by contractor will have a minimum ceiling height of 2.7 m. The rooms required for the main administration building area are as follows:

- changeroom for men with three showers
- changeroom for women with one shower
- large office (14 m^2)
- medium office (12 m^2)
- small office (8 m^2)
- six cubicles
- lunch room for 10 people
- meeting room for 25 people
- meeting room for 10 people
- washrooms
- mechanical room
- computer server room

18.5.1.12 Mine Technical & Mine Supervision & Mine Dry

This building is a two-storey building combined with main administration office, mill office and mine dry areas. The modular building with dimensions defined by contractor will have a minimum ceiling height of 2.7 m. These facilities will have clean and dirty areas designated for men and women and will be complete with showers, sinks, toilets, lockers and overhead laundry baskets. The required rooms for this area are as follows:

- one changeroom for men with 20 showers
- one changeroom for women with three showers
- 250 lockers for dirty items; 250 lockers for clean items
- one large office (14 m^2)
- five medium offices (12 m^2)
- five small offices (8 m^2)

- 37 cubicles
- one lunch room for 20 people
- two meeting rooms for 20 people
- three washrooms
- one mechanical room
- one orientation room for 20 people
- one health and safety mine rescue room
- one nurse room
- one security room
- one janitorial room

18.5.1.13 Security Gatehouse & Access Control

This building is a single-storey modular building with dimensions 10 m long x 3 m wide x 3 m high. This building includes the following:

- two cubicles
- one washroom
- one mechanical room

18.5.1.14 Mill Network and Control Room

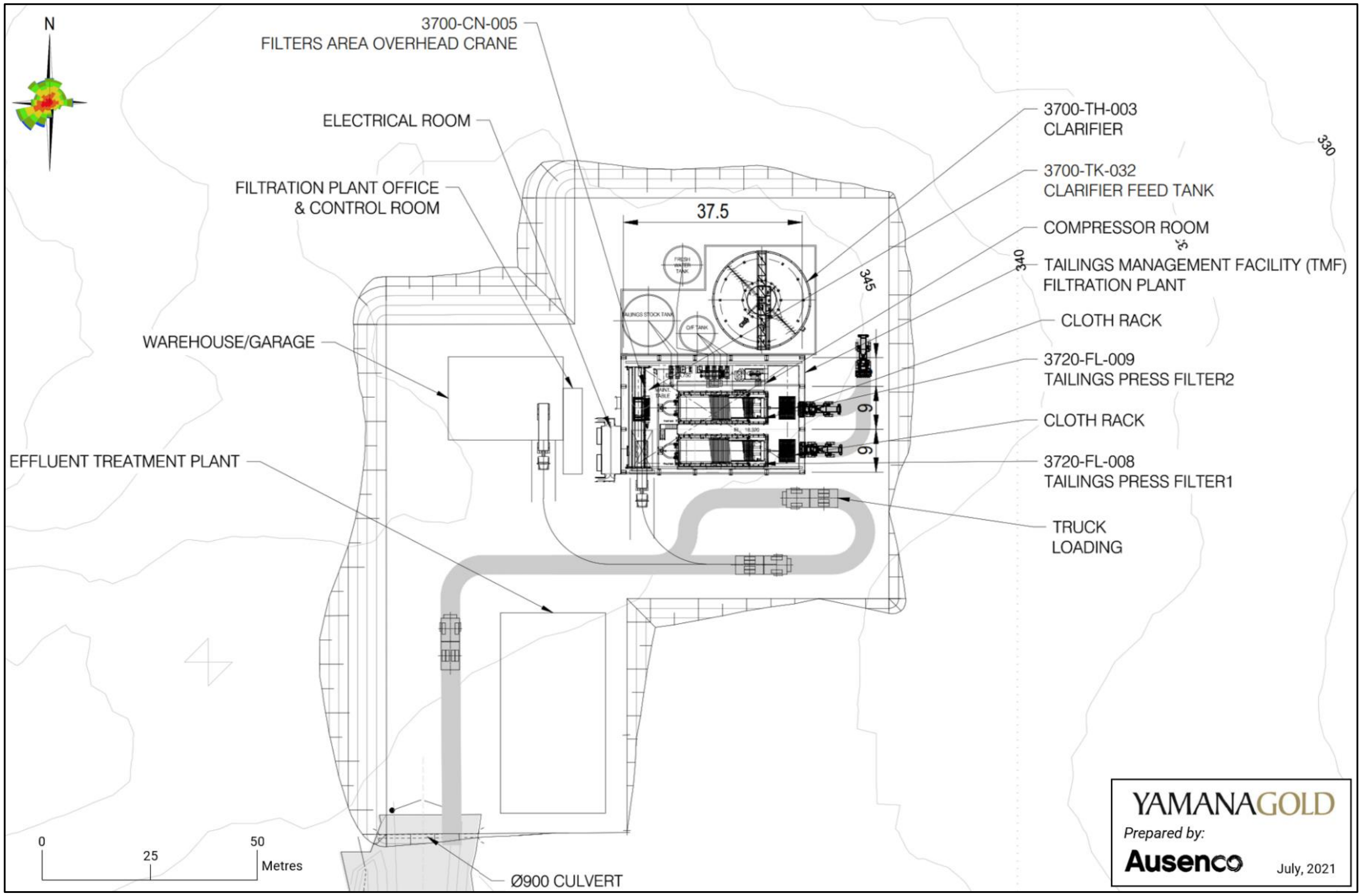
This building is a single-storey modular building that measure 18 m long x 4 m wide x 3 m high. This building includes the following:

- two cubicles
- one washroom
- one mechanical room

18.5.2 Filtration Plant Buildings

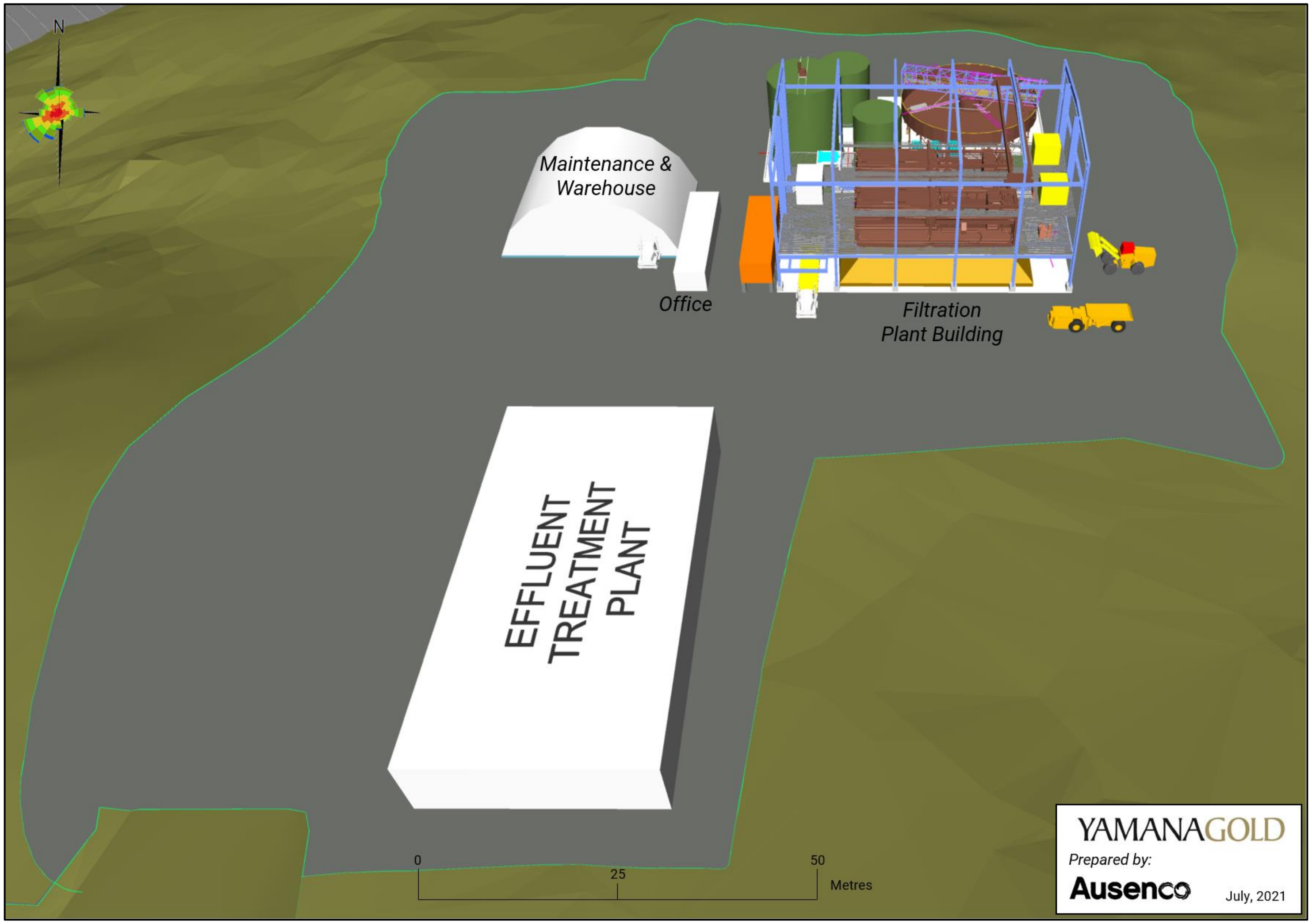
The filtration plant area consists of several buildings as shown in Figure 18-5 and Figure 18-6 and described in the following subsections.

Figure 18-5: General Arrangement - Filtration Plant



Source: Ausenco, 2021

Figure 18-6: 3D Rendering of the Filtration Plant Area



Source: Ausenco, 2021

18.5.2.1 Filtration Plant Office & Control Room

This building is a single-storey modular building that measures 18 m long x 4 m wide x 3 m high (minimum ceiling height of 2.7 m). This building includes the following:

- one office (10 m^2)
- three cubicles
- one lunchroom sized for 6 people
- two washrooms
- one mechanical room

18.5.2.2 Filtration Plant Building

This building is a single-storey pre-engineered rigid frame metal building measuring 35.6 m long x 23.0 m wide x 15.8 m high (internal clearance envelope) with a gable roof (1:12). The building has a 20-tonne overhead crane with a 5-tonne auxiliary hook that runs along the length of the building. There are eight metal personnel doors measuring 0.914 m wide x 2.13 m high and two 5.5 m wide x 3.5 m high overhead doors complete with curtains.

18.5.2.3 Filtration Garage Warehouse

This building is a single-storey pre-engineered rigid frame metal building measuring 37.5 m long x 24.0 m wide x 10.75 m high (internal clearance envelope) with a gable roof (1:12). There are six 0.914 m wide x 2.13 m high metal personnel doors and two 3.0 m wide x 3.0 m high overhead doors complete with curtains.

18.5.2.4 Effluent Treatment Plant

This building is a single-storey pre-engineered rigid frame metal building measuring 42.0 m long x 27.0 m wide x 5.0 m high (internal clearance envelope) with a gable roof (1:12). There are four metal personnel doors measuring 0.914 m wide x 2.13 m high and one 3.0 m wide x 3.0 m high overhead door complete with curtains. The flooring system is a concrete slab-on-grade in one-third of the building (beneath the MBBR), with granular grating on the rest.

18.5.3 Explosives Storage

A 6-m wide access road to the surface explosives storage area connects off the TSF access road. The storage area is located as per provincial quantity-distance guidelines (i.e., 380 m radius from buildings and roads, and 760 m radius from any residence). The gated pad will be approximately 100 m x 28 m, ensuring the two storage magazines are 82 m apart and separated by a berm. The magazines will store 32,000 kg of powdered explosives and 600 cases of explosives caps. The magazines are steel structures, similarly sized as shipping containers, measuring 12.19 m long x 2.44 m wide x 3.66 m high.

18.5.4 Ventilation Raises and Paste Fill Structures

The mine air heat plant sheds are two single-storey pre-engineered rigid-frame metal buildings measuring 7.0 m long x 6.0 m wide x 6.0 m high. The sheds shelter the entry of each heat plant duct elbow into the raises. The structures have a concrete foundation; no heating is required. Electrical lightings and small electrical receptacles are provided as per code.

The borehole collar pre-fabricated shed is a single-storey pre-engineered rigid frame metal building measuring 4.0 m long x 4.0 m wide x 5.0 m high. The structure has a concrete foundation; no heating is required. Electrical lightings and small electrical receptacles are provided as per code.

18.6 Water Systems

18.6.1 Industrial Water Supply & Distribution

Process water from various users in the plant will report to the process water tank. From there it will be distributed to required areas in the plant (e.g., grinding mills, reagent mixing, and vibrating screen spray bars). Excess process water will report to the effluent treatment plant located near the tailings filtration plant.

18.6.2 Fresh Water Supply & Distribution

Fresh water will be sourced from a well and directed to the fresh/fire water tank, from where it will be distributed to required points in the plant. Fresh water will also be used to feed the potable water treatment system, elution circuit, and reagent systems.

Fresh water will be transported via truck to the filtration plant where it will be stored in the filter plant potable/fire water tank prior to use. The bottom section of both the fresh/fire water tank and the filter plant potable/fire water tank will be dedicated to the fire water system.

18.6.3 Potable Water System

The quality requirement for the potable water treatment plant will match the local drinking water guidelines. Fresh water will be sourced from the freshwater intake pump (located in a well) and processed through the potable water treatment skid before being stored in the potable water tank.

Prior to further use, the potable water will be heated by the tepid water heating skid before being distributed to safety showers and other points in the process plant facilities. The distribution piping will be heat traced and insulated wherever it is not inside a heated building. Where necessary, manual drain points will be included.

18.6.4 Fire Water System

All facilities will have a fire suppression system in accordance with the structure’s function. For the most part, fire water will be used with an underground ring main network around the facilities. All buildings will have hose cabinets and handheld fire extinguishers. Electrical and control rooms will be equipped with dry-type fire extinguishers. Ancillary buildings will be provided with automatic sprinkler systems. For the reagents, appropriate fire suppression systems will be included according to their material safety datasheets.

18.6.5 Domestic Effluent and Sanitary System

A domestic effluent and sanitary system package will be supplied at the process plant area to treat all domestic waste collected within the site. The collection network will be underground. Office and domestic waste will be collected and disposed of off site in accordance with applicable regulations.

18.6.6 Water Treatment

An effluent water treatment plant (ETP) designed to treat 200 m^3/h is included in the water systems design for the Wasamac Project. The plant was sized to treat the highest rate anticipated; data from the wettest month (October) was used to select the size. Process water will report to the ETP at rates of 104 m^3/h and 47 m^3/h during operation of the filtration plant or paste backfill plant, respectively (refer to water balance and Figure 18-14), as they operate alternately and according to paste backfill demand. Contact and seepage water collected in the TSF pond will report to the effluent treatment plant at a maximum rate of 96 m^3/h, depending on the time of year. The process water and TSF pond water will be treated before being piped to the future-permitted discharge location west of the TSF pond.

Metals removal by precipitation will reduce contained copper in solution. The precipitate sludge will report to the TSF. A biological treatment method using a moving bed biofilm reactor (MBBR) will subsequently reduce ammonia and cyanide contained in the TSF. MBBR is a tank reactor with nitrifying bacteria attached to mobile media. Blowers will provide the required aeration to complete the nitrification process. Effluent water will be treated according to the Metal and Diamond Mining Effluent Regulations (MDMER) 2021 and Directive 019 discharge regulations. The ETP will be operated year-round, with less water treatment during winter months.

18.7 Site Geotechnical Conditions

Reviewing the existing data (ten boreholes in the mill area) indicates that the subsurface (less than 5 mbgs) soil is mainly composed of clay with significant variability. More importantly, a layer of sandy material (susceptible to liquefaction) was observed in several boreholes. Caution will be taken to remove the unsuitable material for conservative foundation design beneath major loads in the area of the process plant. The foundation design will be assessed with addition of new data from the SCPT program in the first half of 2022. The program has been designed to leverage winter conditions that will allow access to wetland areas.

In the area of the waste rock storage facility (WRSF) and mill basin, limited borehole data exists and demonstrates variability of soil materials. The WRSF and mill basin designs accounted for higher clay content in costing of the foundation preparation. Additional SCPT data will add accuracy to the current designs.

The area of the TSF and TSF pond was investigated with test pits in 2018; two boreholes were completed in the vicinity of the TSF pond. A layer of sandy material at approximately 10-15 mbgs has been identified in both boreholes which is of concern due to the susceptibility of this layer to liquefaction. Variability in thickness of soil horizons, as well as clay contents, were encountered during the investigation. The data in this area will be supported by further geotechnical investigation during the SCPT program in the first half of 2022. The current TSF design mitigates the possibility of variable clay content across the foundation of the facility by ensuring that areas are not vertically stacked too quickly. It is acknowledged that the limited data in the area of the TSF foundation is a risk to the cost estimate and shall be addressed over the coming months upon receipt of the SCPT data.

18.8 Waste Rock Storage Facility

Mine development material will be placed in a waste rock storage facility (WRSF) west of the mill and adjacent to the plant pad. The WRSF will be accessed by trucks across an intersection at the main site entrance, and will be traffic-controlled with boom gates.

Stacking of this facility will commence in Year -1 (pre-production) and be completed in Year 2 of operations. The facility is designed to accommodate 1.83 Mt (914,982 m^3) of waste rock. The current understanding from a limited acid base accounting (ABA) testing program is the waste rock is classified as non-acid-generating due to the ratio of the neutralizing potential verses the acid-generating potential of the minerals in the waste rock. However, additional static testwork and kinetic testwork are required (and underway) to confirm this hypothesis. The assumption is that waste rock will be used as fill and road base material where appropriate, and the timing of earthworks execution has been aligned to optimize this cost-savings opportunity. Contact runoff water reports to the mill basin pond immediately south of the WRSF. Once the WRSF is full in Year 2 of mining, the remaining development waste will be used as a reclamation layer on the progressively reclaimed dry stack tailings storage facility.

18.8.1 Waste Rock Storage Facility Design Criteria

The general design criteria of the WRSF considers the following:

- storage capacity of 1.83 Mt
- in-situ density of 2.0 t/m^3
- bottom-up construction
- 8.5 m lift height
- 6.0 m bench width
- overall WRSF slope of 2.5:1 (H:V)
- lift slope of 1.8:1 (H:V)
- maximum height of 35 m
- access road width of 9 m
- access road gradient of 10%
- design storm event 1:100-year event
- design seismic event 1:2,475-year event

18.8.2 Waste Rock Storage Facility Location and Handling

The WRSF is located immediately west of the process plant pad (refer to Figure 18-1).

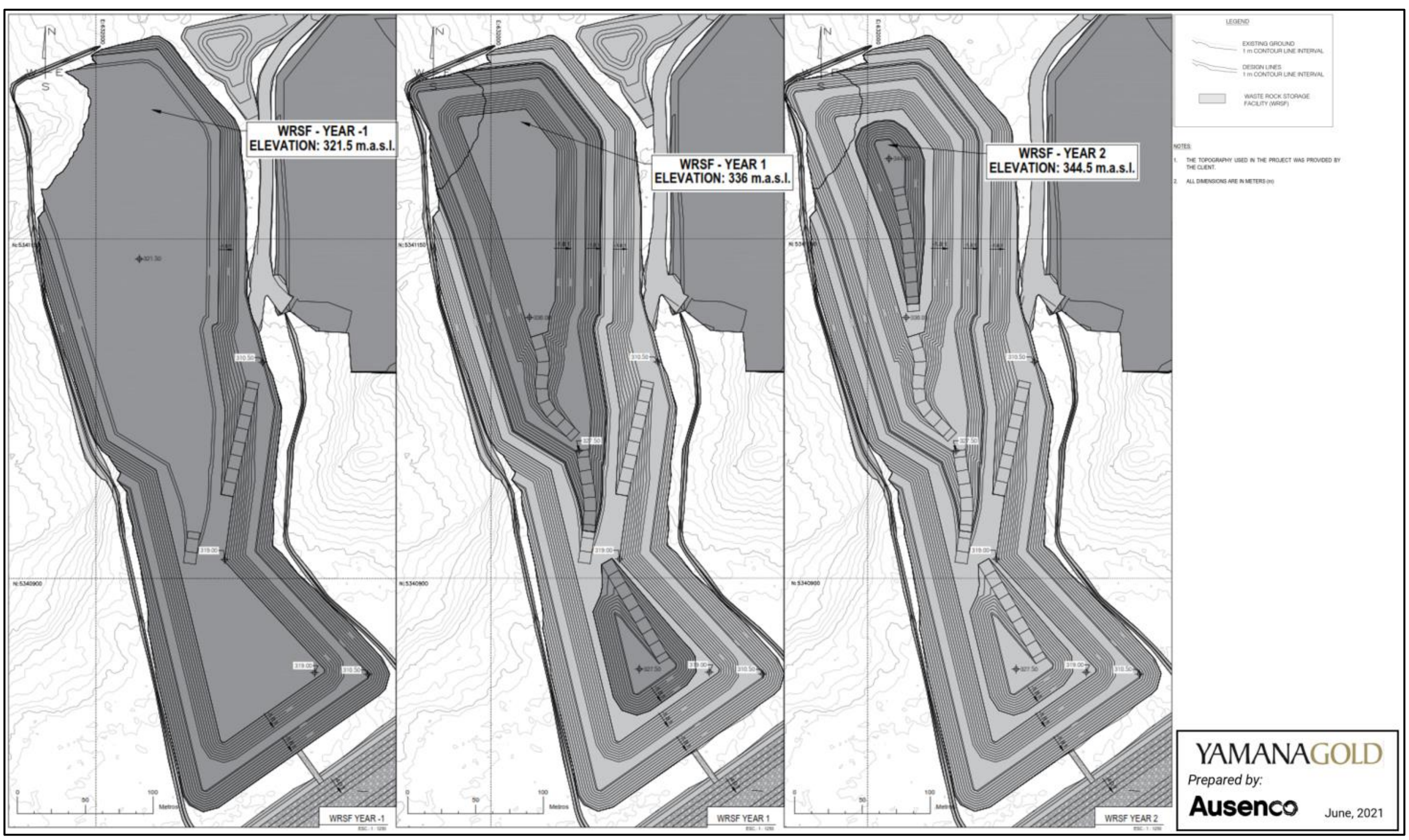
Waste rock will be transported to surface in haul trucks and deposited on the facility. The material will be spread and compacted into the designed lifts using a loader, dozer, and compactor.

The WRSF will be completed over a period of three years. Table 18-3 displays the stacking quantities per year and Figure 18-7 shows the stacking plan from Year -1 to Year 2 of mining.

Table 18-3: Waste Rock Storage Facility Stacking Elevation and Quantities

Stacking Year	Elevation (masl)	Incremental Quantity (m^3)	Cumulative Quantity (m^3)
Year -1	321.5	635,931	635,931
Year 1	336.0	260,805	896,736
Year 2	344.5	18,246	914,982

Figure 18-7: Waste Rock Storage Facility Stacking Plan



Source: Ausenco, 2021

18.8.3 Waste Rock Storage Facility Reclamation Plan

Reclamation of the WRSF will begin in Year -1 and be completed in Year 2 of mining to reduce dust and return the landscape to a natural form as quickly as possible. Reclamation will involve contouring and placement of 1 m of soil and 0.3 m of topsoil, followed by hydroseed. Quantities for each year of reclamation are outlined in Table 18-4.

Table 18-4: Waste Rock Storage Facility Reclamation Surface Area by Mining Year

Mining Year	Surface Area (m^2)
Year -1	32,101
Year 1	32,963
Year 2	4,840

18.8.4 Waste Rock Storage Facility Stability Analysis

The stability analyses for the WRSF were conducted using the limit-equilibrium method in SLIDE 2 (Vers. 9.0) computer program developed by RocScience Inc., Canada (RocScience, 2021). Critical slope through the WRSF and foundation were identified and analysed to aid in the development of the design criteria.

Analyses were undertaken for both static and pseudo-static (earthquake loading) conditions with the calculated factors of safety (FOS) higher than the minimum required values in accordance with design of waste rock facility best practice. The WRSF stability analyses exceeded both static and pseudo-static best practice guidelines.

18.9 Tailings Storage Facility

The Wasamac project will host a dry stack tailings storage facility (TSF) situated 6.3 km northwest of the process plant, proximal to the filtration plant. The TSF is designed to receive 60% of the total production of tailings. The ultimate capacity of the facility is estimated to be approximately 8,570,280 m^3 (14.1Mt) of dry stack tailings with an additional surface layer of approximately 518,779 m^3 (1.04 Mt) of development waste rock. As the stack deposition advances, the TSF will be progressively reclaimed with the waste rock to reduce dust and to return the landscape to a natural state as quickly as possible.

The following standards and regulations were consulted for the design of the Wasamac TSF and its associated infrastructure:

- Directive 019 specific to the mining industry in Quebec (MDDEP 2012, Directive 019 sur l'industrie minière)
- Guide for Mine Site Reclamation and Restoration in Quebec (MERN 2017, Guide de préparation du plan de réaménagement et de restauration des sites miniers au Québec)
- Dam Safety Guidelines produced by the Canadian Dam Association (CDA 2007, 2019)
- *Dam Safety Act* (Loi sur la sécurité des barrages) applied in Quebec and the associated regulation (Règlement sur la sécurité des barrages)
- *Act Respecting Occupational Health and Safety* (Loi sur la santé et la sécurité du travail) applied in Quebec and the associated regulation (Règlement sur la santé et la sécurité du travail dans les mines)
- The Quebec and/or the Canadian Legal framework applied to the environment and water sectors

18.9.1 Tailings Management Facility Location

During the execution of the feasibility study, four potential areas were evaluated for the location and development of the Wasamac Tailings Management Facilities (TMF), as follows:

- acquire terrains close to proposed mill site
- use current Wasamac claims area
- negotiate the use of the Aldermac TSF in rehabilitation
- use RM Nickel claims owned by Yamana (formerly Monarch)

Figure 18-8 shows the footprints of the four proposed locations (Options 1 to 4).

BBA was initially instructed by Yamana to place the Wasamac TMF on the RM Nickel claims. This area of about 1,365 ha could offer different alternatives to construct a tailings storage facility and its related management infrastructure. BBA identified three potential locations, illustrated as Options 1, 2 and 3 on Figure 18-8, on the RM Nickel claims.

An additional option had previously been investigated by BBA and is presented here as Option 4 (Figure 18-8) which involves placing the TMF on crown land directly adjacent to the west of the mill site property. This option offers potential significant savings by eliminating the need for transportation routes, pumps, and pipelines.

The selection criteria of the proposed location of the Wasamac TMF have been defined as follows:

- site characterization based on aerial photos, LIDAR information and regional land use information, including the identification of existing infrastructure such as electric lines, roads, farms, forestry domains, and natural water bodies and environmental protected areas
- volumetric compliance for tailings placement with the targeted volume around 8.6 Mm³ of dry stack tailings
- preliminary analysis of the environmental and social constraints of the selected TMF footprints, including distance and disturbance to public infrastructure and communities, dust and noise control, etc.
- evaluation of sites and selection using a decision-making tool Pugh Matrix. The following principal accounts were analyzed:
  - project economics
  - operational consideration
  - environmental impacts
  - technical consideration
  - social impacts

As a result, Option 1 was recommended for the location of the Wasamac tailings storage facility and its related management infrastructure. The key factors leading to this decision are as follows:

- Proximity to the filtration plant could significantly reduce the tailings haulage distance.
- The proximity of a natural hill and rock outcrops offers the best configuration in terms of stability, dust control, integration with topographic landscape, ease of construction, and flexibility.

- This is the only analyzed area with less than 50% of foundation that could be located on probable sensitive clays and organic soils.
- The TSF elevation is limited to the height of topography.
- This location is at the head of a sub-watershed and will have minimum impact on wetlands and possible fish habitat.
- Despite the possibility of significant savings, Option 4 was not selected due to its proximity to the highway and local communities.

Figure 18-8: Potential Options for the Wasamac TMF

Source: BBA, 2021

18.9.2 Tailings Handling Methodology

Tailings produced at the process plant will be piped to the filtration plant when tailings are not required in the paste backfill plant. It is anticipated that the plants will run for approximately 3 to 4 days consecutively before switching to the alternate system (filtration or paste backfill).

The filtration plant is situated adjacent to the tailings dry stack storage facility, 6.3 km northwest of the plant site. The filtration plant will consist of two filter presses (fast-opening, 62 to 74 chambers, 3.5 m x 2.5 m plates) designed to open directly onto the cement floor beneath for handling. The filter presses each have a 270-bar hydraulic unit. Other equipment in the filtration plant includes a clarifier tank, a clarifier feed tank, and a 20-tonne overhead crane.

Filter cake will deposit on the concrete floor beneath the presses for collection using a front-end loader. The loader will transfer the cake to three 40-tonne haul trucks for transportation and deposition to the TSF facility, adjacent to the filtration plant pad. The body of the haul trucks is heated to assist in prevention of the filter cake sticking to the truck body, and can be lined to assist in leakage from trucks.

18.9.3 Tailings Deposition Strategy

Tailings produced from the Wasamac mine will be transported and deposited in the proposed dry stack TSF and underground stopes. Based on the annual tailings tonnage, a 60/40 split between dry stack tailings and underground paste fill, and the mining plan, a deposition strategy has been developed for the Wasamac TSF. The following considerations and assumptions have been adopted:

- Dry stack technology will be used.
- The filtration plant will be located at the TSF, with a maximum tailings hauling distance of 2 km.
- The expected moisture content on filtered tailings is approximately 10-12%.
- Tailings are considered fine-grained material, with a P_{80} of 60 µm.
- The in-place density of compacted and filtered tailings is expected to range from 1.6 to 1.8 t/m³. An in-place density of about 1.65 t/m³ was used for volumetric calculations, while that of 1.8 t/m³ was used for geotechnical stability analysis.
- The Wasamac TSF has been designed to receive 60% of the total production of tailings. Based on the in-place density mentioned above, the TSF is aimed at storing 8.6 Mm³ of dry stack tailings and 0.5 Mm³ of waste rock as a reclamation cover.
- Final slopes of the stack have been planned with the intention that the dry stack, after reclamation, could be blend with the existing natural slopes in the surrounding area. In combination with the stability analysis (presented in Section 18.9.6), a slope of 7H:1V was used in the design of the TSF.
- The final elevation of the TSF is between 340 and 341 m.

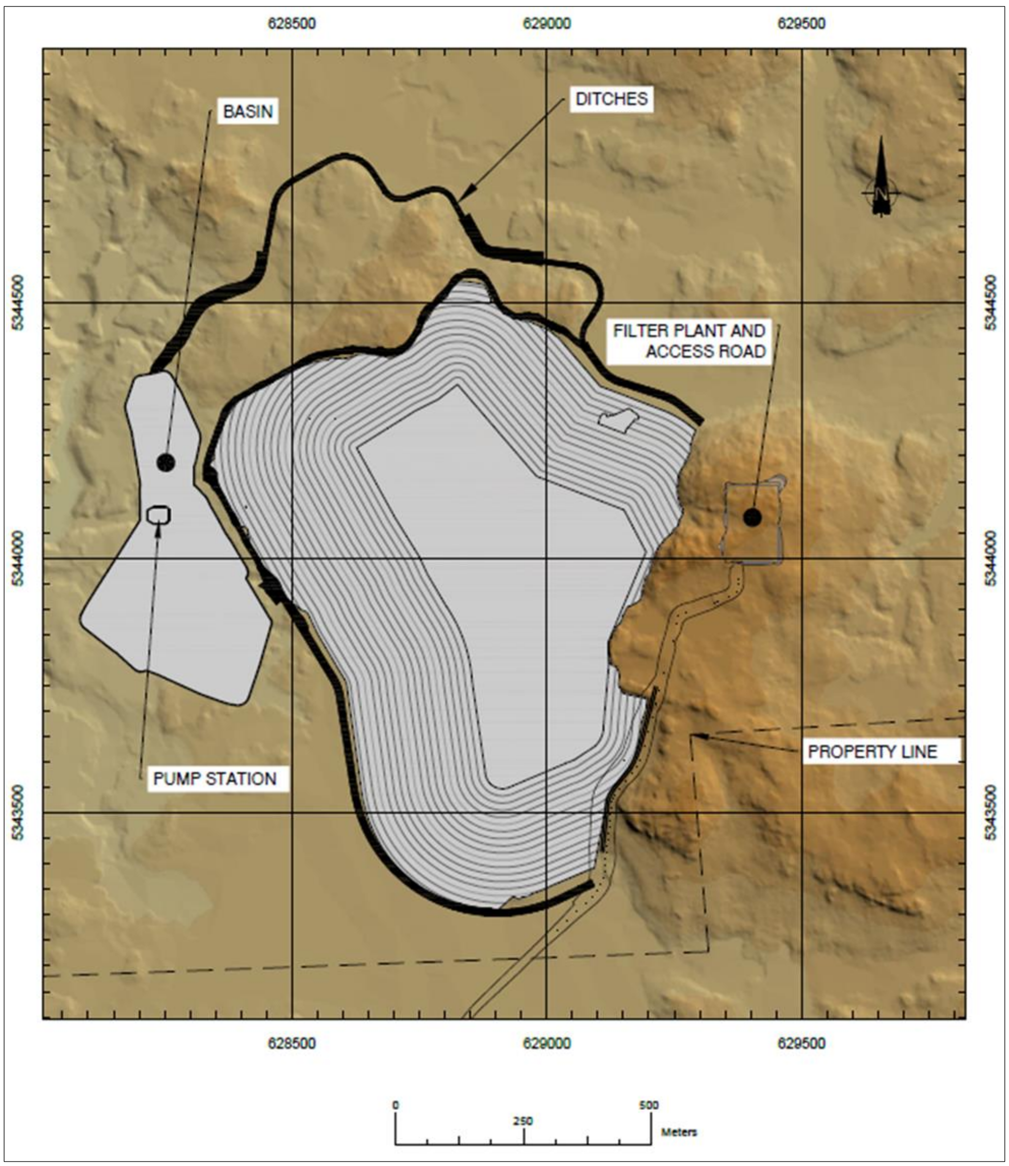
The Wasamac TSF is composed of the following components, identified in Figure 18-9:

- filtration plant (Ausenco)
- pipeline (including civil earthwork) that brings tailings from the mill to the TSF and a second pipeline that returns water to the mill (Ausenco)

- tailings storage area (BBA)
- roads and ditches (Ausenco)
- water management basin (Ausenco)
- pumping station and water treatment plant (Ausenco)

Figure 18-9 presents Wasamac Tailings Storage Facility (TSF) and its related infrastructure.

Figure 18-9: Wasamac Tailings Storage Facility and its Related Infrastructure



Source: BBA, 2021

The tailings deposition sequence was developed based on the following principal factors:

- The dry stack development considers a staged construction to minimize the catchment area and volume of impacted water to be managed and treated.
- Deposition will start at the south of the TSF and expand to the north to allow ditches to drain by gravity into the water management basin.
- Trucks with filtered tailings will start depositing tailings by lifts of 1 m (maximum) in an ascending construction fashion. As construction advances, the final slope of the TSF will be contoured to the design slopes to facilitate surface drainage.
- Dust control will be mitigated by compacting the tailings immediately after placement and by using waste rock for progressive reclamation (as presented in Section 18.9.4).

Table 18-5 summarizes the annual and cumulative tailings to be placed in the Wasamac TSF.

Table 18-5: Cumulative TSF Tailings Storage Capacity over the Mine Life

Period		Produced Ore		Produced Paste Tailings (to Underground)		Produced Tailings (to TSF)		Tailings to TSF per Period	Tailings Cumulative Volume at TSF
(Years)	(Years)	(t/d)	(t/y)	(t/d)	(t/y)	(t/d)	(t/y)	(m³)	(m³)
1	2024	0.00	0	0	0	0	0	0	0
2	2025	0.00	0	0	0	0	0	0	0
3	2026	202.88	74051*	0	0	0	0	0	0
4	2027	4252.16	1,552,040	1,257	458,714	3,198	1,167,378	707,502	707 502
5	2028	6922.72	2,526,791	2,919	1,065,278	4,004	1,461,513	885,766	1 593 267
6	2029	6903.95	2,519,943	2,580	999,279	4,166	1,520,664	921,614	2 514 882
7	2030	6904.44	2,520,121	2,256	941,582	4,325	1,578,539	956,690	3 471 572
8	2031	6881.17	2,511,627	2,672	823,515	4,625	1,688,112	1,023,098	4 494 670
9	2032	6923.29	2,527,000	2,771	975,385	4,251	1,551,615	940,373	5 435 043
10	2033	6903.97	2,519,949	2,630	1,011,382	4,133	1,508,567	914,283	6 349 326
11	2034	6869.93	2,507,526	2,939	960,083	4,240	1,547,443	937,844	7 287 170
12	2035	6903.92	2,519,932	1,969	1,072,682	3,965	1,447,250	877,121	8 164 292
13	2036	3804.61	1,388,683	24,731	718,802	1,835	669,881	405,988	8 570 280
	Total		23,167,664		9,026,702		14,140,963	8,570,280	

Note: * Produced tailings in 2026 to be stored at dry stack TSF in 2027.

Figure 18-10 shows the Wasamac TSF's cumulative storage of tailings and waste rock over the 13-year mine life. The sequence of tailings deposition from Year 4 to Year 13 is illustrated on Figure 18-11.

Figure 18-10: Cumulative TSF Storage Capacity of Tailings and Waste Rock Over the Mine Life

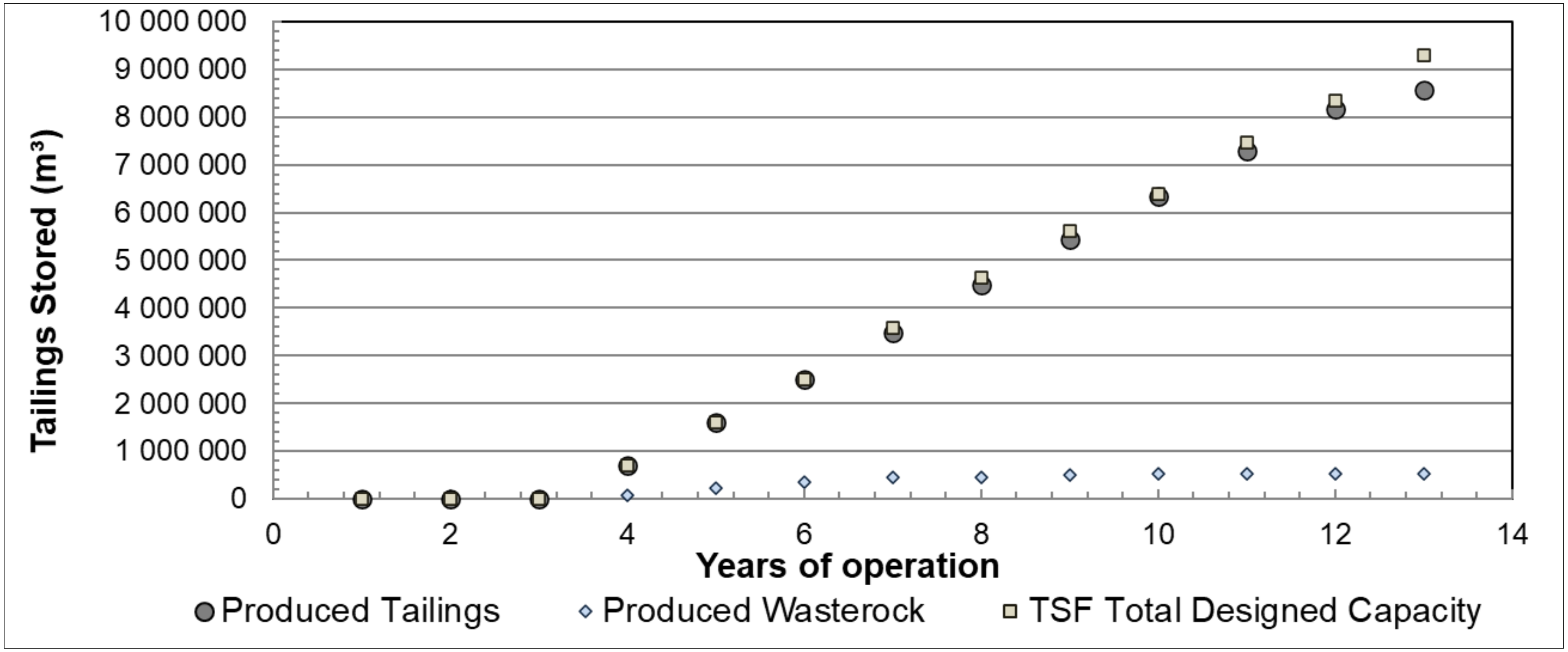
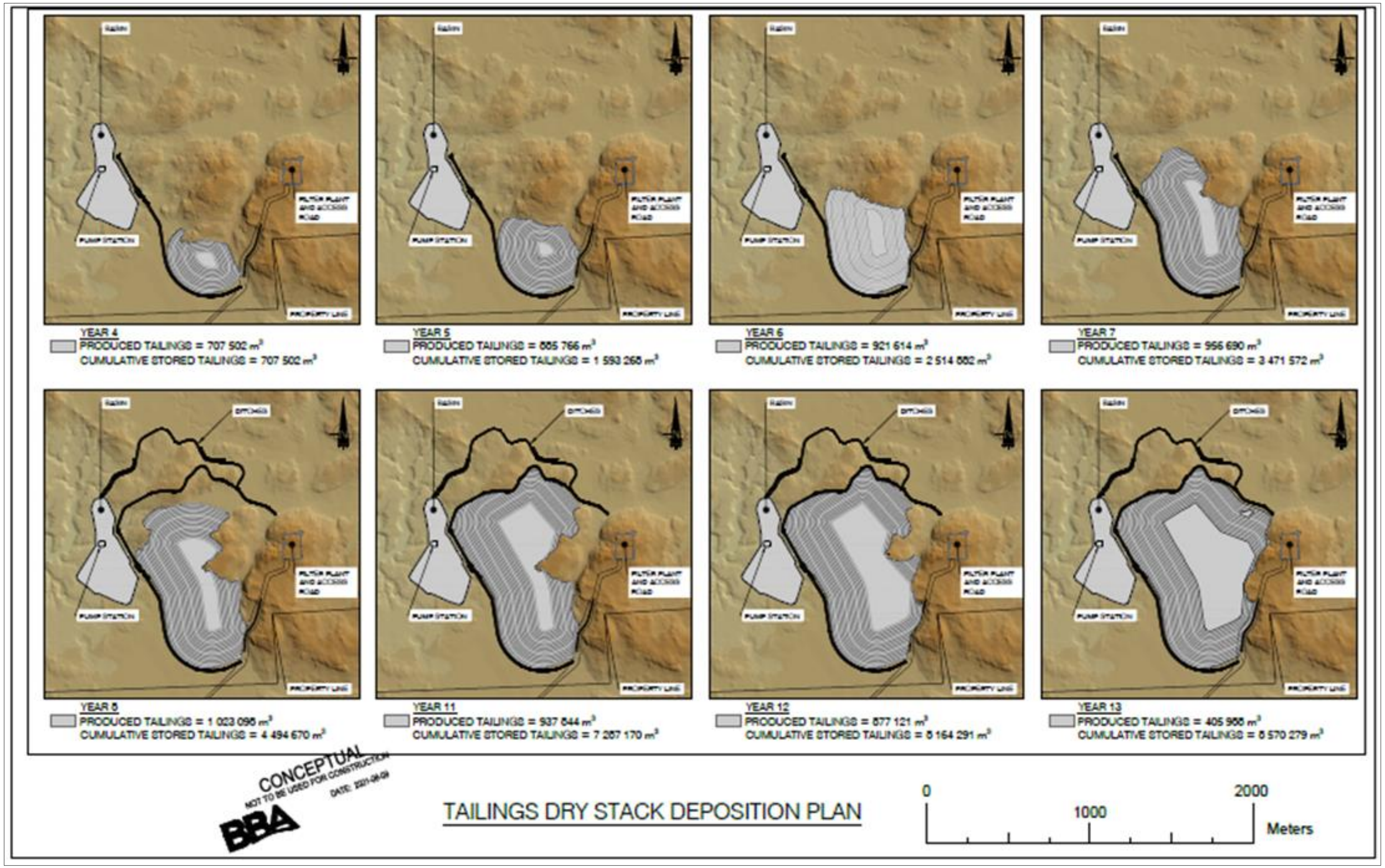


Figure 18-11: Dry Stack Deposition Plan



Source: BBA, 2021

18.9.4 Tailings Reclamation Plan

Dry stack tailings management technology presents the advantage of reducing the required mining footprint and facilitating progressive reclamation. To minimize the impacted areas during TSF development, a progressive reclamation strategy was developed based on the following considerations:

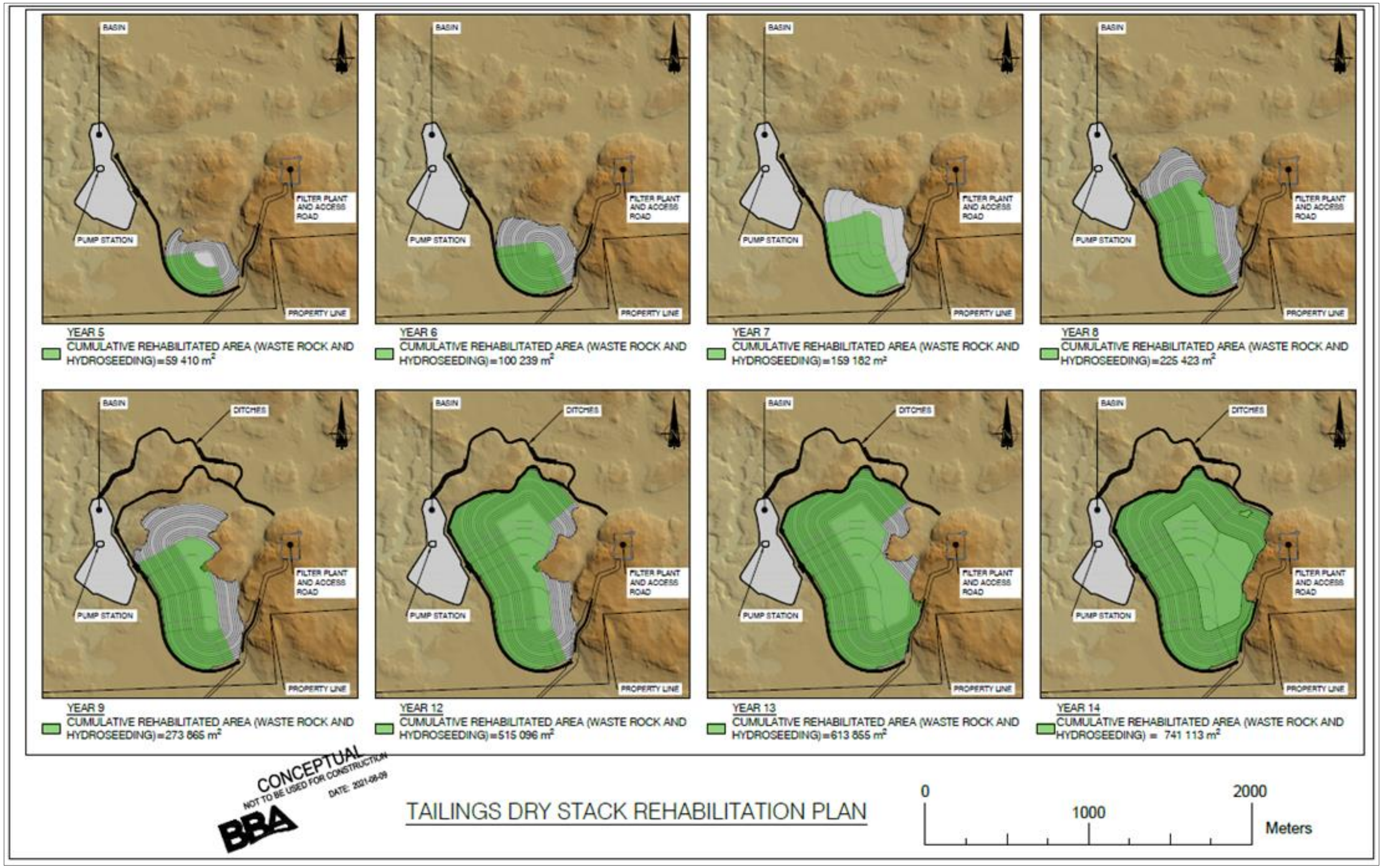
- Dry stack tailings will be placed in TSF as per the deposition plan.
- Progressive reclamation will commence once a section of the TSF has reached its final elevation, or when a portion of the TSF slope is considered final.
- The TSF slope will be covered with a 0.7 m thick layer of waste rock as per the TSF reclamation plan.
- Revegetation will be carried out by hydroseeding over the waste rock. Locally, native trees and shrubs could also be integrated into the final reclaimed surfaces.
- Both the plateau and slopes will be reclaimed concurrently.

The reclamation sequence has been developed based on footprint that needs to be managed at each period of the project, as well as the dry stack deposition plan. Table 18-6 summarizes the reclaimed surfaces that require management. Figure 18-12 presents the different stages of the tailings deposition and reclamation process, from Year 5 to the end of life, since TSF surfaces are not considered final at Years 1 to 4. The areas for reclamation and hydroseeding are shown on Figure 18-12.

Table 18-6: Waste Management and Progressive TSF Reclamation

Period		Produced Waste Rock			Waste Rock Placed at TSF per Period	Waste Rock Cumulative Volume at TSF	TSF Reclaimed Surface
(Years)	(Years)	(t/d)	(t/y)	(m³)	(m³)	(m³)	(m²)
1	2024	0	0	0	0	0	0
2	2025	0	0	0	0	0	0
3	2026	0	0	0	0	0	0
4	2027	362	132,000	66,000	41,587	41,587	59,410
5	2028	866	315,986	157,993	28,580	70,167	40,828
6	2029	725	264,710	132,355	41,261	111,428	58,944
7	2030	512	186,850	93,425	46,368	157,796	66,241
8	2031	12	4,478	2,239	33,909	191,706	48,442
9	2032	331	120,866	60,433	24,460	216,166	34,943
10	2033	35	12,668	6,334	18,106	234,272	25,866
11	2034	0	0	0	126,295	360,567	180,422
12	2035	0	0	0	69,131	429,699	98,759
13	2036	0	0	0	89,080	518,779	127,257
	Total		1,037,558	518,779	518,779		741,113

Figure 18-12: Dry Stack Deposition Plan



Source: BBA, 2021

18.9.5 Tailings Slurry and Reclaim Water Pipelines

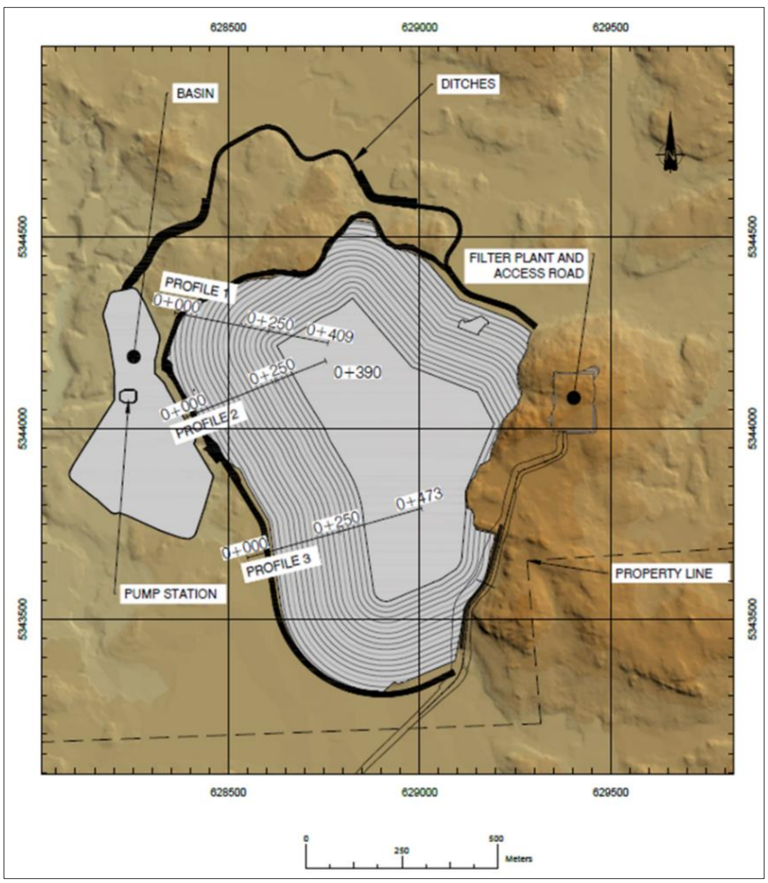
The final tails thickener underflow is pumped to either the underground paste plant or the filtration plant. The HDPE pipeline to the paste plant will buried until it reaches the underground portal, where it will be supported on a pipe rack suspended from the back of the drift. The HDPE pipeline to the filtration plant will run in a trench (fully buried only at vehicle crossings) along the road from the process plant to the filtration plant. Both lines are heat traced and insulated.

An HDPE process water pipeline will run alongside the filtration plant tailings pipeline in the same trench. This pipeline will be bi-directional, pumping process water from the process plant to the effluent treatment plant, or from the filtration plant (clarifier overflow) to the process plant.

18.9.6 Tailings Storage Facility Stability Analysis

A stability analysis has been performed under static and pseudo-static conditions for three critical sections (profiles 1 to 3) of the Wasamac TSF, as illustrated in Figure 18-13.

Figure 18-13: Critical Sections for Slope Stability Analysis of Wasamac TSF



Source: BBA, 2021

The parameters summarized in Table 18-7 were used for the slope stability analysis of the Wasamac TSF. Undrained shear strengths for the soft clay and the crust were obtained based on very limited laboratory tests and an assumed OCR value. Additional tests are required to produce a representative undrained shear strength profile for the entire depth of the clay layer.

The foundation soft clays were defined using stress history and normalized soil engineering properties (SHANSEP) given the progressive construction of the TSF for around 13 years, based on the geotechnical data available and several assumptions. The excess pore pressure of the clay during construction is assumed to be 60% for the short term and pseudo-static analysis, indicating that 40% of clay consolidation is assumed to be complete before the construction of the next stage.

The methodology, design criteria, seismicity of the site, geotechnical data, assumptions, and optimization results are detailed under a separate cover provided in August 2021 (BBA's report 3606029-000000-ERA-41-0001-R00)

Table 18-7: Shear Strength Parameters used in Slope Stability Analysis

Soil Type	γ (kN/m³)	Drained Properties		Undrained Properties		
		C' (kPa)	Φ' (°)	Φ (°)	S_u (kPa)	S_u/σ'_{v0}
Crust Clay	17.8	0	28	0	70	-
Soft Clay	17.8	0	28	0	25	0.38
Dry stack tailings	18	0	28	-	-	-
Rockfill	22	0	35	-	-	-
Till	19	0	32	-	-	-

The results of slope stability analysis of the Wasamac TSF under different loading conditions are summarized in Table 18-8. It was assumed the TSF would be constructed in stages to allow the clay foundation to gain strength through consolidation of the clay. Staged construction allows the use of the SHANSEP method for stability analysis. The obtained factor of safety for the assumed degree of consolidation (40%) during the staged construction shows that the satiability of the TSF in proposed configurations meets the design criteria specified in MERN (2017) and Directive 019 (MDDEP 2012); however, the validity of these assumptions needs to be addressed by more detailed geotechnical tests during detailed engineering.

Table 18-8: Results for slope stability analysis of the Wasamac TSF

Section	Static Short Term	Static Long Term	Pseudo Static	Comment
Profile 1 (with a stabilization berm)	1.6	3.1	1.1	Stage construction
Profile 2	1.5	1.7	1.1	
Profile 3	2.8	2.9	1.9	

18.10 Water Management

The site water management plan for the Wasamac mine will involve piping filtration plant water to the neighbouring effluent treatment plant when mill makeup water is not required. This process water will be treated and discharged at the permitted discharge location northwest of the tailings storage facility (TSF) pond. Water accumulated in the lined mill basin pond will consist of runoff contact water and seepage from the waste rock storage facility, as well as runoff water collected from the plant pad. This water will be piped to the mill process water tank for use or be expelled along the bi-directional water pipeline to the effluent treatment plant. Runoff contact water and seepage accumulated in the lined TSF pond will be pumped to the effluent treatment plant for treatment before discharge to the environment. The effluent treatment plant is designed to treat 200 m^3/h—a rate that will vary if tailings are reporting to the filtration plant or paste backfill plant.

18.10.1 Water Management Strategy

Under the Wasamac water management plan, all clean catchment runoffs shall be diverted away from infrastructure and all contact water of the project site will be intercepted, retained, monitored and treated (if required) prior to discharge to the receiving environment.

The two major water management infrastructures for the Wasamac property include the mill basin, and the tailings storage facility pond. The contact runoff from the waste rock storage facility (WRSF) and process plant site will be collected in the mill basin, and the contact runoff from the tailings storage facility (TSF) will be collected into the TSF pond. The contact water from both mill and TSF ponds will be pumped towards an effluent treatment plant located in proximity of the TSF, on the filtration plant site, for treatment before discharge into the environment west of the TSF pond.

Ponds are designed to retain operational and extreme flood water volumes and provide enough retention time for sediment settlement. The operational flows for these ponds were estimated according to a water balance analysis performed over different components of the water system. The major water component is the runoff (e.g., runoff and snowmelt from natural land covers and WRSF, TSF, process plant, roads). However, other components such as seepage and excess process water also play significant roles during dry seasons.

The extreme runoff and snowmelt components were modelled for the mill basin and TSF pond sub-catchments and surrounding facilities using HEC-HMS (version 4.7).

18.10.2 Plant Site Surface Water management

A drainage ditch surrounds the plant site to report any runoff water from the plant pad to the mill basin.

18.10.3 WRSF Surface Water Management

Two drainage ditches run along the east and west sides of the waste rock storage facility to capture runoff water and seepage to report to the mill basin immediately to the south of the WRSF (see Figure 18-15).

18.10.4 TSF Surface Water Management

A drainage ditch runs along the south side of the TSF in the early stacking years. This ditch is parallel to the road to access the TSF pond (refer to Figure 18-17). As the TSF develops, diversion ditches around the north and west sides allow to prevent runoff water from the natural landscape from entering the TSF. The two diversion ditches also provide road access to the TSF as stacking continues.

18.10.5 Site-Wide Water Balance

A preliminary site-wide water balance analysis was performed for the Wasamac Project, the results of which are summarized in this section.

The process plant water requirement and contributing water from various sources were calculated to manage the water surplus or identify the makeup water requirement. The water components from various sources were as follows:

- process water
- freshwater inflow
- underground mine water
- surface runoff from precipitation
- evaporation from the ponds and surfaces

The mine water balance is designed for two operating scenarios (operation of the filtration plant or paste backfill plant) depending on paste fill demand. The overall flow of different water components for the two operating scenarios is presented in Figure 18-14 on the following page. The environmental contributing volumes are considered for October, the wettest month.

Based on the process mass balance, approximately 4,800 and 3,450 m^3/d of treated contact water is released to the environment for filtration plant and paste backfill plant operating scenarios, respectively. Both operating scenarios require water extraction of 576 m^3/d from a freshwater source to be situated immediately east of the process plant pad. This freshwater volume is necessary for certain pump gland water, reagent mixing, and the elution column.

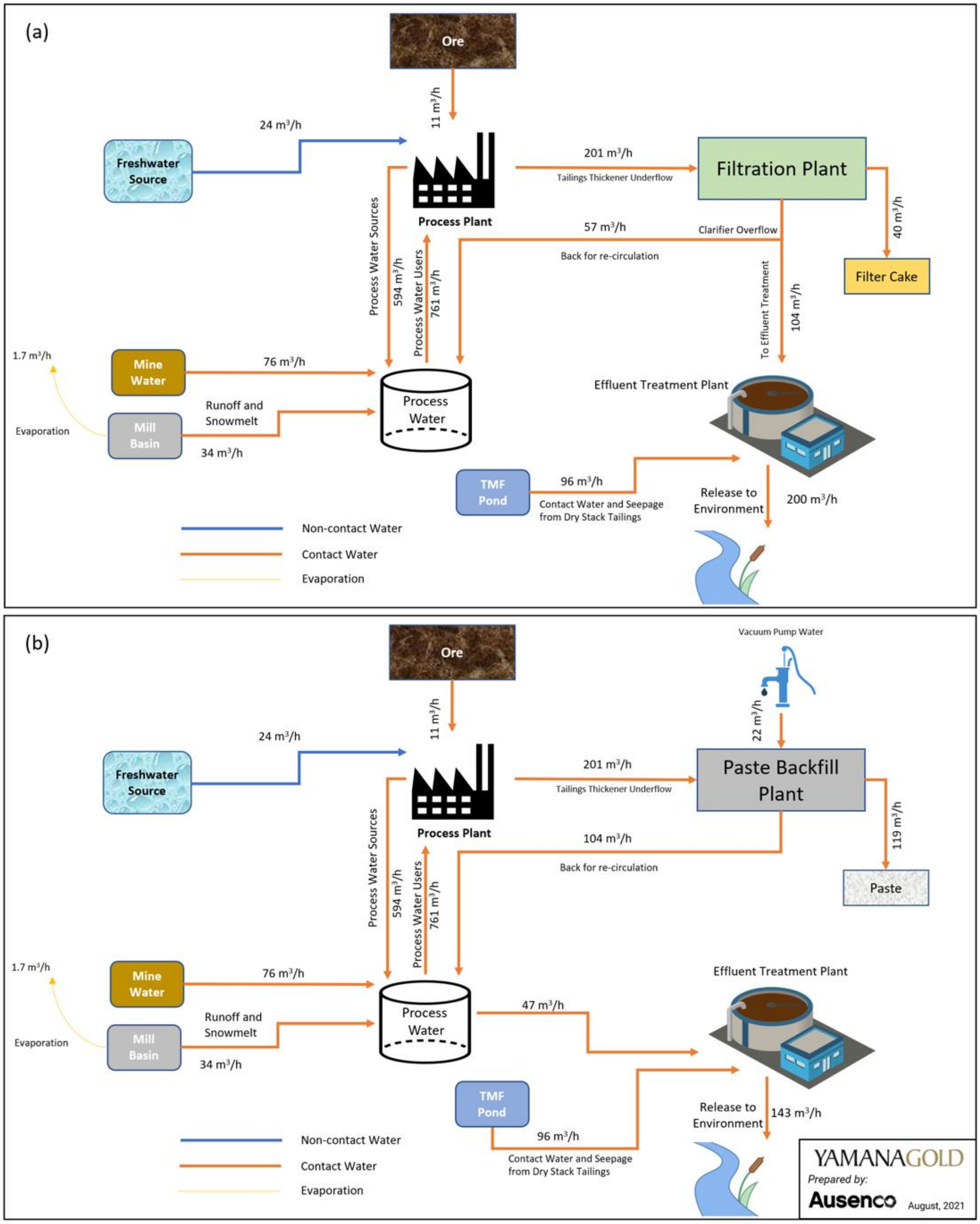
Based on these assumptions, monthly water excess is presented in Table 18-9. It should be noted that these calculations assume the process plant is operating at full capacity. During ramp-up, it is anticipated that there will be less process water in circulation as well as less runoff water collecting in the mill basin and TSF pond, because the size of the contributing facilities is not yet fully developed. The process makeup water demand does not rely on TSF pond water, but mostly on underground mine dewatering water. It is anticipated to have access to this mine water at start-up based on the mine plan development; supplementation from a freshwater source may be necessary if less water is available from the underground mine during ramp-up.

Table 18-9: Makeup and Treated Water from Various Sources and Components

Water Component		Jan	Feb	Mar	Apr	May	Jun	July	Aug	Sep	Oct	Nov	Dec
Fresh and Contact Water to Process Plant													
Net runoff minus evaporation	Plant Site (m^3/d)	-	67	258	279	244	360	293	395	261	372	221	57
	Waste Rock Facility (m^3/d)	-	51	195	211	184	272	221	298	197	281	167	43
	Pond Surface (m^3/d)	59	16	94	80	86	152	124	167	111	158	97	37
Groundwater Mine water (m^3/d)		1,824	1,824	1,824	1,824	1,824	1,824	1,824	1,824	1,824	1,824	1,824	1,824
Freshwater (m^3/d)		576	576	576	576	576	576	576	576	576	576	576	576
Treated Water to Environment													
Treated from Plant (Filter Ops) (m^3/d)		4,800	4,800	4,800	4,800	4,800	4,800	4,800	4,800	4,800	4,800	4,800	4,800
Treated from Plant (Paste Ops) (m^3/d)		3,432	3,432	3,432	3,432	3,432	3,432	3,432	3,432	3,432	3,432	3,432	3,432

Source: Ausenco

Figure 18-14: Wasamac Operational Water Balance for (a) Filtration Operation and (b) Paste Operation



Source: Ausenco, 2021

18.10.6 Start-Up Water

Start-up water will be available from constant dewatering of the underground mine at a rate of 76 m^3/h.

18.10.7 Makeup Water

Makeup water for the process plant is largely provided by the underground mine dewatering water. This water will have had sediment removed before being pumped to the process water tank on the surface.

18.10.8 Design Event

The design storm for sizing the mill and TSF basins is the 1:100-year storm. The hydrologic model used in the runoff analysis was HEC-HMS version 4.7, developed by U.S. Army Corps of Engineers. The US Soil Conservation Service (SCS) unit hydrograph method was applied to determine the runoff hydrograph for several extreme storm events. The SCS Type II distribution was selected to define the distribution of rainfall. A soil with a moderate infiltration rate was chosen for the study area. Antecedent moisture condition III (rain and low temperatures over the last five days causing saturated conditions) was assumed. A weighted average curve number of 92 and 85 was therefore estimated for the mill basin and TSF pond, respectively. Slopes, elevations and channel lengths were taken from topographic maps to estimate the time of concentration for each sub-catchment.

Rainfall intensity-duration-frequency (IDF) data from Kirkland Lake CS climate station was used as the basis for determining the storm rainfall depths for floods of different return periods. Knowing the flood rainfall depths for 2- to 100-year storms, flood intensities for 200- and 100-year storms are calculated using extrapolation equations derived from the IDF tool (Schardong, Simonovic, Gaur, and Sandink, 2020), as shown in Table 18-10.

Table 18-10: Rainfall Intensity Frequency Data at the Wasamac Site

Return Period (Years)	24-h Rainfall Total (mm)
2	44.6
5	57.2
10	65.6
25	76.2
50	84.0
100	91.8
200	100.7
1000	119.9

All sources of runoff, including rainfall, snowmelt, and combined rainfall-snowmelt events, should be considered in determining the design flows.

To account for snowmelt during a rain-on-snow precipitation event, equations developed by the U.S. Corps of Engineers (1960) for snowmelt during rainstorms (US Army Corps of Engineers, 1960) under clear sky are applied, as repeated below:

$$M = (0.1332 + 0.32P)\ T + 1.27$$

Where:

M is the snowmelt (mm/day)
P is the rate of precipitation (mm/day)
T is the temperature of saturated air in degrees Celsius

18.11 Ponds Design

18.11.1 Sediment Management

As the surface contact runoff from disturbed areas of the site, stockpiles, process plant, dry stacks and waste rock storage facilities normally carry loads of suspended solids, sediment ponds are designed to capture and settle the sediment loadings. Both ponds are designed to capture at least a 10-µm particle for the 100-year, 24-hour runoff event.

Based on the required retention volume and to avoid short-circuiting, different length to width ratios were analysed and a 5:1 ratio was selected for design. Overflow structures in the sediment pond system (i.e., spillways) are designed to withstand a minimum 200-year runoff event.

To account for the sediment load, the thickness of the settled sediment in the pond was assumed to be 50 cm and all flood calculations were conducted considering a fully sedimented pond bed of 0.5 metres.

18.11.2 Mill Basin

The mill basin is situated southwest of the process plant pad, immediately to the south of the waste rock storage facility (see Figure 18-15 on the following page). Different components of water excess and loss for the mill basin are presented on a monthly basis in Table 18-11 below. Evaporation losses are relatively small and flow volume from runoff and snowmelt is the key factor determining the month with maximum excess water.

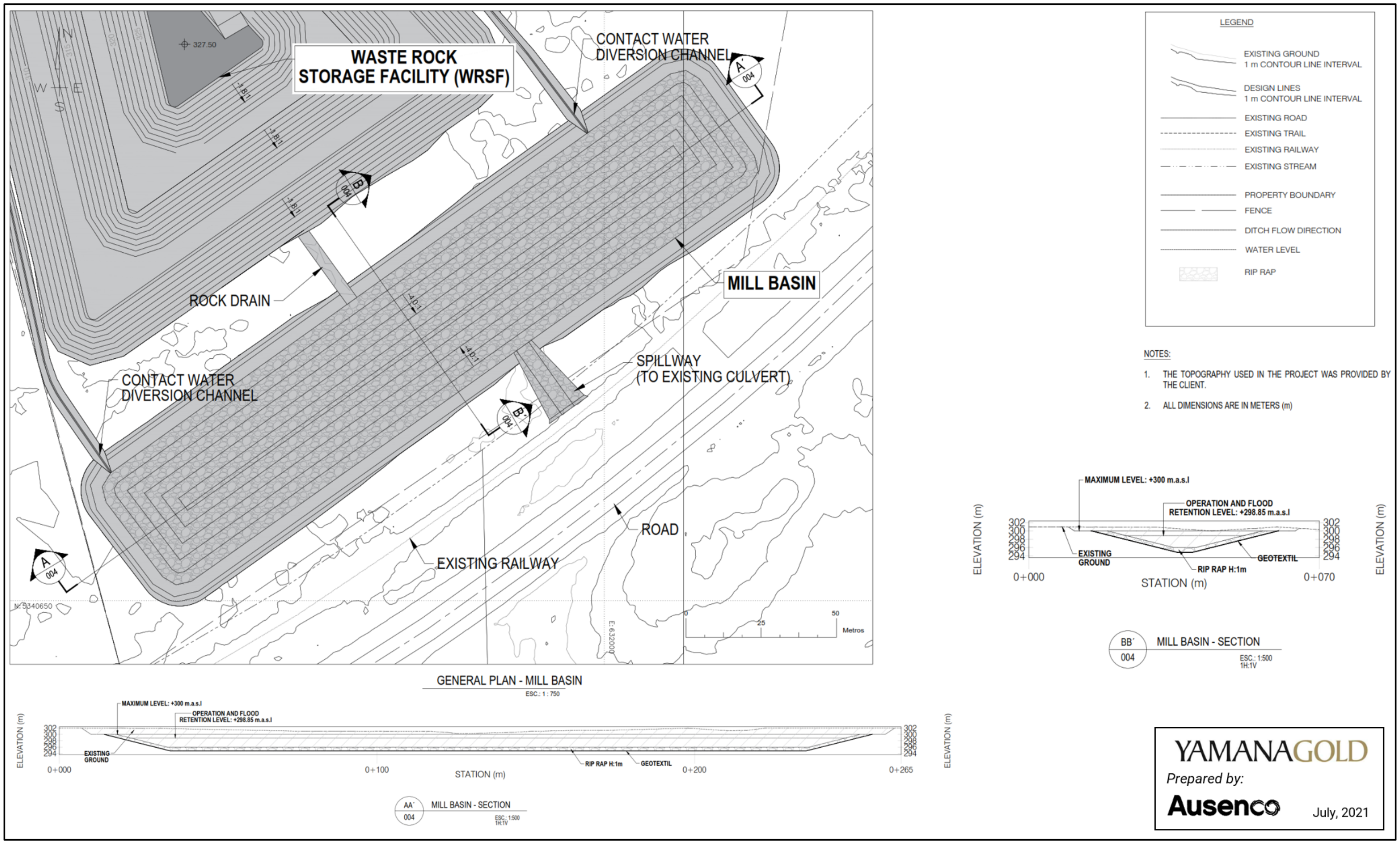
Table 18-11: Mill Basin Water Balance

Component (m^3/d)	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Runoff + Snowmelt	59	134	547	570	514	784	638	860	570	811	485	137
Process Plant	151	151	151	151	151	151	151	151	151	151	151	151
Evaporation Loss	-	-	-	-	100	121	121	94	59	-	-	-
Net Inflow	210	286	699	721	566	815	669	917	662	962	637	288

As shown, October is the month with the highest inflow volume with almost 960 m^3/d rate. It should be noted that the values in Table 18-11 are estimated for the full capacity operation rates. Limited evaporation from the waste rock dump or plant site could result in slightly less net inflow volumes.

To obtain the optimized pond size that permits settlement of suspended particles of at least 10 µm, avoid short-circuiting, and better facilitate removal of accumulate sediment, an optimal length to width ratio of 5 to 1 is estimated for the collection ponds.

Figure 18-15: Mill Basin Pond



Source: Ausenco, 2021

Assuming the specific gravity of the sediment particles to be 2.65 (~density of 2,650 kg/m^3), the kinematic viscosity as 0.01787 cm^2/s, and the particle size to be an average-sized silt sediment (5-micron), the settlement time of particles is approximately 40 minutes. Considering the critical conditions settlement velocity in a transitional flow from turbulent to laminar, the settlement time is five hours and thirty minutes for a depth of 1.2 m, which is significantly less than the 20-hour limit. For shallower effective depths, the settlement time is shorter.

Minimum freeboard requirement from wind setup and wave height were calculated and are presented in Table 18-12.

Table 18-12: Minimum Freeboard Requirement for the Mill Basin Pond

Wind Event Return Period (Year)	Setup (m)	Wave Height (m)	Vertical Run-up (m)	Freeboard Requirement (m)
2	0.019	0.14	0.15	0.17
10	0.026	0.16	0.18	0.21

Using the estimated values for operational flows, design flood, minimum freeboard requirement, minimum sediment load storage and required residence time, the mill basin was sized for three treatment scenarios, as shown in Table 18-13.

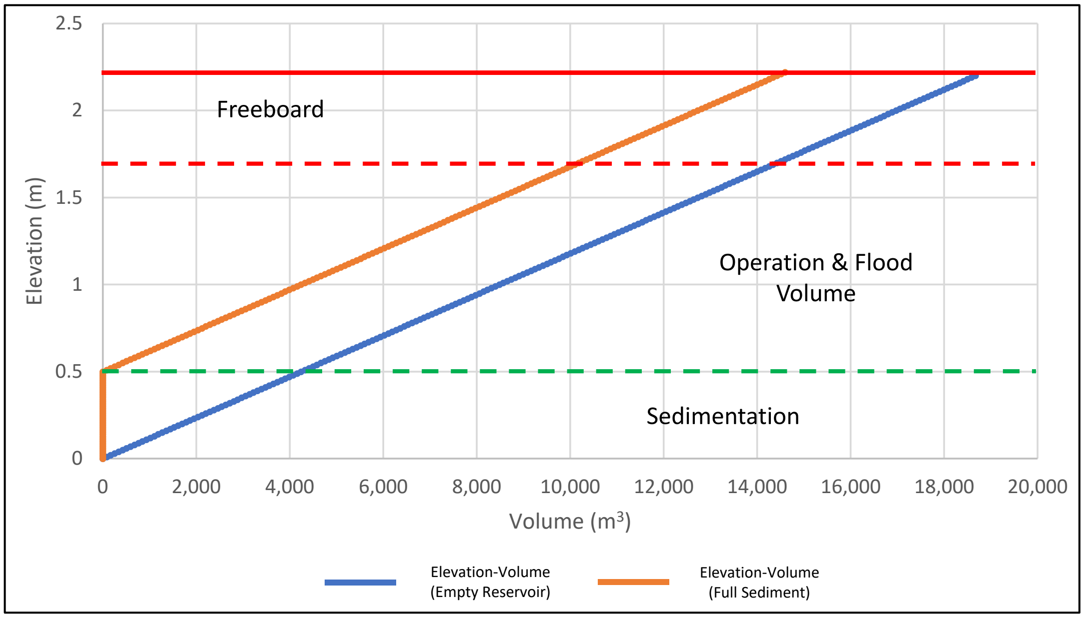
Table 18-13: Scenario Summary and Sizing for Mill Basin Collection Pond

Treatment Capacity (m^3/d)	Required Volume for Operation (m^3)	Required Volume for Extreme Flood (m^3)	Required Volume for Operation and Flood Retention (m^3)	Required Volume for Freeboard and Settled Sediment (m^3)	Total Pond Volume Required (m^3)	Length (m)	Width (m)	Effective Depth (m)	Pond Surface Area (m^2)
700	13,739	12,582	26,321	21,934	48,256	331	66	1.2	21,934
800	2,296	11,817	14,113	11,761	25,873	242	48	1.2	11,761
900	900	9,344	10,244	8,537	18,781	207	41	1.2	8,537

The third treatment scenario (900 m^3/d treatment capacity) was selected as the basis for design of the mill basin.

Based on the design criteria and the volume variation with the elevation changes in the mill basin pond, 50 cm has been allowed for on the bottom of the pond for sedimentation as presented in Figure 18-16.

Figure 18-16: Stage Elevation Curves for the Mill Basin Pond



Source: Ausenco, 2021

18.11.3 Tailings Management Facility Pond

The TSF pond is situated at the toe of the dry stack TSF, 6.3 km northwest of the main plant site. The pond will capture runoff and seepage from the TSF and is equipped with a pumping station to report the pond waters to the effluent treatment plant (see Figure 18-17 on the following page). There is high neutralizing potential for the tailings material in the ABA testwork completed to date, however; the pond design includes a geomembrane liner until tailings kinetic testwork is complete.

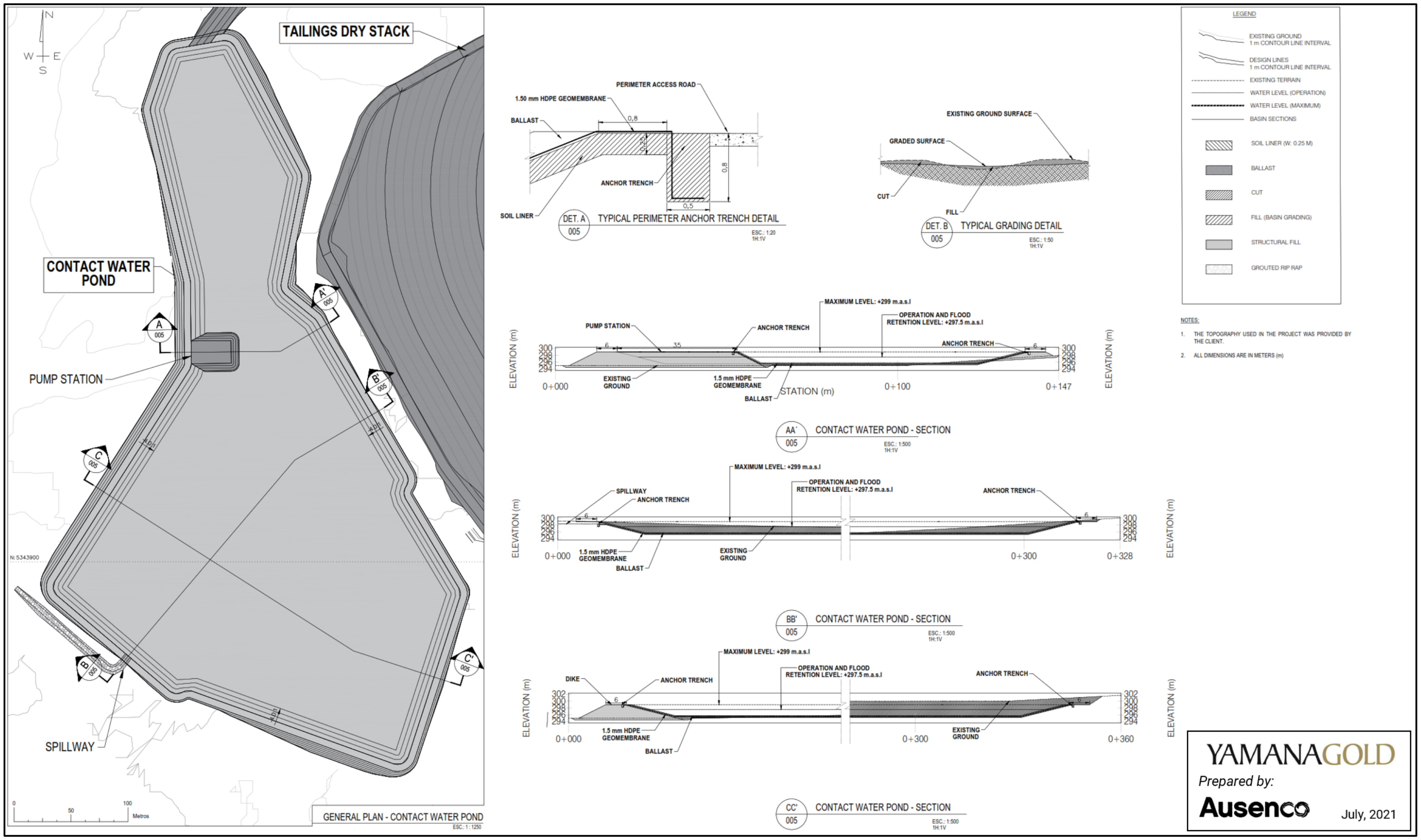
The operational water balance for the TSF pond summarized in this section are calculated and presented in Table 18-14 below. As both components of flow are only driven by meteorological conditions, maximum net inflow occurs in October with a rate of 3,756 m^3/day. Although precipitation rates are high during the cold season, notably in December and January, the net inflow is relatively lower since the water is retained as snowpack and normally adds up to rainfall runoff during the months of April, May and June.

Table 18-14: Tailings Storage Facility Water Balance

Component (m^3/d)	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Runoff + Snowmelt	280	621	2,535	2,637	2,381	3,633	2,957	3,986	2,639	3,756	2,249	636
Process Plant	-	-	-	-	473	570	570	445	278	-	-	-
Net Inflow	280	621	2,535	2,637	1,908	2,063	2,386	3,541	2,361	3,756	2,249	636

It should be noted that the values in the table above are estimated for the full capacity operation rates.

Figure 18-17: Tailings Storage Facility Pond



Source: Ausenco, 2021

The dry stack tailings storage facility extends over almost 84 ha of the project site. The total 100-year, 24-hour flood volume in this catchment is estimated as high as 64,500 m^3. Runoff from floods is retained in this pond, treated, and then discharged to the receiving environment. The same modelling basis as for the mill basin pond was applied to estimate optimum sizes of the TSF collection pond.

Minimum freeboard requirement from wind setup and wave height were also calculated for the TSF pond and are presented in Table 18-15.

Table 18-15: Minimum Freeboard Requirement for the TSF Pond

Wind Event Return Period (Year)	Setup (m)	Wave Height (m)	Vertical Run-up (m)	Freeboard Requirement (m)
2	0.066	0.28	0.31	0.37
10	0.090	0.33	0.36	0.45

The freeboard is set to 50 cm for the TSF pond to meet to minimum requirements.

A summary of modelling for three scenarios (different effluent treatment rates) considering required freeboard or maximum reserved depth for sediment is presented in Table 18-16.

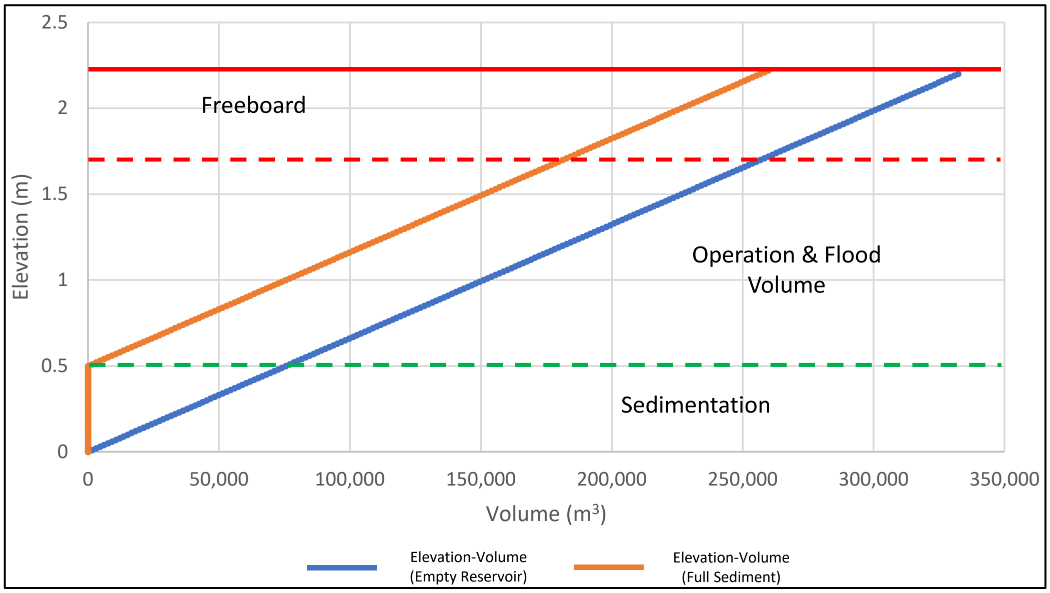
Table 18-16: Scenario Summary and Sizing for Mill Basin Collection Pond

Treatment Capacity (m^3/d)	Required Volume for Operation (m^3)	Required Volume for Extreme Flood (m^3)	Required Volume for Operation and Flood Retention (m^3)	Required Volume for Freeboard and Settled Sediment (m^3)	Total Pond Volume Required (m^3)	Length (m)	Width (m)	Effective Depth (m)	Pond Surface Area (m^2)
2300	116,247	64,480	180,727	150,606	331,334	868	174	1.2	150,606
2400	95,708	64,168	159,876	133,230	293,106	816	163	1.2	133,230
2500	80,408	63,929	144,337	120,281	264,619	776	155	1.2	120,281

By comparing scenarios of treatment plant capacities, as presented above, the collection pond length and width are calculated as 870 m and 175 m, respectively. The first treatment scenario (2,300 m^3/d) treatment capacity was selected as the basis for design of the TSF pond. Considering the effective depth, sediment deposition volume, and other requirements, the total pond volume is estimated as 331,000 m^3.

For these design criteria the volume variation with the elevation changes in the TSF pond, considering 50 cm sedimentation on the bottom of the pond is presented in Figure 18-18.

Figure 18-18: Stage Elevation Curves for the TSF Pond



Source: Ausenco, 2021

18.12 Spillway Design

Overflow structures in the sediment ponds are designed to withstand a minimum 200-year runoff event. Flood routing across the mill basin and TSF pond area was modelled using HEC-HMS, including a routing component for flows through reservoirs to assure short-circuiting is prevented.

The spillway is designed to safely overpass the flood with the 200-year return period for both the mill basin and TSF collection ponds. For this purpose, a trapezoidal spillway with the bottom set at 50 cm below the pond crest elevation is included for the mill basin and the TSF collection ponds. Setting the spillway bottom at 50 cm below the embankment assures maintaining the required freeboard during normal conditions. To account for possible combination of flood and wind waves, a minimum 20 cm freeboard is maintained during the 200-year flood. Summary of these calculations are presented in Table 18-17.

Table 18-17: Optimized Spillway Widths and Overflow Depth

Flood Event	Spillway Width (m)	Maximum Depth (m)	Minimum Freeboard (m)	Peak Inflow (m^3/s)	Peak Discharge (m^3/s)	Maximum Spillway Discharge Capacity (m^3/s)
		During the 200-year 24-hour Flood				
Mill Basin Pond	2.5	0.25	0.25	0.55	0.55	1.64
TSF Pond	15	0.28	0.22	3.6	3.6	8.49

As shown, trapezoidal spillways of 2.5 m and 15 m for the mill basin pond and TSF pond, respectively, can safely overpass the 200-year, 24-hour flood. This design allows a minimum 18- and 42-hour flood retention in the mill basin and TSF ponds which is sufficient for particles larger than 10-µm to settle.

The spillway chute channel is designed as an earth channel with a 2:1 slope, a 1.7 m elevation drop, and riprap to avoid the erosion on the chute and across the apron. As the overflow through the spillway may occur irregularly and probably only once during the lifetime of the mine, the chute channel bottom is considered to have a poor condition with a Manning's coefficient of 0.055. The maximum velocity over the spillway of the mill basin pond reaches 2.3 m/s with a depth of 10 cm. The same velocity and depth are 2.6 m/s and 10 cm for the TSF pond chute.

18.13 Underground Infrastructure

18.13.1 Crusher and Conveyor

The underground crushing plant consists of two areas, the ROM bin and the crushing area. The ROM bin is a steel-lined excavation that is 10 m in diameter and approximately 50 m height. The bin is sized for 7,000 tonnes of ROM, or approximately 24 hours of storage. The truck unloading area above the ROM bin includes a static grizzly and a remotely-operated rockbreaker station. Trucks can access the unloading area from two directions, each connected to the main stopes.

The crushing area is an excavation that measure 30 m x 12 m x 12 m height. The crushing area includes an electrical room, sump pumps, and an overhead crane. The crane services the bin discharge feeder, crusher, dust collector and sacrificial conveyor. Clearance has been allowed around all equipment suitable for either personnel or skid steer access. The area has light vehicle access connected to the main stopes. The underground crusher will be commissioned in 2027; ore will be hauled to the surface until commissioning is complete.

The underground conveyors downstream of the crushing area are situated in dedicated drifts, with additional excavation required for transfer points. The conveyors are suspended from the back of the drift and total approximately 3,000 m of conveyor linear metres. Clearances around the conveying equipment shall be suitable for light vehicle and skid steer access. The drift will also contain water piping, cable trays, and air ducting.

The final underground conveyor daylights at a portal, as the conveyor support system transitions from back-supported conveyor hangers to a bent and truss system.

18.13.2 Mine Air Intake and Heating

Surface roads to support underground mine services for ventilation and the paste backfill plant will be developed off of rang Wasamac, which is accessible from Highway 117 via rang des Cavaliers 1 km northeast of the main project site. The existing rang Wasamac will be extended 1.8 km to provide access to the collar for the paste backfill plant cement borehole and the two ventilation collars upon which the mine air heat plants will be constructed.

Mine air intake and heating will be accomplished from two ventilation raises. The heat plants will sit on concrete pads and be propane fueled; the fans will be at the bottom of the raises to reduce noise.

18.13.3 Paste Plant Cement Discharge and Raise

Binder is fed from a surface borehole directly into the binder silo inside the backfill plant excavation. The binder delivery will be via gravity, which greatly reduces both the noise and time required to unload trucks vs. conventional pneumatic unloading into silos on surface. A building will be required to house the trucks while they are unloading and will require a compressed air system with dryer to facilitate the unloading.

18.13.4 Underground Paste Fill Mixing Plant and Distribution

Thickened tailings slurry is pumped to the underground backfill plant from the mineral processing facility via a pipeline through the declines, which access the orebody. It enters a large, agitated storage tank with approximate six hours of storage capacity. This large tank is sized to allow for continued operation of the backfill plant in the event that upstream interruptions in tailings feed occur. It also serves the purpose of modulating variation in thickener underflow density so that a more consistent slurry can be fed to the filters.

When the plant is operating, tailings slurry is pumped to two of three disc filters (two operating, one standby) and the filter cake discharges onto a weigh conveyor which runs below the filters. The mass of filter cake is measured and used to determine the amount of slurry and binder required to achieve the backfill recipe (solids and binder content). Additionally, a stream of slurry bypasses the filters and reports to a vortex mixer where the dry cement is also added. This serves to help with uniform mixing of cement in the paste backfill as well as reducing cement dust production. The binder is delivered to an underground silo from surface via gravity and is metered into the vortex mixer using a mass flow meter and transported via screw feeders. The cemented paste backfill overflows the twin shaft mixer into a single paste pump which transports it throughout the mine. A single pump was selected based on the plant's anticipated utilization.

A 10-inch diameter distribution system will be installed throughout the mine and will include instrumentation for monitoring of backfill pours. The line will be cleaned with water flushing, foam pigs and compressed air at the end of pours. The filtrate will report to the mine dewatering system via the plant's process water system.

The plant includes a clean water system to provide suitable seal water to the three vacuum pumps, and has its own compressed air system. Emergency power is included in the electrical design for critical equipment.

Figure 18-19 on the following page presents the simplified flowsheet for the paste backfill plant.

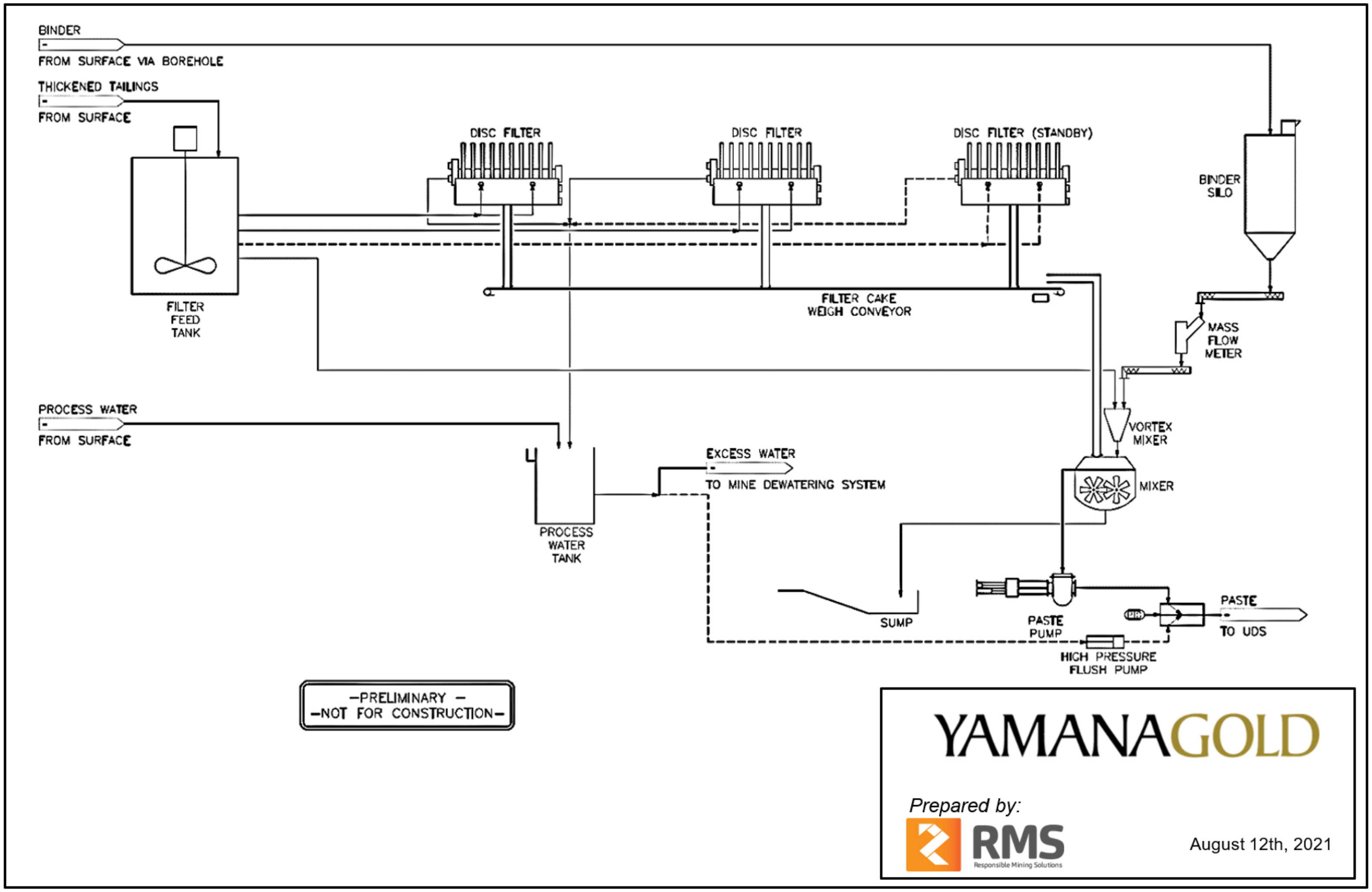
18.13.5 Underground Electrical Distribution

The underground electrical system will be powered by two 13.8 kV cable circuits configured with N-1 contingency and which originates from the 13.8 kV switchgear within the mine portal electrical room. Normal power is supplied by the 18/24 MVA power transformer located at the 120 kV outdoor substation, whereas emergency power is provided by three 1 MW, 13.8 kV standby generators sited at the mine portal electrical room.

Power is distributed to the crushing plant, service bay, paste fill station and ventilation raises via dedicated underground electrical rooms, whereas the ramps across the various zones are serviced by outdoor distribution switchgears and mine power centres (MPC). Each underground electrical room will be outfitted with a 13.8 kV switchgear and a dry-type distribution transformer close coupled to the 600V MCC to allow medium- and low-voltage power distribution. Outdoor switchgears will offer 13.8 kV distribution, whereas MPCs will be equipped with motor starters, power distribution panels, and programmable logic controller (PLC) cabinets to facilitate the low-voltage loads.

Larger electrical consumers such as the crushed ore stockpile conveyor, ventilation fans and underground mine pumps will be supplied at medium voltage (4.16 kV), whereas all other mine pre-production and operational equipment are forecasted to operate at 600 V.

Figure 18-19: Simplified Process Flowsheet Underground Paste Backfill Plant



Source: RMS, 2021

18.14 Francoeur Mine Historic Infrastructure

Infrastructure located on the Francoeur Property associated with the historic mine includes:

- access road to site
- security hut
- headframe and galleries for the underground mine
- administration buildings
- shaft #6 and #7
- ventilation fan with silencer and an acoustic screen
- electrical workshop
- mineral silo and storage facility
- garages and warehouses
- hazardous waste disposal warehouse
- core shack
- treatment pond and waste water treatment system
- waste stockpile and purification fields
- parking

19 MARKET STUDIES AND CONTRACTS

Yamana has not completed any formal marketing studies regarding gold production from mining and processing Wasamac ore into doré bars. Gold production is expected to be sold on the spot market. Terms and conditions included as part of the sales contracts are expected to be typical of similar contracts for the sale of doré throughout the world.

The mineral reserves were calculated at a gold price of US$1,250/oz. For the purpose of economic analysis, a gold price of US$1,550/oz was assumed and a USD:CAD exchange rate of 1.28:1.00 was used.

Yamana plans to contract out the transportation, security, insurance, and refining of doré gold bars; Yamana has not entered nor is currently negotiating any such contracts. Yamana may enter into contracts for forward sales of gold or other similar contracts under terms and conditions that would be typical of, and consistent with, normal practices within the industry in Canada and in countries throughout the world. No company has been contacted to provide a quotation for these services. The feasibility study assumes a refining, transportation and insurance charge of C$2.00/oz of gold and 99.95% payability for gold content, comparable to similar indicative terms reported by other mining operations in the region.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Review

This chapter provides information on the biological, physical and socioeconomic baseline studies for the Wasamac Project. Also discussed are environmental and social impacts that will be assessed through the Environmental Impact Assessment (EIA) process, project permitting requirements, community consultation and engagement requirements, and mine closure planning.

20.1.1 Baseline and Supporting Studies

The Wasamac property is located approximately 15 km west-southwest of the city of Rouyn-Noranda in the Abitibi-Témiscamingue region of western Québec (see Figure 18-1). The property comprises three mining concessions and 11 mining claims covering a total area of 1,149.33 hectares (ha) (WSP, 2020). The baseline data supporting this section were collected within the study area including the Wasamac property and spanning approximately 40 km^2 (Englobe, 2019).

A preliminary baseline study detailing the existing conditions for the biological and physical environment was carried out by AECOM for Richmont Mines (previous property owners) in 2013, and was later accompanied by a complementary study conducted by Englobe in 2019 (AECOM, 2013; Englobe, 2019). Since very little new data in this area have been acquired since the completion of these two studies, this chapter relies heavily upon the information provided in the aforementioned studies. Additional studies are being collected (summer 2021) by WSP to cover gaps created by the change in project layout.

The following subsections describe the existing information on the biological, physical, and socio-economic setting of the project site and surrounding areas.

20.1.1.1 Biological Environment

This section presents the baseline conditions of the terrestrial environment (i.e., wetlands, flora, fauna, species with a special status, and protected areas).

20.1.1.1.1 Wetlands

Regionally, wetlands comprise approximately 2,700 km^2 (8.3%) of the Abitibi and James Bay lowlands. Wetlands cover 10.3% (65.7 ha) of the Wasamac project site. The 2013 AECOM study described four different types of wetlands: ponds, marshes, swamps, and bogs.

20.1.1.1.2 Flora

Forested environments account for 2,706.1 ha of the study area surrounding the Wasamac property. Non-forested areas make up approximately 5% of the study area and are mainly mining areas. The Wasamac property is contained within the Dense Softwood Stands and Light Hardwood Stands subzone of the Softwood Dominated Boreal Vegetation Zone.

The study area is made up of mostly deciduous stands (1,034 ha), but mixed stands (599 ha) and coniferous stands (175 ha) are also present. Deciduous stands are dominated by Trembling Aspen (*Populus tremuloides*) and trees of the genus Betula. Coniferous stands in the area are dominated by Balsam Fir (*Abies balsamea*) and Black Spruce (*Picea mariana*).

The Québec Natural Heritage Data Centre (CDPNQ, 2018a) reported that no plants that are threatened, vulnerable or likely to be designated as such have been detected in the study area or within its radius of influence. However, 21 species of interest are present within the Abitibi Regional County Municipality and could therefore occur within the study area. Of these, one (1) is designated as threatened, one (1) is vulnerable and nineteen (19) others are at risk of being designated as such. None of these species were detected in Englobe's inventory of the area; however, AECOM's 2013 inventory confirmed the presence of the Striated Corallorhiza (*Corallorhiza striata*) and Ostrich Fern (*Matteucia striuthiopteris pensylvania*). Striated Corallorhiza is at risk of being designated as vulnerable or threatened, whereas Ostrich Fern has been designated as vulnerable to harvest. Both occurrences were recorded at the deposit site. The Englobe inventory revealed the presence of Reed Canary Grass (*Phalaris arundinacea*), an invasive plant species whose spread is facilitated by trucking and was found in the majority of open areas.

20.1.1.1.3 Fauna

The Centre du patrimoine naturel du Québec (CPNDQ) indicated the presence of 16 fish species withing the project area(2018b). During the 2019 Englobe inventory, the presence of seven of these species was confirmed between six lake and 10 stream survey stations. These included Brown Bullheaded Catfish (*Ameiurus nebulosus*), Pumpkinseed (*Lepomis gibbosus*), Five-spined Stickleback (*Culaea inconstans*), Finescale Dace (*Chrosomus neogaeus*), Yellow Perch (*Perca flavescens*), Common Logperch (*Percina caprodes*), and an unknown Phoxinus species. The Englobe inventory also recorded the presence of Smallmouth Bass (*Micropterus dolomieu*), which had not been previously recorded in the area. Most species captured in the Englobe inventory were the Brown Bullheaded Catfish and Pumpkinseed, two common species that are tolerant of warm, poorly oxygenated, turbid waters.

The 2013 inventory revealed the presence of five species of frog, one species of snake, and two species of salamander. The Englobe inventory was primarily focused on evaluating the presence of Snapping Turtle (*Chelydra serpentina*); however, no evidence of this species nor any testudines was present. The studied area presents potential for egg-laying habitat on the shoulders of old roads and infrastructures.

Avifauna surveys in 2013 revealed the presence of 79 species, 26 of which were confirmed in the complementary 2019 inventory. As survey efforts were concentrated along the edges of water, most species identified were migratory waterfowl. Forest dwelling, as well as raptor avifauna, were also recorded as these species tend to frequent transitional riparian environments. The 2013 inventory confirmed the presence of two species designated as vulnerable: the Canada Warbler (*Cardellina canadensis*) and the Bald Eagle (*Haliaeetus leucocephalus*).

Mammals present at the project site include both large, small, and micro mammals. Large mammals present include white-tailed deer (*Odocoileus virginianus*), beaver (*Castor*), black bear (*Ursus americanus*), moose (*Alces alces*), wolf (*Canis lupus*), and fox (*Vulpes vulpes*). The smaller mammals present include muskrat, mink, and otter. In terms of micro mammals, several species of mice and squirrel were recorded during the 2013 inventory. Shrews, moles, and voles have the potential to inhabit the study area, according to the Atlas des Micromammifères du Québec (Desrosiers et al., 2002). Of the potential present species, Cooper's Lemming (*Synaptomys cooperi*) and the Rock Vole (*Microtus chrotorrhinus*) are likely to be designated as threatened or vulnerable in Québec.

The 2019 mammalian inventory efforts were primarily concentrated on chiropterans. In total, 31 bat calls were recorded; of these, the Little Brown Bat (*Myotis lucifugus*) and Northern Myotis (*Myotis keenii*) calls were most abundant. Most of these calls were recorded at stations in riparian habitats. Both species have been designated as endangered (COSEWIC, 2018). The Big Brown Bat (*Eptesicus fucus*) was also recorded. Of the 31 calls, eight calls could not be identified due to lack of significant sound characteristics. Due to their low frequencies, the eight unidentifiable calls could be attributed to the Big Brown Bat, Silver-haired Bat (*Lasionycteris noctivagans*), or Hoary Bat (*Lasiurus cinereus*). The latter two have not been confirmed within the project site but have the potential to occur there. No migratory bats were recorded; however, surveys were taken during a time of the year when migratory bats were not likely to be present in the area. Overall, residential bat numbers were also relatively low.

20.1.1.1.4 Special Status Species

The CPNDQ (2018b) found no records of species threatened, vulnerable or likely to be designated as such at the project site; however, data from the 2013 AECOM and 2019 Englobe inventories indicate the potential for occurrence of some special status species. Plant inventories revealed the presence of two special status species: Striated Corallhoriza (at risk of being designated as vulnerable or threatened) and Ostrich Fern (vulnerable to harvest).

The Snapping Turtle (*C. serpentina*) has previously been sighted at Lake Wasa but was not confirmed by the 2019 inventory. Additional studies during the egg-laying season are required to determine the presence of this species at the project site. Avifauna surveys showed the presence of two vulnerable species: Canada Warbler and Bald Eagle. Mammalian surveys showed the presence of two endangered species within the study area: Little Brown Bat and Northern Myotis. Cross-referencing the special status species recorded in the greater Abitibi-Témiscamingue region and the habitat potential of the project site indicates the potential presence of Cooper’s Lemming and the Rock Vole. Supplemental studies occurring during crucial migratory times for migratory bats as well as other species could help confirm land use and timing of land use for many species in the area.

20.1.1.1.5 Protected Areas

According to the *Natural Heritage Conservation Act* (MELCC, 2018), no protected areas are present within the project site. Regional Protected Areas (located within 10 km of the inventoried area) include the following:

- Protected Ruisseau-Clinchamp Ecological Reserve
- Ancient Forest of Lake Dasserat
- Protected Lake Opasatica Biodiversity Reserve
- Lake Évain Containment Area for White-tailed Deer

20.1.1.2 Physical Environment

This section presents the results of the studies of baseline conditions of the physical environment (i.e., air quality and noise, hydrology, and surface water quality).

20.1.1.2.1 Air Quality and Noise

The Air Quality Index is a publicly available tool used to inform residents about the air quality in their area and is based on the following parameters: ozone, $PM_{2.5}$, sulphur dioxide, nitrogen dioxide, and carbon monoxide. Air Quality Index information for 2019 is available for locations near the Wasamac project site, namely the Rouyn-Noranda region and the

Montée du Sourire & Downtown sector. Air quality monitored for the Rouyn-Noranda region showed that air quality was higher overall than at the Montée du Sourire & Downtown sector. In Rouyn-Noranda in 2019, air quality was rated "good" for 180 days, "acceptable" for 152 days and "poor" for 33 days. During this same period in the Montée du Sourire & Downtown sector, air quality was rated as "good" for 163 days, "acceptable" for 152 days and "poor" for 77 days.

The Provincial MEFCC Air Quality Monitoring Network has two air quality monitoring stations in Rouyn-Noranda: Parc Tremblay Station and East Mgr Rhéaume. Information from these stations describing the years of maximum and minimum concentrations of sulphur dioxide, ozone and $PM_{2.5}$. Overall, concentrations of sulphur dioxide and $PM_{2.5}$ at Parc Tremblay have decreased over time, whereas concentrations of ozone have increased. Data from East Mgr Rhéaume Station were only available for sulphur dioxide concentrations, which have also shown to decrease over time.

Current air quality conditions will be monitored over six months or more at sampling stations deemed appropriate for the project site. This data will be used for atmospheric dispersion modelling.

In 2013, Vinacoustik Inc. carried out a noise study focusing on the sound impact of the excavation phase of a ramp, considering the effect of a 38 km-per-hour wind. The study identified four "sensitive points" based on the approximate locations of residential areas near the perimeter of the project site. It was found that the noise limit criteria set by the MEFCC was not exceeded at any of the sensitive points except for Point 2, which was located at the northern portion of project site. However, it was noted that ambient noise levels in this area are equal to or greater than the noise levels from excavation or the project itself.

20.1.1.2.2 Surface Water Hydrology and Water Quality Baseline

The surface water hydrology of the project site is characterized by the presence of several lakes, meandering streams and wetlands, as well as the absence of rivers. Most streams are affected by beaver activity, leading to the formation of ponds and, subsequently, wetlands. Lakes within the area are Arnoux, Mackay, Adéline, Wasa, Chat Sauvage and Hélène. In 2013, AECOM carried out a theoretical evaluation of flows and floods at three lakes within the study area. Peak flows were calculated for annual floods between 2 and 100 years (see Chapter 18 under water management)

In 2019, Englobe carried out a complementary baseline study that evaluated the water quality of six lakes and nine streams within the vicinity of the project site. Water quality was not found to be limiting to aquatic life. Most analyzed parameters did not surpass their applicable criteria and approximately one-quarter of the analyzed parameters did not reach detection limits. Within the lakes assessed, four have exceeded at least one criterion. Lakes Chat Sauvage and Hélène exceeded one criterion: chromium. Chromium levels in these lakes were found to be three times greater than the CCME guideline for the protection of aquatic life; however, all other lakes assessed showed smaller divergence for the chromium criterion. Lake Arnoux showed the greatest number of exceedances, namely in aluminum, chromium, copper, lead, iron and zinc, as well as fecal coliforms. All lakes surveyed exceeded the MEFCC Drinking Water Quality Regulation criterion of 0 colony-forming units per 100 millilitres for fecal coliforms. In addition, all nine streams surveyed exceeded at least one parameter of either CCME or MEFCC criteria. Of all the stations, n˚458, located on the western border of the deposit site, had the worst water quality results. All stations surveyed exceeded metal concentration criteria, most often in iron, aluminum, and manganese. Additionally, four stations exceeded the criteria for lead and chromium, seven stations exceeded criteria for copper, and five stations exceeded the guideline criteria for zinc concentrations. As was the case in lakes, all stream stations exceeded the acceptable levels of fecal coliforms for the Drinking Water Quality Regulations (MEFCC), notably in station n˚551, which had 25 times the criteria threshold of colony-forming units acceptable for the CCME recreational activity criterion. The total suspended solid and turbidity criteria were only exceeded for one sample at station n˚458. Several streams also exceeded the oxygen criteria designated by MEFCC as well as the acceptable pH levels for the CCME and MEFCC criteria of less than 6.5.

20.1.1.2.3 Groundwater Hydrology and Water Quality Baseline

In 2012 and 2014, studies were carried out by Richelieu Hydrogéologie concerning the hydrogeological context of the project site. These studies evaluated aquifers made of clayey-silt of thicknesses between 1.5 m and 9.4 m, aquifers made of loose granular deposits with a base composed of silty to sandy till (0.4 m to 13.6 m thick) and bedrock.

The studies also included a survey of residential wells used for drinking water supply. Residents of rang des Cavaliers were surveyed between the intersection of Highway 117 and the eastern end of Lake Hélène, Wasamac Road and Rideau Boulevard (Highway 117). Within the 45 samples collected, it was found that groundwater in the area was relatively low in minerals except those locations proximal to an old TSF at the southern end of the property. In addition, pH levels were generally alkaline, and at least one aesthetic objective for iron, manganese, and sulphur was exceeded in most wells. The guideline for the potability of drinking water was exceeded in one location for selenium, four locations for lead, and four locations for mercury concentrations. Aromatic hydrocarbon levels (both monocyclic and polycyclic) were found to be below or close to detection levels in most samples except for two samples, which exceeded the organic contaminant criteria for toluene and one sample that exceeded the criterion for benzo(a)pyrene.

According to Richelieu Hydrogéologie, heightened levels of iron and manganese are among the main issues with Québec groundwater as they are usually caused by natural minerals present in soils. It was further concluded that natural minerals are the likely cause for the elevated levels of mercury, lead, and selenium found in the samples. Elevated levels of chlorine found in groundwater samples could be caused by the salt application along Rideau Boulevard (Highway 117). Calcium and sulphate levels were found to be higher in wells proximal to the old TSF. Organic compounds are likely to have anthropogenic origin; however, no such source was found during the 2012 surveys.

20.1.1.2.4 Soil Quality Baseline

A soil quality characterization study was carried out by AECOM in 2013. The following parameters were analyzed: arsenic, aluminum, beryllium, cadmium, cobalt, copper, mercury, molybdenum, lead, nickel, zinc and total cyanides. Results from 33 stations (TF-01 to TF-33) were compared against the Ministère de l'Environnement et de la Lutte contre les changements climatiques (MELCC) criteria for the Soil Protection and Rehabilitation Policy and the government of Québec's regulations respecting the burial of contaminated soils (Règlement sur l'Enfouissement des Sols Contaminés) characterize the soils within the range of criteria A to C (low to high) of the MELCC and the concentration limit of Appendix I of the Règlement sur l'Enfouissement des Sols Contaminés.

The A to B criteria (low to medium concentrations) of the MELCC policy was exceeded for concentrations of the following:

- arsenic at 2 stations
- cobalt at 10 stations
- copper at 22 stations
- nickel at 3 stations
- zinc at 2 stations

The B to C criteria range was exceeded for concentrations of molybdenum at 13 stations. Cadmium, mercury, lead and total cyanides did not exceed MELCC criteria at any sampling station. The stations with the most concentration of elements exceeding the applicable criteria were TF-01, TF-20 and TF-22. No explanation was given for the high concentrations found at these sampling stations.

20.1.1.3 Socio-economic Setting

The Wasamac property lies within the administrative region of Abitibi-Témiscamingue and within the municipality of Rouyn-Noranda. In 2013, AECOM carried out a study on the socio-economic setting of the Wasamac property for Richmont Mines (former owners).

20.1.1.3.1 Employment and Economy

In 2019, the Abitibi-Témiscamingue region counted 147,552 residents according to the Ministry of Municipal Affairs and Housing. In the 2011 Statistics Canada report, it was predicted that a population decrease of 2.7% could be expected in the region between 2006 and 2031, whereas an increase of 15.8% was expected for the province of Québec. The 2011 report also provided the age breakdown comparison between the city of Rouyn-Noranda, the Abitibi-Témiscamingue region, and the province of Québec.

Since the area was first developed, the regional economy of Abitibi-Témiscamingue has always has been largely based on the exploitation of natural resources. In 2006, a higher percentage of workers in the region (13.8%) were employed in the primary sector (agriculture and other resource-related industries) relative to the rest of the province (3.7%). In the 2020 Detailed Project Description provided by WSP, the current primary sector share of workers is now at 14.9% in the Abitibi-Témiscamingue region and at 2.2% in the province of Québec. The rate of unemployment in Abitibi-Témiscamingue was marginally less than that of the provincial rate in 2011. In 2017, the employment rate in the region was recorded as being the third highest in Québec.

A 2018 Statistics Canada survey revealed that residents of the Abitibi-Témiscamingue region regard their personal health conditions to be within the range of very good to excellent and their satisfaction with their level of quality of life to be strong. Despite this, life expectancy in the Abitibi-Témiscamingue region is lower than the rest of Québec. This is correlated with health-related behaviours that are common in the region including smoking tobacco, excessive alcohol consumption, nutritional deficiencies, and an overall sedentary lifestyle. Access to health services within the region is also limited relative to the rest of the province. It was reported that between 2010 and 2011, approximately one-quarter of the regional population did not have access to a family physician. Regardless of the shortage of physicians, according to Statistics Canada, people with the most severe health conditions were more likely to have a general physician.

20.1.1.3.2 City of Rouyn-Noranda

The city of Rouyn-Noranda is considered to be the most important city within the region of Abitibi-Témiscamingue. Rouyn-Noranda also served as a Regional County Municipality comprised of 16 municipalities from 1981 until its dissolution in 2002 following the amalgamation of its component municipalities. Currently, Rouyn-Noranda spans an area of 6,010.5 km^2. According to a survey from the Ministry of Municipal Affairs and Housing, the city of Rouyn-Noranda counted 42,889 residents in 2019. Economically, the city of Rouyn-Noranda is similar to the Abitibi-Témiscamingue region. In 2011, the unemployment rate in Rouyn-Noranda was only 1% greater than that of the province of Québec. In Rouyn-Noranda, approximately 13.1% of the labour force is employed in the primary sector. Rouyn-Noranda has had an upsurge in mining activity since 2010 due to the increase in the value of gold.

20.1.1.3.3 Transport Infrastructure

Highway 117 is a section of the Trans-Canada Highway that crosses between Québec and Ontario. According to the Ministry of Transports Québec, the daily average flow of traffic is 3,500 vehicles per day in the summer, and 2,900 in the winter (approximately 3,200 vehicles per day, annually). Highways 101 (a national highway to the west) and 391 (a collector highway to the east) also border the project site. This daily average flow shows a slight increase from the 2010 Ministry of Transports Québec data, which showed an annual average of 3,100 vehicles per day. Rouyn-Noranda is serviced by two rail companies: Ontario Northland Railway and Canadian National Railway. The Ontario Northland Railway line runs along the north side of Highway 117. The Ontario National Railway and Canadian National Railway lines are both under federal jurisdiction.

20.1.1.3.4 Past, Present and Future Land Uses

The Wasamac property is situated in a rural environment with isolated residential, recreational, and agricultural structures. No protected areas either at the federal level or the provincial level under the *Natural Heritage Conservation Act* are located within 5 km of the Wasamac property. Agricultural activity is concentrated along Highway 117 and Rang des Cavaliers. Residences in the area are primarily located alongside Highway 117, Rang des Cavaliers, and alongside Lake Hélène. In the 2013 AECOM field campaigns, surveys showed that residences and cottages within the area were served by individual wells for drinking water and private septic structures for wastewater.

Recreational structures are present within the study area. A recreational park was developed south of Lake Hélène by the residents of the local neighbourhood; however, this area is not an officially designated municipal park. There is also a hunting camp to the south of the Wasamac property. Although no leases have been granted for cottages, rough shelters, hunting or trapping camps in the area, this does not prevent local residents from developing structures on the land.

Although there are many fishing enthusiasts in the area, it is not a popular spot for sport fishing. Fishing in the region is concentrated around lakes Adéline and Hélène, although fishing at other regional lakes (Wasa and Chat Sauvage) has been reported but not observed. Ice fishing is also common practice on these two lakes. In 2007, more than 32,000 people were documented practicing hunting and trapping in Abitibi-Témiscamingue, making up 7% of all the hunters in Québec (MRNF, 2007). Evidence of deer, moose, black bear, and migratory bird hunting has been documented in the area.

The use of alternative vehicles such as boats, snowmobiles, and all-terrain vehicles for either transport or recreation is very popular. The 2013 AECOM field surveys reported observing many docks near residences and cottages as well as pleasure crafts including canoes, kayaks, motorboats, rowboats, pedal boats, and pontoons. The Snowmobile Club Network has a network of trails spanning over more than 300 km that link the regional centres with northeastern Ontario. An all-terrain vehicle route also runs the length of Highway 117 within the 2013 study area. Another important recreational network in the region is the provincial cycling network, the “Route Verte,” which passes across the study area (2013) along Highway 117 to Ville-Marie.

A site of particular interest within the vicinity of the Wasamac property is the recreational-conservation zone Kekeko Hills, located approximately 10 km from downtown Rouyn-Noranda and to the south of the project site. Kekeko Hills spans approximately 32 km^2 and is untouched by anthropogenic development. A network of trails and pedestrian sidewalks has been in development since 1990. The site is maintained and developed by an organization known as Friends of Kekeko. Given its ecological value for recreation and tourism, the city of Rouyn-Noranda has begun its participatory process to turn the site into a regional park. Numerous steps have taken place to ensure its preservation and public access. No additional social or environmental enhancement or development plans have been identified within the study area.

The Ministry of Culture and Communications has identified 70 archaeological areas of interest within the region of Abitibi-Témiscamingue. To the west of the project site, three sectors of strong archaeological potential have been identified: Opasitica, Dasserat and Buies. The route supposedly taken by the Le Moyne Brothers and Chevalier de Troyes is also located near the project site. No buildings of significant heritage or archaeological sites of importance were identified within the vicinity of the Wasamac property. Additionally, no archaeological site assessments have taken place on the Wasamac property.

20.1.2 Environmental Disclosure

There are no known environmental issues that would materially impact Yamana's ability to extract mineral resources or mineral reserves.

20.2 Waste Management and Water Management

This section presents the project mine waste management and water management approaches, including geochemical characterization of tailings, waste rock and ore.

20.2.1 Geochemical Characterization

20.2.1.1 Previous Testing

There are three known sources of project-related geochemical data:

- an EcoMetrix report, which contains the transcribed and compiled analyses from 2018 (EcoMetrix, 2020)
- an SGS report, which contains only limited useable data to inform the environmental geochemistry (SGS, 2012)
- a base metal labs report from 2018, which contains limited applicable data useable for environmental geochemistry (BaseMet Labs, 2018)

The data are summarized in Table 20-1.

Table 20-1: Material Testing Summary

Material Type	Static Data (No. Tests*)	Whole Rock Analysis / Oxide / Mineralogy (No. Tests)	Metal Mobility (No. Tests)	Kinetic Testing (No. Tests)
Ore	7	4*/4/0	7	0
Tailings	89	14/26/4	2	0
Waste rock	7	0/7/0	7	0

Note: *not including redundancy. Does not include economic metal assays.

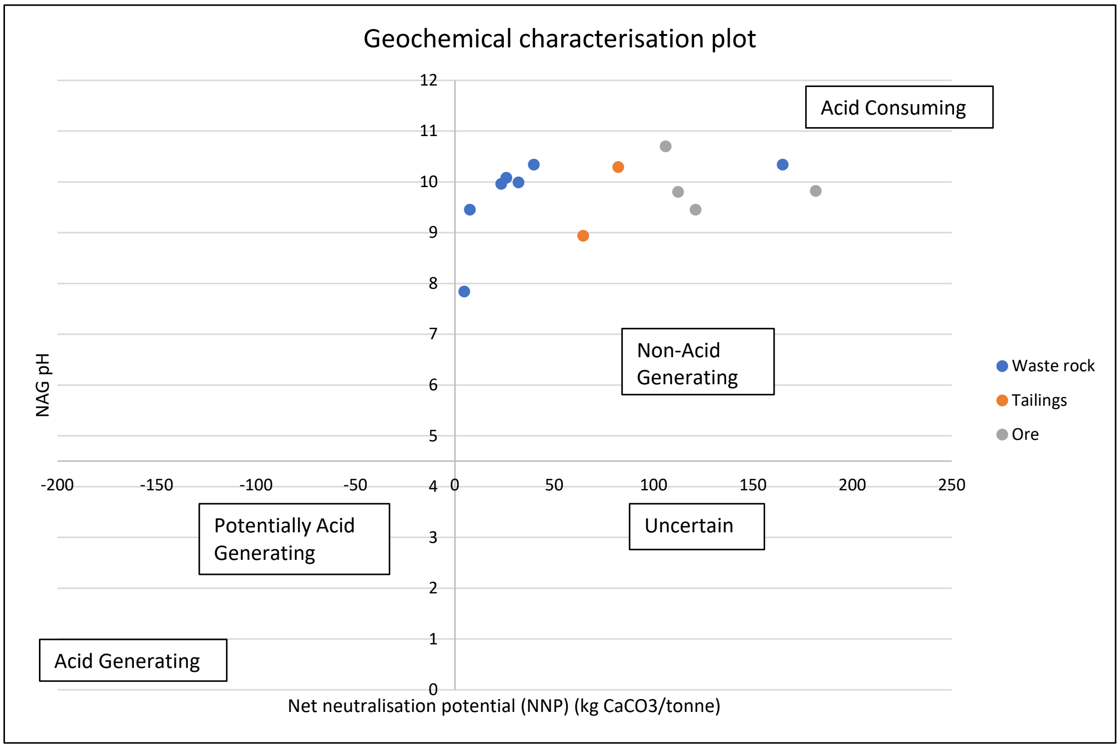
In summary, tailings have been tested extensively, while ore and waste rock have not.

The geochemical characterization (AMIRA 2002) of samples that have had net acid generation testing completed on them is shown in Figure 20-1. As evidenced on the figure, all the samples (ore, waste rock and tailings) tested in the non-acid-generating (and even in the acid-consuming) territory.

20.2.1.2 Tailings Characterization

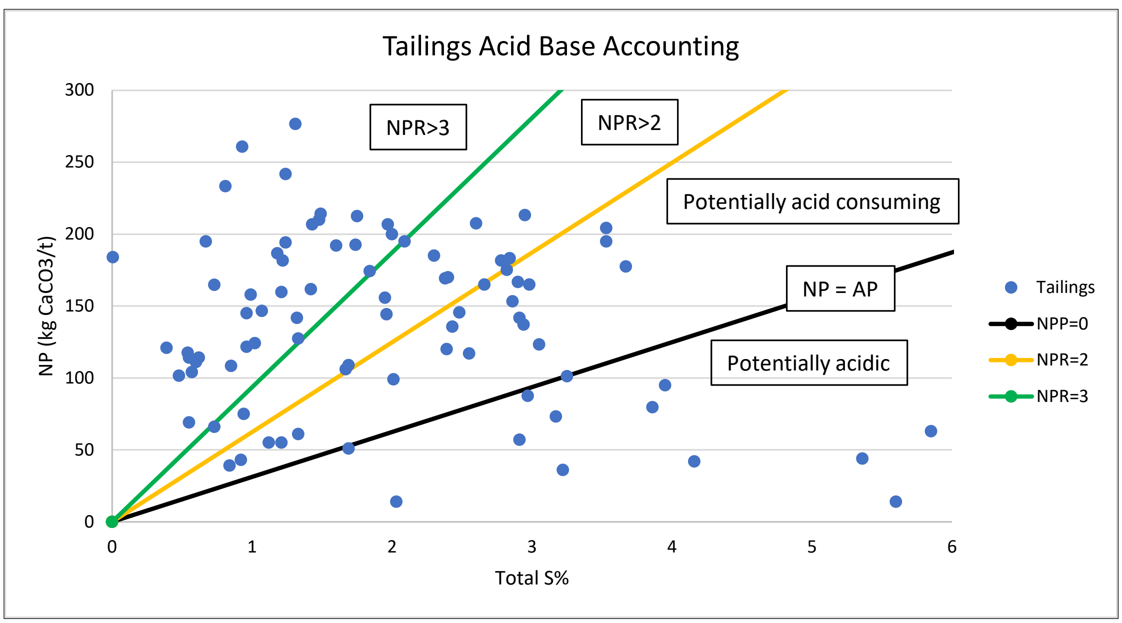
The tailings acid base accounting (ABA) data were compiled and plotted, as shown on Figure 20-2. As evidence on the figure, the tailings samples are broadly distributed, falling into all categories of acidification risk.

Figure 20-1: Geochemical Characterization of All Materials



Source: Adapted from AMIRA, 2002

Figure 20-2: Tailings Acid Base Accounting



Source: Hemmera, 2021

The tailings ABA statistics are presented in Table 20-2. When interpreting tailings ABA characteristics, there is a higher degree of uniformity (homogeneity) with tailings than with other waste, which means that often the median statistic is a reasonable predictor (e.g., neutralization potential ratio based on median acid potential and median neutralization potential). Environmental considerations should be acknowledged for the purposes of understanding acidification risks. The TSF will be a dry-stack facility, hence, there will be significantly less fluid present. The presence of fluid is an important component that helps in producing a homogenous response since it allows aqueous alkalinity to disperse within the interstitial pore space while simultaneously reducing oxygen access. De-saturated, or subaerial deposition may promote some adverse geochemical response driven by oxygen exposure. There is limited data available regarding dry stack facilities, so more conservative statistics are used to estimate the potential for acidification risks of the tailings in the dry-stack facility.

Reactivity and geochemical progression of materials is opportunistic. If all the materials react slowly in a uniform environment, the materials at the highest risk (i.e., materials with highest sulphide content) will eventually react faster, consuming the NP faster, and generally progress to a more aggressive state than the other material with lower sulphide content.

Table 20-2: Tailings ABA Statistics

Statistic	Sulphur (%)	Sulphide (%)	AP (kg $CaCO_3$/t)	Carb-NP (kg $CaCO_3$/t)	NPR	NNP
Count	90	90	90	90		
Median	1.8	1.6	56	143	2.6	80.8
Percentile 95	4.82	4.81	150.1	195.5	0.9	41.5

Note: Carb-NP = carbonate neutralization potential.

For the purposes of predicting tailings reactivity, the median and 95th percentiles of acidity potential (AP) are calculated and compared to the median neutralization potential (as Carb-NP). The method of using incongruous statistics in the ratio is to account for the opportunistic reactivity mentioned earlier. In this case, the median (bulk) response is likely to be relatively benign (Median NPR = 2.6), but it is possible that some areas, if exposed, may consume all the local NP and may generate some acid (95th Percentile NPR = 0.9).

The process of sulphide oxidation will liberate metals and salts, which could cause environmentally adverse effects even if the tailings do not generate acid.

20.2.1.2.1 Modified Shake Flask Testing

ABA testing was completed on the material which was to undergo the modified shake flask extraction testing (MSFE) testing. The results of the ABA work can be seen in Table 20-3.

The purpose of the ABA work was to confirm that sulphide was present prior to committing to the modified shake flask test. The standard shake flask test procedures were modified to include hydrogen peroxide to oxidize sulphide in the reaction vessel. The presence of sulphide is evidence that hydrogen peroxide addition would induce sulphide oxidation in the modified shake flask test.

Table 20-3: ABA Testing on Three Tailings Samples

Sample ID	Paste pH	TIC %	$CaCO_3$ NP	S(T) %	$S(SO_4)$ %	S (Sulphide) %	AP Kg $CaCO_3$/t	Modified NP	Net Modified NP	Fizz Test
Method Code	Sobek	CSB02V	Calc.	CSA06V	CSA07C1	Calc.	Calc.	Modified	Calc.	Sobek
LOD	0.20	0.01	#N/A	0.005	0.01	#N/A	#N/A	0.5	#N/A	#N/A
BL 729-Z1	8.01	2.00	166.7	2.408	0.01	2.40	74.94	149.9	75.0	Slight
BL 729-Z2	7.94	1.97	164.2	2.494	0.02	2.47	77.31	146.5	69.2	Slight
BL 729-ZD	8.25	1.78	148.3	1.443	<0.01	1.44	45.09	136.4	91.3	Slight

20.2.1.2.2 Interpretation of Modified SFE Results

The average sulphide content of sulphide is 2.1% (60 kg $CaCO_3$/t), but the average neutralization potential is 160 kg $CaCO_3$/t. The presence of sulphide indicates that we should expect reactivity in the modified SFE test; and the presence of neutralization potential indicates that we should expect some calcium release, as the carbonate component is consumed by acidity.

The result of this is that the counterions involved in the reaction (Ca^{2+} and SO_4^{2-}) tend to be present at molar equivalence. Note that the reactions presented are only for neutralization potential in the form of calcite, but are applicable as well for dolomite (Ca, Mg)$(CO_3)_2$. In the case that dolomite is the acting neutralization agent, we could expect to observe a SO_4: Ca ratio that favors SO_4.

The results of the molar ratio assessment for Ca and SO_4 for the modified SFE tests is shown in Table 20-4.

Table 20-4: Calcium, Sulphate molar ratios

Sample	SO_4 (mg/L)	SO_4 (mol/L)	Ca (mg/L)	Ca (mol/L)	SO_4/Ca (Molar Ratio)
BL 729-Z1	293	3.1	94.2	2.4	1.3
BL 729-Z2	541	5.6	198	4.9	1.1
BL 729-ZD	47	0.49	24.3	0.61	0.81

20.2.2 Interpretation of Results

The following observations apply to the molar ratio discussion:

- BL 729-Z1 – Molar ratio of 1.3 suggest dolomite may be present in moderate quantities
- BL 729-Z2 – Molar ratio of 1.1 suggest the dominant neutralization agent is calcite
- BL 729-ZD – weakly reacting, and low molar ratio of 0.81 suggest that a substantial portion of the calcium in solution is a result of dissolution, not acid consumption (this is confirmed by observation of the alkalinity in the leachate for this sample)

The most concerning of these is the potential observation that BL 729-Z1 may contain elevated dolomite, and that that mineral may be reactive as a neutralization agent.

Dolomite contains magnesium which has elevated solubility, and with a very low molecular weight, is often underestimated in terms of its impact on solubility of other metals and salts (through electrical effects in solution).

On review of the mineralogical data (4 samples tested in 2018), there is confirmation that there is dolomite present (Table 20-5), note that the mineralogical analyses presented below are not the same samples as those discussed in this section.

Table 20-5: Mineralogical Composition

Mineral/Compound	Chemical Formulae	Z1-MC-T13 (BL348-13) (wt. %)	Z2-MC-T21 (BL248-21) (wt. %)	Z3-MC-T29 (BL348-29) (wt. %)	Z3-MC-T5 (BL348-5) (wt. %)
Quartz	SiO_2	18.9	20.7	24	16.8
Albite	$NaAlSi_3O_8$	20.2	27	45.4	29.2
Anorthite	$CaAl_2Si_2O_8$	2.9	0.3	0.3	0.5
Microcline	$KAlSi_3O_8$	4.5	13.6	1.8	2.4
Chlorite	$(Fe,(Mg,Mn)_5,Al)(Si_3Al)O_{10}(OH)_8$	10.1	2.2	1.9	12.7
Muscovite	$KAl_2(AlSi_3O_{10})(OH)_2$	20.2	14.8	11.2	15.4
Biotite	$K(Mg,Fe)_3(AlSi_3O_{10})(OH)_2$	3.1	2.9	1.9	2.7
Calcite	$CaCO_3$	8.5	4	1.9	9.4
Dolomite	$CaMg(CO_3)_2$	7.7	10.9	9.2	7.2
Pyrite	FeS_2	1.6	1.5	0.7	0.9
Magnetite	Fe_3O_4	0.9	0.8	0.7	1.8
Hematite	Fe_2O_3	1.1	0.6	0.7	0.6
Rutile	TiO_2	0.3	0.6	0.2	0.3

The mineralogical testing confirms the interpretation of the SO_4: Ca molar ratios, and also shows that dolomite is present as the dominant neutralization agent in two of the samples.

20.2.3 General Observations

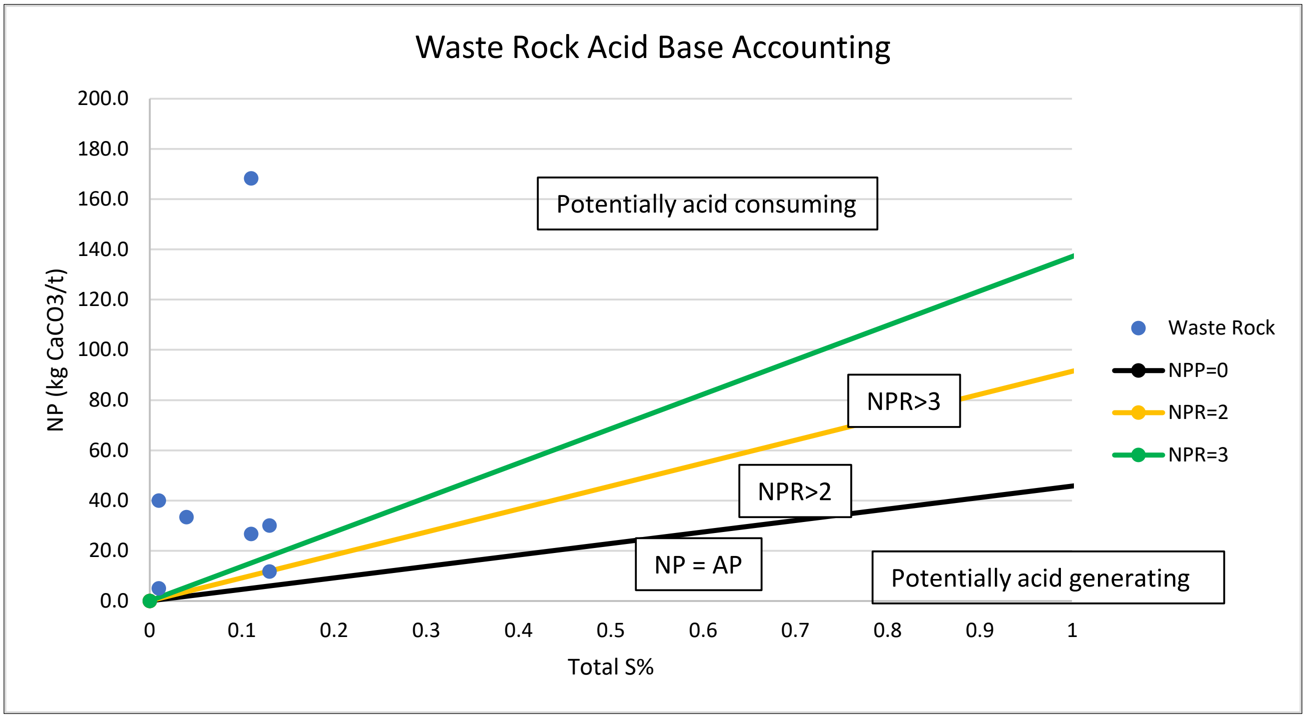
The following general observations are made:

- Results from the modified SFE tests suggest sulphide oxidation will occur but acid build-up (in excess of neutralization capacity) is unlikely to result.
- Dolomite is present in the tailings.
- The results from the modified SFE testing will be used to directly inform the first flush water quality of intercepted water at the tailings facility in the water quality model.
- Tailings kinetic testing is currently underway and will inform subsequent studies.

20.2.3.1 Waste Rock ABA Characterization

The results of waste rock ABA testing are shown in Figure 20-3. The waste rock has not been sufficiently sampled and analyzed to generate an appropriate amount of statistical confidence about the geochemical behaviour of the material.

Figure 20-3: Waste Rock Acid Base Accounting



Source: Hemmera, 2021

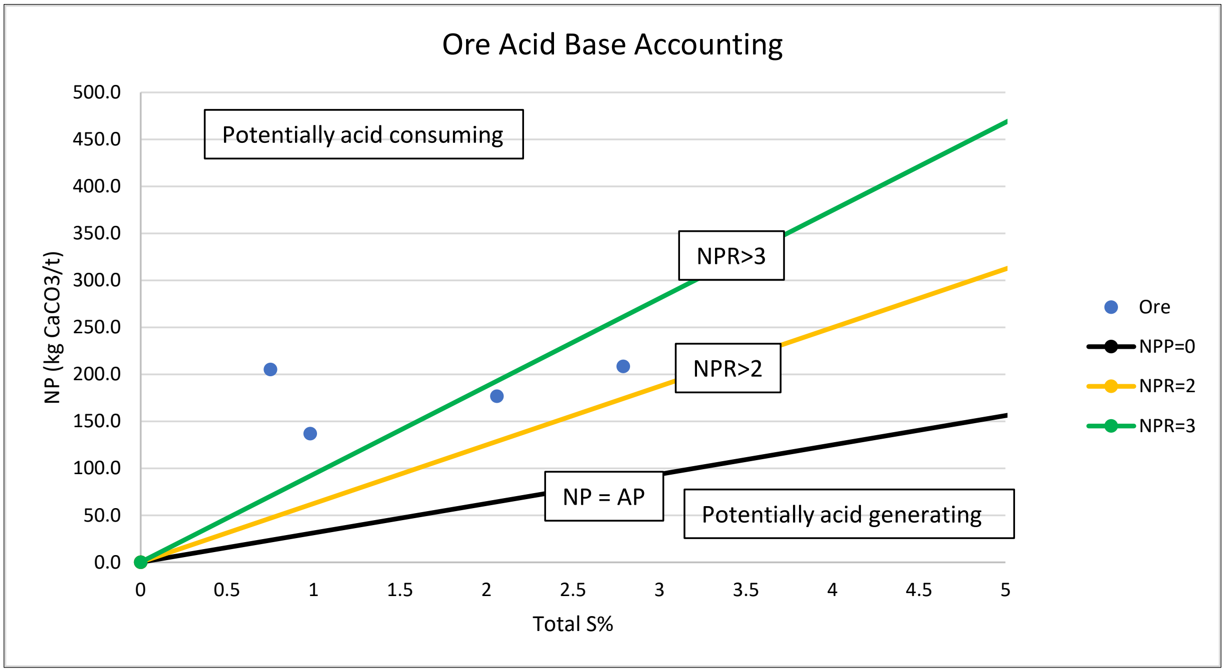
While all waste rock samples plot in the potentially acid consuming (not acid generating) area. Waste rock kinetic testing is underway which will inform future studies including water quality impacts. A more aggressive static program is recommended to generate appropriate data to characterize waste rock.

20.2.3.2 Ore ABA Characterization

Ore samples were submitted for static testing to assess the risk posed by lower-grade material during stockpiling for future processing. In this section, “ore” refers to both high-grade (economically advantageous) and low- and medium-grade (economically disadvantageous) ore material. The acid base accounting for the ore is presented in Figure 20-4.

No sampling or testing has been performed to date on weathered ore or low- and medium-grade ore. There is insufficient data to draw statistically relevant conclusions about the ore. Based on the samples analyzed to date, the ore may be reactive (elevated sulphide), but also contain adequate neutralization potential to avoid acidification. No kinetic testing is underway for ore of any type (low-, medium- or high-grade ore). The current mine plan includes milling/processing of low- and medium-grade ore to avoid storage of this material on site; hence, it is likely that these materials will not be exposed long enough to generate acid.

Figure 20-4: Ore Acid Base Accounting



Source: Hemmera, 2021

20.2.4 Waste Rock and Tailings Management

Waste rock and tailings management will follow a general process of testing and analysis, assess, inform, and direct. The purpose of these processes is to generally assess the material, then use that information (in conjunction with industry best practices) to determine the appropriate handling and management techniques.

Ausenco has identified that information gaps persist, and data generating procedures have been actioned, such as:

- static testing for waste rock and low-grade ore
- kinetic testing for waste rock and low-grade ore
- kinetic testing for tailings

The purpose of the kinetic testing is to determine how quickly the materials in question will advance to an acidic state, if at all. Materials that are predicted to generate acid will be targeted for more conservative handling procedures, such as more rapid processing, less time in exposed environments (weathering), and ideally, ultimately be disposed of in subaqueous environments.

Ausenco assumes, on a preliminary basis, that none of the mine waste streams will generate acid, however additional kinetic testing is underway to confirm the preliminary assumption. It is also possible that other environmental impacts may be realized as a result of waste weathering which are less common—such as neutral mine drainage. Testing is underway which will inform these determinations.

Ausenco geotechnical engineers and geochemists will work together to identify management practices that satisfy both short- and long-term geochemical stability requirements.

The results of the previous geochemical characterization (Section 20.2.3.1and Figure 20-3) demonstrates the bulk of the waste rock material is NAG. Therefore, it should not impact the surface water. During operations and post closure, monitoring and testing of waste rock generated by the project along with collecting and testing water samples from the mill basin pond will be conducted on quarterly basis. If the monitoring and testing program indicates poorer water quality than originally indicated, Yamana will investigate the best practices for managing of waste rock and surface water in accordance with national and provincial regulations.

The results of the previous geochemical characterization (Section 20.2.3.2and Figure 20-4) demonstrates the tailings are both NAG and indeterminate. Therefore, the TSF pond is lined, and water is sent to the treatment plant prior to release to the environment. During operations and post closure, monitoring and testing of tailings generated by the project along with collecting and testing water samples from the TSF pond will be conducted on quarterly basis. If the monitoring and testing program indicates poorer water quality than originally indicated, Yamana will investigate the best practices for managing of tailings and surface water in accordance with national and provincial regulations.

20.3 Permitting

The following subsections summarize the federal and provincial environmental regulatory approvals and permitting regulations that apply to the construction and operation of the project, as currently proposed. Yamana relies on a collaborative approach to ensure the success of Wasamac. In this regard, its environmental assessment process is conducted in collaboration with neighbors and First Nations. A community relations office will soon be opening its doors to ensure constant dialogue and accessibility to the Yamana team as well as to information on the project. A campaign of baseline monitoring and testing is currently underway with the objective of completing the EIA submissions at the provincial and federal levels by the second quarter of 2022.

20.3.1 Federal Environmental Approvals

20.3.1.1 Federal Impact Assessment

The Physical Activities Regulations (also known as the Project List) identifies types of projects that may require a federal impact assessment (IA) under the *Impact Assessment Act, 2019* (IAA). The ore production capacity will be above 5,000 t/d, estimated at 7,500 t/d. When the physical activity associated with the carrying out of a proponent's project is described in the Physical Activities Regulations, the proponent must provide the Agency with an Initial Project Description. The project entered the planning phase with the submission of an Initial Project Description (IPD) on November 16, 2020. Subsequently, the Agency published the Notice of Impact Assessment Decision on November 26, 2020, which confirms that the project is subject to the Federal IA process as per section 16(1) of the Act; the Public Participation Plan, the Cooperation Plan, the Indigenous Engagement and Partnership Plan, and the Permitting Plan were released by the Agency on March 26, 2021. The planning phase ended with the publication of the Tailored Impact Statement Guidelines (TISG) document, which covers the content required for the Environmental Impact Statement (EIS). The TISG were published on March 26, 2021 and essentially represent terms of reference for the EIS.

Yamana is currently producing the EIS following the collection of the necessary studies and assessment of impacts. Once the EIS is submitted, the Agency will assess whether it is administratively complete (includes all the elements included in the TISG). Upon a determination of administrative completeness, the Agency will begin the IA phase which involves detailed review of the EIS.

20.3.1.2 Federal Permitting Requirements

In addition to the requirement for assessment under the IAA, the permitting plan for the project (published by the Agency on March 26, 2021) includes the key federal regulatory instruments or permits that may be required (final determination is made following the EIA process):

- Authorization under paragraphs 34.4(2)(b) and 35(2)(b) of the *Fisheries Act*: An authorization under paragraphs 34.4(2)(b) and 35(2)(b) of the *Fisheries Act* may be required for proposed works, undertakings or activities that could result in the death of fish or harmful alteration, disruption or destruction of fish habitat.

- Authorization to use waters frequented by fish as a tailings impoundment area: Under subsection 5(1) of the Metal and Diamond Mining Effluent Regulations, an authorization may be required for any project where the proposed disposal of mine waste would affect waters frequented by fish.

- Approval of works under the *Canadian Navigable Waters Act*: An approval under the *Canadian Navigable Waters Act* (CNWA) may be required. An approval is required for any major work on navigable waters, whether listed or not in the Schedule to the CNWA (paragraph 5[1][a]).

- Permit under subsection 73(1) of the *Species at Risk Act*: A permit under the *Species at Risk Act* (SARA) may be required if the project can affect wildlife species at risk listed in Schedule 1 of SARA, as an endangered, threatened or extirpated species or on any element of their critical habitat or the residence of their individuals in a manner that is prohibited under subsection 32(1), section 33, subsection 58(1) and section 61 of SARA.

- Licences for explosive factories and magazines under subsection 7(1) of the *Explosives Act*: Facilities for storing explosives are subject to the requirements of the *Explosives Act* and its regulations.

20.3.2 Provincial Environmental Approvals

20.3.2.1 Provincial Environmental Assessment

The proposed project is described in Section 2, Paragraph 22 of Part II of Schedule I of the Regulation respecting the EIA and review of certain projects (c. Q-2, r. 23.1):

> “(2) the establishment of a mine whose maximum daily capacity for extracting any other metal ore is equal to or greater than 2,000 metric tons.”

The projects listed is subject to the EIA and review procedure (EIARP) provided for in Subdivision 4 of Division II of Chapter IV of title I of the *Environmental Quality Act* (EQA and must obtain an authorization from the government.

Yamana is currently in this phase of preparation of the EIS for the Wasamac Project. It is estimated that the EIS will be filed in Q1-Q2 of 2022.

Following the environmental assessment procedure, Yamana will proceed to the authorization requests for the construction and the exploitation of the project with provincial and municipal authorities.

In addition to this, the Wasamac Project was selected as a pilot project by the government, under the authority of an interministerial table composed of the five following: Ministry of Energy and Natural Resources, Ministry of Forests, Wildlife and Parks, Ministry of the Environment and the Fight against Climate Change, Ministry of Municipal Affairs and Housing, and the Ministry of Economy and Innovation. The primary objective of this initiative is to establish a viable interaction system with stakeholders and, in particular, to promote the social acceptability of mining projects. The first meeting took place on December 19, 2019. Sporadic follow-ups are made with the Ministère de l'Énergie et des ressources naturelles (MERN), leader of the table.

20.3.2.2 Provincial Permitting Requirements

A variety of other provincial and municipal permits will also be required depending on the final design of the mine project components. These permits will be requested by Yamana following the EIA phase. Table 20-6 presents the applicable permits as described in the Detailed Project Description (translated).

20.3.3 Federal – Provincial Harmonization

The Cooperation Plan provided by IAA on March 26, 2021 sets out how the Agency will cooperate with the Ministère de l'Environnement et de la Lutte contre les changements climatiques (MELCC) for the Wasamac Project. The provincial and federal environmental impact assessment processes will be harmonized.

Table 20-6: Provincial Permits

Type	Authority	Document to be Filed	Regulatory Reference	Trigger Related to the Project
Government authorization	Ministère de l'Environnement et de la Lutte contre le Changement Climatique (MELCC)	Environmental impact statement prepared in accordance with specific guidelines issued by the MELCC	*Environmental Quality Act* (EQA), s. 31.1 Regulations for the assessment and review of the environmental impacts of certain projects	Establishment of a metalliferous mine with a daily capacity average extraction rate of 2,000 t/d or more.
Specific authorization to erect or modify a structure, undertake the operation of an industry, carry out an activity or use an industrial process that could affect the quality of the environment	MELCC Regional Management	Application for authorization	EQA, s. 22 EQA, r. 3 Directive 019 on the mining industry	The operation of a mine and the use of an industrial process (ore processing plant) are industrial activities that can modify the quality of the environment.
Authorization to establish a water supply intake	MELCC Regional Management	Application for authorization	EQA, s.32 Directive 019 on the mining industry	The project requires the establishment of a water supply intake.
Specific authorization to erect or alter a structure, undertake the operation of an industry, carry on an activity or use an industrial process that may affect a watercourse, lake or wetland	MELCC Regional Management	Application for authorization Compensation plan for the impairment of target environments	EQA, s. 22 EQA, r.9.1 EQA, r. 35	Project activities, infrastructure and facilities will affect wetlands and waterbodies.
Authorization for devices or equipment intended to prevent, reduce or stop the release of contaminants into the atmosphere	MELCC Regional Management	Application for authorization	EQA, s. 22	The project will involve the use of devices and equipment to prevent, reduce or stop the release of contaminants into the atmosphere (e.g., dust collectors). The instrumentation and process detail plans will be defined in a later stage.
Industrial sanitation certificate	MELCC Regional Management	Application for Certification	EQA, s. 31.28 Certification Regulations of sanitation in an industrial environment	Threshold: annual ore extraction capacity exceeding 2,000,000 tonnes per year or annual ore or tailings processing capacity exceeding 50,000 tonnes per year.
Statement	Statement		Regulations for the mandatory reporting of certain airborne contaminant emissions.	Any operator that emits to the atmosphere a contaminant listed in Part I of Schedule A in a quantity that meets or exceeds the reporting threshold listed in that Schedule for that contaminant or class of contaminants.
Authorization or permit for any activity involving the withdrawal of groundwater or surface water (dewatering, keeping dry, water supply, etc.)	MELCC Regional Management	Application for authorization	EQA, s. 31.75 EQA, r.35.2 Water Withdrawal and Protection Regulation	Threshold: 75,000 L per day (75 m3/d)
Authorization to carry out an activity likely to modify wildlife habitat	Ministry of Forests, Wildlife and Parks	Application for authorization	*Wildlife Conservation and Enhancement Act*, s. 128.7 Wildlife Habitat Regulations	The presence of wildlife habitat as defined in the Regulations in the project area was confirmed (fish habitat).
Intervention permit for the cutting of wood for the purpose of carrying out certain mining activities	Ministry of Forests, Wildlife and Parks	Application for a permit	*Sustainable Forest Development Act*, s. 73 Regulation on the sustainable management of forests in the domain of the State	The project requires deforestation.

Type	Authority	Document to be Filed	Regulatory Reference	Trigger Related to the Project
Authorization to construct or improve a multipurpose road	Ministry of Forests, Wildlife and Parks	Application for a permit	Loi sur l'aménagement durable en territoire forestier, s. 41 Règlement sur l'aménagement durable du domaine forestier	The project requires the construction or improvement of a multi-use road.
Permits for construction and site development	City of Rouyn-Noranda	Application for a permit	Regulation No. 2015-847, s. 46 of c. 4	The project requires the construction of buildings and infrastructure.
Mining lease	Ministry of Energy and Natural Resources	Lease application	Mining Act, s. 100 Mineral Substances (Other than Petroleum, Natural Gas and Brine) Regulations, s. 38	Any person who mines mineral substances, except for surface mineral substances, petroleum, natural gas and brine, shall have previously entered into a mining lease with the Minister.
Approval of the tailings site (waste rock and tailings facility) and the mill site	MERN	Application for approval	SI, ss. 240 and 241 Mineral Substances (Other than Petroleum, Natural Gas and Brine) Regulations, s. 124	The project includes the development of tailings storage areas and a mill.
Redevelopment and Restoration Plan Approval	MERN	Application for approval	SI, ss. 232.1 and 232.2 Mineral Substances Regulation other than oil, natural gas and brine, s.109	Must submit a redevelopment and restoration plan to the Minister's approval, projects involving: - any activity related to the extraction of ore or tailings carried out in open air or underground, - processing of ore or tailings; - the development of accumulation areas
Authorization to use public land	MERN	Request for authorization	Law on the Lands of the State Domain, art. 47 Regulation respecting the sale, lease and grant of interests in real property on Crown lands, s. 35	A private use lease is required for areas where surface infrastructures will be built. A specific lease is required for the establishment of a park for to receive the mine tailings.
Explosives permit, including a general permit, permit of deposit and transport permit	Sûreté du Québec	Application for a permit	*Explosives Act*, s. 2-6	The project requires the installation of a powder magazine, the use and transportation of
Authorization for road construction if less than 60 m from a watercourse if more than 300 m long	MELCC	Application for authorization	EQA, s. 22	At this time, the project does not include the construction of roads to within 60 m of a watercourse over 300 m in length.

20.4 Engagement and Consultation

20.4.1 Quebec Public Participation Guidelines

Within the framework of the EIARP in Southern Québec, various mechanisms have been set up to promote public participation and the consideration of public concerns regarding projects likely to have impacts on the physical, biological and human environments. Public participation allows for better identification of project-related issues and ensures informed decision-making by government. It is possible to obtain information and express a point of view on a project during these phases of the EIARP:

- consultation on the issues that the impact statement should address
- public information period
- public hearing, mediation or targeted consultation

20.4.2 Quebec First Nations Engagement and Consultation

Other than what is described in the above section, which is also reiterated in the Ministerial Directive received from the Ministry in November 2019 for the project, Wasamac must give priority to the implementation of specific approaches with the First Nations communities concerned and, to the extent possible, mutually agreed upon with these communities.

In all cases, the proponent's approaches shall remain distinct from the consultations that the Ministry may conduct with First Nations as part of the EIARP.

It should be remembered that the obligation to consult and, if applicable, to accommodation of First Nations communities, which stems from the decisions of the Supreme Court of Canada, is incumbent upon the government of Québec. In this context, the steps taken by the proponent with First Nations communities will not relieve the government of its consultation obligations.

Guidelines for First Nations engagement regarding mining projects can be found in the following document: "Document d'information à l'intention des promoteurs et introduction générale aux relations avec les communautés autochtones dans le cadre de projets de mise en valeur des ressources naturelles", Gouvernement du Québec, 2015.

20.4.3 Federal Consultation and Engagement

The Public Participation Plan, developed by the Agency in March 2021, takes into account the context of the Covid-19 pandemic, to provide the best opportunities for public participation during the entire IA process for the project. This Plan may be adapted as the pandemic or the IA evolves. If this is the case, the Agency will inform the public and stakeholders.

The Agency has compiled a preliminary list that includes communities, associations and other stakeholders whose interests may warrant participation in this project's IA. The Agency has developed this list based on documents provided by the proponent, and on the recommendations of the Conseil régional en environnement de l'Abitibi-Témiscamingue and of the Department of Women and Gender Equality Canada. The Agency also included participants and stakeholders who had already submitted comments during the two consultations period of the Planning Phase.

In the Public Participation Plan, for each phase in the IA process, a table presents, the engagement activities led by the Agency and, to the extent possible, the activities planned by the proponent.

20.4.4 Federal Indigenous Engagement and Participation Plan

Several Indigenous groups have established or potential Aboriginal or Treaty rights in the project's study area. The Government of Canada has the obligation to consult and, if applicable, accommodate Indigenous peoples and communities when contemplating actions that may adversely affect established or potential Indigenous and treaty rights.

This Indigenous Engagement and Partnership Plan (Plan) that was prepared by the Agency for the Wasamac Project in March 2021 outlines opportunities and methods to conduct meaningful consultations between the Agency and potentially affected Indigenous peoples. Meaningful consultations will be conducted throughout the project's IA process, in the spirit of reconciliation toward a renewed nation-to-nation relationship and in accordance with the principles respecting the Government of Canada's relationship with Indigenous peoples.

While IA is not a rights-determination process, the Crown will consult with the Indigenous peoples listed below to understand concerns and potential adverse impacts of the project on their exercise of Aboriginal and treaty rights and, where appropriate, accommodate them. These consultations will also form an integral part of the work that will support the assessment of the project.

This list is subject to change as knowledge of the potential effects and impacts of the project is gained, or if the project or its components are modified, or as a result of any other information gathered during the IA.

20.4.5 Consultation and Engagement Activities Completed

As per the Detailed Project Description (WSP, 2020), various means of communication have been put in place to establish and maintain dialogue with the community and the various stakeholders, including the following:

- written communications to citizens (notification of upcoming activities and work)
- newsletter (published twice a year) distributed by mail to residents of the area and by email to stakeholders
- online discussion forum (tool to be redefined)
- dedicated community relations email address (administered daily by the Vice President of Operations and Community operations and community relations)
- individual meetings with the project's neighbours (mitigation measures and corrections to work done prior to the acquisition by Monarch)
- personalized letters and regular exchanges with the city of Rouyn-Noranda (urban planning department)
- four coffee meetings for the project's neighbours and the City's elected officials (January 22, 2018, October 24, 2018, October 3, 2019 and February 11, 2020)
- mailing of the coffee shop reports to project neighbours
- presentations on the project to various municipal, para-municipal and community organizations

With respect to First Nations, a first meeting with the management of the Abitibiwinni First Nation Council was held in October 2018 during which Monarch representatives presented the project and its progress.

The Abitibiwinni First Nation Council expressed its interest in being informed and involved in the next steps of the project. A second meeting was held on December 5, 2019. No concerns were raised during these meetings.

As a result of contacts made with these First Nations by mail during the spring and summer of 2020, three Indigenous groups (Timiskaming First Nation, Abitibiwinni First Nation Council and Wahgoshig First Nation) directly expressed their interest to Monarch to be involved in the development of the project due to the potential impacts on their traditional territory. Meetings are underway, or planned, with these groups to better identify their interests and expectations as to how to involve them in the preparation of the EIA submissions.

No agreements have been signed to date between Yamana and neighboring communities and/or First Nations. Yamana will continue consulting and engaging with First Nations and neighboring communities as part of the EIA processes.

20.5 Mine Closure

20.5.1 Closure Plan and Cost Estimate and Financial Assurance

Under the *Mining Act*, anyone who engages in mining exploration work or mining operations determined by regulation must submit a reclamation plan (subsequently referred to as "closure plan") for approval by the Ministère de l'Énergie et des ressources naturelles (MERN). Approval is conditional upon a favourable opinion from the Ministère de l'Environnement et de la Lutte contre les changements climatiques (MELCC).

Yamana is currently preparing a conceptual closure plan and cost estimate for the Wasamac Project; this conceptual plan and cost estimate will form part of the EIA. The conceptual closure plan and cost estimate will meet the requirements of the Guide and applicable legislation. The objective of the conceptual closure plan will be to return the site to a satisfactory condition by:

- eliminating unacceptable health hazards and ensuring public safety
- limiting the production and spread of contaminants that could damage the receiving environment and, in the long term, aiming to eliminate all forms of maintenance and monitoring
- returning the site to a condition in which it is visually acceptable (reclamation)
- returning the infrastructure areas (excluding the tailings impoundment and waste rock piles) to a state that is compatible with future use (rehabilitation).

The conceptual nature of the closure plan and cost estimate reflects the fact that the project has not yet been constructed and operated. Thus, the plan and cost estimate are based on all quantifiable information available at the time this report was prepared. During subsequent revisions, the cost estimate will become increasingly accurate. The cost estimate for the conceptual plan is discussed in Section 21.3.3.

Yamana will assess the cost for mine site closure work in current dollars for all areas of land affected at the end of mine life (including the cost of all studies), and the assessment will cover the mining facilities and accumulation areas.

21 CAPITAL AND OPERATING COSTS

Unless stated otherwise, all costs presented in this chapter are in Canadian Dollars (CAD, C$).

21.1 Capital Costs

The estimate conforms to Class 3 guidelines for a feasibility-level estimate with a ±15% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International).

Table 21-1 on the following page provides a summary of the estimate for overall initial capital cost. The estimate includes costs for mining, site preparation, process plant, tailings facility, power infrastructure, Owner's costs, spares, first fills, buildings, roadworks, and off-site infrastructure.

The estimate is based on an EPCM execution approach outlined in Chapter 24. The following parameters and qualifications were considered:

- For material sourced in US dollars (8.8% of initial capex), an exchange rate of 1.23 Canadian dollar per US dollar was assumed, as per the exchange spot rate at the time of bid solicitation.
- No allowance has been made for exchange rate fluctuations.
- There is no escalation added to the estimate.
- A growth allowance was included.
- Data for the estimates have been obtained from numerous sources, including:
    - mine schedules
    - feasibility-level engineering design
    - topographical information obtained from the site survey
    - geotechnical investigations
    - budgetary equipment quotes from Canadian and International suppliers
    - budgetary unit costs from several local contractors for civil, concrete, steel, electrical, piping and mechanical works
    - data from similar recently completed studies and projects

Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs, and Owner's costs) were identified and analysed. Percentage of contingency was allocated to each of these categories on a line-item basis based on the accuracy of the data. An overall contingency amount was derived in this fashion.

As outlined in Table 21-1, the initial capital cost of the project will be approximately C$533 million, with ongoing sustaining capital costs of C$432 million. Of the total capital costs, more than 84% of the project costs were derived from first principles bulk material take-offs and equipment sizing calculations, with supporting quotations for major equipment, and contractor supply/installation rates.

Table 21-1: Summary of Capital Costs (C$M)

WBS	Description	Initial Capital Costs (C$M)	Sustaining Capital Costs (C$M)
1100	Old Wasamac Dewatering & Rehabilitation	1.8	0
1200	Mine Portals	1.2	0
1300	Mine Development	81	150
1400	Underground Mobile Equipment	40	84.5
1500	Underground Infrastructure & Construction	5.3	0.1
1600	Underground Services	36	31
1700	Backfill Plant & Network	0	52
1800	Technical Services & Instrumentation	0.1	0
1900	Waste Rock Storage Facility	2.9	4.1
2800	Underground Plant & System Ore Receival & Crushing	24	0
3100	Crushed Ore Storage & Reclaim Tunnel	5.7	0
3200	Grinding	30	0
3300	Leaching	18	0
3400	Elution and Gold Room	10	0
3500	Tailings Disposal & Pipeline	9.4	0
3600	Reagents	4.2	0
3700	Tailings Filtration Plant	12	0
3800	Process & Tailings Air & Water	4.9	0
3900	Process Buildings	13	0
4100	Bulk Earthworks	7.5	0
4200	High-Voltage Power Switchyard & Power Distribution	12	0
4300	Communications	1.6	0.5
4400	Truck Shop & Fuel Storage	5.5	0
4500	Buildings	8	0.8
4600	Site Services	9.7	6.4
4700	Industrial Water Management	1	0
4800	Tailings Storage Facilities	15	48
5100	Main Access Road	0.7	0
5300	Mine Services Access Roads (to Vent Collars)	0.6	0
6100	Temporary Construction Facilities & Services	21	0
6200	Commissioning Representatives & Assistance	1.1	0
6300	Spares (Commissioning, Initial & Insurance)	0.7	0
6400	First Fills & Initial Charges	1.8	0
6500	Freight & Logistics	0.1	0
7100	Engineering & Construction Management Services	25	0
7200	Underground Mining & Engineering	14	0
8100	Project Management & Home Office	6.3	0
8200	Construction Labour	6.9	0
8300	Pre-Production Labour	4	0
8400	Pre-Production Programs	2.5	0
8500	Finance, Legal, Insurance	3.4	0
8600	Conceptual Closure Costs	0	22
8700	Pre-Production Mining Operating Cost	25	0
8800	Pre-Production Process Operating Cost	2	0
8900	Pre-Production Technology & Innovation	3.7	0
	Subtotal	477	401
9100	Project Contingency	56	31
	Total Project Costs	533	432

Source: Ausenco, 2021

21.2 Basis of Estimate

21.2.1 Exchange Rates

Vendors and contractors were requested to price in native currency. The estimate is prepared in the base currency of Canadian dollars (CAD, C$). Pricing has been converted to Canadian dollars using the exchange rates in Table 21-2. Note that the foreign exchange spot rate at the time of bid solicitation of 1.23 CAD:USD was used in the capital cost estimate. The operating estimates and financial model consider a longer-term forecast of the currency conversion rate of 1.28 CAD:USD. The US dollar contributions to the capital estimate are strictly 8.8% of initial capital costs (no US dollar costs in sustaining). Of that 8.8%, 88% is mechanical equipment, 11% is electrical equipment, and 1% is first fills and spares.

Table 21-2: Estimate Exchange Rates

Code	Currency	Exchange Rate
CAD	Canadian	1.00
AUD	Australian Dollar	0.93
EUR	Euro	1.47
USD	United States Dollar	1.23

Source: Ausenco, 2021

21.2.2 Area 1000 – Mining

InnovExplo provided estimates for all underground mine pre-production capital costs, except for material handling, automation, communications, electrical and maintenance infrastructure costs, which were provided by Ausenco. The total underground mine initial capital cost is estimated at C$168.0 million.

In the underground mine pre-production capital costs contractors will develop the two access ramps, all rapid-development, and the major infrastructures. From mid-2026, the mine development teams will be employed when sufficient working faces are available. These teams will be used in conjunction with the contractors' teams until the second trimester of 2028, after which only three mine jumbos will be required. Pre-production costs reflect all mobile equipment and fixed plant purchases along with associated capitalized costs for the period.

Quotations for the estimates were obtained from various suppliers and suitable quotations included in the cost model. Cost considerations include quotations from industry leading vendors such as Sandvik, MacLean, Howden, CMAC, DSI and Technosub. Pre-production capital costs include material and labour for each category.

21.2.3 Area 2000 – Underground Plant and Systems

The estimate allows for the underground plant outside of the mining scope. This includes the following items:

- rockbreakers
- grizzly feeder and associated comms and instrumentation

- primary crushing unit
- conveyors, including sacrificial conveyor
- all associated platework, mechanical equipment, electrical equipment (including motor control centres or "MCCs")

All major equipment was sized based on the process design criteria outlined in Chapter 17. Once the mechanical equipment list was outlined, the mechanical scopes of work were derived and sent for budgetary pricing by Canadian and International equipment suppliers. Once the budgetary quotations were reviewed and integrated, 98% of the value of mechanical equipment associated with WBS 2000 was sourced from budgetary quotations; the remainder was sourced by benchmarking against other recent Canadian gold projects and studies.

Similar to the above, all major electrical equipment was sized based on the project equipment list. Once the electrical equipment list was outlined, scopes of work were derived and sent for budgetary pricing by Canadian and International equipment suppliers. Once the budgetary quotations were reviewed and integrated, 99% of the value of electrical equipment was sourced from budgetary quotations; the remainder was sourced by benchmarking against other recent Canadian gold projects and studies.

Underground construction works are discussed in Section 20.2.4.

21.2.4 Area 3000 – Process Plant & 4000 – On-Site Infrastructure

All major processing equipment was sized based on the process design criteria outlined in Chapter 17. Once the mechanical equipment list was outlined, the mechanical scopes of work were derived and sent for budgetary pricing by Canadian and International equipment suppliers. Once the budgetary quotations were reviewed and integrated, 91% of the value of mechanical equipment was sourced from budgetary quotations; the remainder was sourced by benchmarking against other recent Canadian gold projects and studies.

Similar to the above, all major electrical equipment was sized based on the project equipment list. Once the electrical equipment list was outlined, scopes of work were derived and sent for budgetary pricing by Canadian and International equipment suppliers. Once the budgetary quotations were reviewed and integrated, more than 95% of the value of electrical equipment was sourced from budgetary quotations; the remainder was sourced by benchmarking against other recent Canadian gold projects and studies.

In support of the major mechanical and electrical equipment packages, the process plant and infrastructure engineering design was completed to a feasibility study level of definition, allowing for the bulk material quantities (steel, concrete, earthworks, piping, cables, instruments, etc.) to be derived for the major commodities.

After the derivation of bulk material quantities for the process plant and infrastructure areas, major construction contracts were formed and tendered to projects from similar projects in Eastern Canada's budgetary pricing bids.

As a total, more than 90% of project costs were derived from first principles, with equipment quotation or contract supply/installation.

21.2.5 Area 5000 – Off Site Infrastructure

21.2.5.1 Main Access Road

The estimate allows for upgrades to the site access road. The main access road from rang Jacques-Paquin to the process plant site has not been developed, so typical road building works will be necessary along this 520 m section. Highway 117 will require deceleration/acceleration lanes in both directions with sufficient length to allow for appropriate cueing. Immediately north of the entrance of rang Jacques-Paquin from Highway 117 is the Ontario Northland Railway. The railway will require upgrading due to heavy truck usage during construction. Signal crossings will also need to be installed as a safety measure due to the higher traffic flow. Roadwork quantities were scoped by the civil/structural department.

Roadworks were quoted as part of the major earthworks package by providing contractors with a bill of quantities for completion of unit rates for each designated task. The returned price schedules included the direct and indirect costs to perform the works. Ausenco's historical pricing was used for the railway crossing requirements.

21.2.5.2 Mine Services Access Road

The estimate allows for earthworks and culverts for the roadways to the vent collars/shafts for the underground ventilation's maintenance. The works were quoted as part of the major earthworks package by providing contractors with a bill of quantities. They returned price schedules with the direct and indirect costs to perform the works.

21.2.6 Area 6000 – Project Indirects

21.2.6.1 Area 6100 – Temporary Construction Facilities and Services

Contractor indirect costs are related to the contractor's direct costs, but cannot easily be allocated to any part of them. These costs are as follows:

- mobilization and demobilization
- site offices and utilities
- construction equipment including mobile equipment, scaffolding, safety supplies, etc.
- head office costs/contribution
- financing charges
- insurances
- profit

Contractors provided indirect costs as part of their pricing schedules.

21.2.6.2 Area 6200 – Commissioning Representatives and Assistance

Vendor representative costs during commissioning and construction includes vendor representative support during the installation of the purchased equipment.

Vendor representative costs have been based on the engineer's evaluation of recommendations and prices provided by equipment vendors during the pricing enquiry process.

21.2.6.3 Area 6300 – Spares (Commissioning, Initial and Insurance)

Commissioning spares quantities were recommended and priced by equipment suppliers. Where equipment pricing was not solicited from vendors, historical information was used to derive a cost for commissioning spares. This resulting cost covers all commission spares for mechanical and electrical and instrumentation (E&I) spares.

Capital spares prices for mechanical, piping, and E&I are based on the prices provided by equipment vendors during the enquiry process. If vendors did not provide a cost for capital spares, a factored allowance was included based on the supply price and benchmarked against Ausenco's in-house database of projects.

21.2.6.4 Area 6400 – First Fills & Initial Charges

Process first fill quantities (e.g., mill media and reagents) and first fill lubricants (e.g., greases, oils, and hydraulic fluids) are calculated based on the engineering design and priced using quotes that were provided by reagent and media suppliers.

21.2.6.5 Area 6500 – Freight and Logistics

Project freight is distributed amongst the individual items as required; a factored rate of 7.5% for inland freight from the USA to site and from Canada to site was applied. This rate was calculated using recent freight quotes of similar Canadian mining projects. Freight for transporting heavy equipment to site was vendor-sourced.

21.2.7 Area 7000 – Project Delivery

This section discusses Area 7100 (Engineering and Construction Management Services) and Area 7200 (Underground Mining and Engineering).

The engineering, procurement, project, and construction management budget was compiled by identifying resources over a defined schedule. Area 7100 was developed by Ausenco, and 7200 by InnovExplo. The EPCM services estimates were developed based on the project delivery strategy described in Chapter 24, and include the following items:

- engineering
- procurement (home office based)
- construction management (site based)
- project office facilities
- staff transfer expenses

- secondary consultants
- field inspection and expediting
- corporate overhead and fees
- travel expenses
- home office expenses
- site office expenses
- commissioning support
- other consulting services (geotechnical, environmental, shipping logistics, surveys, and QA/QC)

The engineering, procurement, project, and construction management estimate has been developed from a deliverables list and by identifying resources over a defined schedule. A detailed assessment of consultants and project general expenses is also included in EPCM costs.

21.2.8 Area 8000 – Owner's Costs

Inputs for major cost contributions to the Owner's Costs estimate were provided by Yamana. These include the staffing plan and fully burdened salaries, the legal costs, the office rental, all environmental and permitting program costs, and community donations/initiatives. Owner's costs include the following:

- Owner's team (including construction, start-up, and commissioning)
- pre-production process and administrative costs
- community agreements
- environmental services
- freight and logistics support
- recruiting, training and site visits
- IT and communications
- insurance, finance, legal, and offices
- closure costs
- operational readiness
- pre-production mining, process and technology and innovation operating costs

21.2.9 Area 9000 – Provisions – Estimate Contingency

Estimate contingency is included to address anticipated variances between specific items in the estimate and the final actual project cost.

Contingency is defined as a monetary allowance that is included, over and above the base cost, to contribute to the success of the project by providing for various cost uncertainties. The level of contingency varies depending on the nature of the contract and the client's requirements. Due to uncertainties at the time the capital cost estimate was developed (in terms of the level of engineering definition, basis of the estimate, schedule development, etc.), it is essential that the estimate include a provision to cover the risk from these uncertainties. The amount of risk was assessed with due consideration of the preliminary level of design work, the way pricing was derived, and the preliminary nature of the plan for project implementation.

The estimate contingency does not allow for the following:

- abnormal weather conditions
- changes to market conditions affecting the cost of labour or materials
- changes of scope within the general production and operating parameters
- effects of industrial disputes

InnovExplo calculated a contingency of C$13.1 million on initial capital associated with mining activities. For the remaining project areas, Ausenco calculated a contingency of C$42.4 million following the percentage allotments by commodity according to Table 21-3.

Table 21-3: Contingency Applied

Commodity Code	Commodity Description	Contingency Applied
A	Architectural	15%
B	Earthworks	10%
C	Concrete	15%
D	Mining	0%*
E	Electrical	15%
F	Platework and Mechanical Bulks	15%
I	Instrumentation	10%
M	Mechanical Equipment	15%
N	Plant & Miscellaneous Equipment	5%
O	Mobile Equipment	15%
P	Pipework	15%
Q	Electrical Bulks	15%
R	Rail	10%
S	Structural Steel	15%
U	Field Indirects	15%
V	Third-Party Packages/Other	15%
W	EPCM, EPC & EP	15%
X	Provisions	0%
Y	Owner’s Costs	5%

*Note: Mining Contingency is allowed for by InnovExplo’s pricing contribution. Source: Ausenco, 2021

21.2.10 Growth Allowance

Each line item of the estimate is developed initially at base cost only. A growth allowance is then allocated to each element of those line item costs to reflect the level of definition of design and pricing strategy.

Estimate growth can be described as follows:

- intended to account for items that cannot be quantified based on current engineering status but which are empirically known to appear
- accuracy of quantity take-offs and engineering lists based on the level of engineering and design undertaken at a feasibility study level
- pricing growth for the likely increase in cost due to development and refinement of specifications as well as re-pricing after initial budget quotations and after finalization of commercial terms and conditions to be used on the project

Where an allowance has been used that is the result of factoring, no growth has been applied, as the factor has been surmised from a total cost.

For WBS 2000, 3000 and 4000, growth has been calculated by commodity and by evaluating the status of the engineering scope definition and maturity and the ratio of the various pricing sources for equipment and materials used to compile the estimate. The capital cost growth allowance is presented in Table 21-4..

Table 21-4: Growth Allowances for WBS 2000, 3000 and 4000

Commodity Code	Discipline	Growth Applied
A	Architectural	5%
B	Earthworks	5%
C	Concrete	5%
E	Electrical	5%
F	Platework	5%
I	Instrumentation	5%
M	Mechanical Equipment	5%
P	Pipework	5%
S	Structural Steel	5%

Source: Ausenco, 2021

21.2.11 Exclusions

The following costs and scope are excluded from the capital cost estimate:

- senior finance charges
- residual value of temporary equipment and facilities
- environmental approvals

- this study or any further project studies
- force majeure issues
- future scope changes
- special incentives (schedule, safety, or others)
- no allowance has been made for loss of productivity and/or disruption due to religious, union, social and/or cultural activities
- management reserve (project contingency)
- Owner's escalation costs
- Owner's foreign exchange exposure
- operating costs
- working capital
- land acquisition
- project-specific risk reserve (not evaluated)

21.3 Basis of Capital Cost Estimate – Sustaining

21.3.1 Area 1000 – Mining

A large portion of sustaining capital costs is attributable to the underground mining operation. Significant sustaining capital is required as mining progresses. Sustaining capital costs include drifts, ventilation raises, ramp extension, underground infrastructure, backfill network, and mobile equipment.

InnovExplo provided estimates for all underground mine sustaining capital costs except for material handling, automation, communications, electrical and maintenance infrastructure costs, which were provided by Ausenco. The underground sustaining capital costs by Ausenco consisted largely of electrical equipment, bulks, and communication equipment necessary to support the ongoing development of the mine each year. The stacking and lease agreement for the waste rock storage facility is also included in the mining sustaining costs.

The same approach as described in Section 21.2.2 was used to estimate sustaining capital cost. Quantities (e.g., equipment, development, supplies) were based on underground mine design and estimates made in Chapter 16. The costs were attribute to the appropriate period based on the underground mine planning.

The total sustaining capital cost estimate is C$321.8 million, as detailed in Table 21-5.

Table 21-5: Mining Sustaining Capital Costs (C$M)

WBS	*Item*	Sustaining Capital Cost (C$M)
1100	Old Wasamac Dewatering & Rehabilitation	0.0
1200	Mine Portals	0.0
1310	Mine Development	109.5
1320	Contracted Mine Development	16.6
1330	Mine Vertical Development	24.2
1400	Underground Mobile Equipment	84.5
1500	Underground Infrastructure & Construction	0.1
1610	Underground Ventilation	3.5
1620	Underground Communications & Electrical	26.5
1630	Underground Water Management	0.8
1700	Backfill Plant & Network	51.9
1800	Technical Services & Instrumentation	0.1
1900	Waste Rock Storage Facility	4.1
	Total Sustaining Capital	321.8

21.3.2 Area 4000 – On Site Infrastructure

On-site infrastructure sustaining capital costs include underground mine communications infrastructure (WBS 4300), underground control room (WBS 4500), process plant and surface support vehicle leases (WBS 4600) and the ongoing expansion of the TSF and associated roadworks (WBS 4800). Equipment, buildings and construction works were priced by vendors and contractors as part of the packages associated with initial capital expenditures.

The associated TSF sustaining capital costs (WBS 4800) are shown by year in Table 21-6.

Table 21-6: Tailings Management Facility (WBS 4000) Sustaining Capital Costs (C$M)

Description (WBS)	Year										LOM Total
	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	
Tailings transport & Placement (4820)	3.4	4.9	4.9	5.5	4.4	4.4	3.9	4.5	3.9	2.1	41.9
Roads & Drainage (4840)						1.6					1.6
Equipment Leasing (4850)	0.9	0.9	0.9	0.9	0.9	-	-	-	-	-	4.7
Total	4.3	5.8	5.8	6.4	5.3	6.0	3.9	4.5	3.9	2.1	48.2

21.3.3 Area 8000 – Owner's Costs (Conceptual Closure Costs)

The conceptual closure cost estimate captures the following areas:

- progressive reclamation of the waste rock storage facility and tailings storage facility
- demolition of site infrastructure

- closure of underground mine ventilation raises
- deconstruction of roadways and removal of culverts
- removal of any contaminated soils in the plant site and filtration plant site areas
- revegetation of infrastructure areas
- water testing and breaching of the mill basin and TSF ponds
- establishment of an artificial wetland at the tailings storage facility
- post-closure monitoring, and administrative and engineering support

A list of key rehabilitation and closure activities for these areas was derived referencing the guidelines from the "Guide de préparation du plan de réaménagement et de restauration des sites miniers au Québec". Progressive reclamation of waste rock storage facility begins and ends in Years 1 and 3, respectively. The tailings storage facility is reclaimed throughout Years 1 to 10 inclusive. At the end of the mine life, the site as a whole will be reclaimed. Site infrastructure demolition encompasses activities associated with the decommissioning and disposal of the process plant and filtration plant infrastructure, including equipment, in-plant piping, and off-plot piping. The deconstruction of roadways and culvert crossings were estimated for the main plant site area, the TSF and explosives storage access roads, as well as the access roads for the ventilation and paste backfill plant cement borehole collar. Reclamation activities include, but are not limited to, surface contouring, placement of a granular layer, an organic soil layer, hydroseeding and revegetation.

The rehabilitation costs were estimated using benchmark unit rates provided by contractors and as per the surface area or volumes of land requiring rehabilitation. The cost of demolition of pieces of equipment was projected as 4% of their capital cost. An allowance of 10% was carried for engineering and project management, as well as a 15% contingency, amounting to a total closure cost of C$24.8 million.

Post-closure monitoring includes allowances for monitoring programs, environmental testwork, and general and administrative costs (a small office space rental, legal costs, safety and security, etc.).

The closure estimate assumes mine waste products are not acid-generating and not metal-leaching as per static testwork completed to date; kinetic testwork in Q3-Q4 2021 will add an understanding of the long-term behaviour of mine waste products.

21.3.4 Area 9000 – Provisions – Contingency

The same methodology described in Section 21.2.9 was applied for sustaining capital contingency.

InnovExplo calculated a contingency of C$19.6 million on initial capital associated with mining activities. For remaining project areas, Ausenco calculated a contingency of C$11.9 million.

21.3.5 Salvage

Salvaging costs have been projected by assuming that all mechanical, electrical, the paste backfill plant and mobile equipment will carry a 10% resale value at the end of the mine life. Total salvaging value was estimated at C$25.2 million.

21.3.6 Growth Allowance

The same methodology described in Section 21.2.10 was applied for sustaining capital.

21.3.7 Exclusions

The same exclusions apply as described in Section 21.2.11.

21.4 Operating Costs

The operating cost estimate is presented in Q2 2021 Canadian dollars (CAD, C$). The estimate was developed to have an accuracy of ±15%. The estimate includes mining, processing, and general and administration (G&A) costs.

The overall life-of-mine operating cost is $1,333 million over 10 years, or an average of $57.53/t of ore milled in a typical year. Of this total, processing and G&A account for C$497.1 million and mining accounts for C$835.9 million.

21.4.1 General Assumptions

Common to all operating cost estimates are the following assumptions:

- Cost estimates are based on Q2 2021 pricing without allowances for inflation.
- For material sourced in US dollars, an exchange rate of 1.28 Canadian dollar per US dollar was assumed.
- Fuel costs and associated taxes were taken as the historical average for the previous six years at nearby Val d'Or, Québec. Estimated retail costs are C$1.158/L for petroleum diesel.
- The annual power costs were calculated using a unit price of C$0.051/kWh. This value was assumed to be the same as in the 2018 Feasibility Study.
- Labour is assumed to come from the local area of highly skilled workers in Rouyn-Noranda.
- Effluent treatment plant costs provide for equipment lease.

21.4.2 Mine Operating Costs

Total operating costs, average cost per year, and costs per tonne for underground production are summarized in Table 21-7. These operating costs are detailed per year in Table 21-8. These costs include material and labour for each individual category. Unit costs used in the estimation of mine development costs are summarized in Table 21-9. The only contingency applied to the mining operating costs is applied to the paste fill cost at a ratio of 10%.

InnovExplo provided estimates for all underground mine operating costs except for material handling costs, which were provided by Ausenco. The total underground mine production cost is $835.9 million.

Table 21-7: Underground Production Costs (C$M)

Activity	Total LOM Cost (C$M)	Average LOM (C$M/a)	Average LOM ($/t mined)
Mine Development	126.0	10.5	5.44
Drill & Blast	128.3	10.7	5.54
Loading & Haulage	141.0	11.7	6.09
Grade Control	6.4	0.5	0.28
Backfill	147.1	12.3	6.35
Underground Ventilation & Heating	21.0	1.7	0.90
Underground Communications & Electrical	24.2	2.0	1.04
Underground Water Management	6.7	0.6	0.29
Supervision	12.6	1.1	0.55
Mine Services	180.6	15.1	7.80
Technical Services	30.0	2.5	1.29
Contingency	12.2	1.0	0.53
Total Mining Operating Cost	835.9	69.7	36.08

Note: The sum of the values may be slightly different from the total value due to rounding.

Table 21-8: Mine Development Costs (C$)

Development	*Metres*	*Cost ($/m)*
Pre-production Capital Cost (Owner Operated)	3,862	3,275
Pre-production Capital Cost (Contractor)	15,824	4,054
Sustaining Capital Cost (Owner Operated)	34,492	3,176
Sustaining Capital Cost (Contractor)	3,086	5,388
Operating cost (Owner Operated)	47,431	2,655
Total	104,696	3,142

Table 21-9: Underground Production Cost per Year (C$M)

Activity	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	LOM
Mine Development	-	-	4.7	20.6	18.3	14.0	11.6	14.0	12.0	10.5	12.7	10.5	0.5	126.0
Drill & Blast	-	-	-	9.6	14.9	13.4	13.3	13.7	12.8	13.3	14.1	14.7	8.4	128.3
Loading & Haulage	-	-	0.2	7.2	14.4	15.3	15.6	15.5	15.1	15.7	15.6	16.0	10.5	141.0
Grade Control	-	-	0.0	0.4	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.4	6.4
Backfill	-	-	-	8.8	15.3	15.6	15.4	16.8	17.9	15.9	15.2	15.9	10.2	147.1
Underground Ventilation & Heating	0.2	1.0	1.4	1.9	2.2	2.2	2.2	2.3	2.4	2.2	2.1	2.1	1.2	21.0
Underground Communications & Electrical	-	-	-	2.2	2.2	2.3	2.4	2.4	2.4	2.6	2.6	2.6	2.4	24.2
Underground Water Management	-	0.0	0.4	0.5	0.6	0.6	0.7	0.7	0.7	0.8	0.8	0.8	0.6	6.7
Supervision	0.2	0.9	1.1	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.2	12.6
Mine Services	0.8	3.1	8.6	17.3	18.2	18.4	18.3	18.3	18.1	18.4	18.5	18.6	15.4	180.6
Technical Services	0.6	2.5	2.8	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	2.7	30.0
Contingency	-	-	-	0.7	1.4	1.3	1.3	1.2	1.3	1.4	1.3	1.5	0.9	12.2
Total Mining Operating Cost	1.8	7.7	19.3	73.5	92.4	88.1	85.8	89.7	87.6	85.7	87.9	87.6	54.4	835.9

Note: The sum of the values may be slightly different from the total value due to rounding.

21.4.3 Underground Systems Operating Costs

The underground systems operating costs have been considered under processing operating costs in the categories of power consumption, maintenance cost, operation and maintenance personnel, and consumables for the primary crusher.

21.4.4 Process Operating Costs

The life-of-mine process operating cost is $499 million over 10 years. A breakdown of the process operating costs for a typical year and the unit costs are presented in Table 21-10. Chapter 22 shows the all-in life-of-mine weighted process operating cost in dollars per tonnes milled, which includes ramp-up.

Table 21-10: Average Annual Process Operating Costs

Cost Centre	C$M/a	C$/t
Reagents and Consumables	14.39	5.63
Plant Maintenance	1.74	0.68
Power	4.77	1.87
Laboratory	0.17	0.07
Labour (O&M)	13.98	5.47
Water Treatment	1.79	0.70
Processing Mobile Equipment	1.29	0.51
Total	38.14	14.93

Source: Ausenco, 2021

21.4.4.1 General Assumptions

Assumptions made in developing the process operating cost estimate are listed below:

- Mill production is set at an average of 7,000 t/d or 2.555 Mt/a.
- Process plant operating costs are calculated based on labour, power consumption, and process and maintenance consumables.
- Off-site gold refining, insurance, and transportation costs are included in the financial model in Chapter 22.
- Oxygen is assumed to be delivered to site as liquid oxygen.
- Operating costs incurred during the pre-production period have been capitalized.
- Most labour rates were provided by Yamana based on comparable salaries at other local mine sites. Other labour rates were estimated based on Ausenco's past project experience.
- General and administrative (G&A) costs were mainly provided by Yamana, the rest were populated from benchmarks.

- Consumables costs are based on direct quotes from vendors, where provided, and otherwise are based on data from quotes for similar projects in Eastern Canada.
- No factor for spare parts has been applied to adjust for consumption of less spare parts in early years of operation.
- Grinding media consumption rates have been estimated based on the ore characteristics.
- Reagent consumption rates have been estimated based on the metallurgical testwork results at a nominal basis.
- Mobile equipment costs provide for fuel and maintenance, not for purchase or vehicle lease.

21.4.4.2 Consumables

Individual reagent consumption rates were estimated based on the metallurgical testwork results, Ausenco's in-house database and experience, industry practice, and peer-reviewed literature. Major reagent costs were obtained from vendor quotations to Rouyn-Noranda, including SAG and ball mill media, sodium cyanide, hydrated lime, flocculant, activated carbon, and sodium metabisulphite (SMBS). Other reagent cost was obtained through benchmarking for similar projects performed by Ausenco. A detailed description of the reagents required for the process is provided in Chapter 17.

Other consumables (e.g., liners for the primary crusher, SAG mill, ball mill, and ball media for the mills) were estimated using:

- metallurgical testing results (Bond abrasion testing)
- Ausenco's in-house calculation methods, including simulations
- forecast nominal power consumption

Reagents and consumables represent approximately 38% of the total process operating cost at C$5.63/t milled.

21.4.4.3 Maintenance

Annual maintenance consumable costs were calculated based on a total installed mechanical capital cost by area using a weighted average factor from 3% to 4%. The factor was applied to mechanical equipment, platework, and piping. The total maintenance consumables operating cost is C$0.68/t milled, or approximately 4% of the direct mechanical capital cost, which is equivalent to approximately 5% of the total process operating cost.

21.4.4.4 Power

The processing power draw was based on the average power utilization of each motor on the electrical load list for the process plant and services previously summarized in Section 1. Power will be supplied by the Hydro-Québec grid to service the facilities at the site.

21.4.4.5 Laboratory & Assays

Operating costs associated with laboratory and assay activities were estimated according to the anticipated number of assays per day and per year, estimated by Ausenco. Assay costs include environmental sampling and assaying. Assay costs associated with processing mine grade control samples or exploration samples are included in the mine operating costs. The laboratory and assays comprise approximately 0.45% of the total process operating cost, and the forecasted annual requirement for internal assays will be around 16,000 for the processing plant. Approximately 3,100 samples per year are required for the environmental sampling schedule.

21.4.4.6 Mobile Equipment

Vehicle costs are based on a scheduled number of light vehicles and mobile equipment (including fuel, maintenance, spares and tires, and annual registration and insurance fees).

21.4.4.7 Labour

The personnel requirement was estimated by benchmarking against similar projects. The labour costs incorporate personnel requirements for plant operation, such as management, metallurgy, operations, maintenance, site services, assay lab, and contractor allowance. The total operational labour averages 96 employees.

Individual personnel were divided into their respective positions and classified as either 8-hour or 12-hour shift employees. Salaries were provided by Yamana, who obtained salaries for similar positions at other mine sites in the area. Yamana also confirmed the specific benefits and bonuses to be allocated. Thus, the rates were estimated as overall rates, including all burden costs.

An organizational staffing plan outlining the labour requirement for the process plant is shown in Table 21-11. The G&A staffing plan is summarized in Table 21-12.

Table 21-11: Operations and Maintenance Staffing Plan

Labour / Contractor Summary	#/Shift	# Shifts	Quantity
Process Upper Management			
Senior Metallurgist	1	1	1
Plant and Site Maintenance Superintendent	1	1	1
Maintenance Planner	1	1	1
Chief Assayer	1	1	1
Mill Trainer	1	1	1
Chief Metallurgist/Process Superintendent	1	1	1
Mill Operations			
Shift Foreman	1	4	4
Control Room Operator	1	4	4
Crusher Operator	1	4	4
Grinding Operator	1	4	4
Leach/Reagents Operator	1	4	4
Filter Plant Operator	2	4	8
Filter Plant Operator Assistant	1	4	4
TSF Truck Driver	4	2	8
Gold Room Foreman	1	2	2
Gold Room Operator	1	2	2
Technical Services			
Graduate Metallurgist	1	2	2
Metallurgical Technician	1	2	2
Assay Laboratory Technician	2	4	8
Mill Maintenance			
Maintenance Foreman	1	1	1
Electrical Foreman	1	1	1
Electrician	2	2	4
Millwright/Fitter	4	4	16
Mechanical Apprentice	1	2	2
Electrical Apprentice	1	2	2
Instrument Technician	2	2	4
Electrician Technician	2	2	4
Total – Process	38	64	96

Table 21-12: G&A Staffing Plan

Labour / Contractor Summary	No. per Shift	No. Shifts	Quantity
Management			
General Manager	1	1	1
Mine Manager	1	1	1
Human Resources Manager	1	1	1
Health & Safety Manager	1	1	1
Chief Engineer	1	1	1
Process Manager	1	1	1
Finance Manager	1	1	1
Community & Social Manager	1	1	1
Maintenance Manager	1	1	1
Operational Excellence manager	1	1	1
IT			
IT Superintendent	1	1	1
IT Technician	1	1	2
Procurement			
Procurement Superintendent	1	1	1
Purchaser	1	2	2
Warehouse Supervisor	2	1	2
Warehouse Technician	3	2	6
Safety & Security			
Safety Superintendent	1	1	1
Training Coordinator	1	1	1
Safety Coordinator	2	1	2
Security Personnel	2	4	8
Nurse	1	2	2
Community & Social			
Community & Social Superintendent	1	1	1
Permitting & Environment			
Environment Superintendent	1	1	1
Environment Technician	4	1	4
Human Resources			
Human Resources Superintendent	1	1	1
Human Resources Coordinator	1	1	1
Payroll Administrative Clerk	1	1	1
Finance			
Controller	1	1	1
Planning Specialist	1	1	1
Accountant	1	1	1
AP Clerk	1	1	1
Operational Excellence			
Operational Excellence Coordinator	2	1	2
Administration			
Assistant to General Manager	1	1	1
Reception	1	1	1
Surface Crew			
Surface Supervisor	1	1	1
Surface Personnel	3	1	3
Total - G&A	47	43	59

21.4.5 Infrastructure Operating Costs

Infrastructure operating costs are included under processing operating costs in the category of power consumption.

21.4.5.1 Tailings Storage Facility Operating Costs

Tailings management facility (TSF) operating costs include labour for personnel operating equipment at the TSF and fuel for the equipment. These costs are carried under the process operating costs. Development of the TSF for ongoing placement of filter cake is carried as a capital cost, as well as the heavy equipment leases.

21.4.5.2 Effluent Treatment Operating Costs

Water treatment costs are expenses not directly related to the production of gold and include expenses not included in mining, processing, external refining, and transportation costs. These costs were developed from first principles by Ausenco and checked alongside effluent treatment plant vendors regarding required power for operation and consumables. A breakdown summary of effluent treatment operating costs is shown in Table 21-13..

Table 21-13: Effluent Treatment Plant Operating Cost Summary

Cost Centre	C$M/a	C$/t
Plant Maintenance	0.082	0.03
Labour	0.055	0.02
Power	0.169	0.07
Leasing	0.900	0.35
Reagent Consumables	0.579	0.23
Total	1.79	0.70

21.4.6 General and Administrative Operating Costs

General and administrative (G&A) costs are expenses not directly related to the production of gold and include expenses not included in mining, processing, external refining, and transportation costs. These costs were developed with input from Yamana, as well as Ausenco's in-house data on existing Canadian operations.

A bottom-up approach was used to develop estimates for G&A costs over the life of mine. The G&A costs were determined for a 10-year mine life with an average cost of C$5.18/t milled. These costs were assembled according to the following departmental cost reporting structure:

- G&A maintenance (includes snow-clearing, surface grading, and watering during the summer)
- G&A personnel
- human resources (includes recruiting, training, and community relations)
- infrastructure power (includes power, fuel, and heat)

- site administration, maintenance and security (includes subscriptions, professional memberships and dues, external training, advertising and promotional material, first aid, office supplies and equipment, sewage and garbage disposal, bank and payroll fees)
- assets operation (includes non-operation-related vehicles)
- health and safety (includes personal protective equipment, hospital service costs and first aid supplies)
- environmental (includes sampling and TSF operation)
- IT and telecommunications (includes hardware and satellite link)
- contract services (includes insurance, consulting, sanitation, auditing, licenses, and legal fees)
- cyanide code fees

The G&A labour costs were estimated by developing a headcount profile for each department. Labour rates provided by Yamana were applied to develop the total G&A labour cost.

G&A labour resources include 59 employees. A breakdown summary of life-of-mine G&A costs is shown in Table 21-14.

Table 21-14: Annual Average G&A Operating Cost Summary

Cost Centre	C$M/a	C$/t
G&A Labour	7.63	2.99
G&A Expenses	5.25	2.05
Site Maintenance (including mobile equipment)	0.35	0.14
Total	13.23	5.18

22 ECONOMIC ANALYSIS

22.1 Forward-Looking Information Cautionary Statements

The results of the economic analyses discussed in this chapter represent forward-looking information as defined under Canadian securities law. This report has included certain non-IFRS performance measures, such as: Cash cost and All-in sustaining cost ("AISC"). These non-IFRS performance measures do not have a standardized meaning, and therefore may not be comparable to similar measures employed by other issuers. Cash cost is calculated by summing mining cost, processing cost, G&A, refining charges, and royalties, and dividing it by payable gold ounces. AISC is calculated by summing Cash cost, sustaining capital, closure cost, and salvage value and dividing it by payable gold ounces. The results of the economic analysis are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Forward-looking information includes the following:

- mineral reserve estimates
- assumed commodity price and exchange rates
- proposed mine production plan
- projected mining and process recovery rates
- assumptions about mining dilution and the ability to mine in areas previously exploited using underground mining methods as envisaged
- sustaining costs and proposed operating costs
- interpretations and assumptions regarding joint venture and agreement terms
- assumptions as to closure costs and closure requirements
- assumptions about environmental, permitting, and social risks

Additional risks to the forward-looking information include:

- changes to costs of production from what is assumed
- changes in the estimated timing and quantity of production
- unrecognized environmental risks
- unanticipated reclamation expenses
- unexpected variations in quantity of mineralized material, grade or recovery rates

- geotechnical or hydrogeological considerations during mining being different from what was assumed
- failure of mining methods to operate as anticipated
- failure of plant, equipment, or processes to operate as anticipated
- changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis
- ability to maintain the social license to operate
- accidents, labour disputes, and other risks of the mining industry
- changes to interest rates
- changes to tax rates
- changes in government regulation of mining operations
- potential delays in the issuance of permits and any conditions imposed with the permits that are granted

22.2 Methodologies Used

The project has been evaluated using a discounted cash flow (DCF) analysis based on a 5% discount rate. Cash inflows consist of annual revenue projections. Cash outflows consist of capital expenditures, including pre-production costs, operating costs, taxes, and royalties. These are subtracted from the inflows to arrive at the annual cash flow projections. Cash flows are taken to occur at the mid-point of each period. It must be noted that tax calculations involve complex variables that can only be accurately determined during operations and, as such, the actual post-tax results may differ from those estimated. A sensitivity analysis was performed to assess the impact of variations in metal prices, discount rate, foreign exchange rates, operating costs and capital costs.

The capital and operating cost estimates developed specifically for this project are presented in Chapter 21 in 2021 Canadian dollars. The economic analysis has been run on a constant dollar basis with no inflation.

22.3 Financial Model Parameters

A base case gold price of US$1,550/oz is based on consensus analyst estimates and recently published economic studies. The forecasts are meant to reflect the average metal price expectation over the life of the project. No price inflation or escalation factors were taken into account.

The economic analysis was performed using the following assumptions:

- construction starting December 1, 2024
- commercial production starting on October 1, 2027

- mine life of 9.7 years
- exchange rate of 1.28 (USD:CAD)
- cost estimates in constant Q3 2021 Canadian dollars with no inflation or escalation
- 100% ownership with 1.5% NSR
- capital costs funded with 100% equity (no financing costs assumed)
- all cash flows discounted to December 1, 2024 using mid period discounting convention
- gold is assumed to be sold in the same year its produced
- no contractual arrangements for refining currently exist

22.3.1 Taxes

The project has been evaluated on a post-tax basis to provide an approximate value of the potential economics. The tax model was compiled with assistance from Yamana's taxation professionals. The calculations are based on the tax regime as of the date of the feasibility study. At the effective date of the cashflow analysis, the project was assumed to be subject to the following tax regime:

- The Canadian corporate income tax system consists of 15% federal income tax and 11.5% provincial income tax.
- The mining tax rate in Québec is calculated using progressive tax rates, with each rate applied to a portion of the operator's annual profit.

Table 22-1 shows the mining tax rate that applies to each portion of the operator's annual profit margin segment.

Table 22-1: Mining Tax Rates in Québec

Profit Margin		Tax Rate
First Segment	0% to 35%	16%
Second Segment	More than 35%, up to 50%	22%
Third Segment	More than 50%	28%

At the base case gold price assumption, total tax payments are estimated to be C$426.8 million over the life of mine.

22.3.2 Royalty

A 1.5% net smelter returns royalty has been assumed for the project, resulting in approximately C$50.3 million in royalty payments over life of mine.

22.3.3 Refining & Transport Cost

Mine revenue is derived from the sale of gold doré into the international marketplace. No contractual arrangements for refining exist at this time. However, the parameters used in the economic analysis are consistent with current industry rates. A refining and transport charge of C$2.00/oz was assumed with 99.95% gold payability resulting in a C$3.4 million cost over the life of mine.

22.4 Economic Analysis

The economic analysis was performed assuming a 5% discount rate. The pre-tax $NPV_{5\%}$ is C$610 million; the IRR is 21.7%; and payback period is 3.6 years. On a post-tax basis, the $NPV_{5\%}$ is C$326 million; the IRR is 16.1%; and the payback period is 4.0 years.

A summary of project economics is shown graphically in Figure 22-1 and listed in Table 22-2. The analysis was done on monthly, quarterly, and annual cashflow basis, but the cashflow output is shown on an annualized basis in Table 22-3.

Figure 22-1: Project Economics Summary

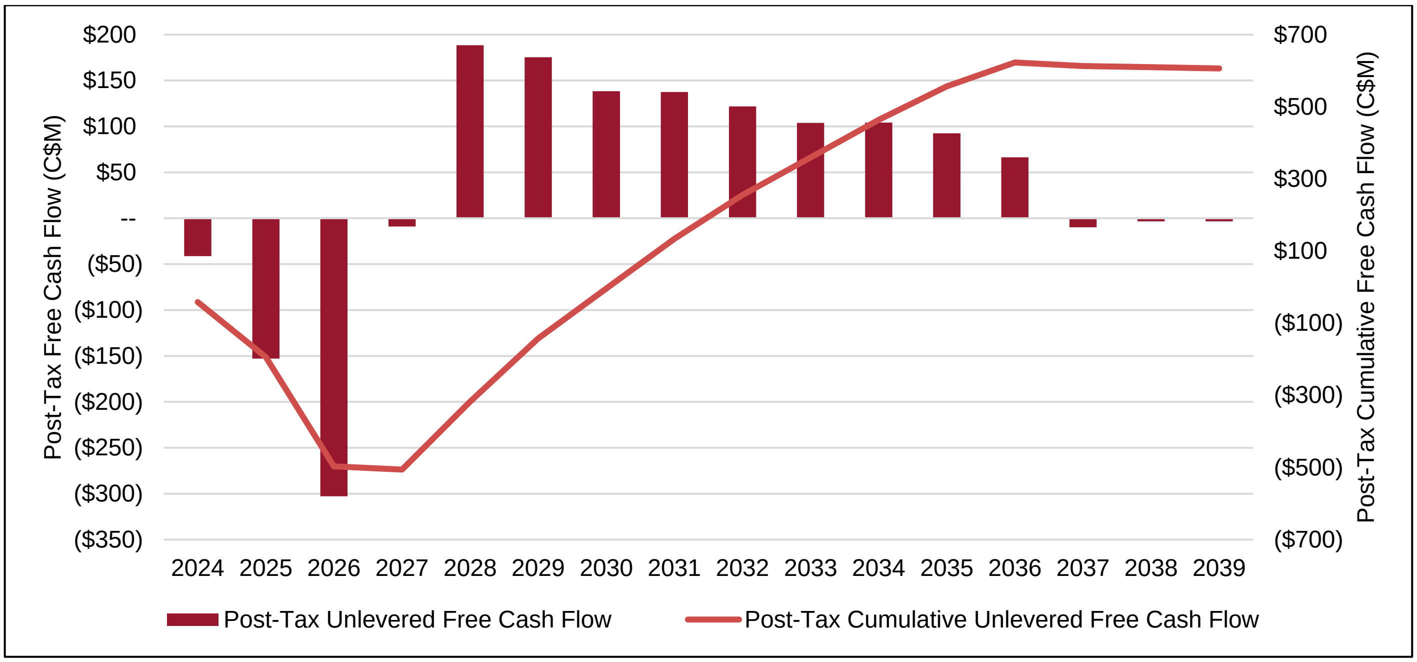


Table 22-2: Summary of Project Economics

Description		LOM Total / Avg.
General		
Gold Price (US$/oz)		$1,550
Mine Life (years)		9.7
Total Mill Feed Tonnes (kt)		23,168
Production		
Mill Head Grade (g/t)		2.56
Mill Recovery Rate (%)		88.7%
Total Mill Ounces Recovered (koz)		1,694
Total Average Annual Production (koz)		169
Operating Costs		
Mining Cost (C$/t Milled)		$36.08
Processing Cost (C$/t Milled)		$15.70
G&A Cost (C$/t Milled)		$5.75
Refining & Transport Cost (C$/oz)		$2.00
Total Operating Costs (C$/t Milled)		$57.53
Cash Costs (US$/oz Au)		$640
AISC (US$/oz Au)		$828
Capital Costs		
Initial Capital (C$M)		$533
Sustaining Capital (C$M)		$406
Closure Costs (C$M)		$25
Salvage Costs (C$M)		($25)
Financials	**Pre-Tax**	**Post-Tax**
NPV (5%) (C$M)	$610	$326
IRR (%)	21.7%	16.1%
Payback (years)	3.6	4.0

Note: Cash cost, and AISC are non-IFRS performance measures. Refer to Forward-Looking Cautionary Statements.

Table 22-3: Project Cash Flow on an Annualized Basis

Cash Flows Discounted to December 31, 2024	Units	Sum/Avg	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040
Macro Assumptions																			
Gold Price - Flat	US$/oz	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550
Foreign Exchange	USD:CAD	$1.28	$1.28	$1.28	$1.28	$1.28	$1.28	$1.28	$1.28	$1.28	$1.28	$1.28	$1.28	$1.28	$1.28	$1.28	$1.28	$1.28	$1.28
Free Cash Flow Valuation																			
Revenue	C$mm	$3,359	--	--	$5	$238	$400	$396	$392	$396	$372	$340	$333	$307	$180	--	--	--	--
Operating Cost	C$mm	($1,333)	--	--	($6)	($117)	($144)	($139)	($137)	($141)	($139)	($137)	($139)	($139)	($97)	--	--	--	--
Refining Charges	C$mm	($3)	--	--	($0)	($0)	($0)	($0)	($0)	($0)	($0)	($0)	($0)	($0)	($0)	--	--	--	--
Royalties	C$mm	($50)	--	--	($0)	($4)	($6)	($6)	($6)	($6)	($6)	($5)	($5)	($5)	($3)	--	--	--	--
EBITDA	C$mm	$1,972	--	--	($1)	$118	$250	$250	$249	$249	$228	$198	$189	$164	$80	--	--	--	--
Initial Capital Cost	C$mm	($533)	($47)	($163)	($323)	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Sustaining Capital Cost	C$mm	($406)	--	--	($5)	($126)	($36)	($43)	($43)	($36)	($37)	($35)	($23)	($16)	($6)	--	--	--	--
Closure Capital Cost	C$mm	($25)	--	--	--	($1)	($1)	($0)	($0)	($0)	($0)	($0)	($0)	($0)	($0)	($14)	($5)	($5)	--
Salvage Value	C$mm	$25	--	--	--	--	--	--	--	--	--	--	--	--	$25	--	--	--	--
Change in Net Working Capital	C$mm	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Pre-Tax Unlevered Free Cash Flow	C$mm	$1,033	($47)	($163)	($329)	($9)	$213	$207	$205	$213	$190	$163	$166	$147	$99	($14)	($5)	($5)	--
Pre-Tax Cumulative Unlevered Free Cash Flow	C$mm	$1,033	($47)	($210)	($539)	($548)	($335)	($128)	$78	$291	$481	$644	$810	$957	$1,056	$1,043	$1,038	$1,033	$1,033
Federal Tax	C$mm	($133)	--	--	--	--	--	($4)	($21)	($23)	($21)	($19)	($19)	($18)	($10)	$2	$1	$1	--
Quebec Tax	C$mm	($101)	--	--	--	--	--	--	($16)	($19)	($17)	($15)	($15)	($14)	($8)	$2	$1	$1	--
Quebec Mining Duty	C$mm	($193)	$5	$10	$27	($0)	($25)	($27)	($30)	($33)	($30)	($25)	($27)	($24)	($14)	--	--	--	--
Post-Tax Unlevered Free Cash Flow	C$mm	$606	($41)	($153)	($303)	($9)	$188	$175	$138	$138	$122	$104	$104	$92	$66	($10)	($3)	($3)	--
Post-Tax Cumulative Unlevered Free Cash Flow	C$mm	$606	($41)	($194)	($497)	($506)	($317)	($142)	($4)	$134	$256	$360	$464	$556	$623	$613	$610	$606	$606
Production Profile																			
Production Summary																			
Total Mineral Reserve Mined	kt	23,168	--	--	74	1,552	2,527	2,520	2,520	2,512	2,527	2,520	2,508	2,520	1,389	--	--	--	--
Total Waste	kt	7,144	124	581	1,261	909	646	657	671	585	622	412	364	311	1	--	--	--	--
Total Material Mined	kt	30,312	124	581	1,335	2,461	3,173	3,177	3,191	3,097	3,149	2,932	2,872	2,831	1,390	--	--	--	--
Percent of Reserve Depleted	%	100.0%	--	--	0.1%	6.7%	10.9%	10.9%	10.9%	10.8%	10.9%	10.9%	10.8%	10.9%	6.0%	--	--	--	--
Project Life	yrs	9.7																	
Mill Feed	kt	23,168	--	--	33	1,594	2,527	2,520	2,520	2,512	2,527	2,520	2,508	2,520	1,389	--	--	--	--
Mill Head Grade	g/t	2.56	--	--	2.75	2.65	2.81	2.79	2.76	2.78	2.57	2.43	2.32	2.18	2.26	--	--	--	--
Contained Gold	koz	1,910	--	--	3	136	228	226	223	224	209	197	187	176	101	--	--	--	--
Mill Recovery	%	88.7%	--	--	85.6%	88.6%	88.4%	88.4%	88.4%	89.0%	89.9%	87.3%	89.8%	87.8%	89.9%	--	--	--	--
Recovered Gold	koz	1,694	--	--	2	120	202	199	198	200	188	172	168	155	91	--	--	--	--
Gold % Payable	%	99.95%	--	--	99.95%	99.95%	99.95%	99.95%	99.95%	99.95%	99.95%	99.95%	99.95%	99.95%	99.95%	--	--	--	--
Payable Gold	koz	1,693	--	--	2	120	201	199	198	200	188	172	168	155	91	--	--	--	--
Revenue	C$mm	$3,359	--	--	$5	$238	$400	$396	$392	$396	$372	$340	$333	$307	$180	--	--	--	--
Operating Costs																			
Mine Operating Costs	C$mm	$836	--	--	$3	$74	$92	$88	$86	$90	$88	$86	$88	$88	$54	--	--	--	--
Mill Operating Costs	C$mm	$364	--	--	$2	$30	$38	$38	$38	$38	$38	$38	$38	$38	$29	--	--	--	--
G&A Costs	C$mm	$133	--	--	$1	$13	$13	$13	$13	$13	$13	$13	$13	$13	$13	--	--	--	--
Operating Costs per tonne Processed	*C$/t Processed*	*$58*			*$183*	*$73*	*$57*	*$55*	*$54*	*$56*	*$55*	*$54*	*$55*	*$55*	*$70*	--	--	--	--

Cash Flows Discounted to December 31, 2024	Units	Sum/Avg	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040
Refining, Transport Cost & Royalties																			
Refining & Transportation	C$mm	$3	--	--	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	--	--	--	--
NSR Royalty																			
Total Revenue	C$mm	$3,359	--	--	$5	$238	$400	$396	$392	$396	$372	$340	$333	$307	$180	--	--	--	--
Less: Refining & Transport Costs	C$mm	$3	--	--	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	--	--	--	--
Total Net Revenue	C$mm	$3,356	--	--	$5	$238	$399	$395	$391	$395	$372	$340	$333	$307	$180	--	--	--	--
NSR Royalty	%	1.5%	--	--	1.5%	1.5%	1.5%	1.5%	1.5%	1.5%	1.5%	1.5%	1.5%	1.5%	1.5%	--	--	--	--
Royalties	C$mm	$50	--	--	$0	$4	$6	$6	$6	$6	$6	$5	$5	$5	$3	--	--	--	--
Cash Costs																			
Cash Cost *	US$/oz Au	$640	--	--	$1,912	$782	$581	$570	$566	$576	$602	$648	$671	$725	$862	--	--	--	--
All-in Sustaining Cost (AISC) **	US$/oz Au	$828	--	--	$3,519	$1,607	$724	$738	$738	$715	$757	$809	$779	$807	$696	--	--	--	--
Capital Expenditure																			
Total Initial Capital	C$mm	$533	$47	$163	$323	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Mining	C$mm	$195	$15	$49	$131	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Underground Plant & Systems	C$mm	$24	--	--	$24	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Process Plant	C$mm	$106	$2	$52	$52	--	--	--	--	--	--	--	--	--	--	--	--	--	--
On-Site Infrastructure	C$mm	$60	$1	$20	$38	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Off-Site Infrastructure	C$mm	$1	$0	$1	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Project Indirects	C$mm	$23	$2	$1	$19	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Project Delivery	C$mm	$39	$15	$10	$13	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Owner's Costs	C$mm	$29	$5	$10	$14	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Provisions	C$mm	$56	$6	$18	$32	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Total Sustaining Capital	C$mm	$406	--	--	$5	$126	$36	$43	$43	$36	$37	$35	$23	$16	$6	--	--	--	--
Sustaining Capital Cost	C$mm	$406	--	--	$5	$126	$36	$43	$43	$36	$37	$35	$23	$16	$6	--	--	--	--
Closure Cost	C$mm	$25	--	--	--	$1	$1	$0	$0	$0	$0	$0	$0	$0	$0	$14	$5	$5	--
Salvage Value	C$mm	($25)	--	--	--	--	--	--	--	--	--	--	--	--	($25)	--	--	--	--
Total Capital Expenditures including Salvage Value	C$mm	$940	$47	$163	$328	$127	$37	$43	$43	$36	$37	$35	$23	$16	($19)	$14	$5	$5	--

Note: Cash cost, and AISC are non-IFRS performance measures. Refer to Forward-Looking Cautionary Statements.

22.5 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV and IRR of the project using the following variables: gold price, discount rate, foreign exchange, operating cost, initial capital cost, and head grade. Pre-tax sensitivity results are shown in Table 22-4 and Figure 22-2; Table 22-5 and Figure 22-3 show post-tax sensitivity results. The analysis revealed that the project is most sensitive to changes in gold price, foreign exchange, and head grade and less sensitive to discount rate, operating cost, and initial capital cost.

Table 22-4: Pre-Tax Sensitivity Analysis

Pre-Tax NPV Sensitivity to Discount Rate

Gold Price (US$/oz)

Discount Rate	$1,300	$1,400	$1,550	$1,700	$1,800
1.0%	$434	$633	$933	$1,232	$1,431
3.0%	$320	$495	$757	$1,020	$1,195
5.0%	$225	$379	$610	$842	$996
8.0%	$111	$240	$433	$626	$754
10.0%	$50	$165	$337	$509	$623

Pre-Tax IRR Sensitivity to Discount Rate

Gold Price (US$/oz)

Discount Rate	$1,300	$1,400	$1,550	$1,700	$1,800
1.0%	11.9%	16.1%	21.7%	26.7%	29.9%
3.0%	11.9%	16.1%	21.7%	26.7%	29.9%
5.0%	11.9%	16.1%	21.7%	26.7%	29.9%
8.0%	11.9%	16.1%	21.7%	26.7%	29.9%
10.0%	11.9%	16.1%	21.7%	26.7%	29.9%

Pre-Tax NPV Sensitivity to FX

Gold Price (US$/oz)

FX	$1,300	$1,400	$1,550	$1,700	$1,800
1.36	$350	$514	$760	$1,005	$1,169
1.32	$288	$447	$685	$924	$1,083
1.28	$225	$379	$610	$842	$996
1.24	$163	$312	$536	$760	$909
1.20	$100	$244	$461	$678	$822

Pre-Tax IRR Sensitivity to FX

Gold Price (US$/oz)

FX	$1,300	$1,400	$1,550	$1,700	$1,800
1.36	15.3%	19.4%	25.0%	30.1%	33.3%
1.32	13.7%	17.8%	23.3%	28.4%	31.6%
1.28	11.9%	16.1%	21.7%	26.7%	29.9%
1.24	10.1%	14.3%	19.9%	25.0%	28.1%
1.20	8.2%	12.5%	18.1%	23.2%	26.3%

Pre-Tax NPV Sensitivity to Opex

Gold Price (US$/oz)

Opex	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	$416	$571	$802	$1,033	$1,187
(10.0%)	$321	$475	$706	$937	$1,091
--	$225	$379	$610	$842	$996
10.0%	$129	$284	$515	$746	$900
20.0%	$34	$188	$419	$650	$805

Pre-Tax IRR Sensitivity to Opex

Gold Price (US$/oz)

Opex	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	17.0%	20.7%	25.9%	30.6%	33.6%
(10.0%)	14.5%	18.4%	23.8%	28.7%	31.8%
--	11.9%	16.1%	21.7%	26.7%	29.9%
10.0%	9.2%	13.6%	19.4%	24.7%	27.9%
20.0%	6.1%	10.9%	17.1%	22.6%	26.0%

Pre-Tax NPV Sensitivity to Initial Capex

Gold Price (US$/oz)

Initial Capex	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	$326	$480	$711	$943	$1,097
(10.0%)	$276	$430	$661	$892	$1,046
--	$225	$379	$610	$842	$996
10.0%	$175	$329	$560	$791	$945
20.0%	$124	$278	$510	$741	$895

Pre-Tax IRR Sensitivity to Initial Capex

Gold Price (US$/oz)

Initial Capex	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	16.5%	21.0%	27.2%	32.8%	36.4%
(10.0%)	14.0%	18.4%	24.2%	29.5%	32.9%
--	11.9%	16.1%	21.7%	26.7%	29.9%
10.0%	10.1%	14.0%	19.4%	24.3%	27.3%
20.0%	8.4%	12.3%	17.4%	22.1%	25.0%

Pre-Tax NPV Sensitivity to Head Grade

Gold Price (US$/oz)

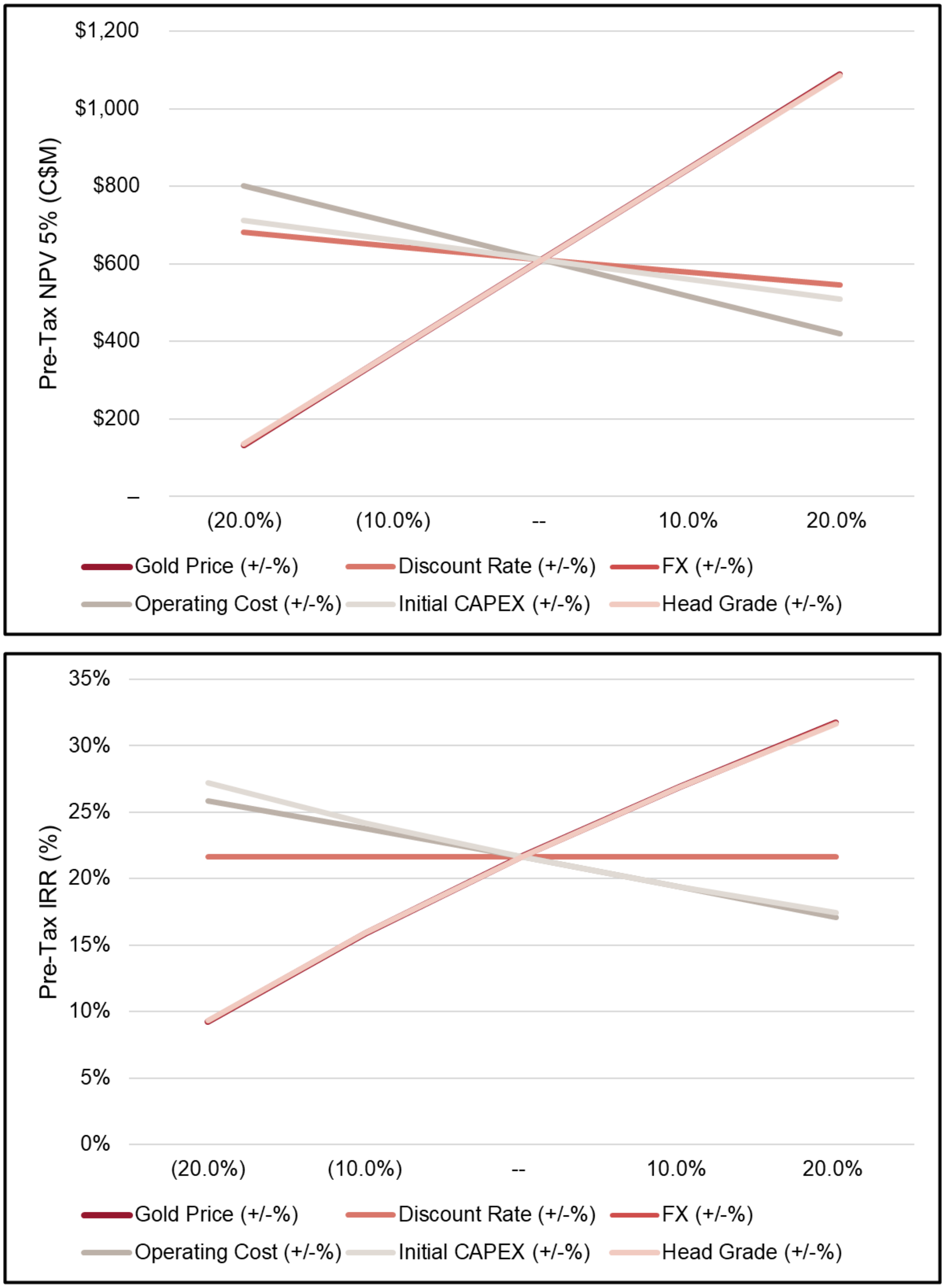
Head Grade	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	($174)	($50)	$135	$320	$444
(10.0%)	$26	$165	$373	$581	$720
--	$225	$379	$610	$842	$996
10.0%	$424	$594	$848	$1,102	$1,272
20.0%	$624	$809	$1,086	$1,363	$1,548

Pre-Tax IRR Sensitivity to Head Grade

Gold Price (US$/oz)

Head Grade	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	-%	3.2%	9.3%	14.5%	17.7%
(10.0%)	5.9%	10.2%	15.9%	21.0%	24.1%
--	11.9%	16.1%	21.7%	26.7%	29.9%
10.0%	17.2%	21.3%	26.8%	32.0%	35.2%
20.0%	21.9%	26.0%	31.6%	36.8%	40.1%

Figure 22-2: Pre-Tax NPV & IRR Sensitivity Results



Source: Ausenco, 2021.

Table 22-5: Post-Tax Sensitivity Analysis

Post-Tax NPV Sensitivity to Discount Rate — Gold Price (US$/oz)

Discount Rate	$1,300	$1,400	$1,550	$1,700	$1,800
1.0%	$238	$360	$540	$718	$835
3.0%	$157	$265	$423	$580	$683
5.0%	$89	$185	$326	$465	$556
8.0%	$8	$88	$207	$324	$401
10.0%	($36)	$36	$143	$248	$317

Post-Tax IRR Sensitivity to Discount Rate — Gold Price (US$/oz)

Discount Rate	$1,300	$1,400	$1,550	$1,700	$1,800
1.0%	8.3%	11.6%	16.1%	20.2%	22.8%
3.0%	8.3%	11.6%	16.1%	20.2%	22.8%
5.0%	8.3%	11.6%	16.1%	20.2%	22.8%
8.0%	8.3%	11.6%	16.1%	20.2%	22.8%
10.0%	8.3%	11.6%	16.1%	20.2%	22.8%

Post-Tax NPV Sensitivity to FX — Gold Price (US$/oz)

FX	$1,300	$1,400	$1,550	$1,700	$1,800
1.36	$167	$267	$416	$561	$657
1.32	$128	$226	$371	$513	$607
1.28	$89	$185	$326	$465	$556
1.24	$50	$143	$280	$416	$505
1.20	$11	$101	$235	$366	$453

Post-Tax IRR Sensitivity to FX — Gold Price (US$/oz)

FX	$1,300	$1,400	$1,550	$1,700	$1,800
1.36	11.0%	14.3%	18.8%	22.9%	25.5%
1.32	9.7%	13.0%	17.5%	21.6%	24.2%
1.28	8.3%	11.6%	16.1%	20.2%	22.8%
1.24	6.9%	10.2%	14.7%	18.8%	21.4%
1.20	5.4%	8.8%	13.3%	17.4%	19.9%

Post-Tax NPV Sensitivity to Opex — Gold Price (US$/oz)

Opex	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	$206	$300	$437	$573	$663
(10.0%)	$148	$243	$382	$519	$610
--	$89	$185	$326	$465	$556
10.0%	$29	$126	$268	$408	$501
20.0%	($32)	$66	$210	$351	$444

Post-Tax IRR Sensitivity to Opex — Gold Price (US$/oz)

Opex	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	12.4%	15.4%	19.5%	23.3%	25.7%
(10.0%)	10.4%	13.5%	17.9%	21.8%	24.3%
--	8.3%	11.6%	16.1%	20.2%	22.8%
10.0%	6.1%	9.6%	14.4%	18.6%	21.2%
20.0%	3.7%	7.5%	12.5%	16.9%	19.6%

Post-Tax NPV Sensitivity to Initial Capex — Gold Price (US$/oz)

Initial Capex	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	$190	$286	$426	$565	$657
(10.0%)	$140	$235	$376	$515	$606
--	$89	$185	$326	$465	$556
10.0%	$39	$134	$275	$414	$505
20.0%	($12)	$84	$225	$364	$455

Post-Tax IRR Sensitivity to Initial Capex — Gold Price (US$/oz)

Initial Capex	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	13.2%	16.9%	22.0%	26.6%	29.5%
(10.0%)	10.6%	14.1%	18.8%	23.1%	25.8%
--	8.3%	11.6%	16.1%	20.2%	22.8%
10.0%	6.4%	9.5%	13.8%	17.7%	20.1%
20.0%	4.6%	7.7%	11.8%	15.5%	17.8%

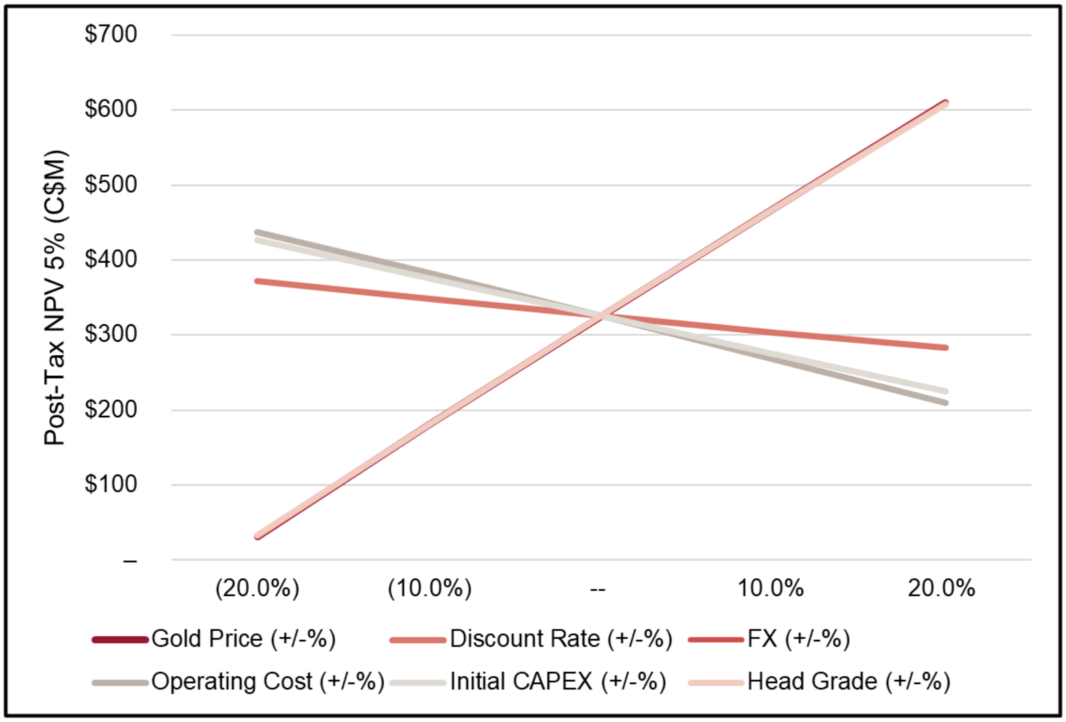
Post-Tax NPV Sensitivity to Head Grade — Gold Price (US$/oz)

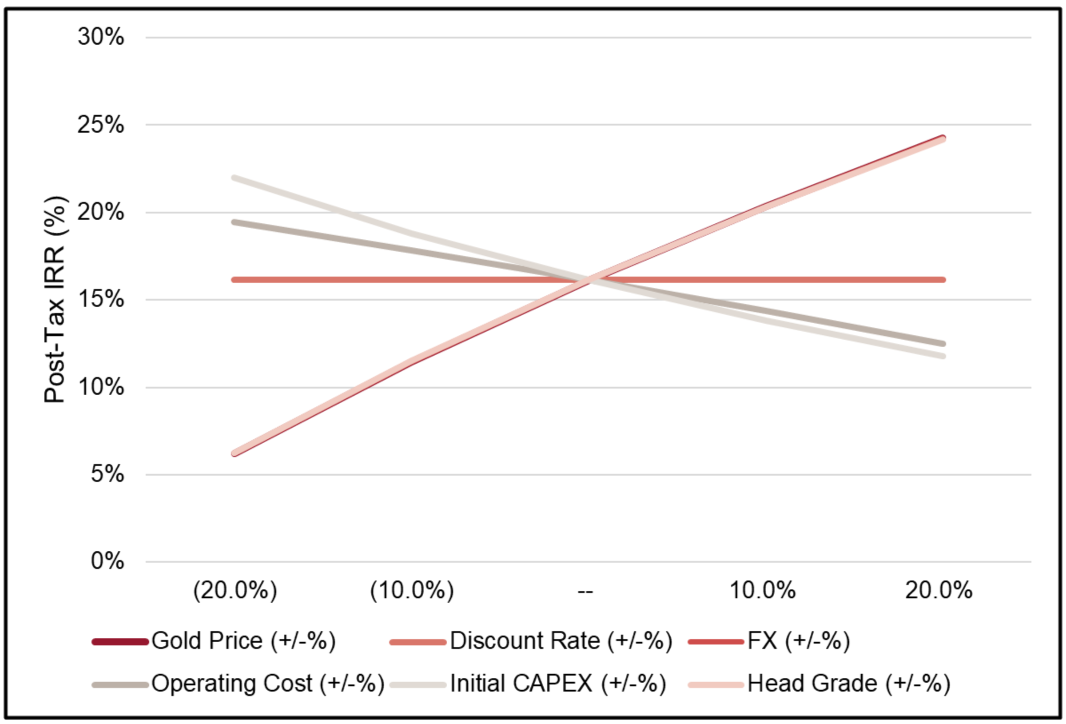
Head Grade	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	($165)	($85)	$33	$149	$225
(10.0%)	($36)	$51	$181	$308	$392
--	$89	$185	$326	$465	$556
10.0%	$212	$315	$468	$618	$717
20.0%	$333	$445	$608	$770	$877

Post-Tax IRR Sensitivity to Head Grade — Gold Price (US$/oz)

Head Grade	$1,300	$1,400	$1,550	$1,700	$1,800
(20.0%)	-%	1.6%	6.3%	10.4%	13.0%
(10.0%)	3.6%	6.9%	11.5%	15.6%	18.1%
--	8.3%	11.6%	16.1%	20.2%	22.8%
10.0%	12.5%	15.8%	20.3%	24.4%	27.0%
20.0%	16.4%	19.6%	24.2%	28.4%	31.0%

Figure 22-3: Post-Tax NPV & IRR Sensitivity Results





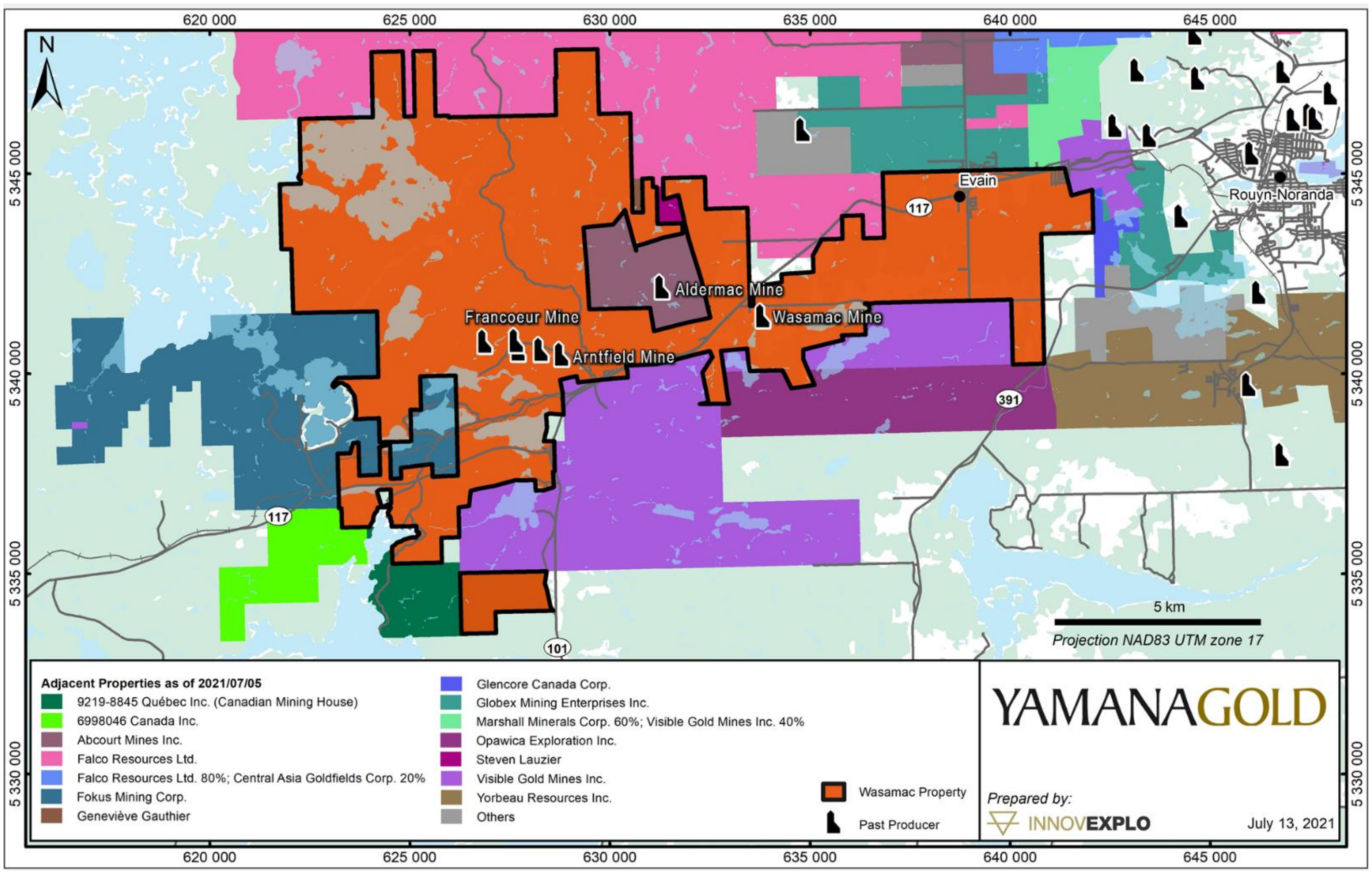
Source: Ausenco, 2021.

23 ADJACENT PROPERTIES

Presented below are the adjacent properties that are significant and of interest to the Wasamac Project. Yamana maintains a significant land position in the Rouyn-Noranda Mining District (see Figure 23-1).

The QP has not verified mineral resource estimate or published geological information pertaining to other adjacent properties. The information about mineralization on adjacent properties is not necessarily indicative of mineralization on the property.

Figure 23-1: Adjacent Properties to the Property



23.1 Aldermac Mine

The past producing Aldermac copper mine occurs in between the historical Wasamac and Francoeur mines to the north. The property is currently owned by Abcourt Mines Inc.

The following is taken from Barrett et al. (1991):

> "The original Aldermac mine near Noranda contained several Cu–Zn massive sulphide lenses hosted by felsic to mafic volcanic rocks of the late Archean Blake River Group. The original Nos. 3–6 orebodies, which consisted of massive pyrite, with lesser magnetite, pyrrhotite, chalcopyrite, and sphalerite, contained 1.87Mt of Cu-Zn ore that averaged 1.47% Cu (Zn was not recovered). The orebodies occurred within felsic breccias and tuffs up to 10 m thick that are stratigraphically overlain by an extensive dome of mainly massive rhyolite and rhyodacite (up to 250 m thick and at least 550 m across). Most of the volcanic rocks that laterally flank and overlie the felsic dome are dacitic to andesitic flows, breccia, and tuff, with minor rhyolites, and associated subvolcanic sills of quartz-feldspar porphyry and gabbro.
>
> The new massive sulphide deposit, discovered in 1988, lies 150 to 200 m east of the mined-out orebodies, at a similar stratigraphic level within altered felsic breccia and tuff. The sulphides are mainly in the No. 8 lens, which contains 1.0Mt at an average grade of 1.54% Cu, 4.12% Zn, 31.2 g/t Ag, and 0.48 g/t Au".

This estimate is considered to be historical in nature and should not be relied upon.

23.2 Wasa Creek

The Wasa Creek property of Visible Gold Mines Inc. is less than 5 km south of the Wasamac deposit. The mineralization consists of finely disseminated pyrite in quartz veins, silicified zones and in-sheared zones hosted in basalts and andesitic basalts. Locally, gold is visible in places in the mineralized and silicified zones.

In 2011, 13 drill holes over 8,820 metres were drilled by Visible Gold on claims south and west of the Wasamac deposit. Twelve drill holes were drilled on their Wasa Creek property, targeting the Cadillac fault or parallel structures, and one drill hole was drilled on a claim adjacent to the property over the Wasa lake. The best results were 21.75 g/t Au over 4.1 m in drill hole WC-12-01 and 3.45 g/t Au over 5.95 m and 3.22 g/t Au over 7.5 m in drill hole WC-12-05. A follow-up drilling campaign was carried out in 2012 with several gold intervals with more than 1 g/t Au.

23.3 Galloway Project

The property is located 30 km west of Rouyn-Noranda. Historical drilling on the Galloway occurrence showed multiple drill intersections of 50 to 100+ metres over 1 g/t Au. A magmatic hydrothermal system with lode and disseminated Cu-Au-Mo mineralization intersecting volcanic rocks and a porphyry syenite intrusion. On September 5, 2012, Vantex Resources, the previous owner of the property, publicly disclosed a mineral resource estimation for the Galloway Pitchvein deposit (now considered historical: MacInnis et al., 2012). Fokus Mining Corp. is the operator of the Galloway Project.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution and Organization

This Project Execution Plan (PEP) is the governing document that establishes the means to execute, monitor, and control the Wasamac project. The plan serves as the main communication tool to ensure the project team understands project objectives and how they will be accomplished.

The PEP includes, but is not limited to, the following:

- an overview of the project
- the scope of work and services for the underground mine development and major facilities
- the project schedule with key activities and target dates identified
- an organizational chart

The PEP will be supported by the following sub-plans:

- Health, Safety and Environment Management Plan
- Engineering Execution Plan
- Procurement Strategy and Management Plan
- Contracting Strategy and Management Execution Plan
- Construction Execution Plan
- Commissioning Execution Plan
- Project Controls Plan
- Project Quality Plan
- Risk Management Plan
- Logistics and Materials Management Plan
- Site Requirements for Construction
- Commercial Management Plan

24.1.1 Objectives

Yamana aims to bring the Wasamac mine into operation while satisfying the following objectives:

- zero harm to personnel involved with construction, operation, and maintenance of the facilities, and 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
- maintain positive community relations and consciously design the project to reduce the impact to local surrounding neighbourhoods
- target a reduced carbon footprint by application of automation and technology, also geared to increase operational safety

24.1.2 Execution Strategy

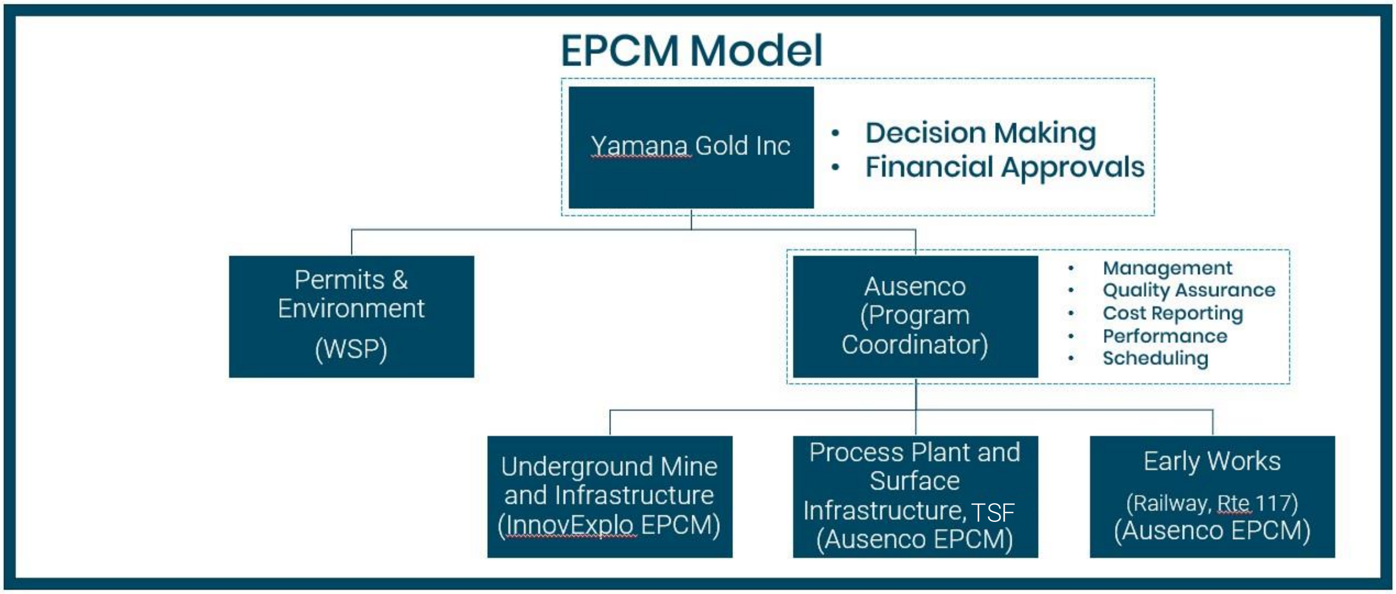
Three EPCM contracts will be employed to deliver the detailed engineering and execution phases of the project with the structure displayed in Figure 24-1. The three contracts are described as follows:

1. EPCM contract led by Ausenco for early works that involves the expansion of Highway 117 for deceleration lanes and upgrades to the railway crossings at the main entry, as well as the secondary access to the tailings management facility
2. EPCM contract led by Ausenco, that encompasses the process plant, filtration plant, TSF, and on-site infrastructure
3. EPCM contract led by InnovExplo that generally encompasses the development of the underground mine and underground infrastructure

Ausenco will assist as program coordinator as in the overarching management of the three EPCM contracts. The coordinator role involves project-wide management, quality assurance, cost reporting, performance monitoring and scheduling activities. All decision-making and financial approvals will be performed by Yamana.

Yamana is performing or managing the project scope related to permitting, environmental work, and connection to the Hydro Quebec transmission line.

Figure 24-1: General Execution Structure and EPCM Model



24.1.3 EPCM Delivery Strategy

The EPCM delivery strategy is summarized as follows:

- Engineering and design for construction will be completed by contract leads. For the process plant and surface infrastructure EPCM, as well as early works, EPCM Ausenco will work from the GTA office and a satellite office in Rouyn-Noranda, with support from global subject matter experts (SMEs) as needed. The underground mine EPCM work will be performed by InnovExplo in Rouyn-Noranda.
- Procurement of equipment and services, expediting and contract management will be performed by Yamana. Ausenco and InnovExplo will advise Yamana on vendor and contractor selection through the development of contracting strategies, production of specification and contractor packages, and by performing technical and commercial bid evaluations. Yamana will perform commercial management following package awards.
- Ausenco and InnovExplo will provide technical supervision and support on site as required. Site teams will report to the Ausenco program coordinator, who will report to Yamana's project management personnel. The construction team will be based in Rouyn-Noranda; the construction team will be supported from home offices for cost, schedule and progress reporting, and invoice processing and payment.

24.1.4 Project Organization

The project team is organized based on an integrated team approach, minimizing the duplication of roles and activities between the Owner's team and their major delivery partners. The organization for the engineering and procurement phase and the construction phase is depicted in Figure 24-2 and Figure 24-3, respectively.

Figure 24-2: Engineering and Procurement Phase Organization Chart

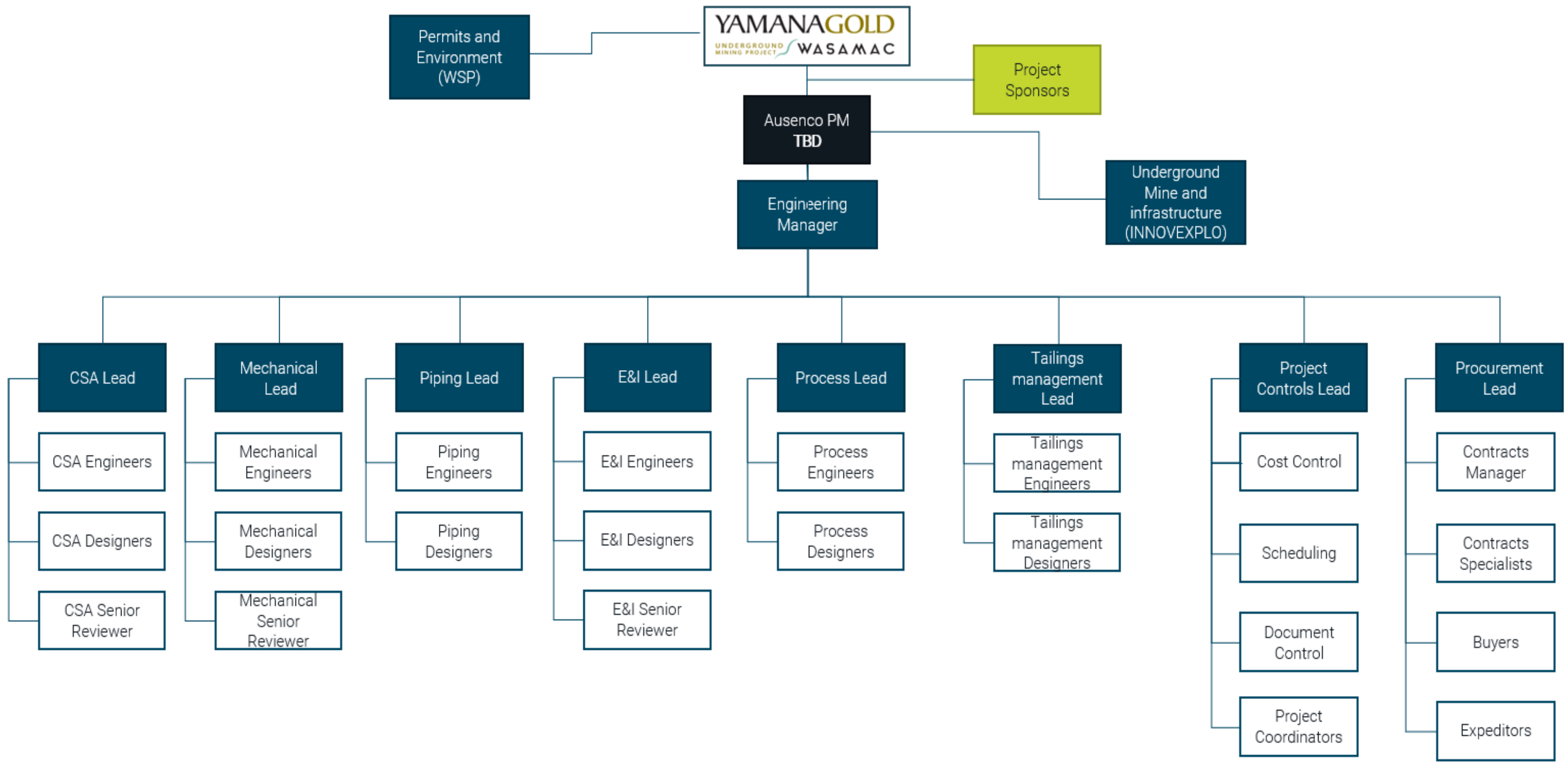
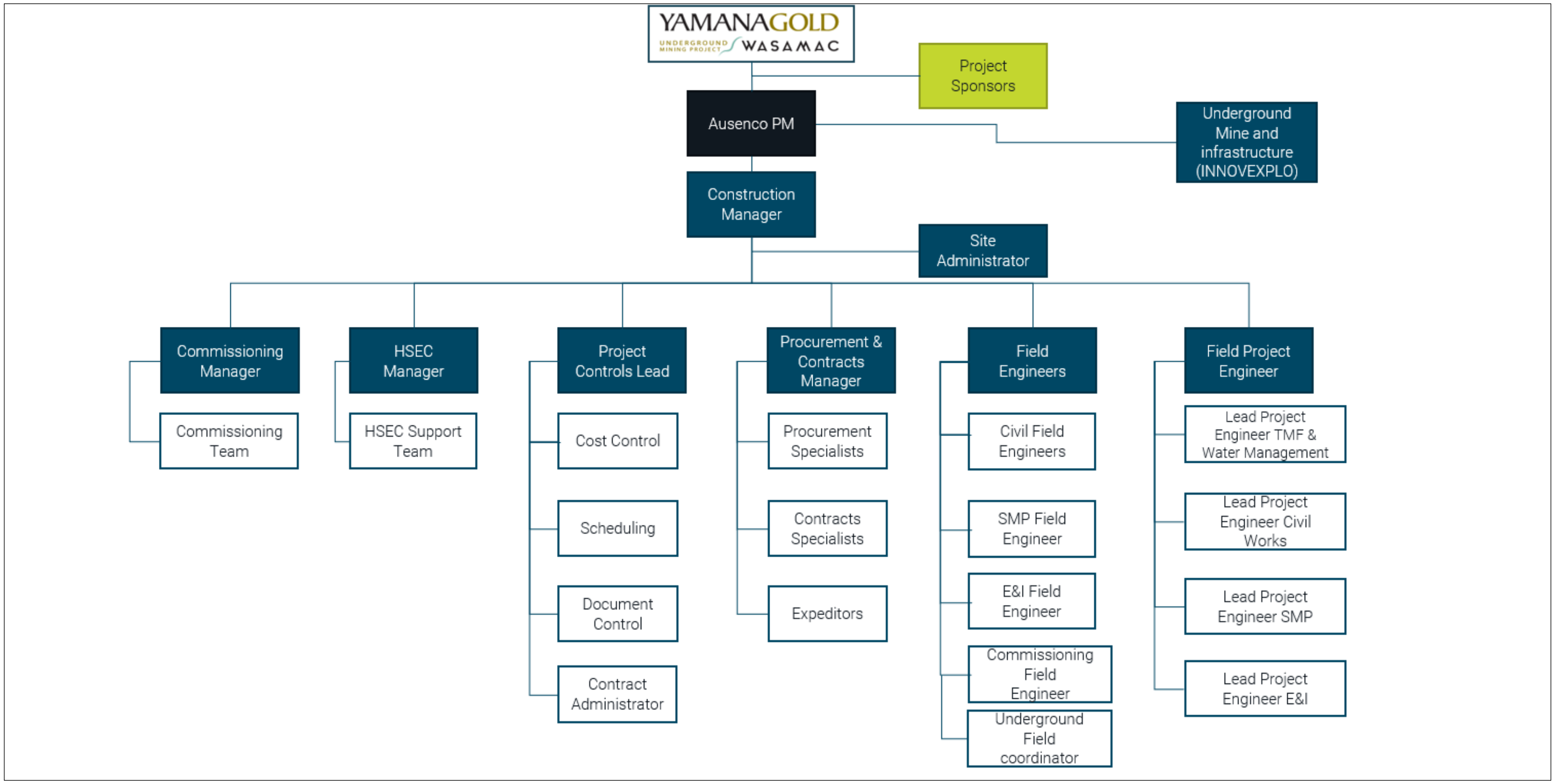


Figure 24-3: Construction Phase Organization Chart



24.1.5 Construction Execution Strategy

24.1.5.1 Construction Sequencing

This section outlines the high-level execution sequencing constraints that were evaluated to determine the execution schedule baseline for the feasibility study. An overall master execution schedule is provided in Section 24.1.5.

Early works are anticipated to commence in Q4 2022. The three early work scope areas include the following:

- permitting of Highway 117 expansion with the Ministry of Transport Quebec (MTQ)
- detailed engineering, design, and execution of the highway expansion
- detailed engineering, design, and execution of the railway upgrades

These early works activities will all be completed prior to first mobilization to site for the other works, planned for August 2024. This date is predicated on Yamana filing and receiving the appropriate environmental/construction permits to allow breaking ground to occur. No site works—including mobilization and staging equipment on site, early site preparations and stockpiling of construction materials—will progress until these permits are acquired. If required, local townships can be utilized to stage equipment away from the project property.

Once the permits are acquired, the civil works contractors will mobilize first to carry out clearing and grubbing of the main access road and specific site works boundaries. As the clearing and grubbing activities continue, the heavy civil work will follow to strip the topsoil and organics and stockpile it in designated areas for future remediation works. Temporary water management catchments and ditches will also be developed as the civil works continue for process plant pad development, filtration plant pad development, waste rock storage facility pad development, and development of the access road to the TSF and initial TSF footprint.

After completion of early civil works, there will be three main work fronts on the project property, as follows:

- The mining works will continue ramp development, generating and stockpiling waste rock material that will be crushed/screened via a contract crushing/screening plant and used for construction materials.
- The TSF works will place and compact hauled waste rock to raise the pond walls, finishing with crushed/screened material. The geomembrane liner will be installed.
- Process plant works will begin concrete work in spring 2025 to build/install major equipment foundations. Construction will be continuous until commissioning activities begin in Q4 2026 prior to first gold in December 2026.

24.1.5.2 Winter Construction

Project construction will continue through the winters of 2024, 2025, and 2026. To mitigate downtime and loss of productivity, the following considerations were included in the execution schedule:

- Bulk earthworks activities will not be carried out during winter months. Communications with several earthworks contractors in the area indicated the reduced availability and significantly higher costs for earthworks activities in the winter.
- Concrete works are scheduled to be performed during summer months. The construction sequence is such that pre-engineered buildings will be fully constructed and cladded prior to the winter seasons. This will allow installation to continue within the buildings, sheltered from any inclement weather.

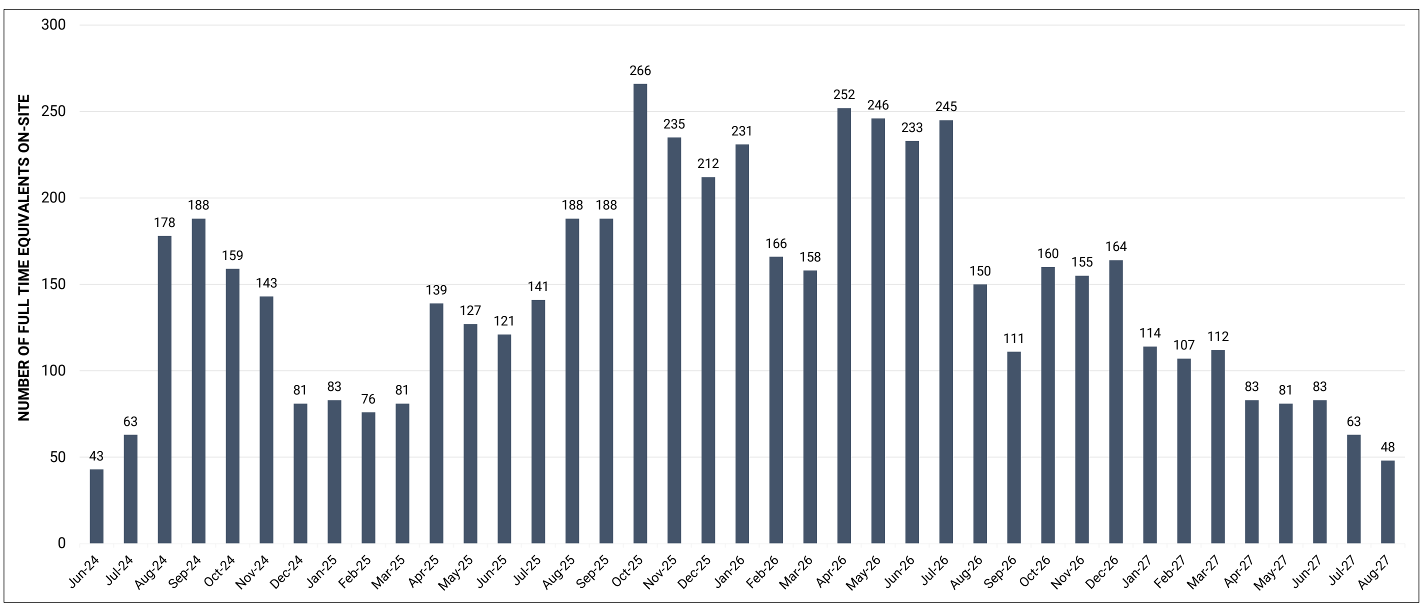
24.1.5.3 COVID-19 Considerations

As project construction is expected to begin in Q3 2024, complications in the working environment as a result of Covid-19 are expected to be minimal given current immunization rates. Following development of the safety plan, a mandatory proof of vaccination may be embedded in the contracts as a requirement for any worker entering the site. Necessary protocol and procedures shall be included in the site safety policy plan.

24.1.5.4 Construction Staffing

A labour loading forecast was developed for the construction phase of the Wasamac project (see Figure 24-4). The construction labour forecast was developed utilizing the labour hours that were received from the contractors that provided budgetary pricing for the feasibility study and tallied on full-time equivalents, as well as construction support teams from the EPCM contract leads and the Owner's team.

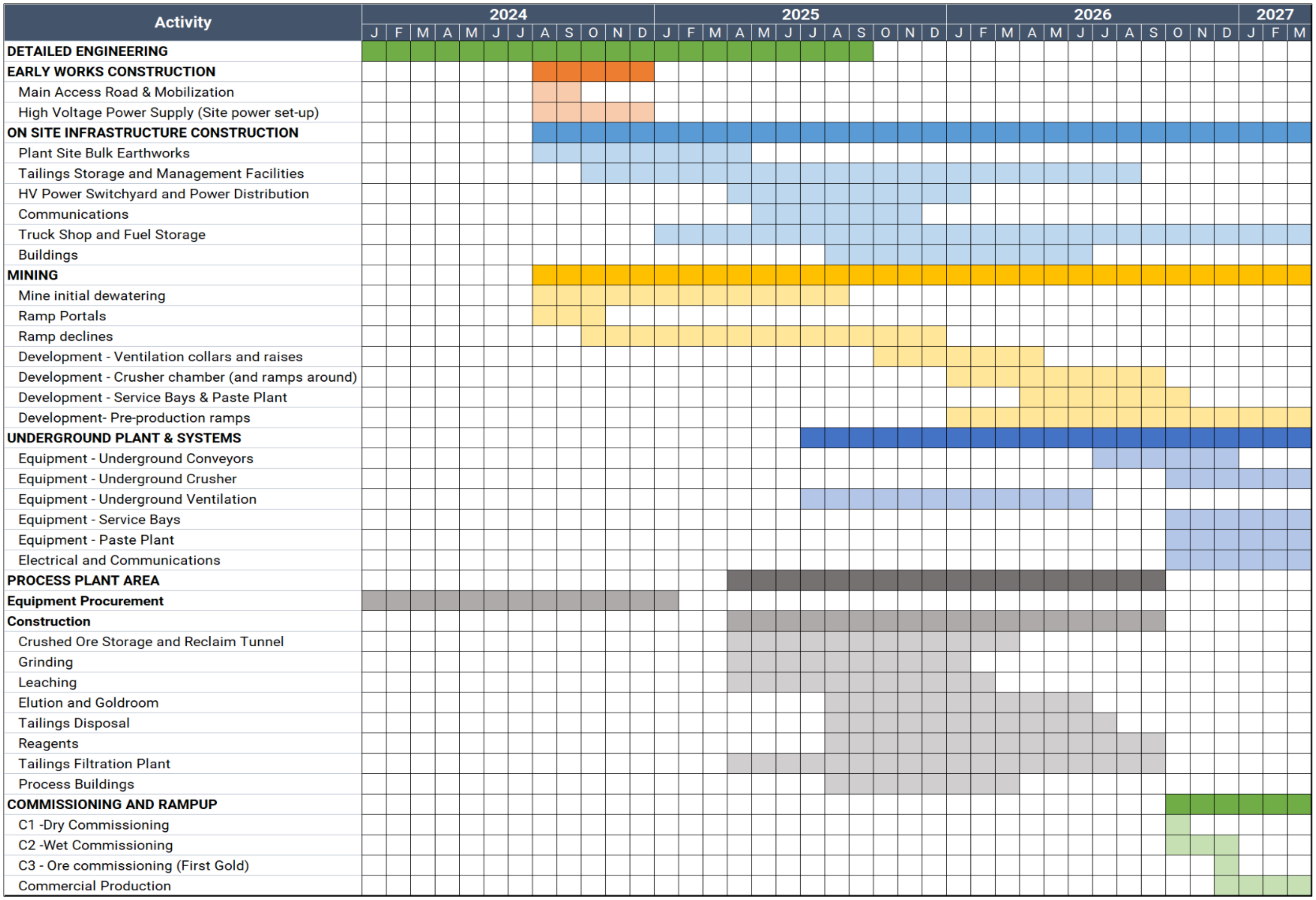
Figure 24-4: Number of Full-Time Personnel On-Site during Construction Period



24.1.6 Project Execution Schedule

The project execution schedule developed for the feasibility study is presented in Figure 24-5.

Figure 24-5: Project Execution Schedule



25 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QPs have provided the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this report.

25.2 Property Description & Location

The Wasamac property is located in the Abitibi-Témiscamingue administrative region of the Province of Québec, Canada, approximately 15 km west-southwest of the city of Rouyn-Noranda. The property covers an area of 10,268.56 ha, extending 20 km east-west and 15 km north-south, and underlies parts of Beauchastel and Dasserat township.

The property is subdivided into six claim blocks: the Wasamac Block, Wasamac NE Block, Teck JV Block, R.M. Nickel Block, Consolidated Francoeur Block, and the Western Buff Block, which together comprise 6 mining concessions, 281 mineral claims and 5 mining leases, for a total of 292 mineral titles.

The Issuer holds 100% ownership of the mineral titles for the property, except for the Teck JV Block, in which the Issuer holds 60% ownership of five claims.

25.3 Data Verification and Mineral Resources

The deposit is continuous for over 900 m vertically and 2.7 km along strike, and remains open at depth and along its projected lateral extensions. Gold mineralization is typically associated with finely disseminated pyrite and stockwork of pyrite-rich micro-veinlets hosted in albite-sericite-ankerite alteration zones confined within the Francoeur-Wasa shear zone, a subsidiary fault of the Cadillac–Larder Lake fault zone.

The QPs from InnovExplo conducted a site visit that included a review and validation of the data used for the mineral resource estimate. They also validated the geology and mineralization associated with the deposit.

Yamana updated the mineralization model supported by the alteration assemblages and by the interpreted units and structures of the lithological model, which are based on the available geological and analytical information in the database.

The QPs believe that the information presented in this report provides a fair and accurate picture of the deposit's potential. The QPs conclude the following:

- The database supporting the mineral resource and mineral reserve estimates is complete, valid, and up to date.
- The geological and grade continuity of the deposit has been demonstrated
- The mineral resource model is primarily based on significant changes made to the 3D geological model (modified mineralized zones) supported by new geostatistical analysis, new interpolation strategy, new assumptions for mineral resources classification, and the creation of a potentially mineable shape to constrain the mineral resource estimate.
- For an underground scenario, the deposit contains, exclusive of the mineral reserves, an estimated indicated mineral resource of 5.769 kt grading 1.76 g/t Au for 326 koz of gold and an estimated inferred mineral resource of 3.984 kt grading 2.01 g/t Au for 258 koz of gold.

- It is likely that with the ongoing exploration and infill drilling campaign by Yamana, drilling at depth, to the east, to the west, or within gaps will increase the inferred mineral resource tonnage and upgrade some of the inferred mineral resources to the indicated category.
- In addition to the direct potential extensions of the current deposit, the entire property is characterized by additional exploration potential supported by the presence of known mineral occurrences and past producers.

25.4 Mineral Reserves Estimate

Mineral reserves were classified in compliance with CIM's "Definition Standards for Mineral Resources and Mineral Reserves". Mineral reserves for the deposit incorporate dilution and mining recovery factors based on the selected mining method and design. Mineral reserves comprise the estimated tonnage and grade of ore that is considered economically viable for extraction.

The mineral resource block model update was used as the basis for estimating the mineable tonnage considered in the mine plan. Once cut-off grades for the assorted mining areas were estimated, the stope shapes were optimized based on various parameters, such as geometry and dilution. The final mineral reserve estimate was obtained after completing the stope and underground mine designs, economic validation, and considering additional factors, such as mining recovery.

The cut-off grade calculations were based on parameters from benchmarks as well as Yamana and InnovExplo estimates. Due to the variation in metallurgical recoveries for different parts of the deposit, four cut-off grades were used for stope optimization designs.

Internal dilution was considered when optimizing stope shapes and converting them into planned mineable stope shapes. External dilution was also considered during stope optimization by using appropriate ELOS values (see Chapter 16) based on stope size, location, and rock mechanics properties. Backfill dilution was added afterwards, based on the location of each stope and the mining sequence. The external waste dilution is estimated to be 11%. When considering the backfill dilution, the average dilution of the project is estimated to be 13%.

The estimated mining recoveries for the project range from 86% to 95%. Recovery varies mainly according to blasting method and the associated challenges, as well as rock mechanics conditions, such as sill pillar recovery. The average mining recovery for the project is 93.6%.

Table 25-1 summarizes the total mineral reserves estimated for the project. The tonnes and grades shown have the appropriate mining factors applied (i.e., mine dilution and recovery).

Table 25-1: Wasamac Estimate of Mineral Reserves as of June 30, 2021

Category	Tonnage (kt)	Grade Au (g/t)	Contained Gold (koz)
Proven	---	---	---
Probable	23,168	2.56	1,910
Total Proven & Probable	23,168	2.56	1,910

Notes: 1. The QPs for the mineral reserve estimate are Mr. Denis Gourde, P.Eng. and Sébastien Tanguay, P.Eng. (InnovExplo). The mineral reserve estimate conforms to the 2014 CIM Definition Standards on Mineral Resources and Reserves and follows 2019 CIM definitions and guidelines. 2. Mineral reserve estimate has an effective date of June 30, 2021. 3. The metallurgical recoveries varies with the metallurgical domain: 92.0% for the eastern part of the main zone; 81.6% for the central part of the main zone; 86.2% for the western part of the main zone; 92.7% for zones 3 and 4. 4. Estimated at US$1,250/oz Au using an exchange rate of US$1.32:C$1.00, variable cut-off value related to the metallurgical domain: 1.45 g/t for the eastern part of the main zone; 1.68 g/t Au for the central part of the main zone; 1.63 g/t for the western part of the main zone; 1.62 g/t for zones 3 and 4. 5. Mineral reserve tonnage and mined metal have been rounded to reflect the accuracy of the estimate and numbers may not add due to rounding. 6. The total refining, processing, mining, tailing management and general and administration cost is estimated at 63.21$/t. 7. Mineral reserves include both internal and external dilution. The internal mining dilution varies with the metallurgical domain: 11.9% for the eastern part of the main zone; 15.1% for the central part of the main zone; 17.4% for the western part of the main zone; 25.9% for zones 3 and 4. The external dilution is estimated to be 11%. the average dilution of the project is estimated to be 13%. 8. The estimated mining recoveries for the site range from 86% for the sills to 95% for the stopes. The average mining recovery factor was set at 93.6% to account for mineralized material left in each block in the margins of the deposit. For the purpose of the COG calculation, mine recovery was set at 95%. 9. The qualified person responsible for this section of the technical report is not aware of any mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the mineral reserve estimate.

25.5 Mining Method and Mine Plan

The Wasamac Project will be an underground operation with a proposed mining method optimized to the deposit's geometry and employing both longitudinal and transverse long-hole stoping. The project is planned mainly beneath and east of the historical Wasamac mine workings. Geomechanical engineering design criteria were developed to support the underground mining production sequence and design.

To access the deposit, two 3-kilometre-long ramps will be developed; one for material handling and the second for personnel and equipment. These two ramps are designed to be used as emergency exits.

Levels are to be spaced every 25 m and connected by decline ramps. Levels include all the necessary infrastructures required for large-scale mechanical long-hole stoping. Two main levels are designed to be used as centres of operation for major infrastructures (e.g., like the service bay and main hub); they also provide connecting access ramps between all mining areas.

The main hub is to be located between L425 and L500. The full rockwork includes a rockbreaker at the grizzly level (including two dumping points), a 7500-tonne circular ore bin, a crushing station, and conveyor transfers. From the underground crusher, ore will be transported to the crushed ore stockpile on the surface using a conventional, two-segment conveyor system. The waste material will be hauled by the 63-tonne trucks to the closest backfilling activities, or to a waste pad at surface via the service ramp. The haul trucks will be automated to allow haulage to continue between shifts. Modern equipment and methodologies will be used to optimize the drill and blast patterns and to minimize blast vibration and where possible mining equipment will be operated remotely form surface.

Increased mineral reserves, the reduction in development metres, and the optimized materials handling system will allow a 7,000 t/d production level to be sustained. Wide stopes (typically 10 to 15 m), shallow depth, underground conveyor system, and adoption of modern technology are expected to establish the project as a low-cost underground mining operation. The underground mine design includes two raises to surface to provide fresh air to the mine. All stope without a few exceptions will be backfilled, and the paste plant will be localized underground.

The first ore is scheduled to be mined during the last trimester of 2026. The commercial production period is slated to begin in October 2027, when the mine reaches 7,000 t/d of production. To achieve this target date, an average of 8,700 m of horizontal development per year, with a maximum of 15,800 m in 2027, will be necessary. During the pre-production period, major infrastructures like the services bay, underground crusher, paste plant and main raises are to be excavated, and all associated equipment installed and commissioned.

The life-of-mine plan specifies a rapid ramp-up in production in the first year, with production rising to approximately 200,000 ounces after mill recovery per year for the subsequent four years. Average gold production is expected to be 169,000 ounces after mill recovery per year over the life of mine of 10 years. Based on current mineral reserves, Wasamac has a mine life to October 2036, but potential conversion of mineral resources and exploration potential could possibly extend the mine life.

A maximum of 71 mobile pieces of equipment will be bought by Yamana and used underground (including seven 60-tonne trucks, ten production LHDs, four jumbos and three production drills). Up to 307 personnel will be employed by Yamana to work underground in various departments (e.g., operations (216 persons); maintenance (66 persons); and technical services (25 persons).

25.6 Metallurgical Testwork

Historical testwork data was heavily relied upon in this feasibility study update. Additional testwork was completed on whole ore leach, oxygen uptake, flotation, and SMC test feed size analysis to validate historic testwork data and conclusions.

Leaching testwork showed that a leach residence time of 35 hours should be sufficient for the Wasamac ore, compared to 48 hours in the 2018 Feasibility Study. There may be scope to further reduce the 35-hour residence time with confirmation through additional testwork. SMC test results used in this study were adjusted by JKTech to account for different test feed sizes. This adjustment allowed for an increased Axb value of 39.3 to be used in the evaluation compared to 32.2 in the 2018 feasibility study, indicating a less conservative design and decrease in the cost of the SAG mill.

Flotation flowsheet testing confirmed that additional gold recovery was obtained for the Z1 and Z2 samples compared to whole ore leach processing. No improvement was seen in the ZP sample. The increase in recoveries were not likely high enough to offset associated higher capital and operating costs.

Oxygen uptake testing showed the samples to have moderate to high oxygen demand. However, additional testwork is needed on a wider range of samples to confirm oxygen supply requirements.

25.7 Recovery Methods

The process flowsheet follows a conventional primary crushing, SAG + ball mill circuit followed whole ore leach, carbon adsorption, stripping, and regeneration, cyanide destruction, tailings dewatering, and tailings filtration. The flowsheet was selected for a design throughput of 7,500 t/d (2.7Mt/a), with a nominal throughput of 7,000 t/d.

Compared to the 2018 feasibility study, the following changes to the flowsheet have been implemented:

- the primary crusher has been moved underground for noise and dust mitigation for the surrounding residents
- the quantity of leach tanks has decreased from five to three
- the eight pumpcell CIP tanks have been decreased to seven conventional CIP tanks
- the pre-detox thickener has been removed
- the quantity of detox tanks has decreased from two to one

In the next stage of study, thickening and tailings filtration testing should be vendor specific for possible equipment selection with process guarantees. Additional testwork should be completed to determine the silver content of the ore, as increased silver grades may decrease the efficiency of the CIP and carbon elution circuits.

25.8 Site Infrastructure

The main site infrastructure consists of an underground mine, a dry-stack TSF, a waste rock storage facility, two contact water ponds, access roads to the main plant site and to the TSF, a truck shop, a paste backfill plant, and an effluent treatment plant. The mine site areas will be fenced, and the main plant access area will be gated for security.

The extent of the plant pad is retracted in the east to reduce blasting costs of the shallow bedrock in this area and retain the outcrop as a natural sound barrier. The filtration plant pad is optimized to reduce disturbance to the natural landscape and incur cost savings in earthworks.

The waste rock storage facility (WRSF) is designed to store 1.83Mt of waste rock and strategic phasing of lifts allows for progressive reclamation as early as Year -1 of mining. A light/heavy traffic security system will be installed at the entry to the WRSF and site access to the processing plant pad to provide security for the cross traffic between the underground mining trucks and light vehicles. Access to the WRSF at this location allows for isolation of light and heavy vehicles, improving site safety.

The water balance for plant processes is optimized to source as little freshwater as possible for make-up water. The mill basin is reduced in size by reporting the underground mine waters (following removal of sediment) directly to the process plant instead of the mill basin.

25.9 Environmental, Social and Permitting

There are sufficient tailings samples to adequately characterize the acidification risk of the tailings as 'low'; however, depending on the tailings material distribution in situ and local environmental conditions, there may be variable responses observed.

Yamana is currently preparing a conceptual closure plan and cost estimate for the Wasamac Project, concurrently with the EIA submissions. The conceptual closure plan will meet the requirements of the Guide and applicable legislation. The objective of the closure plan will be to return the site to a satisfactory condition by:

- eliminating unacceptable health hazards and ensuring public safety
- limiting the production and spread of contaminants that could damage the receiving environment and, in the long term, aiming to eliminate all forms of maintenance and monitoring
- returning the site to a condition in which it is visually acceptable (reclamation)
- returning the infrastructure areas (excluding the tailings impoundment and waste rock piles) to a state that is compatible with future use (rehabilitation)

The Physical Activities Regulations (also known as the Project List) identifies types of projects that may require a federal impact assessment (IA) under the *Impact Assessment Act, 2019* (IAA). The ore production capacity will be above 5,000 t/d, estimated at 7,500 t/d. When the physical activity associated with the carrying out of a proponent's project is described in the Physical Activities Regulations, the proponent must provide the Agency with an Initial Project Description. The project entered the planning phase with the submission of an Initial Project Description (IPD) on November 16, 2020. Subsequently, the Agency published the Notice of Impact Assessment Decision on November 26, 2020, which confirms that the project is subject to the Federal IA process as per section 16(1) of the Act. Yamana is currently producing the federal EIS following the collection of the necessary baseline studies and assessment of impacts to be submitted by Q2 2022. [

On the Provincial side, the project is subject to the EIA and review procedure provided for in Subdivision 4 of Division II of Chapter IV of title I of the Environmental Quality Act (EQA) and must obtain an authorization from the government.

Following the environmental assessment procedure, Yamana will proceed to the authorization requests for the construction and the exploitation of the project with provincial and municipal authorities.

In addition to this, the Wasamac Project was selected as a pilot project by the government, under the authority of an inter-ministerial table composed of the five following ministries: Ministry of Energy and Natural Resources, Ministry of Forests, Wildlife and Parks, Minister of the MEFCC, Ministry of Municipal Affairs and Housing, and the Ministry of Economy and Innovation. The primary objective of this initiative is to establish a viable interaction system with stakeholders and, in particular, to promote the social acceptability of mining projects. The first meeting took place on December 19, 2019. Sporadic follow-ups are made with the Ministère de l'Énergie et des ressources naturelles (MERN), the leader of the table.

Under the Provincial EIA process, a Project Notice for Wasamac was submitted on November 19, 2019. Yamana is in the preparation phase of the EIS for the Wasamac Project. It is estimated that the EIS will be filed by Q2 2022. The federal and provincial EIA processes will be harmonized.

In addition to the Environmental Assessment processes, the Wasamac Project will need multiple permits at various instances such as a federal Fisheries Act authorization for impacts to fish habitat and a provincial Wetland Compensation Plan for impacts to wetland habitat. The detail and content of these regulatory instruments will be determined following the EIA process into the permitting phase.

Environmental impacts anticipated to be encountered during construction and operations are captured in the EIA.

25.10 Capital Cost Estimates

The total initial capital cost for the Wasamac Project is C$533 million and life-of-mine sustaining costs are C$432 million. The sustaining capital cost includes closure costs of C$22 million. See Table 21-1 for a summary of capital costs.

25.11 Operating Cost Estimates

Overall operating costs for mining ($36.08/t mined, $835.9 million), processing (average $14.93/t milled, $366 million), and G&A (average $5.18/t milled, $133 million) is $56.19/t milled or $1,334.9 million over the life of mine.

25.12 Economic Analysis

An engineering economic model was developed to estimate the project's annual pre-tax and post-tax flows and sensitivities based on an 5% discount rate.

The analysis uses the following key inputs:

- mine life of 9.7 years
- exchange rate of 1.28 (USD:CAD)
- base case gold price of US$1,550/oz
- cost estimates in constant Q3 2021 Canadian dollars with no inflation or escalation considered
- results are based on a 100% ownership with a 1.5% NSR
- capital costs funded with 100% equity (no financing costs assumed)

The economic analysis was performed assuming a 5% discount rate. The pre-tax $NPV_{5\%}$ is C$610 million; the IRR is 21.7%; and payback period is 3.6 years. On a post-tax basis, the $NPV_{5\%}$ is C$326 million; the IRR is 16.1%; and the payback period is 4.0 years.

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV, and IRR of the project using the following variables: gold price, discount rate, foreign exchange, operating cost, initial capital cost, and head grade. The analysis revealed that the project is most sensitive to changes in gold price, foreign exchange, and head grade and less sensitive to discount rate, operating cost, and initial capital cost.

25.13 Project Opportunities

A summary of opportunities is presented in the following subsections.

25.13.1 Mining

Table 25-2 on the following page outlines opportunities and benefits for the underground mine area.

Table 25-2: Summary of Underground Mining Opportunities

Opportunity Explanation	Benefit
1. Geomechanical detailed assessment; improve knowledge of ground conditions in order to validate and optimize design parameters, ground support and scheduling criteria.	1. Better estimation for dilution, recovery, support requirements, drilling pattern, infrastructure location, capital and operating costs, etc.
2. Fully electric vehicle fleet (trucks & LHDs); when BEV with automation is made available, modify mine plan to remove all unnecessary diesel production equipment	2. Decrease environmental impact and CO_2 emissions; increase social acceptability with a zero-emission mine plan; lower maintenance and ventilation costs, increase efficiency.
3. Optimize underground paste plant location and network.	3. Ensure optimal paste fill distribution, validate the best cost-effective system to distribute paste fill in all mining area; social acceptability regarding surface installation.
4. Blasting optimization and vibration modelling; using various models and external services, ensure the best drilling and blasting methods are used.	4. Ensure optimal dilution and recovery; ensure that prediction of surface vibration will permit pre-emptive mitigation for problematic blasts.
5. Optimization of the number of trucks and LHDs; get a more accurate estimation of the number of trucks and LHDs required for the life of mine.	5. Complete a detailed analysis considering Monte Carlo simulation for traffic, efficiency, maintenance, and other parameters of the complete material handling system for the first three years of production.
6. Ventilation network optimization: should be reassessed in future phases to ensure optimal disposition and equipment selection.	6. Optimization to ensure sufficient airflow, while avoiding over design and unnecessary costs.
7. Optimize the centres of production to maximize productivity, especially during the early stages of the project.	7. Increase flexibility in mine plan, thus ensuring continuous and assured production targets throughout the project. When at full production, centres of production could be planned larger to decrease development, backfill and others to optimize efficiency and minimize mining cost.
8. Replace propane with natural gas.	8. A trade-off analysis between propane and natural gas is recommended to evaluate potential savings and environmental impact reduction.
9. Optimization of backfill utilization in combination with surface activities.	9. Optimize backfill costs and underground planning while minimizing surface footprint and tailing/waste disposal strategies.
10. Waste, paste fill or tailing disposal in the old workings.	10. Minimizing surface footprint.
11. Proximal inferred resource conversion	11. Development cost reduction because of higher tonnage per metre.

25.13.2 Metallurgical Testwork

The underlying reasons for the low recoveries from Zone 1 and Zone 2 have not been determined. Additional investigative testing using new samples that have comprehensive geological logging and assaying is required for this program.

An opportunity exists to decrease the size of the elution circuit by changing the mine plan to deliver ore of a more consistently lower grade to the plant.

Further reductions in the leach retention time are possible with additional testing to quantify leach recoveries between 24 and 35 hours.

There is a potential opportunity to decrease the size of the comminution circuit with updated testwork. As is explored in Chapter 13, the comminution circuit sizing was based on adjusted testwork results.

The flotation flowsheet showed superior recoveries for Z1 and Z2 samples compared to the whole ore leach flowsheet. As part of a holistic review of optimizing production from Zones 1 and 2, installing a flotation and concentrate leach circuit after start-up can provide improved recoveries. This would require initial production from the Zone Principal and Zone 3 with the whole ore leach flowsheet.

25.14 Project Risks

25.14.1 Risk Workshop Methodology

A risk workshop was held during the feasibility study to identify and quantify risk likelihood and consequence, and to define mitigating strategies. Workshop attendees included experts from all relevant parties contributing to the study. Risks were identified for the following categories/areas and are discussed in more detail in subsections 0 to 25.14.5 :

1. Natural Hazards
2. Environmental Impact
3. External Issues
4. Fire or Explosion
5. Layout
6. Chemical
7. Plant
8. People
9. Health Hazards
10. Working Environment
11. Other

Risks not properly mitigated may negatively impact the projected economic outcomes.

"Consequence" and "likelihood" risk components were evaluated quantitatively, and considered together when applying a rating to the individual risk. Risk owners were identified as the responsible parties for executing mitigating measures. The risk levels are defined in Table 25-3.

Table 25-3: Risk Categories

Risk Level	Definition
Very High	Unacceptable Risk – additional mitigation and risk reduction measures must be generated and implemented as soon as possible.
High	Unwanted Risk – Implementation of mitigating measures required, as well as re-evaluation of risk at regular intervals.
Moderately High	Acceptable risk with control – Risks must be reduced to lowest possible level.
Moderate	Acceptable Risk
Low	Negligible Risk

25.14.2 Risk Workshop Outcomes

The workshop identified 68 notable risks. The distribution after considering mitigating measures is presented in Table 25-4. Of the 68 risks, 4 are exclusive to the execution phase, 39 exclusive to operation, and the remainder to both. After taking mitigating measures into consideration, no risks remained in the "very high" category. Six remained in the "high category", 16 in the "moderately high" category, and the remaining were considered to be "moderate" and "low".

Table 25-4: Risk Distribution Following Consideration of Mitigating Measures

Likelihood		Consequence Severity				
		Insignificant	Minor	Moderate	Major	Severe
		1	2	3	4	5
Almost Certain	5	0	0	0	0	0
Likely	4	0	1	3	0	0
Possible	3	0	1	4	9	6
Unlikely	2	0	1	10	17	4
Rarely	1	0	0	3	5	4

25.14.3 Summary of Key Risks

The risks presented in Table 25-5 remain "high" and will require ongoing monitoring, with new mitigating measures applied as possible. Risk owners are responsible for implementing mitigation measures and ensuring proper monitoring.

25.14.4 Summary of Mining Risks

Additional mining risks, impacts, and mitigations that were identified are summarized in Table 25-6.

Table 25-5: Summary of Risks Rated as "High" Following Consideration of Mitigating Measures

Area	Category	Hazard	Causes	Consequences	Schedule (Execution/Operation)	Mitigating Measures	Risk Ranking	Risk Owner
Tailings Facility	Environmental Impact	Erosion	- Wind erosion in the dry stack tailings facility area	Contamination in the landscape surround the TMF	Operations	-Water truck intended to manage active working areas for dust -Active reclamation plan targeting coverage as soon as reasonably possible	High	Yamana
Mining	Natural Hazards	Rock failure	- Lower competency rock areas - Seismic event	- Significant underground failure - Failure of the crown pillar	Both	- Design review/practice - Geotechnical monitoring instruments - Seismic monitoring system and vibration control in design - Emergency planning	High	InnovExplo
Mining	Natural Hazards	Rock failure	- Lower competency rock areas - Seismic event	- Failure of the crown pillar beneath Route 117	Both	- Design review/practice - Geotechnical monitoring instruments - Seismic monitoring system and vibration control in design - Emergency planning	High	InnovExplo
Process Facility	External Issues	Sabotage	- Theft of information - Instigate change to systems - Ransom	- Material damage to finances of project - Public disclosure of material information - Environmental damage if loss of controls of system	Operations	- Firewalls and security systems - System audits - Employee hacker training	High	Yamana
Off-Site Infrastructure	Layout	Vehicles	- Collision at entry to main access road off of 117	- Limited access for emergency vehicles to reach the collision site - Injury to people	Both	- Ensure railroad crossing is gated and has signage / lights - Ensure road widening considers cue of cars waiting for the train to pass, on both sides of 117	High	Ausenco
Off-Site Infrastructure	External Issues	Proximity to transport corridors	- Waiting at rail crossing causing cue of cars to build up	- Collision	Both	- Initiate studies for timing of rail activity - Evaluate offset in shift change to reduce conflict - Ensure sufficient cue lane for entry to Main Access Road off of Highway 117 from both directions following rail timing study	High	Yamana

Table 25-6: Summary of Mining Risks

Area	Risk Description and Potential Impact	Mitigation Approach
Geology & Mineral Resources	1. Potential lack of grade continuity of the inferred mineral resource due to local wide drilling spacing.	1. Risk can be reduced through future infill drilling campaigns; it will reduce the spacing between samples informing the inferred mineral resource
Underground Mine	1. Uncertainties related to geomechanical and hydrogeological knowledge could lead to delays in production, additional dilution or additional support requirement.	1. Additional information will be gathered through additional drilling campaigns (Phase 1 and Phase 2); ground support and sequence will be validated with new information.
	2. Crown pillar and overburden validation is required for the two main ramps near the surface.	2. Phase 2 of the drilling campaign will validate the overburden depth and rock quality; and validate appropriate excavation method and support.
	3. Stope sequencing and diverging from planned sequencing could cause excessive induced stress and production challenges.	3. Constant follow-up of strict planning criteria and planning validation to minimize induced stress, while satisfying long-term production requirements.
	4. Seismic activities due to production and mining operations.	4. Microseismic monitoring combined with proactive seismic protocol and constant improvement of mining sequence, design, and ground support.
	5. Technical knowledge of the historic mine is incomplete or outdated. This deficiency could lead to problematic voids, hazardous inflows, and water- or pillar-management problems.	5. Additional drilling will be conducted during pre-production development to validate void locations; pillar evaluation will be complete for stopes in close proximity to former mine workings
	6. Delays in pre-production development and infrastructure construction could postponed production start.	6. Maximize rapid development through contractors and promote innovative, efficient, and improved methods and technologies. Efficient construction techniques and optimized construction vs haulage in ramp prioritization.
	7. Paste fill plant location may be revised depending on surface noise and infrastructure nuisance to the community.	7. Sound analysis and mitigation evaluation will be completed. If required a paste fill plant relocation will be considered.
	8. Tailings pipes though the main ramp to the underground paste fill plant could be a hazard if a spill occurs.	8. Additional engineering and possible alternative options, need to be evaluated (e.g., resistant piping, pipe relocation, pipe or protective measures). A surface location for the paste fill plant may be considered.
	9. Recovery of challenging stopes (notably large upper stopes, some secondary stopes and sill pillars) could be lower than anticipated and/or have higher dilution.	9. Use of innovative technologies, in combination with continuous improvement during production, will ensure optimal recovery of all types of stopes and sill pillars.
	10. Major maintenance to main crusher or main conveyor could negatively impact mill input	10. Planning for maintenance versus ore stockpile at surface should be optimized to ensure maximal mineral reserve accessibility at surface. Haulage by truck through the main ramps will mitigate the remaining deficit.
	11. Poor management of ventilation systems, increasing ventilation cost and operational challenges.	11. Ensure proper VOD implementation with experienced personnel.
	12. Long term stability of major vertical development (such as ore pass and silo) and their degradation due to rock impact, and difficult maintenance due to non-entry excavation type.	12. Utilization of best practices during construction and adequate ground support.
	13. Competent and skilled workforce could be limited in the future, which could be an impediment to achieving the high production targets in the first years.	13. Entice competent and experienced workforce by promoting higher than median advantages; share competent personnel with other projects owned by Yamana.
	14. Equipment and material availability and overall risks associated with supply management could results in delays or prices increase.	14. Optimal supply management from the owner is required; orders need to be prepared as soon as possible; collaboration between nearby mines will help to ensure constant material supply
	15. Social acceptability of the neighbouring communities is paramount for project success; blast-induced vibration, excessive surfaces activities, unplanned visible infrastructures, sounds, dust, and other nuisances are all factors that need to be mitigated.	15. In addition to many future studies regarding noise and vibration control and mitigation, a close collaboration with the community is important before and throughout the project.

25.14.5 Summary of Geochemical Risks

Geochemical data gaps still exist (limited waste rock static and kinetic data); however, tailings have been adequately characterized. The most significant geochemical risk identified is elevated sulphide in tailings. While the tailings also contain adequate buffering capacity to offset the acid generation, there is risk that metals and salts could be liberated and cause substantial impacts to receiving systems (either within mine boundary or external).

As the project progresses, waste rock kinetic testing will come online (WSP), which will address the most significant data gaps. It is expected that geochemical characterization will proceed according to best practice, which includes regular sampling of all materials.

26 RECOMMENDATIONS

26.1 Overall

The results presented herein demonstrate that the Wasamac Project is technically and economically viable. In light of these results, InnovExplo recommends a bulk sample program for Yamana to de-risk the investment. This bulk sample program would target specific mining areas for geomechanical and metallurgical assessments. Also, the geological continuity and grade would be confirmed with development and stope excavations. It is also recommended to complete a geotechnical investigation of the subsurface of major facilities given the known variability of clay content at short range. This will allow for more accuracy in foundation design, which was carried as conservative in this study considering limited geotechnical information. Additional characterization of the site hydrogeology and geochemistry are also recommended to support development of the site-wide water balance, the groundwater model, and the water quality model for both project and permitting initiatives.

In the qualified person's opinion, the character of the property is of sufficient merit to justify the program recommended.

Analysis of the results and findings from each major area of investigation completed as part of this UFS suggests numerous recommendations for further investigations to mitigate risks and/or improve the base case designs. The following sections summarize the key recommendations arising from this UFS. Each recommendation is not contingent to a subsequent one. Table 26-1 on the following page presents a summary of recommended tasks, detailed in the subsections that follow.

26.2 Underground Mining

The activities, optimizations and reviews listed in Table 26-2 are recommended to ensure an optimal mine design considering all aspects of a sustainable project.

26.2.1 Bulk Sample Program

The bulk sample will consist of excavating approximately 10,500 m of access ramps and development to access the top portion of both metallurgical main zone central and east (mining area E38) with low recoveries to test with more material (25,000 t) and validate or improve recovery. The results of the program could have a material impact on the finance of the project. Geological continuity and geomechanics properties will also be assessed. Minimal surface support infrastructure (e.g., temporary offices, dry, shop, water supply, power and water treatment facilities) will be required for 30 months.

26.2.2 Detailed Geomechanical Assessment

To validate ground conditions, dilution, and recovery factors (and other parameters such as efficiencies of equipment, drilling patterns and location of major infrastructures), a two-phase geomechanical program has been established. Phase 1 has been completed. An analysis of Phase 1 data and the completion of Phase 2 is recommended for each of the main mining areas. Phase 1 consisted of a geomechanical drilling campaign in the footwall of mining area E29 and the western part of E38, close to major underground infrastructures. Phase 2 will comprise similar work directed at the other mining areas of the project that had to be postponed due to permitting delays.

Table 26-1: Summary of Recommended Tasks

Section	Item	Cost (C$)
26.2.1	Bulk Sample Program	90,000,000
26.2.2	Geomechanical Detailed Assessment	300,000[1]
26.2.3	Crown Pillar Monitoring during Dewatering	65,000
26.2.4	Hydrogeological Investigations	80,000
26.2.5	Fully Electric Fleet – Trucks & LHDs	30,000
26.2.6	Optimize Underground Paste Plant Location & Network – Surface vs. Underground	300,000
26.2.7	Blasting Optimization & Vibration Modelling	150,000
26.2.8	Ventilation Network Optimization	50,000
26.2.9	Replace Propane by Natural Gas[2]	100,000
26.2.10	Additional Engineering	300,000
Subtotal - Mining		91,375,000
26.3.1	Infill Drilling	11,000,000
26.3.2	"Near-Deposit" Exploration	1,500,000
26.3.3	Francoeur Deposit Confirmation Drilling and Near-Mine Exploration	1,500,000
26.3.4	Greenfield and Brownfield Exploration on the Wasamac Property	1,000,000
Subtotal - Geology		15,000,000
26.4	Metallurgical testwork	511,000
26.5	Site Infrastructure	239,000
26.6	Water Quality	32,000
26.7	Tailings Kinetic Testwork	64,000
26.8	Environment, Permitting, and Community Relations	1,401,000
	Total	108,622,000

Notes: 1. Considering the utilization of exploration drill holes for the geomechanical campaign. 2. The replacement of propane with natural gas should be studied in combination with the utilization of natural gas for surface infrastructures.

The geomechanical program targets major lithologies affecting the infrastructure and production centres of project. Results will be used to validate the properties of geomechanical domains in all principal mining areas. This program will also allow for detailed engineering of the portals and crown pillars for both ramps underneath Highway 117 and the railway parallel to Highway 117. The geomechanical field campaign includes complete joint-set and rock-mass logging, laboratory testwork, and updated rock mechanics analysis of the project, etc. Major structures such as faults will also be surveyed.

26.2.3 Crown Pillar Monitoring during Dewatering

A crown pillar monitoring program should be established prior to mine dewatering and production activities. The repercussions and impacts of these operations at surface are paramount to social acceptability. A tight monitoring system will ensure complete knowledge of the effects on the rock and soil horizons during the dewatering operation. Considering that little technical information regarding the former mine is available, additional monitoring and investigation could be used to optimize the design and the operations around the old workings, while demonstrating to the community the active accountability of the mine.

26.2.4 Hydrogeological Investigation

A two-phase hydrogeological testing program has been established. Phase 1 work has been completed, but the data has yet to be analysed. The tested drill holes are mainly in undisturbed areas, with no obvious faulting. The main shear zones (Francoeur-Wasa shear zone) and other transversal faults have not yet been tested to validate preferential flow zones. Phase 2 will require testing of additional drill holes to be drilled in the Francoeur-Wasa shear zone and transverse faults. The same testing procedures as for phase 1 is required, and should include slug tests, injection tests, and profile tracer tests, among others. The drill holes to be tested could be the same as future geomechanical drill holes if they are located in appropriate geological structures. The central and the eastern parts of the project area should be prioritized for this work, as these are the least understood due to a lack of historic data.

Once completed, the groundwater flow model should be updated accordingly, and new inflow and environmental impact predictions carried out. This model should be built on an unstructured mesh system so the faults can be properly represented in 3D.

26.2.5 Fully Electric Fleet – Trucks & LHDs

Considering Yamana's commitment to minimize its carbon footprint, the evaluation of a full battery fleet should be completed. One of the main aspects of the production plan is automation; however, battery electric vehicles (BEV) are not currently up to the task, as the equipment batteries discharge too rapidly. According to Sandvik, fully automated BEVs at a size compatible with the Wasamac project should be available by 2025. Artisan's product line will also soon be able to provide the Z65 truck (65 tonne) and LHD A18 (18 tonne), which should include the automation options.

Apart from environmental benefits, an electric fleet would considerably decrease project fuel consumption, ventilation requirements, and maintenance costs. Sandvik is also developing new technologies such as trolley lines for electric trucks and automated battery swapping, which could be utilized at the project in the coming years.

26.2.6 Optimize Underground Paste Plant Location and Network

Surface sound management south of Highway 117 is crucial for social acceptability. Cement delivery by truck is one of the possible sources of noise and complaints that will need to be proactively mitigated. Various solutions include repositioning the underground paste plant to move the dumping station as far as possible from the rang des Cavaliers and soundproofing the delivery process. A study considering the impact to the neighbourhood of cement delivery at surface should be completed. A trade-off study to assess various underground paste plant locations is under consideration. This will also be a good opportunity to reach a final decision on the surface vs. underground paste plant location deliberation.

26.2.7 Blasting Optimization and Vibration Modelling

To minimize and predict vibrations due to production blasts, additional analysis and testing is recommended. Well-developed technologies specializing in explosive vibration modelling and control that would provide representations of seismic behavior and vibration attenuation, seismic level prediction, and follow-up and monitoring during exploitation and production blasting, would be an asset.

This monitoring would ensure respect for the vibration level standards, which in turn would help to mitigate the impact of mining activities on the neighbourhood. Field campaigns for vibration attenuation modelling could be implemented immediately and make use of forthcoming exploration or geotechnical drill holes for data acquisition and baseline studies.

26.2.8 Ventilation Network Optimization

Optimization of the ventilation network should be reassessed in later phases to ensure optimal disposition and equipment selection. Ventilation designs should also be reviewed if the final mining equipment fleet is modified (i.e., additional BEV added, elimination of fuel engine, etc.). Design should be optimized to ensure sufficient airflow, while avoiding over-design and unnecessary costs.

26.2.9 Replace Propane by Natural Gas

A trade-off analysis between propane and natural gas is recommended to evaluate the potential savings and environmental impact reduction that would be achieved by replacing propane with natural gas (notably for the surface burners at the ventilation raises). Although a natural gas line does not currently pass along rang des Cavaliers, discussions are ongoing with Energir, Quebec's natural gas supplier. Additional benefits would include no propane tank at surface or daily propane truck deliveries, which would be favourable for the community along Rang des Cavaliers.

26.2.10 Additional Detailed Engineering

Additional detailed engineering on the following will be required during subsequent phases to optimize the mine design:

- underground electrical network (electrical station design, installation plan, etc.)
- construction works (ventilation wall design, concrete construction, etc.)
- mine-pumping network (pump installation, drain hole drilling plan and network, etc.)
- crusher/ore bin/chute and conveyors arrangements

26.3 Drilling & Geology

Based on the information presented in this technical report, including the results of the 2021 mineral resource and mineral reserve estimates, the qualified persons recommend that drilling and exploration programs should continue to be carried out with the objectives described below and summarized in Table 26-2.

Table 26-2: Drilling & Geology – Proposed Recommendations and Estimated Costs

Section	Item	Cost (C$)
26.3.1	Infill Drilling	11,000,000
26.3.2	"Near-Deposit" Exploration	1,500,000
26.3.3	Francoeur Deposit Confirmation Drilling and Near-Mine Exploration	1,500,000
26.3.4	Greenfield and Brownfield Exploration on the Wasamac Property	1,000,000
	Total	15,000,000

26.3.1 Infill Drilling

Infill drilling is recommended to better delineate the mineral resources and increase confidence in grade, improve mine planning, and provide further geotechnical and metallurgical data. The delineation drilling program should focus first on the areas expected to be developed in the first three years of production and subsequently on the remaining mineral resource. A budget of C$11 million to complete 68,000 metres of drilling is recommended for this program.

26.3.2 "Near-Deposit" Exploration

Drilling is recommended to expand the current mineral resource envelopes of the deposit to depths below the established mineral resource and to test for mineralization in the poorly explored gaps between defined zones. The focus of this exploration effort is to delineate secondary zones, such as Wildcat, and test high-priority extensions of the Francoeur-Wasa Shear Zone at depth, and to the east and west of the Horne Creek Fault. Approximately 10,000 metres of drilling corresponding to C$1.5 million is recommended for this program.

26.3.3 Francoeur Deposit Confirmation Drilling and Near-Mine Exploration

Drilling is recommended to confirm the current mineral resource at the Francoeur deposit and expand it by testing high-potential targets adjacent to, and down-dip from, historical mining operations and along the Arntfield-Francoeur segment of the Francoeur-Wasa Shear Zone. A projected 10,000 metres of drilling corresponding to C$1.5 million is recommended for this program.

26.3.4 Greenfield and Brownfield Exploration on the Wasamac Property

Development of a long-term pipeline of greenfield and brownfields exploration discoveries through geophysical surveying and testing of exploration targets is recommended, focusing first on the secondary gold-bearing shear zones to the Arntfield-Francoeur segment of the western Francoeur-Wasa Shear Zone (which includes Lac Fortune) and concentrating subsequently on other known occurrences. A budget of 6,000 metres of drilling corresponding to C$1 million is recommended for this program.

26.4 Metallurgical Testwork

The underlying reasons for the low recoveries from Zone 1 and Zone 2 samples have not been established. Additional samples should be collected to represent grade ranges, and spatial and lithological variability in these zones. Incorporating diagnostic leach test protocols may assist in determining the underlying cause of the poor recoveries.

The testing program should also include testing to better define the optimum leaching time by conducting fixed duration leach tests for 8, 12,16, 24 and 36 hours on representative samples.

It is recommended to generate additional leached and cyanide detoxified slurry to provide samples for vendor thickener and filtration tests. Thickening and tailings filtration testing completed to date is not vendor specific and may not be suitable for equipment selection with process guarantees.

Flotation testing for this study confirmed that no additional gold recovery could be expected by adding flotation to the process flowsheet. It is recommended to conduct tests on samples generated from the proposed test mining program to validate the selected flowsheet. A more detailed financial analysis may show that the flotation flowsheet economics are superior to the whole ore leach flowsheet. If so, it is recommended to conduct the required testing to support any engineering design required.

It is recommended to conduct tests on samples generated from the proposed bulk sampling program to validate the selected flowsheet.

The above testwork is estimated at C$511,941, which includes laboratory testwork costs, management, and interpretation of the results.

26.5 Site Infrastructure

It is recommended to pursue further data collection of the subsurface in the areas of major infrastructure. A seismic cone penetration test (SCPT) program has been designed and is planned for execution in the first half of 2022. The SCPT program will investigate the shallow subsurface in the area of the TSF, the WRSF, the plant pad area, the mill basin and TSF pond. Data collected will investigate variability in soil horizons and confirm foundation loading. Targets were selected with aerial photography. The program has been designed to leverage winter conditions that will allow access to wetland areas. The program is quoted to cost C$48,900.

It is recommended to optimize the storage of waste rock underground in the paste backfill versus on the surface in the WRSF to reduce the surface expression of mine waste product. It is also recommended to consider alternative locations for the waste rock, given that the study location impacts wetlands and as a result will require a more costly design. The estimated cost to complete a trade-off on the waste rock location is C$190,000, which includes geotechnical investigations in other viable areas.

The classification of the waste materials (waste rock and tailings) tested to date indicates a tendency for acid consumption; this is not the same as waste characterization, which is an important component of a waste management plan. It is recommended to characterize the waste rock material with kinetic testwork, which is in progress with WSP.

26.6 Water Quality

A site-wide water quality model was a prior recommendation and is in progress with Ausenco, with a target completion date in Q4 2021. The remaining cost of the water quality model is C$31,808.

26.7 Tailings Kinetic Testwork

There are sufficient tailings samples to adequately classify the tailings as having a 'low' risk of acidification (Price, 2009); however, depending on the tailings material distribution in situ and local environmental conditions, there may be variable responses observed. Preparation of tailings samples for kinetic testwork is in progress for four samples; the program is quoted to cost C$64,056, which includes laboratory testwork costs, management, and data interpretation.

Recommendations for the TSF area as per the SCPT program are discussed in Section 26.5.

26.8 Environment, Permitting, and Community Relations

Recommendations for environment, permitting, and community relations are as follows:

- complete ongoing additional baseline studies remaining in 2021
- complete the detailed site-wide water balance
- complete the EIS for submission to the province and the Impact Assessment Agency
- continue public and First Nations consultation activities, addressing and documenting concerns
- complete a conceptual closure plan and update the closure cost estimate for the project
- support engineering during permitting

The estimated cost for the above-mentioned tasks is C$1,400,797, which excludes work spent on baseline studies already completed in 2021.

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