NI 43-101 Technical Report

PREFEASIBILITY STUDY FOR THE **KIENA MINE COMPLEX PROJECT**

Val-d'Or, Québec, CANADA

Prepared for:

Wesdome Gold Mines Ltd.

WESDOME

EFFECTIVE DATE: April 12, 2021
SIGNATURE DATE: June 7, 2021

By qualified persons:

Charlotte Athurion, P. Geo. ..... BBA Inc.
Navin Gangadin, P. Eng. ..... BBA Inc.
Pierre-Luc Richard, P. Geo. ..... BBA Inc.
Michael Stochmal, P. Eng. ..... BBA Inc.
Frank Palkovits, P. Eng. ..... Mine Paste Ltd.
Pierre Roy, P. Eng. ..... Soutex Inc.
Sheila Daniel, P. Geo. ..... Wood Canada Ltd.
Narendra Verma, P. Eng. ..... Wood Canada Ltd.

BBA MPL MINE PASTE LTD SOUTEX wood.

DATE AND SIGNATURE PAGE

This report is effective as of the 12th day of April 2021.

"Signed and sealed original on file"	June 7, 2021
Charlotte Athurion, P. Geo. BBA Inc.	Date

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Navin Gangadin, P. Eng. BBA Inc.	Date

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Pierre-Luc Richard, P. Geo. BBA Inc.	Date

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Michael Stochmal, P. Eng. BBA Inc.	Date

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Frank Palkovits, P. Eng. Mine Paste Ltd.	Date

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Pierre Roy, P. Eng. Soutex Inc.	Date

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Sheila Daniel, P. Geo. Wood Canada Ltd.	Date

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CERTIFICATE OF QUALIFIED PERSON

Charlotte Athurion, P. Geo.

This certificate applies to the NI 43-101 Technical Report titled "Prefeasibility Study for the Kiena Mine Complex Project, Val-d'Or, Québec, Canada" (the Technical Report), prepared for Wesdome Gold Mines Ltd. (Wesdome) dated June 7, 2021 and effective as of April 12, 2021.

I, Charlotte Athurion, P. Geo., do hereby certify that:

1. I am a Geologist with BBA Inc. located at 1034, 3rd avenue, Suite 202, Val-d'Or, Québec J9P 1T6 Canada.
2. I graduated with an equivalent of a bachelor's degree in geology (B.Sc.) from Université Joseph Fourier (Grenoble, France) in 2010. In addition, I obtained a M.Sc. from the Institut National de la Recherche Scientifique (INRS, city of Québec, Québec) in 2013.
3. I am a member in good standing of the Ordre des Géologues du Québec (OGQ Member No. 1784) and the Association of Professional Geoscientists of Ontario (APGO Member No. 3122).
4. I have worked in the exploration and mining industry for more than 8 years. My expertise has been acquired with Les Mines J.A.G. Ltd., Explorateurs-Innovateurs de Québec Inc., Canadian Malartic (exploration branch) and, since November 2016, with numerous companies through my career as a consultant.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am co-author and responsible for Chapters 12 and 14. I am also co-author for the relevant portions of Chapters 1, 2, 3, 25, 26 and 27 of the Technical Report.
8. I personally visited the Kiena Property that is the subject of this Technical Report on October 16, 2020 and November 17, 2020, as part of this current mandate and on several other occasions in the last 4 years.
9. I have had prior involvement with the property that is the subject of the Technical Report by being involved in several different technical support mandates in the last 4 years and by being on of the QPs of the Technical report issued on January 28, 2021 and effective as of October 31, 2020 titled "Update of the Mineral Resource Estimate for the Kiena Mine Complex Project, Val-d'Or, Québec, Canada".
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 7th day of June 2021.

"Signed and sealed original on file"

Charlotte Athurion, P. Geo.
BBA Inc.

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Navin Gangadin, P. Eng.

This certificate applies to the NI 43-101 Technical Report titled "Prefeasibility Study for the Kiena Mine Complex Project, Val-d'Or, Québec, Canada" (the Technical Report), prepared for Wesdome Gold Mines Ltd. (Wesdome) dated June 7, 2021 and effective as of April 12, 2021.

I, Navin Gangadin, P. Eng., do hereby certify that:

1. I am a Senior Mechanical Engineer and Project Manager with BBA E&C located at 10 Carlson Court, Suite 420, Toronto Ontario, M9W 6L2.
2. I graduated with a degree from the University of Waterloo, Waterloo, Ontario, Canada, with a Bachelor of Applied Science in Mechanical Engineering in 1997.
3. I am a member of Professional Engineers of Ontario (PEO) and the Professional Engineers and Geoscientist Newfoundland and Labrador (PEGNL).
4. I have been employed in mining related consulting engineering for 24 years. I have provided engineering and project management support for both engineering studies and execution mandates for mine infrastructure, processing, smelting and refining facilities.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am author and responsible for Chapters 1, 2, 3, 19, 22, 24, 25, 26 and 27, and for Sections 18.1,18.3 to 18.10, 21.1.1 to 21.1.6, 21.1.12 to 21.1.15, 21.2.1, 21.2.2, 21.2.7, and 21.2.8 of the Technical Report.
8. I personally visited the property that is the subject to the Technical Report on two separate occasions, the latest on February 18, 2020.
9. I was project manager for the preparation of the Preliminary Economic Assessment filed in 2020 for the property that is the subject of the Technical Report. I have no other prior involvement with the property.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 7th day of June 2021.

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Navin Gangadin, P. Eng.
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Pierre-Luc Richard, P. Geo., M.Sc.

This certificate applies to the NI 43-101 Technical Report titled "Prefeasibility Study for the Kiena Mine Complex Project, Val-d'Or, Québec, Canada" (the Technical Report), prepared for Wesdome Gold Mines Ltd. (Wesdome) dated June 7, 2021 and effective as of April 12, 2021.

I, Pierre-Luc Richard, P. Geo., M.Sc., do hereby certify that:

1. I am a Principal Geologist with BBA Inc. located at 2020 Robert-Bourassa Blvd, Suite 300, Montréal, Québec, Canada, H3A 2A5.
2. I am a graduate of Université du Québec à Montréal in Resource Geology in 2004. I also obtained a M.Sc. from Université du Québec à Chicoutimi in Earth Sciences in 2012.
3. I am a member in good standing of the Ordre des Géologues du Québec (OGQ Member No. 1119), the Association of Professional Geoscientists of Ontario (APGO Member No. 1714), and the Northwest Territories Association of Professional Engineers and Geoscientists (NAPEG Member No. L2465).
4. I have worked in the mining industry for more than 17 years. My exploration expertise has been acquired with Richmont Mines Inc., the Ministry of Natural Resources of Québec (Geology Branch), and numerous companies through my career as a consultant. My mining expertise was acquired at the Beaufor mine and several other producers through my career. I managed numerous technical reports, mineral resource estimates and audits as a consultant for InnovExplo from February 2007 to March 2018 and as a consultant for BBA since.
5. I have read the definition of "qualified person" set out in NI 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am author and responsible for the preparation of Chapters 4 to 11 and 23, and co-author of Chapters 12 and 14. I am also co-author for the relevant portions of Chapters 1, 2, 3, 25, 26 and 27 of the Technical Report.
8. I personally visited the Kiena Property that is the subject of this Technical Report on August 6 to 8, 2020, and on other occasions during the course of other mandates.
9. I have had no prior involvement with the property that is the subject of the Technical Report, except for authoring Technical Reports in the past.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 7th day of June 2021.

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CERTIFICATE OF QUALIFIED PERSON

Michael J. Stochmal, P. Eng.

This certificate applies to the NI 43-101 Technical Report titled "Prefeasibility Study for the Kiena Mine Complex Project, Val-d'Or, Québec, Canada" (the Technical Report), prepared for Wesdome Gold Mines Ltd. (Wesdome) dated June 7, 2021 and effective as of April 12, 2021.

I, Michael J. Stochmal, P. Eng., do hereby certify that:

1. I am a Senior Mining Engineer and Project Manager with BBA Inc. located at 1010 Lorne Street, Unit 103, Sudbury, Ontario, P3C 4R9, Canada.
2. I graduated from the University of Toronto in Toronto, Ontario, Canada in 1996 with a B.A.Sc. in Civil Engineering. In 1999, I graduated from Laurentian University in Sudbury, Ontario, Canada with an MEng in Mineral Resources Engineering.
3. I am a member of the Professional Engineers of Ontario (90495664).
4. I have been employed in consulting engineering and mining operations for over 21 years. I have provided engineering support and conducted engineering studies for clients in consulting roles and worked at mining operations in technical roles as Engineer, Senior Engineer and Engineering Supervisor responsible for both execution and long range planning duties.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am author and responsible for the preparation of Chapters 15 and 16, except for Section 16.8.9 (Paste Backfill Facilities), and for Sections 21.1.7 and 21.2.3. I am also co-author for the relevant portions of Chapters 1, 2, 3, 25, 26 and 27 of the Technical Report.
8. I personally visited the property that is the subject to the Technical Report on September 29, 30 and October 1, 2020.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 7th day of June 2021.

"Signed and sealed original on file"

Michael J. Stochmal, P. Eng.
BBA Inc.

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CERTIFICATE OF QUALIFIED PERSON

Frank Palkovits, P. Eng.

This certificate applies to the NI 43-101 Technical Report titled "Prefeasibility Study for the Kiena Mine Complex Project, Val-d'Or, Québec, Canada" (the Technical Report), prepared for Wesdome Gold Mines Ltd. (Wesdome) dated June 7, 2021 and effective as of April 12, 2021.

I, Frank Palkovits, P. Eng., do hereby certify that:

1. I am a Mining Engineer and President of Mine Paste Ltd., located at 26 Windsor Crescent, Sudbury, ON, Canada, P3E 1Z5.
2. I am a graduate of Laurentian University, Sudbury, ON, Canada, graduated in 1988 with B.Eng., in Mining Engineering. Also have a Geological Engineering Technician Diploma, 1981 from Cambrian College, Sudbury, ON., Canada.
3. I am a member of Professional Engineers Ontario.
4. I have been employed in the mining industry since 1980, beginning with field exploration programs, then with INCO in 1981 as an underground mining/geological technician, as an underground miner before obtaining my degree. Subsequently next 15 years up to 2000 at INCO in Sudbury as mining engineer to Chief Engineer, underground foreman. From 2000 to 2010, joined Golder Paste Technology as VP followed by President/Owner of Kovit Engineering Ltd (Paste backfill, tailings), then with sale to Outotec as Global Lead on Mining and Backfill. Currently Owner of Mine Paste Ltd, and co-owner of RMSCorp (both mine backfill and tailings management).
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am author and responsible for the preparation of Sections 16.8.9 (Paste Backfill Facilities), 21.1.11 and 21.2.6. I am also co-author for the relevant portions of chapters 1, 2, 3, 25, 26 and 27 of the Technical Report.
8. I personally visited the property that is the subject to the Technical Report on November 24, 2020.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed this 7th day of June 2021.

"Signed and sealed original on file"

Frank Palkovits, P. Eng.
Mine Paste Ltd.

CERTIFICATE OF QUALIFIED PERSON

Pierre Roy, P. Eng.

This certificate applies to the NI 43-101 Technical Report titled "Prefeasibility Study for the Kiena Mine Complex Project, Val-d'Or, Québec, Canada" (the Technical Report), prepared for Wesdome Gold Mines Ltd. (Wesdome) dated June 7, 2021 and effective as of April 21, 2021.

I, Pierre Roy, P. Eng., do hereby certify that:

1. I am an engineer with Soutex inc. located at 1990 rue Cyrille-Duquet, Local 204, Québec, Qc, G1N 4K8, Canada.
2. I am a graduate of Université Laval (Québec, Québec, Canada) with a B.Sc. in Mining Engineering in 1986, and a M.Sc. in Mining in 1989.
3. I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, (OIQ Licence: 45201), Professional Engineer registered with the Professional Engineers of Ontario, (PEO Licence: 100110987.
4. I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in mining operations, engineering, and financial evaluations for 33 years. During this time, I have been involved in mineral processing and environmental coordination at Kiena mine for six (6) years and Troilus mine for nine (9) years. I have also worked as a consultant for the mineral processing industry for two (2) years at CRM in Québec and with Soutex inc. in Québec for sixteen (16) years. As consultant I have been involved in many projects in iron, base metals, and gold mining sectors.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am author and responsible for the preparation of Chapters 13 and 17, and Sections 18.12, 21.1.8, 21.1.9, 21.2.4 and 21.2.5. I am also co-author for the relevant portions of Chapters 1, 2, 3, 25, 26 and 27 of the Technical Report.
8. I personally visited the property that is the subject of the Technical Report on November 16 to 19, 2020.
9. I have had no prior involvement, except the fact that I had worked as a staff employee from 1990 to 1996 at the mill department, with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed this 7th day of June 2021.

"Signed and sealed original on file"

Pierre Roy, P. Eng.
Soutex Inc.

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CERTIFICATE OF QUALIFIED PERSON

Sheila Ellen Daniel, P.Geo.

I, Sheila Ellen Daniel, P.Geo., am employed as a Principal Geoscientist, Discipline Lead Mining Environmental Management Approvals at Wood Environment & Infrastructure Americas, a Division of Wood Canada Limited, located at 2020 Winston Park Drive, Suite 600, Oakville, Ontario, Canada, L6H 6X7.

This certificate applies to the technical report titled "*Prefeasibility Study for the Kiena Mine Complex Project, Val-d'Or, Québec, Canada*" (the Technical Report), prepared for Wesdome Gold Mines Ltd. (Wesdome) dated June 7, 2021 and has an effective date of April 12, 2021.

My qualifications and relevant experiences are that:

1. I graduated with a M.Sc. from McMaster University in 1990 and with a B.Sc. (Honours) from the University of Western Ontario in 1988.
2. I am a registered Professional Geoscientist of Ontario (#0151).
3. I have worked as a mining environmental consultant continuously for 30 years experience.
4. As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).
5. I have not visited the property.
6. I am author and responsible for the preparation of Chapter 20, and co-author of Section 21.1.10. I am also co-author for the relevant portions of Chapters 1, 2, 3, 25, 26 and 27 of the Technical Report.
7. I am independent of Wesdome as independence is described by Section 1.5 of NI 43–101.
8. I have no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.
10. As of the date of this Certificate, 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: June 7, 2021

"Signed and sealed original on file"

Sheila Daniel, M.Sc., P.Geo.
Principal Geoscientist
Discipline Lead, Mining Environmental Management and Approvals

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CERTIFICATE OF QUALIFIED PERSON

Narendra S. Verma, P.Eng.

I, Narendra S. Verma, P.Eng., am employed as a Principal Geotechnical Engineer at Wood Environment & Infrastructure Americas, a Division of Wood Canada Limited, located at 2020 Winston Park Drive, Suite 600, Oakville, Ontario, Canada, L6H 6X7.

This certificate applies to the technical report titled "*Prefeasibility Study for the Kiena Mine Complex Project, Val-d'Or, Québec, Canada*" (the Technical Report), prepared for Wesdome Gold Mines Ltd. (Wesdome) dated June 7, 2021 and has an effective date of April 12, 2021.

My qualifications and relevant experiences are that:

1. I am a graduate of Ph.D. in Civil Engineering, University of Ottawa, Ottawa, Ontario, Canada (1977), M.Tech. in Soil Mechanics and Foundation Engineering, Indian Institute of Technology, Delhi, India (1967) and B.Sc. in Civil Engineering, Panjab University, Chandigarh, India (1965).
2. I am a member of Ordre des Ingénieur du Quebec, Professional Engineers Ontario, Canadian Dam Association, and American Society of Civil Engineers.
3. I have worked as a geotechnical engineer continuously for 56 years experience with focus on tailings management facilities and water management dams.
4. As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).
5. I have not visited the property.
6. I am author and responsible for the preparation of Sections 18.2 and 18.11, and co-author of Section 21.1.10. I am also co-author for the relevant portions of Chapters 1, 2, 3, 25, 26 and 27 of the Technical Report.
7. I am independent of Wesdome as independence is described by Section 1.5 of NI 43–101.
8. I have previously performed safety assessment of the Kiena Mine Tailings Storage Facility in 1997.
9. I have read NI 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.
10. As of the date of this Certificate, 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: June 7, 2021

"Signed and sealed original on file"

Narendra S. Verma, Ph.D., ing., P.Eng., F.ASCE, D.GE.
Principal Geotechnical Engineer
Wood Environment & Infrastructure Americas,
a Division of Wood Canada Limited

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LIST OF ABBREVIATIONS AND UNITS OF MEASURE

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
$/t	dollar per metric tonne
%	percent
”	inch
°C	degree Celsius
°F	degrees Fahrenheit
µm	micron
2D	two dimensional
3D	three dimensional
a	annum (year)
A x b	SMC parameters
AA	atomic absorption
AACE	Association for the Advancement of Cost Engineering
Ag	silver
AGB	Abitibi Greenstone Belt
AGI	Advanced Geosolutions Inc.
Ai	abrasion index
AISC	all-in sustaining cost
ARD/ML	acid rock drainage/metal leaching
Au	gold
B	billion
BBA	BBA Inc.
BEV	battery electric vehicles
BMA	bulk mineralogical analysis
BWi	Bond work index
C	carbon
CAD or C$	Canadian dollar
CaO	lime
CAPEX	capital expenditure
CCTV	closed circuit television
CDA	Canadian Dam Association
CF	Callahan Fault
CG	Cadillac Group
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	carbon-in-pulp
CN	cyanide

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
CN_{WAD}	weak acid dissociable cyanide
CRF	cemented rockfill
CRMs	certified reference materials
CSZ	Callahan Shear Zone
CTRI	*Centre Technologique des Résidus Industriels*
Cu	copper
CVAAS	cold vapour atomic absorption spectrometry
CWi	crushing work index
d	day (24 hours)
DAP	delivered at place
DDH	diamond drillhole
deg. or °	angular degree
DF	Dubuisson Formation
DFO	Fisheries and Oceans Canada
DPFZ	Destor-Porcupine Fault Zone
DSM	deep soil mixing
DWi	drop-weight index
EA	environmental assessment
EBIT	earnings before interest and tax
ECCC	Environment and Climate Change Canada
EDF	environmental design flood
EM	electromagnetics
EPCM	engineering, procurement and construction management
EPs	engineering providers
EQA	Environmental Quality Act
et al.	and others
ETP	effluent treatment plant
FA	fire assay
Fe	Iron
FS	feasibility study
ft	feet (12 inches)
ft^2	square feet
FTE	full-time equivalent
FW	foot wall
g	gram
G&A	general and administrative
g/t	gram per (metric) tonne

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
GHG	greenhouse gas
GISTM	Global Industry Standard on Tailings Management
GPa	gigapascal
GSC	Geological Survey of Canada
GSI	Geological strength index
GTL	GRAVITY tails leaching
h	hour (60 minutes)
ha	hectare
HADD	harmful alteration, disruption or destruction
HDPE	high density polyethylene
HF	Héva Formation
hp	horsepower
HSE	health, safety and environment
HW	hanging wall
ICP-MS	inductively coupled plasma-mass spectrometry
ID	Identification
ID^2	inverse distance squared
IDF	inflow design flood
IP	induced polarization
IRR	internal rate of return
JF	Jacola Formation
K	thousand (000)
KCFM	thousand cubic feet per minute
KCN	potassium cyanide
kg	kilogram
kg/t	kilogram per tonne
km	kilometre
KNA	kriging neighbourhood analysis
kPa	kilopascal
KSZ	K Shear Zone
kt	kilotonne
kWh	Kilowatt-hour
L	litre
lb	pound
LG	Louvicourt Group
LHD	load haul dump
LLCFZ	Larder Lake-Cadillac Fault Zone

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
LOM	life of mine
LVF	La Motte-Vassan
m	metre
M	million
m a.s.l.	metres above sea level
m^2	square metre
m^3	cubic metre
Ma	mega annum (million years)
MBBR	moving bed biofilm reactor
MDMER	Metal and Diamond Mining Effluent Regulations
MELCC	*Ministère de l'Environnement et de la Lutte contre les changements climatiques*
MERN	*Ministère de l'Énergie et des Ressources naturelles*
MF	Marbenite Fault
MFFP	*Ministère des Forêts, de la Faune et des Parcs*
MG	Malartic Group
mg	milligram
min	minute (60 seconds)
Mine Paste	Mine Paste Ltd.
mm	millimetre
MOWL	maximum operating water level
MPa	megapascal
mpd	meters per day
MRE	mineral resource estimate
MT	magnetotelluric
Mt	million tonne
MTO	material take-offs
Mtpa	million tonne per annum
MVA	megavolt-ampere
MW	megawatt
Na_2S	sodium sulphide
NaCN	sodium cyanide
NBCC	National Building Code of Canada
NF	Norbenite Fault
Ni	nickel
NN	nearest neighbour
No.	number
NOWL	normal operating water level

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
NPAG	non-potentially acid generating
NPV	net present value
NSR	net smelter return
NTS	National Topographic System
NTU	nephelometric turbidity units
Ø	diameter
OK	ordinary kriging
OPEX	operating expenses
oz	troy ounce
oz/t	troy ounces per tonne
oz/y	troy ounces per year
P_{80}	80% passing - product size
Pb	lead
PEA	preliminary economic assessment
PF	Parfouru Fault
PFS	prefeasibility study
PG	Piché Group
PGA	peak horizontal ground accelerations
pH	potential of hydrogen
PMF	probable maximum flood
PO	Pontiac Group
ppm	parts per million
PRVs	pressure reducing valves
PS	Pontiac Subprovince
psi	pounds per square inch
Q1	first quarter
Q4	fourth quarter
QA/QC	Quality Assurance / Quality Control
QP(s)	Qualified Person(s)
RHF	Rivière Héva Fault
RQD	rock quality designation
RWi	Bond rod mill work index
s	second
S	sulphur
SAG	Semi-autogenous (mill)
SCSE	SAG circuit specific energy
SD	standard deviation

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
SEDAR	System for Electronic Document Analysis and Retrieval
SG	specific gravity
SMBS	sodium metabisulfite
SMC	SAG mill comminution (test)
SO_2	sulphur dioxide
Soutex	Soutex Inc.
st	short ton (2,000 lbs)
t	tonne (1,000 kg) (metric ton)
t_a	JK abrasion parameter
TFS	tailings filter stack
TMF	tailings management facility
TOC	total organic carbon
tpa	tonne per annum
tpd	tonne per day
tph	tonne per hour
tpy	tonne per year
TSF	tailings storage facility
U/G	underground
UCS	uniaxial compressive strength
UDS	underground distribution system
U-Pb	uranium-lead
USD or US$	United States dollar
UTM	Universal Transverse Mercator
VAT	value added taxes
VDF	Val-d'Or Formation
VIPs	value improving practices
VLF	very low frequency
VoIP	Voice over Internet Protocol
vs.	versus
w/w	weight per weight
WBS	work breakdown structure
WEC	work element coding
Wesdome	Wesdome Gold Mines Ltd.
WG	water gauge
WOL	whole ore leach
Wood	Wood Canada Ltd.
wt%	weight percent

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
WTP	water treatment plant
XRD	X-ray diffraction
y	year (365 days)
yd	yard (36 inches)
Zn	Zinc

1. SUMMARY

1.1 Introduction

Wesdome Gold Mines Ltd. (Wesdome) commissioned BBA Inc. (BBA) to prepare a technical report (the "Report") of the Prefeasibility Study, herein also referred to as the "PFS" or the "Study", for the Kiena Mine Complex project (the "Project"). The Kiena Mine Complex is an existing underground mine that has been on care and maintenance since mid-2013, located in Val-d'Or, Quebec, Canada. Following new gold discoveries, Wesdome is looking to restart operations at the Kiena Mine Complex.

This Report was completed by BBA with the assistance of several specialized consultants, including Mine Paste Ltd., Soutex Inc., and Wood Canada Ltd. This Report was prepared according to the guidelines set out under the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") to support the results of the Study as disclosed in Wesdome's press release entitled "Wesdome Announces Positive Pre-Feasibility Study and Restart Decision for the Kiena Mine in Val d'Or, Quebec; IRR of 98%" dated May 26, 2021.

1.2 Contributors

The major prefeasibility study contributors and their respective areas of responsibility are presented in Table 1-1.

Table 1-1: Report contributors

Qualified Person / Consulting Firm	General overview of responsibilities
BBA Inc. (BBA)	
▪ Charlotte Athurion, P. Geo. ▪ Navin Gangadin, P. Eng. ▪ Pierre-Luc Richard, P. Geo. ▪ Michael Stochmal, P. Eng.	▪ Property description, location and accessibility; ▪ Historical data review; ▪ Geological setting, mineralization, deposit types, exploration and drilling; ▪ Sample preparation and QA/QC, and data verification; ▪ Geological modelling and mineral resource estimate; ▪ Mineral reserve estimate; ▪ Underground mine design, underground infrastructure, ventilation, production scheduling, underground capital costs and operating costs; ▪ Surface infrastructure design and capital costs; ▪ General and administration operating costs; ▪ Market studies and contracts; ▪ Financial analysis and overall NI 43-101 integration.

Qualified Person / Consulting Firm	General overview of responsibilities
Mine Paste Ltd. (Mine Paste)	
▪ Frank Palkovits, P. Eng.	▪ Paste backfill plant design, capital costs and operating costs.
Soutex Inc. (Soutex)	
▪ Pierre Roy, P. Eng.	▪ Metallurgical testwork analysis and process design criteria; ▪ Site water balance; ▪ Process plant and tailings pumping design, capital and operating costs; ▪ Water treatment plant design, capital and operating costs.
Wood Canada Ltd. (Wood)	
▪ Narendra Verma, P. Eng. ▪ Sheila Daniel, P. Geo.	▪ Tailings storage facility upgrade and dam raising design, capital costs; ▪ Environmental studies, permitting and closure costs.

1.3 Key Project Outcomes

The reader is advised that the results of the PFS summarized in this Report are intended to provide an initial, high-level review of the proposed optimized project configuration and revised design options. The PFS mine plan, execution plan and economic model include numerous assumptions. There is no guarantee that the Project's economics described herein will be achieved.

The key outcomes of this PFS are the following:

- The Project's mineral resource estimate (2020 MRE) inclusive mineral reserves is estimated at 1.791 M indicated tonnes at an average grade of 13.8 g/t (0.796 Moz) and 3.421 M inferred tonnes at an average grade of 6.0 g/t (0.658 Moz);
- The Project's mineral resource estimate (2020 MRE) exclusive mineral reserves is estimated at 0.644 M indicated tonnes at an average grade of 7.6 g/t (0.157 Moz) and 3.404 M inferred tonnes at an average grade of 5.9 g/t (0.649 Moz);
- The Project's Probable Mineral Reserve is estimated at 1.574 M diluted ore tonnes at an average diluted gold head grade of 11.89 g/t (0.601 Moz contained gold);
- The life of mine (LOM) plan supports an average annual rate of recovered gold ounces approximating 83,574 oz/y over an estimated 7 year mine life (0.592 Moz recovered gold);
- LOM project capital costs (±25% accuracy) total C$231.8M;
- The LOM operating cost estimated at C$187.71/t ore is represented by the mining cost estimated at C$119.38/t ore (including paste backfill), process plant operating cost estimated at C$31.80/t ore (including water treatment), and general and administrative cost estimated at C$36.53/t ore;

- All-inclusive sustaining cost of production at US$676/oz recovered gold over LOM;
- Base case (C$1,600/oz gold; USD:CAD exchange rate of 1.32) project NPV of C$366.6M at a 5% discount rate and an IRR of 98% after mining and income taxes. Payback period is 2.7 years.

1.4 Property Description and Ownership

The Project is in the Province of Quebec in the Abitibi-Témiscamingue administrative region within the limits of the municipality of Val-d'Or and 100 km east of Rouyn-Noranda. It lies to the northwest of the urban centre of Val-d'Or and covers 7,047 ha.

The coordinates for the approximate centre of the Project are latitude 48°08' N and longitude 77°54' W (284105E and 5335715N: NAD 83 / UTM Zone 18).

As of April 12, 2021, the Kiena Mine Complex property consists of a contiguous group of 183 contiguous electronic map designated mining claims and one mining concession. Of the 183 mining claims, Wesdome has 169 claims registered 100% in their name. Six claims are held by Mines Dynacor (50%) and Wesdome (50%), which represents the Maufort property. Eight claims are held by Wesdome (75%) and 9264-7890 Québec inc. (25%), which represents the Siscoe Extension property.

Some of the mining titles comprising the Project are subject to certain agreements and royalties.

The Project includes the milling and tailings facilities of the Kiena mine, nine shafts including the Parker Shaft, and related underground workings from past producers and exploration projects, and various surface facilities. Other than the exploration offices and underground exploration development, the principal infrastructure of the Project has been under care and maintenance since mid-2013.

1.5 Geology and Mineralization

The Kiena Mine Complex is located in northwestern Quebec, straddling the southern part of the Abitibi Greenstone Belt (AGB) and the northern part of the Pontiac Subprovince:

- The AGB comprises E-trending successions of folded and faulted volcanic and sedimentary rocks and intervening domes of intrusive rocks. An important geologic feature of the AGB is the occurrence of major, E-trending ductile-brittle fault zones. These zones cut across the entire belt from the Kapuskasing structural zone in the west to the Grenville front in the east, dividing the supracrustal rocks and intervening domes into distinct lozenge-shaped domains. The most two important fault zones in the southern AGB are Destor-Porcupine Fault Zone (DPFZ) in the north and Larder Lake-Cadillac Fault Zone (LLCFZ) in the south.
- The Pontiac Subprovince (PS) consists principally of a turbiditic succession composed of graywacke and mudstone with minor intercalated conglomerate and basalt.

The southern Abitibi Greenstone Belt in the area of the Kiena Mine Complex consists of 2714–2700 Ma volcano-plutonic assemblages, including the Malartic and Louvicourt groups, intruded by calc-alkaline plutonic rocks. The Malartic Group comprises mainly komatiitic and tholeiitic basalt flows and sills, with minor sedimentary rocks, which are interpreted as an oceanic floor in an extensional environment related to mantle plumes, whereas the Louvicourt Group is mainly composed of mafic to felsic volcanic rocks that formed in a subduction-related arc setting. From south to north, the Kiena Mine Complex area is underlain by the lithologies of the Pontiac Group (PO), the Piché Group (PG), the Cadillac Group (CG), and formations belonging to the Louvicourt Group and the Malartic Group.

The area of the Kiena Mine Complex has a series of large-scale shear zones and related subsidiary faults trending ESE-WNW to SE-NW, subparallel to stratigraphy and dipping steeply to the north. They are, from south to north: the Larder Lake-Cadillac Fault Zone (LLCFZ), the Parfouru Fault (PF), the Marbenite Fault (MF), the Norbenite Fault (NF), the Callahan Fault (CF), the K Shear Zone (KSZ) and the Rivière Héva Fault (RHF). The area of the Kiena Mine Complex is cut by all of them. The shear zones contain dykes or stocks of monzonitic or tonalitic composition that vary widely in age (pre-, syn- or post-tectonic) and are spatially associated with gold mines (Norlartic, Marban, Kiena, Sullivan, Goldex, Siscoe, Joubi, Sigma and Lamaque). The observed diversity in the styles and ages of gold mineralization related to these large-scale shear zones demonstrates that several distinct episodes of mineralization occurred.

Gold mineralization on the Kiena Mine Complex shares many geological attributes with other vein-type gold deposits of the Val-d'Or district and with orogenic gold deposits (also known as lode gold, greenstone-hosted quartz-carbonate vein, or mesothermal deposits) in terms of host rock composition, mineralogy and hydrothermal alteration.

Gold mineralization on the property occurs in all rock types except the Proterozoic dykes but is more common in intrusive bodies and basalt as these acted as competent rock units that promoted fracturing during deformation. Gold mineralization is concentrated where there is a marked competency contrast between these competent units and the adjacent deformed komatiite and/or chlorite-talc schists. There are at least two main gold mineralizing events in the region: young deposits in which the gold mineralization did not experience much deformation after its emplacement; and early mineralization in which mineralized bodies are commonly affected by D1 asymmetric folds, are highly strained and are locally dismembered. In a few deposits, both generations are present. Gold-bearing veins in the region exhibit a great variety of orientations, mineralogy and crosscutting relationships.

At least 63 mineralized lenses have been observed on the property. In general, mineralized zones on the property occur near a large-scale fault. They are often associated with a subsidiary shear zone that may be proximal, adjacent or host to the mineralization. Alteration minerals are dominantly albite, carbonates and pyrite with lesser chlorite and silica.

A total of 52 mineralized lenses were interpreted for the purpose of the 2020 MRE.

The Kiena Deep A Zone was first intersected in December 2007 and is localized within the Marbenite Fault deformation corridor and is divided into three main lenses and a fourth smaller lens. The lenses occur along an isoclinal fold associated with the MF and a subsidiary fault.

1.6 Status of Exploration and Drilling

The mineral resource drill database for the Project, as of September 18, 2020, consisted of 6,823 diamond drillholes (DDH) (951,602.6 m) including 213 DDH from the 2019-2020 drilling program. As of April 12, 2021, Wesdome has drilled an additional 143 DDH (41,780.8 m) from underground since the drill database close-out date, which are not included in the 2020 MRE. The QPs, Charlotte Athurion and Pierre-Luc Richard, are of the opinion that while the addition of these new drillholes would increase knowledge and confidence on the Project and have a positive impact on the 2020 mineral resource estimate, it does not have a material impact on the current study presented in this Report.

The QPs, Charlotte Athurion and Pierre-Luc Richard, reviewed the drilling, sample preparation, analytical and security procedures, as well as insertion rates and the performance of blanks, standards, duplicates and check assays for the 2019-2020 drilling programs, and concluded that the observed failure rates are within expected ranges and that no significant assay biases are present. The QPs are of the opinion that the protocols in place are adequate and followed. The database for the Project is of good overall quality and adequate to industry standards. The QPs are of the opinion that the database is appropriate for the purpose of the 2020 MRE and that the sample density allows for a reliable estimate to be made of the size, tonnage and grade of the mineralization in accordance with the level of confidence established by the mineral resource categories in the CIM Standards.

1.7 Mineral Processing and Metallurgical Testing

Metallurgical testwork was carried out for the Project in 2018 and 2019 by the Centre Technologique des Résidus Industriels (CTRI) in 2018 and by SGS Lakefield in 2019. Following the PEA, a metallurgical testwork program was developed and carried out at Base Metallurgical Laboratories to further characterize the ore from the Kiena Deep deposit and VC Zone in a context of a prefeasibility study.

The testwork was realized on composite samples made from drill cores coming from four zones called A Zone, A1 Zone, A2 Zone and VC Zone. Gold content was measured between 1.4 g/t and 41.9 g/t. Silver levels between 0.6 g/t and 5.8 g/t, low levels of total organic carbon and nickel, copper, and zinc in varying minor amounts were measured. Free gold particles were identified by screened metallic assays. A full ICP-MS scan, a Sulphur (S2-) assay by Leco, as well as a Clay XRD were also completed on each composite.

Bulk Mineralogical Analysis (BMA) at 106 µm using QEMSCAN were completed on the composites. Pyrite and pyrrhotite were identified as the predominant sulphur bearing minerals. The analysis on sulphides composition indicates the presence of pentlandite and chalcopyrite. Quartz, chlorite, amphibole, talc, feldspars, and dolomite were the predominant non-sulphide gangue minerals, and varied quite significantly between samples.

Comminution testing consisting of SMC and bond ball mill work index tests were completed on the composites. The composites are characterized as relatively soft from SAG milling and ball milling perspectives, with A x b values of between 52 and 81, and Bond work indexes of 10.7 kWh/t to 12.8 kWh/t, at a closing screen size of 106 µm. Comparative work indexes indicated a wide range of hardness for the variability samples.

Leaching tests were carried out for the four composite samples (A, A1, A2 and VC) and the variability samples. From the initial stage of optimization testing, WOL at a grind of 80 µm with 75 g/t of lead nitrate addition was chosen as the final conditions. Under these conditions, the three composite samples from Kiena Deep (A, A1, A2) had very similar gold recovery trends with their final gold recovery percentage ranging from 98.9% to 99.3% after 48 hours. The VC zone composite had a lower recovery percentage than the three Kiena Deep composites at 97.8%.

There was significant coarse, visible gold recovered to the gravity concentrate at low mass recovery. Gold recovery varied from 20% to 71% in gravity concentrate. No significant final gold recovery difference between WOL and GTL was observed on the composite samples after 48 hours of leaching and minor cyanide consumption reductions, if any, could be achieved. By considering a leaching of 48 hours, the installation of a gravity circuit in the plant is not recommended.

In the WOL tests, the majority of the gold was extracted after 24 hours, but an additional 1.2 to 29.7 percent gold extraction was measured between 24 and 48 hours. Tests at the finer primary grind of 80 µm had increased gold extraction of 1% to 3% for over 120 µm for the three composites. Lead nitrate was critical to improving leach kinetics. The sodium cyanide consumption ranged from 0.42 kg/t to 0.75 kg/t and lime consumption from 0.22 kg/t to 0.78 kg/t for the composite samples. It is suggested to keep the final grind size at around 80 µm, the addition of lead nitrate at a dosage of 75 g/t and the NaCN addition at around 500 ppm.

Cyanide destruction testing using SO_2/air consistently reached a residual CN_{wad} of about 2 ppm to 3 ppm even at a SO_2 ratio of 7 SO_2:CN, most likely due to the presence of nickel. The cyanide destruction tests showed good copper elimination, but nickel concentrations remained high. SO_2 ratio of 8 SO_2:CN and the addition of 50 ppm of copper as copper sulfate is recommended for the PFS along with additional testwork at these conditions.

Sedimentation tests were performed on the leach feed, the whole cyanide destruction tails and on classified cyanide destruction tails which were screened to -53 µm. A flocculant dosage of 20 g/t gave the best results on the pre-leach sample while similar results were obtained for floc dosages between 20 g/t and 40 g/t on the whole cyanide destruction tails. Higher flocculant dosages were required for the classified cyanide destruction tails resulting in lower final settled densities.

Filtration tests were carried out and showed good filtering properties. The final moisture content for the unclassified tailings was between 10% and 12% water for thickened cyanide destruction tails, and between 12% and 16% water for un-thickened cyanide destruction tails. Pressure filtration on the classified tailings produced final cakes of between 14% and 19% moisture.

A bulk sample of 7,032 tonnes from the Kiena Deep A and A1 Zone was processed in the Kiena mill in November and December 2020. At a feed rate of around 950 tpd and a P_{80} of around 67 µm, the combined SAG and ball mill power draw was in the 925 kW to 950 kW range, or 20 kWh/t to 24 kWh/t. Assuming a 90% power transmission to the pinion, the operating Bond work index value is between 15 kWh/t and 18 kWh/t. These values must be used for the grinding power evaluation as the throughput of the grinding circuit will be in the same range as it was during the bulk sample processing. The cyanide concentration was maintained at 500 ppm (NaCN) and the pH was controlled by lime titration that was between 500 ppm and 700 ppm (CaO). Lead nitrate addition was set at 75 g/t. Final Au tails were between 0.15 g/t and 0.25 g/t most of the time with higher values from time to time.

1.8 Mineral Resource Estimate

The October 31, 2020 Kiena Mine Complex Mineral Resource Estimate (2020 MRE) was prepared by Karine Brousseau, P. Eng. (Wesdome) and audited by Pierre-Luc Richard, P. Geo., and Charlotte Athurion, P. Geo. (BBA) using all available information including historical and recent diamond drillholes.

The 2020 MRE includes drill data as of September 18, 2020 and includes an additional 213 diamond drillholes, for a total of 60,865 m drilled since the close-out date of the 2019 MRE.

The 2020 MRE covers the entire Kiena Mine Complex project.

The geological wireframes were constructed in Leapfrog Geo™ by Charlotte Athurion and Clovis Auger (BBA) and Karine Brousseau and Bruno Turcotte (Wesdome). Leapfrog Geo v 5.1.1 was also used to generate mineralized intercepts. Datamine Studio RM version 1.6.87 was used for the compositing, 3D block modelling, interpolation and classification. Statistical studies were undertaken using Excel and Snowden Supervisor v. 8.11.

The Project mineral resource drill database, as of September 18, 2020, consisted of 350 diamond drillholes collared on surface and 6,473 diamond drillholes collared underground with a cumulative length of 951,602.62 m (Figure 14-2). The average length of a diamond drillhole is 139.5 m. Of these 6,823 DDH, a subset of 1,145 diamond drillholes cut across the mineralized zones.

The mineral resources are not mineral reserves as they do not have demonstrated economic viability. The estimate is categorized as Indicated and Inferred Resources based on data density, geological and grade continuity, search ellipse criteria, DDH density and specific interpolation parameters. The effective date of the 2020 MRE is October 31, 2020 based on the compilation status and resource cut-off grade parameters. The Indicated and Inferred 2020 Mineral Resource Estimate (inclusive of mineral reserves) for the Kiena Mine Complex is presented in Table 1-2.

Table 1-2: Indicated and Inferred 2020 Mineral Resource Estimate per zone (inclusive of mineral reserves)

Zone	Cut-off grade (Au g/t)	Indicated Resources			Inferred Resources		
		Tonnage (t)	Grade (Au g/t)	Ounces (Au oz)	Tonnage (t)	Grade (Au g/t)	Ounces (Au oz)
Kiena Deep A	2.8	1,280,000	17.50	720,000	325,700	11.49	120,400
S50	2.8	146,600	4.54	21,400	100,000	3.73	12,000
VC	2.8	137,700	4.79	21,200	169,500	5.25	28,600
ZB	2.8	-	-	-	74,000	4.12	9,800
South	2.8	63,200	4.15	8,400	226,800	3.85	28,000
Presqu'ile	2.8	-	-	-	255,600	6.70	55,100
Dubuisson	2.8	-	-	-	744,600	6.69	160,200
Martin	2.8	163,100	4.77	25,000	109,100	4.29	15,000
North West	2.8	-	-	-	285,800	4.03	37,100
Wesdome	3.6	-	-	-	1,129,400	5.27	191,400
Total		**1,790,600**	**13.83**	**796,000**	**3,420,500**	**5.98**	**657,600**

Notes:

1. The independent qualified persons for the 2020 MRE, as defined by National Instrument ("NI") 43-101 guidelines, are Pierre-Luc Richard, P. Geo., and Charlotte Athurion, P. Geo., both of BBA Inc. The effective date of this MRE is October 31, 2020.
2. These mineral resources are not mineral reserves as they do not have demonstrated economic viability.
3. These mineral resources are inclusive of mineral reserves.
4. The mineral resource estimate follows CIM definitions (2014) for mineral resources.
5. Results are presented in situ and undiluted and considered to have reasonable prospects for economic extraction, below a 100 m crown pillar.
6. The resources include 46 lenses with a minimum true thickness of 3.0 m (2.4 m for Wesdome zones) using the grade of the adjacent material when assayed or a value of zero when not assayed. High-grade capping varies from 20 g/t to 265 g/t Au (when required) and was applied to composited assay grades for interpolation using an Ordinary Kriging interpolation method (ID^2 for Dubuisson zones 1220 and 1230) based on 1.0 m composite and block size of 5 m x 5 m x 5 m, with bulk density values of 2.8 (g/cm^3). A three-step capping strategy was applied, where capping value decreased as interpolation search distance increased, in order to restrict high-grade impact at greater distance. Indicated resources are manually defined and encloses areas where drill spacing is generally less than 30 m, blocks are informed by a minimum of three drillholes, and reasonable geological and grade continuity is shown.
7. The estimate is reported for potential underground scenario at cut-off grades of 2.8 g/t Au (>40 degree dip) and 3.6 g/t Au (<40 degree dip, Wesdome zones only). The cut-off grades were calculated using a gold price of US$1,450 per ounce, a USD:CAD exchange rate of 1.32 (resulting in C$1,914 per ounce gold price); mining cost C$100/t (>40 degree dip); C$150/t (<40 degree dip); processing cost C$40/t; G&A C$25/t.
8. The number of metric tonnes and ounces were rounded to the nearest hundred and the metal contents are presented in troy ounces (tonne x grade/31.10348). Rounding may result in apparent summation differences between tonnes, grades and contained metals content.
9. The QPs are not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues, or any other relevant issues not reported in this Report that could materially affect the mineral resource estimate.

The Indicated and Inferred 2020 Mineral Resource Estimate (exclusive of mineral reserves) for the Kiena Mine Complex is presented in Table 1-3. In this case, mineral resources included in the mineral reserve designs were removed from the mineral resource estimate.

Table 1-3: Indicated and Inferred 2020 Mineral Resource Estimate per zone (exclusive of mineral reserves)

Zone	Cut-off grade (Au g/t)	Indicated Resources			Inferred Resources		
		Tonnage (t)	Grade (Au g/t)	Ounces (Au oz)	Tonnage (t)	Grade (Au g/t)	Ounces (Au oz)
Kiena Deep A	2.8	281,400	11.65	105,400	311,200	11.22	112,200
S50	2.8	69,300	3.82	8,500	99,300	3.72	11,900
VC	2.8	137,700	4.79	21,200	169,500	5.30	28,600
ZB	2.8	-	-	-	74,000	4.10	9,800
South	2.8	63,200	4.15	8,400	226,800	3.85	28,000
Presqu'ile	2.8	-	-	-	255,600	6.70	55,100
Dubuisson	2.8	-	-	-	744,600	6.70	160,200
Martin	2.8	92,100	4.38	13,000	108,200	4.28	14,900
North West	2.8	-	-	-	285,800	4.00	37,100
Wesdome	3.6	-	-	-	1,129,400	5.30	191,400
Total		**643,700**	**7.56**	**156,500**	**3,404,400**	**5.94**	**649,200**

Notes:

1. The independent qualified persons for the 2020 MRE, as defined by National Instrument ("NI") 43-101 guidelines, are Pierre-Luc Richard, P. Geo., and Charlotte Athurion, P. Geo., both of BBA Inc.
2. These mineral resources are not mineral reserves as they do not have demonstrated economic viability.
3. These mineral resources are exclusive of mineral reserves.
4. The mineral resource estimate follows CIM definitions (2014) for mineral resources.
5. Results are presented in situ and undiluted and considered to have reasonable prospects for economic extraction, below a 100 m crown pillar.
6. The resources include 46 lenses with a minimum true thickness of 3.0 m (2.4 m for Wesdome zones) using the grade of the adjacent material when assayed or a value of zero when not assayed. High-grade capping varies from 20 g/t to 265 g/t Au (when required) and was applied to composited assay grades for interpolation using an Ordinary Kriging interpolation method (ID^2 for Dubuisson zones 1220 and 1230) based on 1.0 m composite and block size of 5 m x 5 m x 5 m, with bulk density values of 2.8 (g/cm^3). A three-step capping strategy was applied, where capping value decreased as interpolation search distance increased, in order to restrict high-grade impact at greater distance. Indicated resources are manually defined and encloses areas where drill spacing is generally less than 30 m, blocks are informed by a minimum of three drillholes, and reasonable geological and grade continuity is shown.
7. The estimate is reported for potential underground scenario at cut-off grades of 2.8 g/t Au (>40 degree dip) and 3.6 g/t Au (<40 degree dip, Wesdome zones only). The cut-off grades were calculated using a gold price of US$1,450 per ounce, a USD:CAD exchange rate of 1.32 (resulting in C$1,914 per ounce gold price); mining cost C$100/t (>40 degree dip); C$150/t (<40 degree dip); processing cost C$40/t; G&A C$25/t.

8. The number of metric tonnes and ounces were rounded to the nearest hundred and the metal contents are presented in troy ounces (tonne x grade/31.10348). Rounding may result in apparent summation differences between tonnes, grades and contained metals content.
9. The QPs are not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues, or any other relevant issues not reported in this Technical Report that could materially affect the mineral resource estimate.

1.9 Mineral Reserve Estimate

The Project Probable Mineral Reserves are estimated at 1.574 M ore tonnes at an average gold head grade of 11.89 g/t (0.601 M oz contained gold). The mineral reserve estimate is dependent on Indicated Mineral Resources represented in the 2020 MRE and converted based on technical feasibility and economic viability. Three mineralized zones were considered in the mine schedule representing the LOM plan for the PFS. Most of the ore tonnes as well as gold ounces to be mined are sourced from the Kiena Deep A Zone supplemented by the S50 and Martin Zones. Please refer to Table 1-4 for the mineral reserve estimate by mineralized zone.

Table 1-4: Mineral reserve estimate

Probable Mineral Reserves				
Zone	**Cut-off Grade Considered (Au g/t)**	**Diluted Ore Tonnes (t)**	**Diluted Ore Grade (Au g/t)**	**Contained Metal (Au oz)**
Kiena Deep A	3.15	1,387,100	12.95	577,300
S50	3.20	95,100	4.12	12,600
Martin	2.40	92,200	4.00	11,900
Total		**1,574,400**	**11.89**	**601,800**

Notes:

1. CIM Definitions Standards on Mineral Resource and Reserves (2014) were followed.
2. Underground mineral reserves are diluted tonnes and grades; the reference point is the mill feed at the primary crusher.
3. Cut-off grade considers a gold price of C$1,914 per ounce (US$1,450 per ounce at 1.32 Exchange Rate USD to CAD) for mine design purposes, a 97% metallurgical processing recovery for both the S50 and Martin Zones and 98.5% for the Kiena Deep A Zone, life of mine operating cost of C$122.92/t mining, C$28.25/t processing, C$36.53/t General and Administration. An incremental cut-off grade excluding the mine operating cost was not considered for lateral drift development required through mineralization and would present opportunity to increase the mineral reserve estimate.
4. A minimum mining width of 3.0 m, and minimum footwall angle of 45° was used in the creation of all mineral reserve solids. Longitudinal long hole stoping is the predominant method considered for production mining. The life of mine mining recovery factor is 90% and combined planned and unplanned dilution factor is 27%.
5. A bulk density of 2.8 t/m^3 was used for both ore and waste rock.
6. The application to expand the mining concession to include Martin Zone is being pursued by Wesdome. Martin Zone is scheduled to begin development and production mining in 2024 to provide time for the permitting process to be approved.
7. Diluted ore tonnes and gold ounces were rounded to the nearest hundred. Numbers may not add due to rounding.

1.10 Mining Methods

The spatial location as well as the planned development and production associated with the three mining zones are illustrated in Figure 1-1. The North Zone located under a surface crown pillar approximately 100 m thick below Lac De Montigny was mined historically. As an act of due diligence, the bulk of the voids are to be backfilled to enhance ground stability.

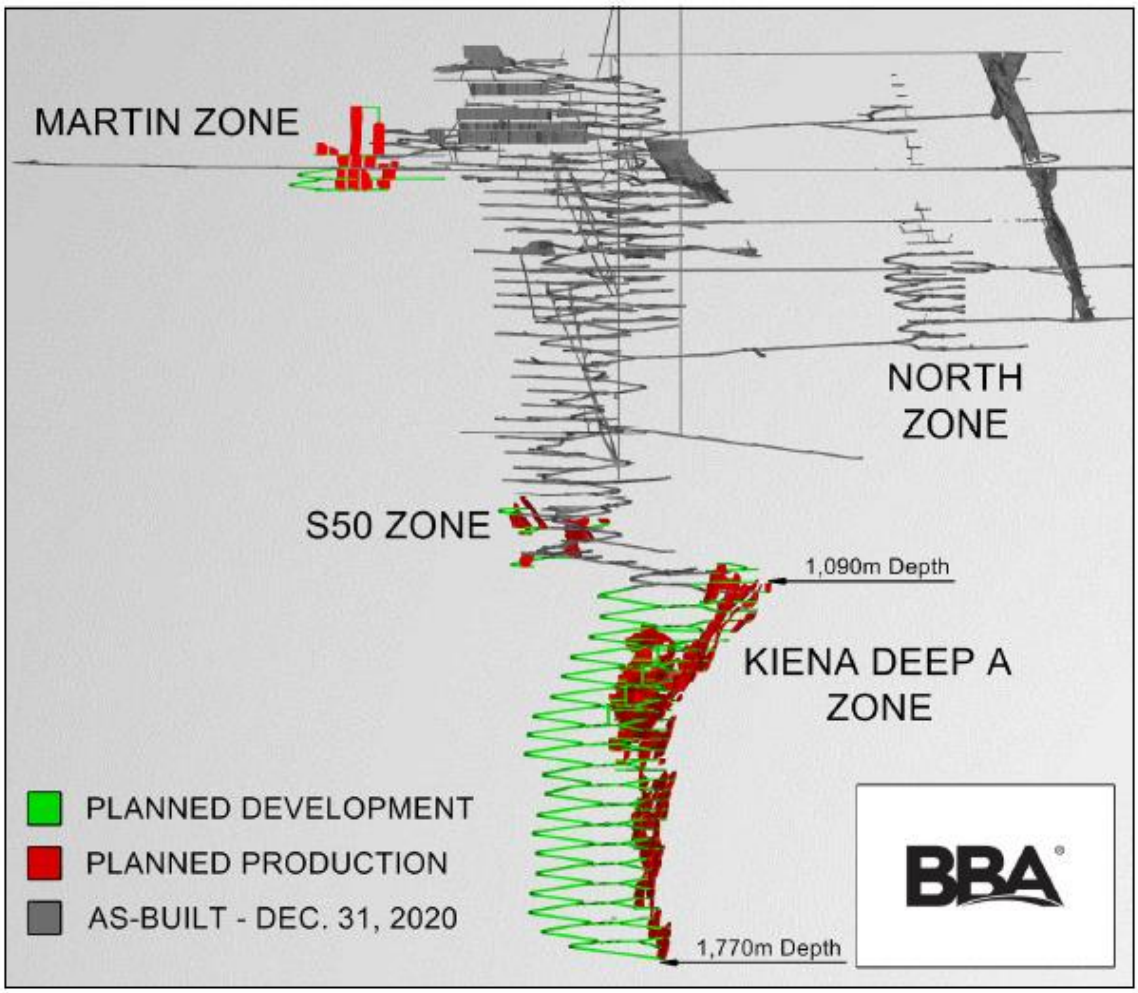


Figure 1-1: Isometric long section of the mining zones considered in the LOM plan (looking northwest)

1.10.1 Development

Conventional drilling and blasting methods will be used for lateral drift and vertical raise development. Ground support designs and installations to reinforce planned development reflect the size of the opening and the anticipated rock mass quality.

1.10.2 Production

Most of the production mining will utilize the overhand long hole stope method, historically used at the Kiena Mine Complex. In cases where production mining is expected under previously mined and backfilled stopes, the underhand long hole stope method will be utilized.

Long hole production stopes will be mined on a longitudinal retreat basis. Stope empirical designs and mine sequencing were established for each of the three mining zones accounting for key factors influencing stability including rock mass quality, the orientation of the principal structures, in situ stress and the geometry of the stope. Cemented rockfill, rockfill and paste backfill were considered for supporting the rock mass after extraction. Mining recovery will be maximized and production stope cycle time optimized in the Kiena Deep A Zone by making paste backfill the predominant backfill type.

1.10.3 Mining Recovery and Dilution

The LOM mining recovery factor is 90% and combined planned and unplanned dilution factor is 27%. Both the mining recovery and dilution factors are weighted and account for both development and production mining combined.

1.10.4 Infrastructure

Both existing and new infrastructure are required to support the LOM plan. The key existing infrastructure includes the hoisting shaft extending to a depth of 930 m with an average hoisting capacity of 2,000 tpd, ore and waste pass systems, the underground crusher, underground ventilation raise and ramp system, service repair, fueling, refuge, process water, mine dewatering, compressed air and electrical facilities.

Beginning in 2021 and continuing as the underground mine extends at depth, new infrastructure will be required both on surface and underground. The key infrastructure to be purchased, installed and constructed includes hoist drives, a 13.8 kV electrical transformer and power line feeding the Kiena Deep A Zone, a new fleet of mobile equipment including three battery powered haulage trucks with charging stations, fresh and exhaust mine air ventilation fans and burners located on surface, a surface paste backfill plant and underground distribution system, mine air ventilation exhaust raise controls, second egress, service repair, fueling, refuge, latrine, fuse magazine, explosives magazine, process water, mine dewatering, compressed air, communications and electrical facilities.

1.10.5 Life of Mine Plan

The life of mine (LOM) plan representing the PFS is scheduled at approximately 7 years, starting in 2021 and ending in January 2028. Productivity rates are influenced by the established labour headcount, new fleet of mobile equipment, existing and new fixed infrastructure and mining cycle constraints. Due consideration was given to hoist shaft conveyance and milling throughput capabilities. The Kiena Mine Complex LOM plan is presented in Table 1-5 and the annual ore tonnes and recovered gold ounces are illustrated in Figure 1-2.

Table 1-5: LOM plan

	Unit	2021	2022	2023	2024	2025	2026	2027	2028	Total
Total Lateral Development	m	4,186	4,071	4,311	4,088	3,123	1,005	169	-	**20,953 m**
Lateral Development Waste	m	3,313	2,683	3,052	2,998	2,297	745	116	-	**15,204 m**
Lateral Development Waste	t	188,583	156,406	178,027	168,858	130,704	33,323	6,480	-	**862,381 t**
Lateral Development Ore	m	872	1,388	1,259	1,090	826	261	53	-	**5,749 m**
Lateral Development Ore	t	35,450	56,403	51,162	42,262	33,571	9,870	2,154	-	**230,872 t**
Lateral Development Ore Grade	g/t	11.12	10.71	9.47	9.90	12.13	3.52	3.20	-	**10.18 g/t**
Vertical Development Ore	m	-	-	-	141	-	-	-	-	**141 m**
Vertical Development Ore	t	-	-	-	3,678	-	-	-	-	**3,678 t**
Vertical Development Ore Grade	g/t	-	-	-	5.3	-	-	-	-	**5.34 g/t**
Vertical Development Waste	m	230	262	267	304	259	72	-	-	**1,394 m**
Vertical Development Waste	t	5,794	6,599	6,727	7,649	6,533	806	-	-	**34,108 t**
Total Development Waste	t	194,376	163,005	184,754	176,507	137,237	34,129	6,480	-	**896,488 t**
Longhole Drilling Metres	m	11,595	32,604	41,964	67,635	63,486	73,927	89,455	2,158	**382,824 m**
Longhole Production Ore	t	40,581	104,763	146,947	231,567	224,300	245,741	335,030	10,955	**1,339,884 t**
Longhole Production Ore Grade	g/t	9.91	14.92	11.30	12.18	14.40	13.70	9.69	5.52	**12.20 g/t**
Total Rockfill	t	-	-	-	-	-	5,189	13,384	-	**18,573 t**
Total Cemented Rock Backfill	t	28,451	38,154	-	-	-	5,477	31,883	-	**103,965 t**
Total Wet Paste Backfill (1.82 t/m^3)	t	-	37,716	107,591	142,988	166,210	167,268	176,417	15,055	**813,245 t**
Total Backfill	t	28,451	75,870	107,591	142,988	166,210	177,934	221,685	15,055	**935,783 t**
Total Ore	t	76,031	161,166	198,109	277,507	257,871	255,612	337,183	10,955	**1,574,434 t**
Average Ore Grade	g/t	10.48	13.44	10.82	11.74	14.10	13.30	9.65	5.52	**11.89 g/t**
Mineable Gold Ounces	oz	25,612	69,659	68,935	104,756	116,933	109,323	104,616	1,944	**601,778 oz**
Recovered Gold Ounces	**oz**	**25,204**	**68,614**	**67,901**	**103,102**	**115,179**	**107,641**	**102,829**	**1,915**	**592,385 oz**

Figure 1-2: Annual ore tonnes and recovered gold ounces

1.10.6 Paste Backfill

The paste backfill design was based on the expectation of high talc tailings and with potentially difficult implications for a tailings “only” paste backfill. The flowsheet was therefore developed using 65% sand with 35% tailings mix, a proven method with gold tailings in the region. This enables lower capital cost and nominally similar operating cost with a tailings solution, with the caveat that more tailings must be disposed of on surface. The recipe enables rapid cycle time with backfill strengths suitable for 7-day turnaround after filling is complete, an important element in mine productivity. The system will use a paste thickener, with the high-density underflow when combined with excavated sand will have the appropriate mass balance to make dense 175 mm slump paste. Since the mine is relatively deep though very constrained strike length, low slump paste backfill will be needed to control the velocity and thereby reduce wear in the lined boreholes from surface to mining areas. Binder use of ~2.3% by weight of backfill has been calculated based on laboratory test results; this includes single or double width stopes (5 to 10 m) with wall exposures of 10 m. It also includes underhand mining of sill pillars, which are few and many months later, allowing for fill strengths to continue to rise.

1.11 Recovery Methods

The principal process steps include grinding, cyanide leaching, gold adsorption and desorption on activated carbon, electrolysis and cyanide destruction. A complete review of the equipment has been made to verify the capacity and condition with respect to the expected mill throughput, reliability, operation schedule and metallurgical performances. From this review, it has been found that the existing equipment allows reaching the throughput and performance objectives that were confirmed from the processing of the bulk sample.

The grinding circuit consists of an open-circuit SAG mill followed by a ball mill in closed circuit with two stages of cyclones. The ground ore goes to a thickener. The thickener overflow feeds three carbon columns and the water coming out of the columns is used as grinding solution.

The thickener underflow is sent to the leaching circuit consisting of three tanks with agitators. Air is sparged from two compressors. Lime, sodium cyanide and lead nitrate are added at the SAG mill feed and leaching circuit feed. The slurry coming from the leaching circuit feeds the CIP circuit consisting of five tanks. The retention time in the leaching and CIP circuits is approximately 48.3h.

The CIP tails go to cyanide destruction tank where sodium metabisulfite and copper sulfate are added. The cyanide destruction tails go to a thickener then are sent to the tailings management facility (TMF) or paste backfill plant where approximately 15% of the tailings will be used for paste backfill on average over the life of the mine. The water needs for the process will be supplied by fresh water from the lake, process water and mine water. Internal recirculation of water will come from the thickeners overflow and additional water will be reclaimed from the TMF. Water recirculation will be maximized.

The loaded carbon taken from the CIP tanks and carbon columns is sent to the carbon stripping circuit, which allows desorption and electrowinning leading to melting and casting of doré bars. This circuit consists of an elution vessel, barren and pregnant solution tanks and an electrowinning cell. The carbon washing and regeneration circuit consists of an acid washing tank as well as a vertical natural gas regeneration furnace.

The production target is set at 1,000 t of dry ore per day. The operating schedule will be adjusted according to the mine throughput to maintain that production target. The gold recovery for Kiena Deep is 98.5% and 97.0% for S50 and Martin zones. The average operating cost for the processing plant is C$28.42/t mined.

1.12 Project Infrastructure

1.12.1 Surface Infrastructure

The Kiena Mine Complex produced gold up until June 2013 and has been under care & maintenance since then. Wesdome has maintained a site presence since 2013 and development was and is still being performed providing access to underground geological drilling platforms while using the existing infrastructure. It is therefore expected that most of the surface infrastructure can be reused where required to meet any needs similar to those during development.

However, a number of modifications and additions to the surface infrastructure will need to be made to support the current Project. Minor modifications to the on-site roads are required to better manage the traffic around the new paste backfill plant and mine air ventilation system. The existing mine air ventilation system and hydraulic backfill plant will be demolished, and new foundations will have to be built for the new ventilation system, paste plant, effluent treatment plant, and transformers.

The site power requirements are expected to increase from the historical peak of 7.8 MVA to a maximum of 12.7 MVA in 2025 and beyond. This increase is imputable to the ventilation system upgrade, the new effluent treatment plant and paste plant, and significant additions to the underground loads. Two new transformers will be installed on surface, one for the underground loads, and one for the effluent treatment plant. The existing power distribution infrastructure can be reused to feed the new ventilation system and paste plant. To meet the power requirements above 10 MVA, Hydro-Québec indicated that a new dedicated 25 kV line with an estimated length of 13 km will have to be added to their distribution network.

1.12.2 Tailings Management

The existing tailings storage facility (TSF) is contained by perimeter dykes and comprises of North Cell and South Cell. The starter perimeter dykes comprise zoned embankments with wide central low permeability silty clayey core and upstream and downstream rockfill shells. The first two phases of the dyke raises were performed by downstream construction method. Some of the later dyke raise(s) were constructed by upstream method on the tailings beach. Subsurface soil conditions at the TSF site are characterized by the presence of varved grey silty clay stratum underlain by a silt and sand stratum and then till. The tailings had been deposited into the TSF by slurry spigotting. Therefore, the tailings within the TSF (which act as the foundation of some of the dyke raises) is in loose to very loose condition and susceptible to both static and seismic liquefaction.

Dam safety assessment of the existing TSF indicated that most of the perimeter dykes require upgrade to meet current dam safety guidelines and standards. The proposed upgrade involves construction of downstream toe buttress where possible and ground improvement combined with toe buttress where toe area is limited due proximity of the lake shore. Based on review of potential ground improvement options, deep soil mixing (DSM) was chosen as the preferred ground improvement method. It is also noted that the existing TSF does not have an emergency spillway, which is required to satisfy current dam safety guidelines and standards.

Based on the TSF existing conditions, upgrade requirements and closure consideration, the TSF will be raised to maximize tailings deposition in the North Cell and allow water management within the South Cell. Emergency spillway will be constructed at the South Cell. An internal spillway will be constructed to allow for passive flow from North to South Cell.

The tailings deposition plan was developed to store 1.3 Mt of tailings by slurry deposition. This provides flexibility with the backfill operation by providing additional tailings storage of 0.20 Mt beyond the PFS LOM tailings to TSF of 1.10 Mt. Based on the tailings deposition planning, Years 2021-2025 tailings will be stored in North Cell with the construction of a 3 m high upstream tailings berm over compacted tailings beach. Remaining Years 2026-2028 tailings will be stored at South Cell by raising the South Cell perimeter dykes by 1.0 m. the operating pond and environmental design flood (EDF) will be managed within the South Cell.

Water will be pumped from South Cell using a pump system to a water treatment plant located at the northeast end of the polishing pond. The pump system will be located within the South Cell with access road connecting to Dyke 10. Treated water will be discharged into the polishing pond, which will then be released to environment via the existing discharge outlet. The reclaim water for mill operation will be from the TSF South Cell via the pump system.

1.12.3 Effluent Treatment Plant

The cyanide destruction process is to be restarted, which will produce cyanate ions and ammonia. The residual concentrations of those contaminants are usually high enough to exceed lethality limits of rainbow trout and daphnia magna therefore, a water treatment plant (WTP) will be required.

The initial WTP design was based on the removal of dissolved metal with co-precipitation with ferric sulphate solution and the removal of nitrogen compounds with the MBBR system (moving bed biofilm reactor) process. Both processes produce sludge that was to be returned to the tailings pond. This initial design had a high probability of having over capacity resulting in a higher than needed investment. Furthermore, it is not possible to accurately predict the behaviour of the contaminants, and therefore to identify what the right design should be.

As a result, a mobile treatment plant based on filtration and reverse osmosis is to be used. It will allow the removal of heavy metals cyanide complexes and reject a stream containing contaminants back to the South Cell. This solution is to be implemented for the first year while an extensive sampling plan is conducted to determine the design criteria of the permanent water treatment plant, if it is needed, or if the mobile plant can be kept in operation. This PFS assumes that the mobile plant is used for the life of mine.

1.13 Environmental and Permitting

Preliminary focused natural and social environment studies were completed in 2020 to update existing information. There were no findings that were in conflict with the proposed design. Wesdome is completing additional studies as needed to support obtaining the necessary environmental approvals. The necessary environmental approvals are in the process of being obtained by Wesdome. Proposed re-opening of the mine will provide local economic benefits and jobs but will also cause air quality and noise emissions and require the discharge of treated effluent to the environment. The environmental effects expected are typical of mining operations in these settings. Stabilization of the existing TSF will reduce the potential for environmental damage from this older facility. Reclamation of the mine progressively during operations and on eventual closure of the mine will return the site to a more naturalized condition.

1.14 Capital and Operating Costs Estimates

1.14.1 Capital Costs

The total capital cost (+/-25%) over the LOM of the Project including indirect costs, contingencies and funding of reclamation activities is estimated to be C$231.8M. Items such as escalation, risk events, sunk costs, and financing costs are not included in the cost estimate. The calculated contingency is C$22.2M corresponding to 10.6% of the overall capital costs, to which an amount of C$0.3M is added for risk mitigation measures, for a total contingency amount of C$22.5M. Table 1-6 presents an overview of capital costs by WBS.

Table 1-6: Capital costs summary by WBS (in thousands C$)

WBS Description	Grand total
Infrastructure	3,668
Administration and Services	104
Mine	130,260
Processing Plant	2,419
Tailings Storage Facility & Water Management	53,418
Total Direct Costs	**189,869**
Owner's Costs	4,657
Project Indirect Costs	14,736
Project Contingency	22,544
Total Indirect Costs	**41,937**
Grand total	**231,806**

1.14.2 Operating Costs

The operating cost estimate for the Project includes all expenses incurred to operate the mine and process plant from the start of 2021 through 2028. The estimate was developed to have an accuracy of ±25% and is presented in Q4 2020, Canadian dollars (CAD). It is anticipated that the Project's workforce requirements will average 205 employees over the life of mine. The total operating cost estimates for the LOM are provided in Table 1-7. The overall LOM operating cost is C$295.5M over 7.1 years, or C$187.71/t of ore mined.

Table 1-7: Operating costs summary (in C$)

Item	Total LOM (C$M)	Average LOM (C$/ore tonne mined)	OPEX (%)
Mining	179.2	113.79	60.6
Processing, Lab & Tailings Management	44.5	28.25	15.0
Water Treatment	5.6	3.55	1.9
General and Administration	57.5	36.53	19.5
Paste Plant	8.8	5.58	3.0
Total	**295.5**	**187.71**	**100.0**

1.15 Project Economics

A financial analysis for the Project was carried out using a discounted cash flow approach. The internal rate of return (IRR) on total investment was calculated based on 100% equity financing, even though Wesdome may decide in the future to finance part of the Project using alternative sources of capital. The net present value (NPV) was calculated from the cash flow generated by the Project based on a discount rate of 5%. The payback period based on the undiscounted annual cash flow of the Project was also indicated as a financial measure.

No inflation or escalation exists in the economic model. The taxation calculations for the Project were made based on a simplified spreadsheet model that was provided by Wesdome. The model calculates pre-tax and after-tax returns and includes the Canadian Corporate Income Tax, consisting of the federal income tax and the provincial (Quebec) income tax; and the Quebec Mining Tax. The model applies 0% royalties on net smelter returns across the life of mine, as there are no royalties applicable to the Kiena Mine Complex project. The model includes provisions for doré transportation, refining and payable charges. The mine is currently expected to start commercial production from the third quarter (Q3) of 2021; however, the financial analysis assumes that all capital expenditures in 2021 are sustaining and that there is no preproduction period. The major inputs and assumptions used for the development of the financial model are listed in Table 1-8.

Table 1-8: Financial model inputs

Execution Plan	
Preproduction Period	0 months
Mine Life (after preproduction)	7.1 years
LOM Ore Tonnes Mined (millions)	1.574
LOM Gold Head Grade (Au g/t)	11.89
Average Annual Gold Production Payable (oz)	83,574
Metal Pricing and FX	
Gold Price (US$/oz)	1,600
USD to CAD exchange rate	1.32
Cost and Tax Criteria	
Estimate Basis	Q4 2020
Inflation/Currency Fluctuation	None
Leverage	100% Equity
Income Tax	Federal, provincial (QC)
Mining Tax	Quebec
Royalties	
Royalty on Net Smelter Return (NSR)	0%
Gold Transportation, Refining, and Payable Charges	
Gold (C$/oz)	3
Payable Terms	
Gold	99.975%

Table 1-9 presents the results of the prefeasibility study.

Table 1-9: Summary of prefeasibility study results

Description	Unit	Value
Total Ore Tonnes Mined	Mt	1.574
Average Gold Head Grade	g/t	11.89
Total Gold Contained	oz	601,653
Total Gold Payable	oz	592,113
Average Annual Gold Production Payable	oz	83,574
Total Preproduction Capital Cost	C$M	0.0
Sustaining Capital	C$M	229.8
Site Restoration Cost	C$M	2.0
Operating Cost	C$/t	187.71
All-in Sustaining Costs (AISC)	US$/oz	676
Total LOM NSR Revenue	C$M	1,250.1
Total LOM Operating Costs	C$M	295.5
Total LOM Pre-tax Cash Flow	C$M	722.8
Average Annual Pre-tax Cash Flow	C$M	102.0
LOM Royalties	C$M	0.0
LOM Mining Taxes	C$M	77.9
LOM Income Taxes	C$M	174.2
Total LOM After-tax Free Cash Flow	C$M	470.6
Average Annual After-tax Free Cash Flow	C$M	66.4
Pre-tax Summary		
Pre-tax NPV (@ 5% Discount Rate)	C$M	569.1
Pre-tax IRR	%	135
Pre-tax Payback (after start of operations)	year	2.2
After-tax Summary		
After-tax NPV (@ 5% Discount Rate)	C$M	366.6
After-tax IRR	%	98
After-tax Payback (after start of operations)	year	2.7

The pre-tax internal rate of return (IRR) is 135% and the pre-tax net present value (NPV) using a 5% discount rate over the mine life is C$569.1M. The after-tax IRR is 98% and the after-tax NPV at a discount rate of 5% over the mine life is C$366.6M.

The results of the after–tax sensitivity analysis performed are summarized in Figure 1-3 and Figure 1-4. This sensitivity analysis shows that the economic viability of the Project will not be significantly impacted by variations in the capital or operating costs, and that the NPV and IRR of the Project are positive over the range of values used for the sensitivity analysis when analyzed individually.

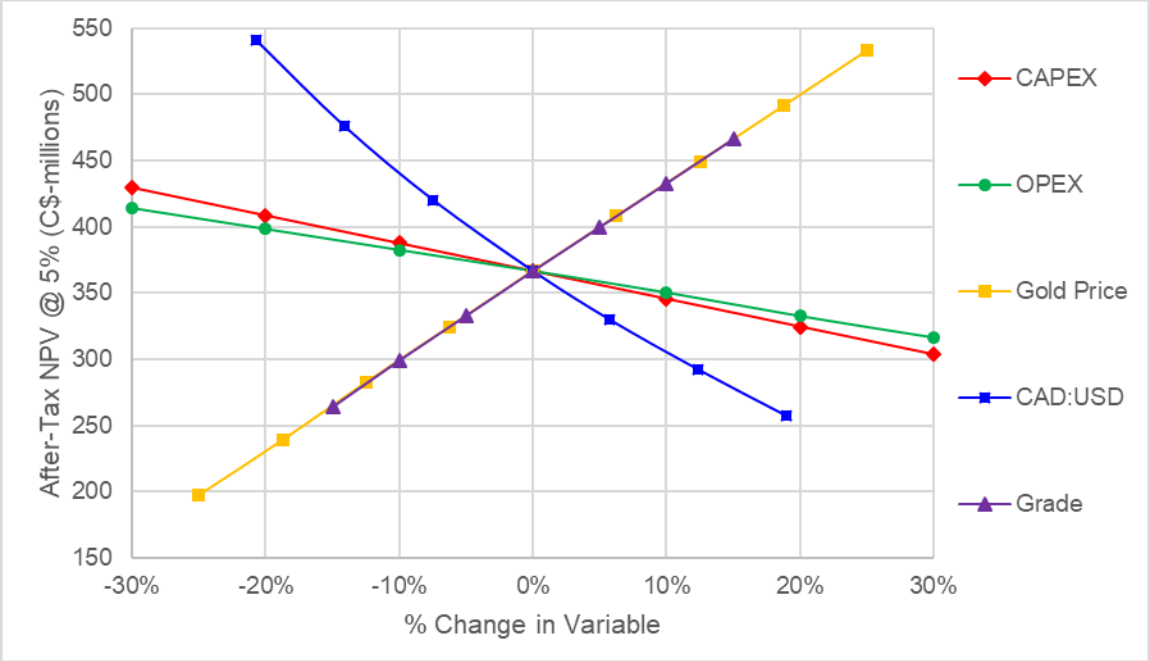


Figure 1-3: After-tax sensitivity analysis for project net present value (NPV @ 5% discount rate)

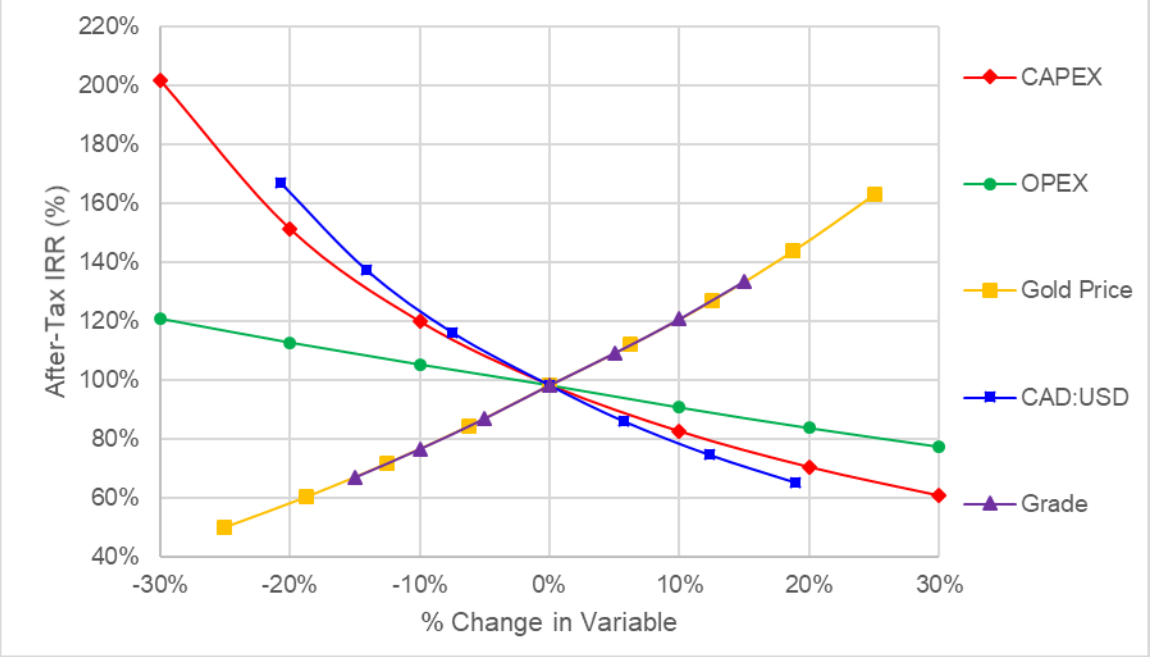


Figure 1-4: After-tax sensitivity analysis for project internal rate of return (IRR %)

1.16 Project Schedule and Organization

An execution schedule for permitting, engineering, procurement, and construction of the Project was developed as part of the prefeasibility study. This schedule is preliminary and will be further refined as the Project progresses. The major project activity milestones are presented in Figure 1-5.

Activity ID	Activity Name	Remaining Duration	Start	Finish	Total Float
Kiena Mine - Execution Schedule		412	1-Feb-21 A	9-Nov-22	0
Project Milestones		373	1-Feb-21 A	9-Nov-22	0
A3880	Permitting Activities Start	0	1-Feb-21 A		
A3710	Limited Notice to Proceed	0	22-Feb-21 A		
A2970	Upgraded Powerline Requested from Hydro-Quebec	0	7-Jun-21		42
A3700	Wesdome Notice to Proceed	0	1-Jul-21*		0
A4250	Boreholes Complete	0		4-Oct-21*	280
A3900	Cyanide Destruction Commissioning Complete	0		29-Oct-21*	0
A3040	Tailings Lift 1 Complete	0		9-Mar-22	175
A3000	Effluent Treatment Plant Commissioning Complete	0		22-Mar-22	166
A2990	Paste Plant Commissioning Complete	0		3-Jun-22*	20
A3720	Mobile Equipment Procurement Complete	0		27-Jul-22	75
A2960	Upgraded Powerline Complete	0		1-Sep-22*	42
A3010	Process Plant Commissioning Complete	0		5-Oct-22	25
A3030	Ventilation Upgrades Complete	0		5-Oct-22	25
A2910	Construction Complete	0		9-Nov-22	0
A3440	Commissioning Complete	0		9-Nov-22*	0
Permitting		231	1-Feb-21 A	1-Mar-22	181
Detailed Engineering		307	5-Apr-21 A	15-Jun-22	85
Procurement		342	23-Feb-21 A	3-Aug-22	70
Construction		412	23-Feb-21 A	9-Nov-22	0
Commissioning		263	4-Oct-21	5-Oct-22	25
Mining		109	1-Sep-21	1-Feb-22	201

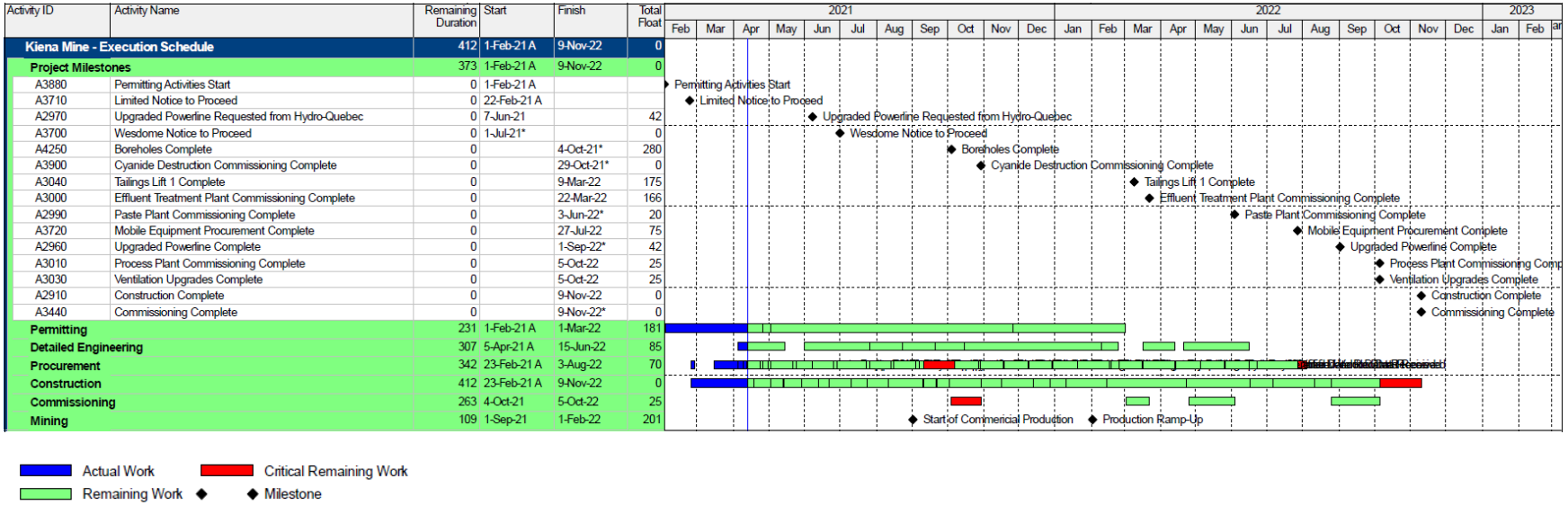


Figure 1-5: Project milestones

1.17 Interpretation and Conclusions

This Report was prepared by a group of independent consultants (QPs) to demonstrate the economic viability of restarting operations at the Kiena Complex based on the reserves estimated for the Kiena Mine Complex project.

This Report provides a summary of the results and findings from each major area of investigation to a level that is considered to be equivalent to and normally expected for a PFS of a resource development project. Standard industry practices, equipment and processes were used in this study. The authors of this report, on the date of publication, are not aware of any unusual or significant risks or uncertainties that could materially affect the reliability or confidence in the Project based on the information available.

The results of the Study indicate that the proposed Project is technically feasible and economically viable using the base case assumptions.

Internal and external risks that could affect the economic viability of the Project have been identified as part of this study. Measures have been proposed to mitigate internal risks; notable internal risks are mostly schedule risks deriving from the aggressive execution timeline. The main external risks to the Project are linked to the ongoing COVID-19 crisis, which is causing disruptions to the supply chains and labour, cost increases, delayed deliveries and lower productivities. The risks associated with COVID-19 have been deemed business risks and as such have not been considered in this study.

1.18 Recommendations

A decision has been made by Wesdome to proceed directly to the execution phase of the Project from the completion of the PFS. While the Project shows positive returns over the range of input variables used in the sensitivity analysis, the reader is advised that the typical risks of executing a project based on PFS level engineering and planning still apply.

It is recommended to proceed with the next steps as per the execution plan presented in Chapter 24, namely detailed engineering and procurement. To that end, it is recommended that Wesdome assembles the execution team, engages the detail engineering firms, and collects firm bids for critical items to meet the schedule.

2. INTRODUCTION

This Report was prepared and compiled by BBA Inc. (BBA) at the request of Wesdome Gold Mines Ltd. (Wesdome or the "Company"). The purpose of this Report is to summarize the results of the Prefeasibility Study (PFS) of the Kiena Mine Complex project (the "Project") in accordance with the guidelines of the Canadian Securities Administrators National Instrument 43-101 (NI 43-101) and Form 43-101F1.

BBA is an independent engineering consulting firm headquartered in Mont-Saint-Hilaire, Quebec, with mining groups based in Montreal, Vancouver, Toronto, Val-d'Or, and Sudbury. The Toronto team led the overall integration of this Report. This report was prepared under PTAG Inc. Project Manager, Mr. Lester Bélanger, and PTAG's team acting on behalf of the Company, based on contributions from a number of independent consulting firms including Mine Paste Ltd. (Mine Paste), Soutex Inc. (Soutex), and Wood Canada Ltd. (Wood).

2.1 Wesdome Gold Mines Ltd.

Wesdome Gold Mines Ltd. is a mining, exploration and development company based in Toronto, Ontario, focused primarily on gold. The Company has three ongoing projects in Canada: the Eagle River Complex in Wawa, Ontario, with two mineral deposits and a processing plant currently operating; the Moss Lake deposit in Thunder Bay, Ontario, at the exploration stage; and the Kiena Mine Complex in Val-d'Or, Quebec. The Company is in the process of divesting of its Moss Lake gold deposit.

The Kiena Mine Complex is an underground mine that has been on care and maintenance since mid-2013, with a 2,000 tpd mill and a 930 m hoisting shaft. A total of 1.75 million ounces of gold has been produced at the Kiena Mine Complex since 1981. Following new gold discoveries, Wesdome is looking to restart operations at the Kiena Mine Complex.

2.2 Basis of Technical Report

The following Report presents the results of the PFS for the development of the Kiena Mine Complex project. As of the date of this Report, Wesdome is a Canadian gold producer trading on the TSX under the trading symbol (WDO), with its head office situated at:

220 Bay St, Suite 1200
Toronto, Ontario
M5J 2W4

This Report, titled "Prefeasibility Study for the Kiena Mine Complex Project" (the "Report"), was prepared by Qualified Persons (QPs) following the guidelines of the NI 43-101 and in conformity with the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Reserves.

2.3 Report Responsibility and Qualified Persons

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional institutions.

- Charlotte Athurion, P. Geo. — BBA Inc.
- Navin Gangadin, P. Eng. — BBA Inc.
- Pierre-Luc Richard, P. Geo. — BBA Inc.
- Michael Stochmal, P. Eng. — BBA Inc.
- Frank Palkovits, P. Eng. — Mine Paste Ltd.
- Pierre Roy, P. Eng. — Soutex Inc.
- Sheila Daniel, P. Geo. — Wood Canada Ltd.
- Narendra Verma, P. Eng. — Wood Canada Ltd.

The preceding QPs have contributed to the writing of this Report and have provided QP certificates, included at the beginning of this Report. The information contained in the certificates outlines the sections in this Report for which each QP is responsible. Each QP has also contributed figures, tables and portions of Chapters 1 (Summary), 2 (Introduction), 3 (Reliance on Other Experts), 25 (Interpretation and Conclusions), 26 (Recommendations), and 27 (References). Table 2-1 outlines the responsibilities for the various sections of the Report and the name of the corresponding Qualified Person.

Table 2-1: Qualified Persons and areas of report responsibility

Chapter	Description	Qualified Person	Company	Comments and exceptions
1.	Executive Summary	N. Gangadin	BBA	All QPs contributed based on their respective scope of work and the Chapters/Sections under their responsibility.
2.	Introduction	N. Gangadin	BBA	All QPs contributed based on their respective scope of work and the Chapters/Sections under their responsibility.
3.	Reliance on Other Experts	N. Gangadin	BBA	All QPs contributed based on their respective scope of work and the Chapters/Sections under their responsibility.
4.	Project Property Description and Location	P-L. Richard	BBA	All Chapter 4
5.	Accessibility, Climate, Local Resource, Infrastructure and Physiography	P-L. Richard	BBA	All Chapter 5
6.	History	P-L. Richard	BBA	All Chapter 6
7.	Geological Setting and Mineralization	P-L. Richard	BBA	All Chapter 7

Chapter	Description	Qualified Person	Company	Comments and exceptions
8.	Deposit Types	P-L. Richard	BBA	All Chapter 8
9.	Exploration	P-L. Richard	BBA	All Chapter 9
10.	Drilling	P-L. Richard	BBA	All Chapter 10
11.	Sample Preparation, Analyses and Security	P-L. Richard	BBA	All Chapter 11
12.	Data Verification	P-L. Richard	BBA	Shared responsibility for all Chapter 12
		C. Athurion	BBA	Shared responsibility for all Chapter 12
13.	Mineral Processing and Metallurgical Testing	P. Roy	Soutex	All Chapter 13
14.	Mineral Resource Estimate	P-L. Richard	BBA	Shared responsibility for all Chapter 14
		C. Athurion	BBA	Shared responsibility for all Chapter 14
15.	Mineral Reserve Estimate	M. Stochmal	BBA	All Chapter 15
16.	Mining Methods	M. Stochmal	BBA	All Chapter 16 excluding Section 16.8.9
		F. Palkovits	Mine Paste	Section 16.8.9
17.	Recovery Methods	P. Roy	Soutex	All Chapter 17
18.	Project Infrastructure	N. Verma	Wood	Sections 18.2 and 18.11
		P. Roy	Soutex	Section 18.12
		N. Gangadin	BBA	Sections 18.1 and 18.3 to 18.10
19.	Market Studies and Contracts	N. Gangadin	BBA	All Chapter 19
20.	Environmental Studies, Permitting, and Social or Community Impact	S. Daniel	Wood	All Chapter 20
21.	Capital and Operating Costs	M. Stochmal	BBA	Sections 21.1.7 and 21.2.3
		P. Roy	Soutex	Sections 21.1.8, 21.1.9, 21.2.4 and 21.2.5
		N. Verma	Wood	Shared responsibility for section 21.1.10
		S. Daniel	Wood	Shared responsibility for section 21.1.10
		N. Gangadin	BBA	Sections 21.1.1 to 21.1.6, 21.1.12 to 21.1.15, 21.2.1, 21.2.2, 21.2.7, and 21.2.8
		F. Palkovits	Mine Paste	Sections 21.1.11 and 21.2.6
22.	Economic Analysis	N. Gangadin	BBA	All Chapter 22
23.	Adjacent Properties	P-L. Richard	BBA	All Chapter 23
24.	Other Relevant Data and Information	N. Gangadin	BBA	All Chapter 24
25.	Interpretation and Conclusions	N. Gangadin	BBA	All QPs contributed based on their respective scope of work and the Chapters/Sections under their responsibility.
26.	Recommendations	N. Gangadin	BBA	All QPs contributed based on their respective scope of work and the Chapters/Sections under their responsibility.
27.	References	N. Gangadin	BBA	All QPs contributed based on their respective scope of work and the Chapters/Sections under their responsibility.

2.4 Effective Dates and Declaration

The Report has several cut-off dates for information:

- Effective date of the Kiena Mine Complex project Mineral Resource Estimate used as the basis for the life of mine (LOM) plan: October 31, 2020;
- Drill Database close-out date: September 18, 2020 (used as the basis for the Mineral Resource Estimate);
- Date of last supply of laboratory testwork and investigations: December 17, 2020;
- Cut-off date for sunk costs: December 31, 2020;
- Date of the final LOM plan: April 12, 2021.

This Report was prepared as National Instrument 43-101 (NI 43-101) Technical Report for Wesdome Gold Mines Ltd. by QPs from the following firms: BBA Inc., Mine Paste Ltd., Soutex Inc., and Wood Canada Ltd.; collectively the "Report Authors". The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on: i) information available at the time of preparation, ii) data supplied by outside sources; and iii) the assumptions, conditions, and qualifications set forth in this Report. This Report is intended for use by Wesdome subject to the terms and conditions of its respective contracts with the Report Authors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other use of this Report by any third party is at the sole risk of that party.

As of the effective date of this Report, the QPs are not aware of any known litigation potentially affecting the Project. The QPs did not verify the legality or terms of any underlying agreement(s) that may exist concerning the Project ownership, permits, off-take agreements, license agreements, royalties or other agreement(s) between Wesdome and any third parties.

The results of this Report are not dependent upon prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings with Wesdome and the QPs. The QPs are being paid a fee for their work in accordance with the normal professional consulting practice.

The opinions contained herein are based on information collected throughout the course of the investigations by the QPs, which in turn reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results can be significantly more or less favourable.

2.5 Sources of Information

2.5.1 General

This Report is based in part on internal company reports, maps, published government reports, company letters and memoranda, and public information, as listed in Chapter 27 “References” of this Report.

This PFS has been completed using available information contained in, but not limited to, the following reports, documents and discussions:

- Technical discussions and emails with Wesdome personnel;
- QPs’ personal inspection of the Project site(s);
- Project team members inspection of the Project site(s);
- Report of mineralogical, metallurgical and grindability characteristics of the Kiena Mine Complex deposits, conducted by industry recognized metallurgical testing laboratories on behalf of Wesdome;
- Internal and commercially available databases and cost models;
- Various reports covering site hydrology, hydrogeology, geotechnical, and geochemistry;
- Various reports covering site physical and biological environment;
- Additional information from public domain sources.

The QPs have no known reason to believe that any of the information used to prepare this Report and evaluate the mineral resources presented herein is invalid or contains misrepresentations.

The authors have sourced the information for this Report from the collection of documents listed in Chapter 27 (References).

2.5.2 BBA

The following individuals provided specialist input to Charlotte Athurion, QP, and Pierre-Luc Richard, QP:

- Karine Brousseau, Senior Engineer - Mineral Resources, Wesdome, provided the 3D modelling and block models to be audited and described in Chapter 14 (Mineral Resource Estimate);
- Bruno Turcotte, Senior Project Geologist, Wesdome, provided information regarding the QA/QC and drilling, and sampling protocols;
- Frank Gagnon, Environment Coordinator of Wesdome at the Kiena Mine Complex provided information regarding the Project’s environmental status, permits, and Social and Community Impact.

The following individuals provided specialist input to Navin Gangadin, QP:

- Sean McCormack, General Manager, Wesdome, and Marc-André Pelletier, COO, Wesdome, provided incurred project capital costs for the development of the Project's cost estimates as well as salaries for operating personnel used to develop labour operating costs found in Chapter 21 (Capital and Operating Costs);
- Jocelyn Marcoux, BBA, provided the industrial standards, norms and factors for the various material, manpower and construction costs used in the development of the capital costs (Chapter 21);
- Scott Gilbert, CFO, Wesdome, provided input parameters to be used in the tax calculations of the financial model used in the development of the economic analysis (Chapter 22);
- Lester Bélanger, Project Manager, PTAG, provided information regarding the Project's execution plan found in Chapter 24 (Other Relevant Data and Information);
- Neil O'Connor, TMG, provided the information regarding the Project execution schedule found in Chapter 24 (Other Relevant Data and Information).

In achieving the mandates represented by Chapter 15 (Mineral Reserve Estimate), Chapter 16 (Mining Methods), and Chapter 21 (Capital and Operating Costs), the following individuals provided specialist input (geology, mining and operational) to Michael Stochmal, QP:

- Marc-André Pelletier, COO, Wesdome, provided historic and operational information pertaining to the site and specifically the underground mine, input into mine schedule rates, mining recoveries and dilution factors, costs for consumable material and mobile equipment considered in Chapters 15, 16 and 21;
- Sean McCormack, General Manager, Wesdome, provided historic and operational information pertaining to the site and specifically the underground mine, input into mine schedule rates, mining recoveries and dilution factors, costs for consumable material and contractor unit costs for mine development, labour salaries, and costs for mobile equipment considered in Chapters 15, 16 and 21;
- Karine Brousseau, Senior Engineer Mineral Resources, Wesdome, provided historic and technical information pertaining to mining recoveries and dilution factors, mineral resource block models, diamond drillhole logs and pictures considered in Chapters 15 and 16;
- Bruno Turcotte, Senior Project Geologist, Wesdome, provided the annual diamond drillhole lengths and contracted unit cost, mineral resource block models, diamond drillhole logs and pictures considered in Chapters 15, 16 and 21;
- Dany Pomerleau, Engineering Coordinator, Wesdome, provided historic and technical information pertaining to mine operations, mine as-built solids, planned mine development solids, production stope solids, and the mine schedule considered in Chapters 15 and 16;
- Simon Fontaine, Mining Engineer, Wesdome, provided mine as-built solids, planned mine development solids, production stope solids, and the mine schedule considered in Chapters 15 and 16;

- Alex Coté Gagné, Mining Technician, Wesdome, provided Ventsim models and mine air ventilation information used to assess the requirement for mine air ventilation infrastructure upgrades and to assess the requirement for mine air ventilation cooling considered in Chapters 16 and 21;
- Paul Robitaille, Logistics/Maintenance Superintendent, Wesdome, provided operational and technical information pertaining to the underground mine hoisting, electrical, communications, process water, underground service repair and mine dewatering systems considered in Chapter 16;
- Billy Lamothe, Mill Superintendent, Wesdome, provided historic information of the hydraulic tailings backfill, surface plant and mill considered in Chapter 16;
- Martin Gauthier, Contract Manager, Wesdome, provided cost estimates and secured pricing for consumable material considered in both capital and operating cost estimations in Chapter 21.

2.5.3 Mine Paste

The following individuals provided specialist input to Frank Palkovits, QP:

- Laboratory testing of mine backfill materials conducted by RMS Corporation located in Burnaby, BC.

2.5.4 Soutex

The following individuals provided specialist input to Pierre Roy, QP:

- Marc-André Pelletier, COO, Wesdome, and Billy Lamothe, Superintendent, Wesdome, provided processing plant capital and operating costs used in the development of the Project's cost estimates found in Chapter 21 (Capital and Operating Costs);
- Pierre Leblanc, Senior Project Manager, ASDR, provided ETP capital costs used in the development of the Project's cost estimates found in Chapter 21 (Capital and Operating Costs).

2.5.5 Wood

The following individuals provided specialist input to Sheila Daniel, QP:

- Guillaume Angers, Environmental Superintendent, Wesdome, provided information regarding consultation activities completed by Wesdome to date related to the Kiena Mine, environmental approval requirements based on his regulatory discussions and reclamation costing found in Chapter 20 (Environmental Studies, Permitting, and Social or Community Impact).

The following individuals provided specialist input to Narendra Verma, QP:

- Ganan Nadarajah, P.Eng., developed the TSF design and capital cost estimates with assistance of Nam Pham, P.Eng., Norman Schwartz, P.Eng., and Steven Gouritin, and provided required information.

2.6 Site Visits

The following bulleted list describes which Qualified Persons visited the Project site that is the subject of this Report, the date of the site visit, and the general objective of the site visit:

- Charlotte Athurion, P. Geo. (BBA) visited the Kiena Mine Complex that is the subject of this Report on October 16, 2020, and on November 17, 2020 as part of the Mineral Resource Estimate mandate. The purpose of the visits was to review the Project with the Wesdome team. The visits included an overview of the general geological conditions, a tour of the core storage facility, and visual inspections of select mineralized drill core samples. A review of assaying, QA/QC and drillhole procedures was also completed. In the past two years, Charlotte Athurion also visited the Wesdome office several times while supporting Wesdome's geologist with the geological interpretation and modelling. From August to November 2020, she provided geology support during the underground bulk sampling.
- Navin Gangadin, P. Eng. (BBA) visited the Kiena Mine Complex that is the subject of this Report on September 5, 2019 and on February 18, 2020. The objective of the site visits was to review the Project scope with Wesdome, tour the site and mill, and to visually inspect the site infrastructure and administrative facilities.
- Pierre-Luc Richard, P. Geo. (BBA) visited the Kiena Mine Complex that is the subject of this Report from August 6 to 8, 2019 and on other occasions as part of previous mandates. The purpose of the visits was to review work with the Wesdome team, discuss general geological conditions, tour the core storage facility, conduct visual inspections of select mineralized drill core samples and conduct an underground visit. A review of assaying, QA/QC and drillhole procedures was also completed.
- Michael Stochmal, P.Eng. (BBA) visited the Kiena Mine Complex that is subject of this Report from September 29 to October 1, 2020. The objective of the site visits was to inspect surface facilities associated with mining, tour the underground mine and gather technical data pertaining to surface facilities, primarily backfill, as well to the underground mine.
- Frank Palkovits, P.Eng. (Mine Paste) visited the Kiena Mine Complex that is the subject of this Report from November 24, 2020 to November 25, 2020. The objective of the site visits was to inspect the surface facilities including the mill, thickening area, hydraulic fill plant, mining plans, tailings storage facility and general knowledge of the area.
- Pierre Roy, P.Eng. (Soutex) visited the Kiena Mine Complex that is the subject of this Report from November 16 to 19, 2020. The purpose of the visits was to provide support for the bulk sample processing. The visits included a review of the Project with the Wesdome team and a tour of the processing plant.

As of the effective date of this Report, the following QPs have not visited the Project site:

- Sheila Daniel, P. Geo. (Wood);
- Narendra Verma, P. Eng. (Wood).

2.7 Currency, Units of Measure, and Calculations

Unless otherwise specified or noted, the units used in this Report are metric. Every effort has been made to clearly display the appropriate units being used throughout this Report.

- Currency is in Canadian dollars (CAD or C$), unless otherwise noted;
- All ounce units are reported in troy ounces, unless otherwise stated:
  1 oz (troy) = 31.1 g = 1.1 oz (Imperial);
- All metal prices are expressed in US dollars (USD or US$);
- A United States dollar (USD) to Canadian dollar (CAD) exchange rate of 1.32 CAD for 1.00 USD was used;
- All cost estimates have a base date of the fourth quarter (Q4) of 2020.

This Report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs consider them immaterial.

2.8 Acknowledgement

BBA and the other contributors to the study would like to acknowledge the following individuals for their general support provided during this assignment:

The Project benefited from the specific input of Marc-André Pelletier, Sean McCormack, Scott Gilbert, Guillaume Angers, Billy Lamothe, Paul Robitaille, Michael Michaud, Bruno Turcotte, Karine Brousseau, Dany Pomerleau, Simon Fontaine, Alex Cote Gagne, Lester Bélanger, Mario Robalo, Tareq Chehadeh, Neil O'Connor, Ashanie Gordon Bailey, Jocelyn Marcoux, Marc-André Sioui, Jérôme Martin, Serge Lebeuf, Cameron Bruin, David Bleiker, Manon Dussault and Ganan Nadarajah. Their contributions are greatly appreciated.

3. RELIANCE ON OTHER EXPERTS

3.1 Introduction

The Qualified Persons (QPs) relied on reports, information sources and opinions provided by Wesdome and external experts for certain aspects of the Project, such as the Project's mineral rights, 3rd party agreements, surface rights, property agreements, royalties, environmental status, and fiscal situation.

As of the date of this Report, Wesdome indicates that there are no known litigations potentially affecting the Kiena Mine Complex project.

A draft copy of the Report has been reviewed for factual errors by Wesdome. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are neither false nor misleading at the date of this Report.

3.2 Mineral Tenure and Surface Rights

Wesdome supplied information regarding mining titles, options' agreements, royalty agreements, environmental liabilities and permits. Pierre-Luc Richard, QP, consulted the GESTIM online claim management system for the latest status regarding ownership and mining titles. Although Mr. Richard has the available claim status documents, he is not qualified to express any legal opinion with respect to the property titles, current ownership, or possible litigations. A description of such agreements, the property, and ownership thereof, is provided for general information purposes only. In this regard, Mr. Richard has relied on information supplied by Wesdome and the work of experts they understand to be appropriately qualified.

This information is used in Chapter 4 of the Report. The information is also used in support of the Mineral Resource Estimate in Chapter 14.

3.3 Taxation and Accounting Rules

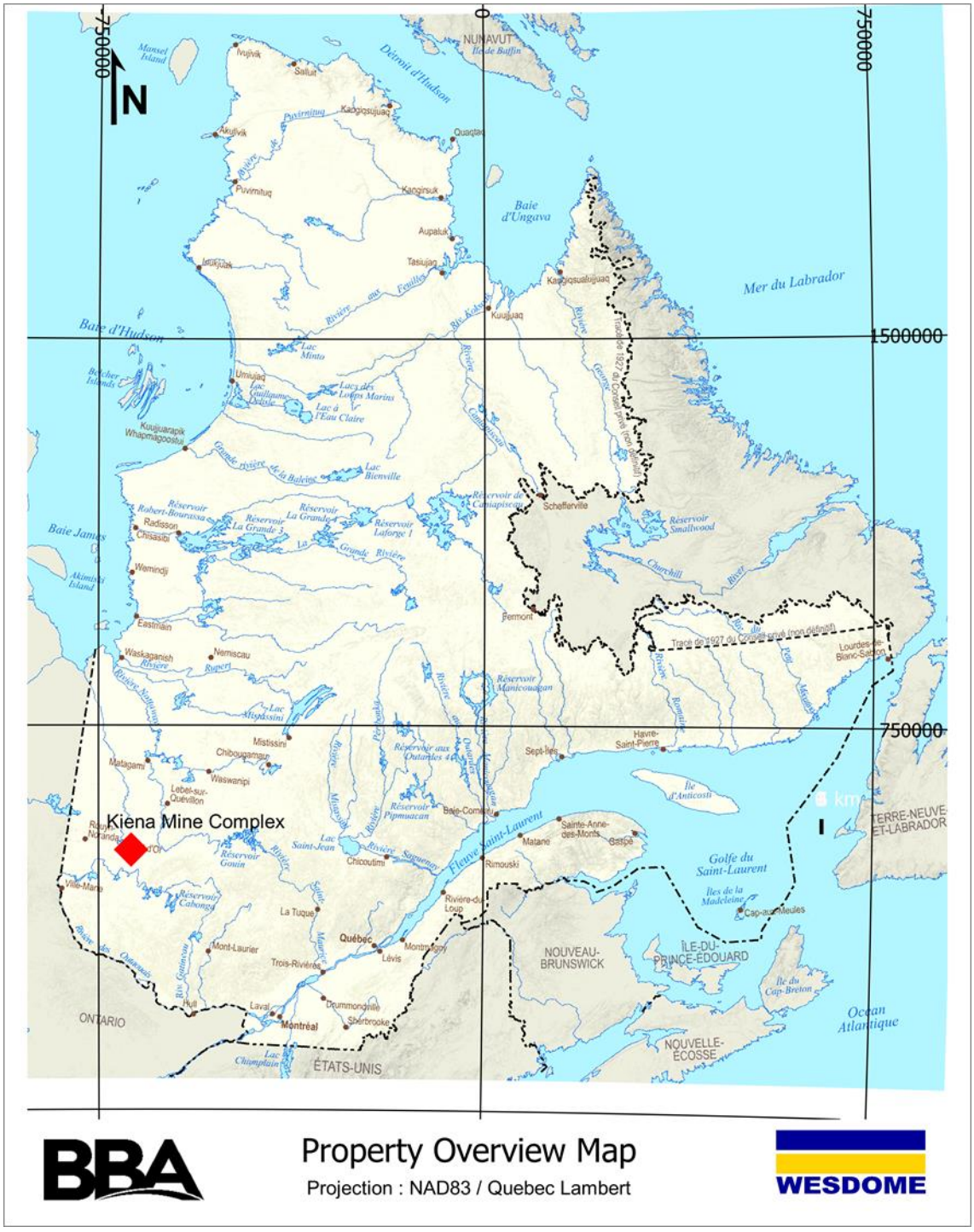
Navin Gangadin, QP, has fully relied upon, and disclaims responsibility for, information supplied by Wesdome staff and experts retained by Wesdome for information related to taxation and accounting rules as applied to the financial model. This information is used in support of the financial analysis in Chapter 22.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Property Description and Location

The Kiena Mine Complex project is located in the Province of Quebec in the Abitibi-Témiscamingue administrative region within the limits of the municipality of Val-d'Or and 100 km east of Rouyn-Noranda (Figure 4-1). It lies to the northwest of the urban centre of Val-d'Or and covers 7,047 ha.

The coordinates for the approximate centre of the Project are latitude 48°08' N and longitude 77°54' W (284105E and 5335715N: NAD 83 / UTM Zone 18). The Project lies in the townships of Dubuisson and Vassan on NTS map sheets 32D/01 and 32C/04.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Kiena Mine Complex
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Montréal
Laval
Sherbrooke
Trois-Rivières
Lévis
Rimouski
Matane
Sept-Îles
Havre-Saint-Pierre
Chibougamau
Chapais
Mistissini
Matagami
Val-d'Or
Rouyn-Noranda
Ville-Marie
Mont-Laurier
Gatineau
Hull
Drummondville
Saint-Jérôme
Chicoutimi
La Tuque
Baie-Comeau
Sainte-Anne-des-Monts
Gaspé
Cap-aux-Meules
Îles de la Madeleine
Île d'Anticosti
Île du Cap-Breton
Golfe du Saint-Laurent
Fleuve Saint-Laurent
Rivière Saguenay
Rivière Ottawa
Rivière Gatineau
Rivière Rupert
Rivière La Grande
Rivière Eastmain
Rivière Broadback
Rivière Harricana
Rivière Nottaway
Rivière Bell
Rivière Turgeon
Rivière Matamec
Rivière Manicouagan
Rivière aux Outardes
Rivière Pentecôte
Rivière Moisie
Rivière Romaine
Rivière Natashquan
Rivière Kaniapiskau
Rivière Caniapiscau
Rivière Koksoak
Rivière aux Feuilles
Rivière George
Rivière Whale
Rivière Arnaud
Rivière Payne
Rivière Innuksuac
Rivière Povungnituk
Rivière Kogaluk
Rivière Eastmain
Rivière Petite Baleine
Rivière Grande Baleine
Rivière Churchill
Rivière Mécatina
Rivière Romaine
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Rivière George
Rivière Whale
Rivière Arnaud
Rivière Payne
Rivière Innuksuac
Rivière Povungnituk
Rivière Kogaluk

4.2 Mineral Tenure

Pierre-Luc Richard verified the status of the mineral claims using the Quebec government online claim management tool GESTIM. As of April 12, 2021, the Kiena Mine Complex property consists of a contiguous group of 183 contiguous electronic map designated mining claims and one mining concession (Figure 4-2).

From the 183 mining claims, Wesdome has 169 claims registered in their name at 100%. Six claims are held be Mines Dynacor (50%) and Wesdome (50%), which represents the Maufort property. Eight claims are held by Wesdome (75%) and 9264-7890 Québec inc. (25%), which represents the Siscoe Extension property.

The Project is located in the Dubuisson and Vassan Township, near the town of Val-d'Or in the Province of Quebec. The total area of the Project is 7,047 hectares. A detailed list of the Project mineral claims is shown in Appendix B.

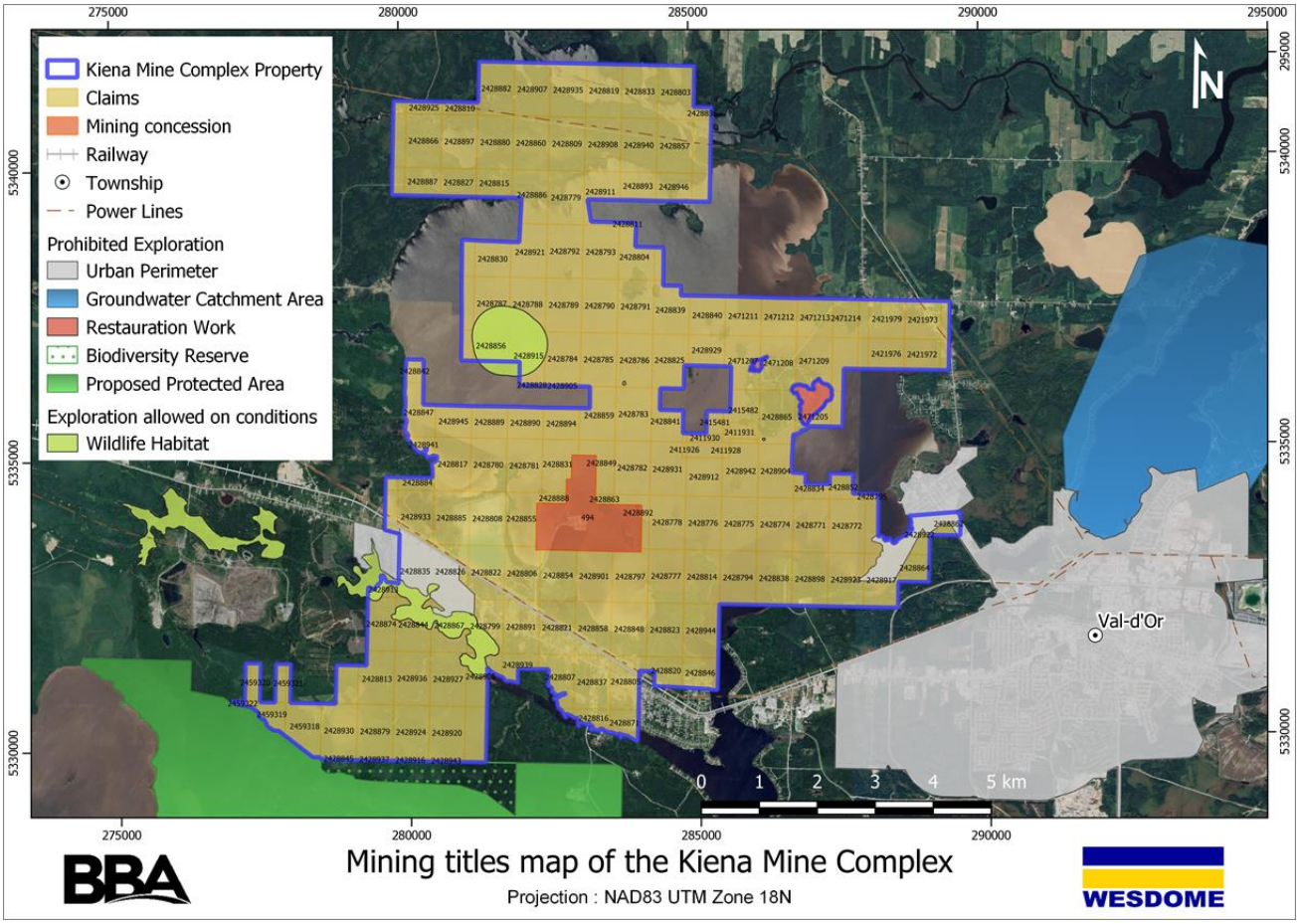


Figure 4-2: Kiena Mine Complex titles as of April 12, 2021

4.3 Royalties, Agreement and Encumbrances

Some of the mining titles comprising the Project are subject to certain agreements and royalties. Figure 4-3 shows the historical properties that were amalgamated to form the current Project, some of which have active royalties. Table 4-1 provides the details of these royalties.

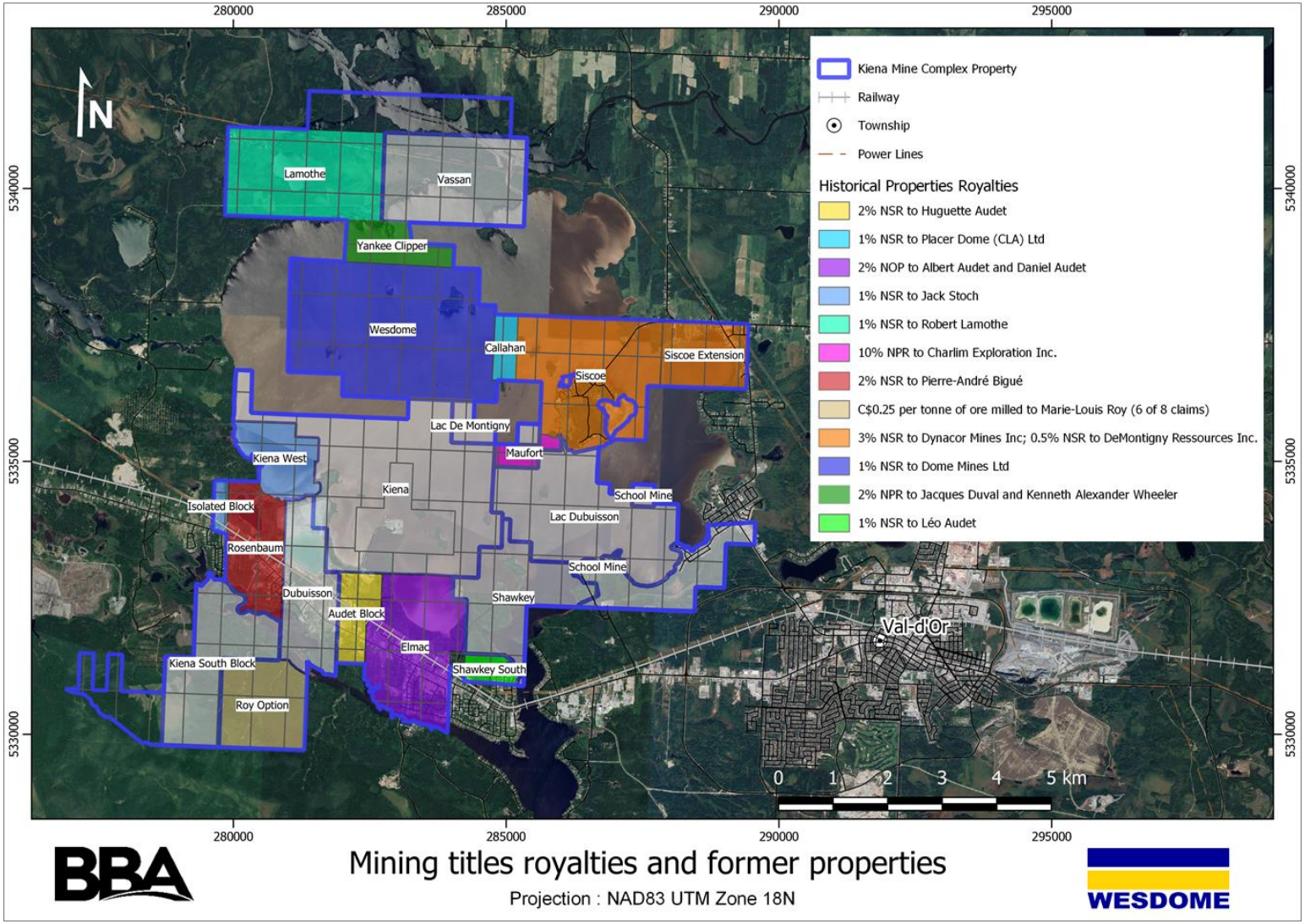


Figure 4-3: Kiena Mine Complex mining royalties and historical property names

Table 4-1: List of historical properties with their applicable royalties

Name	Royalties	Date of agreement	Details
Audet Block	2% NSR to Huguette Audet	N/A	
Callahan	1% NSR to Placer Dome (CLA) Ltd.	N/A	
Dubuisson	No royalty		
Elmac	2% NOP to Albert Audet and Daniel Audet	N/A	
Kiena	No Royalty		
Kiena South Block	No Royalty		
Kiena West	1% NSR to Jack Stoch	N/A	
Lac De Montigny	No Royalty		
Lac Dubuisson	No Royalty		
Lamothe	1% NSR to Robert Lamothe	January 15, 1998	
Maufort	10% NPR to Charlim Exploration Inc.	N/A	
Rosenbaum	2% NSR to Pierre-André Bigué	N/A	1% can be bought back for $1,000,000
Roy Option	CAD0.25 per tonne of ore milled to Marie-Louis Roy (6 of 8 claims)	N/A	
School Mine	No Royalty		
Shawkey	No Royalty		
Shawkey South	1% NSR to Léo Audet	N/A	
Siscoe and Siscoe Extension	3% NSR to Dynacor Mines Inc.	November 9, 1999	1% can be bought back for $500,000
	0.5% NSR to Demontigny Resources Inc.	N/A	0.5% can be bought for $500,000
Vassan	No Royalty		
Wesdome	1% NSR to Dome Mines Ltd.	November 21, 1997	
Yankee Clipper	2% NPR to Jacques Duval and Kenneth Alexander Wheeler	February 25, 1981	

4.4 Environmental Liabilities

Several studies were conducted before and during operations on portions of the Project related to former operations and other projects within its boundaries. The Project includes the milling and tailings facilities of the Kiena mine, nine shafts including Parker Shaft, and related underground workings from past producers and exploration projects (Figure 4-4), and various surface facilities. The remaining infrastructure components at the former Kiena mine site now constitute the principal infrastructure of the Project; these components are listed below:

- The former collar of the Parker Shaft is located close to the actual mill but is not accessible;
- A functional hoisting shaft of 930 m depth;

- A dewatered underground ramp system from 170 m (17 Level) to 1,110 m (111 Level);
- A 2,000-tpd processing plant;
- A tailing management facility;
- Core shacks;
- Offices;
- A guard house and security gate;
- Workshops and warehouses;
- A Hazmat storage facility.

Other than the exploration offices and underground exploration development, the principal infrastructure of the Project has been under care and maintenance since mid-2013. Effluent is being controlled and analytical results are submitted to federal and provincial authorities who have determined that the effluent, based on the available data, complies with regulation requirements. A recent study conducted by Stantec (Stantec, 2017) to assess the chemical behaviour of the waste rock pile concluded that the waste rocks are neither acid generating nor do they have metal leaching potential under the conditions present at the time of the study.

The environmental permits and the social acceptability of the Project, regardless of current status, will need to be reviewed should any operations resume. Moreover, both federal and provincial authorities have amended their regulations since 2013.

Any mining related work would be subject to environmental studies and permit applications. Access is easy as the Project lies within an inhabited area and some data may already have been collected through previous studies. The Project is located near a lake since the 1980's and Wesdome's representatives mentioned to the QP that there has been no social resistance or counteraction.

The last version of the Kiena Closure and Rehabilitation Plan was accepted by the MERN in September 2015. The total cost was estimated at $7.0M. The financial guarantee was completed on September 28, 2017.

In the summer of 2018, a geotechnical study (Stantec, 2018) was conducted to investigate the subsurface conditions of the area surrounding the Kiena waste rock pile. Stratigraphic profiles, cone penetration tests, shear resistance profiles, test pits, and boreholes were done for the characterization.

As of December 2018, Wesdome has kept the municipal authorities informed on their mining and exploration activities but no official consultation has been conducted with other interested stakeholders and neighbors. The main acknowledged concern is groundwater quality as it is the source of drinking water for nearby dwellings. Wesdome continued to monitor final effluent, surface water and groundwater wells after cessation of operations in 2013.

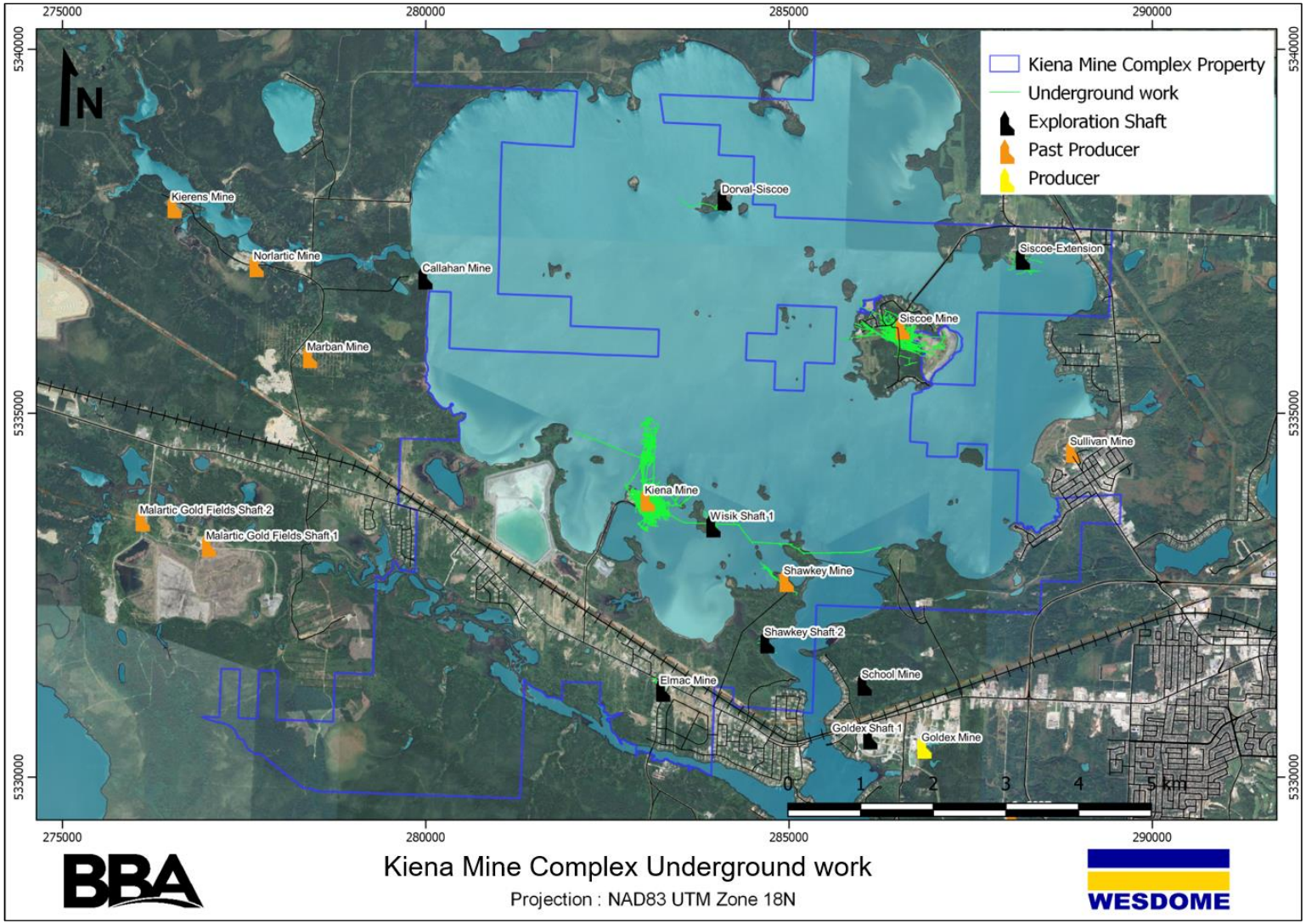


Figure 4-4: Surface projection of the Kiena Mine Complex underground work, production and exploration shafts as of end of 2018

4.5 Permitting

Wesdome has various authorization certificates for their property. Table 4-2 shows the list of certificates they currently hold with the conditions to respect. All certificates of authorization were provided by Mr. Frank Gagnon, Environment Coordinator from Wesdome at the Kiena Mine Complex. According to Mr. Gagnon, the certificates of authorization are valid and active.

Table 4-2: Wesdome certificates of authorization details

Certificate of authorization number	Title	Type	Date	Conditions
7610-08-01-70065-23	Reopening of "Les mines d'or Kiena Ltée"	Authorization certificate	1981-07-29	▪ Extraction of 1,000 tpd of ore; ▪ Truck transportation to the Lamaque Mill; ▪ Hydraulic backfill with the Lamaque Mill waste; ▪ Enlargement of the island with the waste rock; ▪ Disposal of the waste rock in a sequence; ▪ Dust control of the access roads and unpaved traffic areas; ▪ Final effluent monitoring according to the Directive 019 with additionally cadmium at 0.05 mg/l and mercury at 0.001 mg/l; ▪ All mitigation measures of the: ▪ Environmental Impact Assessments (Nov. 21, 1980): ▪ Letter of January 14, 1981; ▪ Letter of May 27, 1981.
		Cession of the authorization certificate from "Les mines d'or Kiena Ltée" to "Mc Watters"	1997-09-12	▪ Same conditions as the authorization certificate of July 29, 1981.
		Modification	2001-02-23	▪ Extraction change of 1,000 tpd to 2,040 tpd of ore.
		Cession of the authorization certificate from "Mc Watters" to "Wesdome inc."	2004-05-25	▪ Extraction of 2,040 tpd of ore.
	Expansion of the waste rock pile	Modification	2004-12-20	▪ From 160,000 t to 418,000 t; ▪ On lots 24B and 25B, road A, Dubuisson township; ▪ Extraction of 10,000 t per month of waste rock; ▪ Kiena Mine waste rock and development of the ramp towards Shawkey and Wesdome; ▪ Waste rock will be haul by truck towards the waste pile; ▪ Actual waste pile of 160,000 t, 11,800 m^2 with a height of 8 m;

Certificate of authorization number	Title	Type	Date	Conditions
				▪ Temporary extension of the waste pile to 258,000 t for 31,357 m^2 at 4.5 m high; ▪ Some waste rock will be used for raising the tailings pond, building berms and roads; ▪ Slope of 2H/1V.
7610-08-01-70065-24	Operation of the ore plant at the Kiena mine	Authorization certificate	1984-08-02	▪ Located on lots 19B to 25B, road B, Dubuisson township; ▪ Treatment plant of 1,250 tpd; ▪ Direct cyanidation and activated carbon recovery; ▪ Hydrocarbon sensor installation where they are likely to be present; ▪ Installation of leak proof structures to store a volume equivalent to the largest tank of the same group for all tanks of toxic and dangerous products; ▪ Installation of a dry or wet scrubber to capture the emissions of gas or dust generated by the treatment operations; ▪ Installation of equipment to eliminate or reduce dust emissions from storage, disposal, transhipment and transportation areas in accordance with the Regulation on the Quality of the Atmosphere; ▪ Construction of a tailings transportation line from the plant to the park; ▪ Development of a tailings pond; ▪ Construction of a polishing pond; ▪ Digging of diversion channels for drainage water at the periphery of the tailings pond; ▪ Installation of a continuous recording system for pH and flow; ▪ Cyanide treatment at the mill effluent; ▪ Effluent monitoring according to the standards established in the authorization certificate.

Certificate of authorization number	Title	Type	Date	Conditions
	Modification of the authorization certificate of Aug. 2, 1984 for the tailings pond construction	Modification	1987-07-27	▪ Lots 19B to 25B, road A, Dubuisson township; ▪ Use of the waste rocks from "Malartic Goldfields" to rise the pond; ▪ Installation of a geotextile into the structure of the pond embankment.
	Modification at the final effluent standards	Modification	1989-04-04	▪ Modification of the effluent discharge standards for the tailings pond pursuant to guidelines from Directive 019.
	Operation of the ore plant at the Kiena mine	Modification	1996-11-27	▪ Addition of two polishing ponds of 32,500 m^2 at the south end of the current pond.
		Cession of the authorization certificate from "Les mines d'or Kiena Ltée" to "Mc Watters"	1997-09-12	▪ All the conditions of the authorization certificate remain unchanged.
		Modification	2001-03-19	▪ Change of 1,250 tpd to the treatment of 2,040 tpd.
		Cession of the authorization certificate from "Mc Watters" to "Wesdome inc."	2004-05-25	▪ All the conditions of the authorization certificate remain unchanged.
7610-08-01-70065-28	Tailings pond expansion	Authorization certificate	1992-05-19	▪ Natural degradation of the cyanide with the SO_2-Air process; ▪ Technical description of the cyanide content of the hydraulic backfill; ▪ Installation of five piezometers at the tailings pond (P-6 to P-10) and a control piezometer for background (P-11); ▪ Commitment to comply with Directive 019; ▪ Regular inspections of embankments and pipes; ▪ Pumping of 125,000 m^3 per year of fresh water from Demontigny Lake; ▪ The overburden will be kept.
		Cession of the authorization certificate from "Les mines d'or Kiena Ltée." to "Mc Watters."	1997-09-12	▪ All the conditions of the authorization certificate remain unchanged.

Certificate of authorization number	Title	Type	Date	Conditions
		Cession of the authorization certificate from "Mc Watters" to "Wesdome inc."	2004-05-25	▪ All the conditions of the authorization certificate remain unchanged.
		Modification	2005-10-28	▪ Upgrading of the northern cell of the tailings pond from elevation 3,048 m to 3,050 m; ▪ Lots 18B to 22B, road A, Dubuisson township.
7610-08-01-70065-29	Clay quarry exploitation	Authorization certificate	1994-08-26	▪ Exploitation area of 40,000 m^2, with an average thickness of 1.5 m and maximum of 2 m; ▪ Lots 21B, 22B and 23B, rang A, Dubuisson township.
		Cession of the authorization certificate from "Les mines d'or Kiena ltée" to "Mc Watters"	1997-09-12	▪ All the conditions of the authorization certificate remain unchanged.
		Cession of the authorization certificate from "Mc Watters" to "Wesdome inc."	2004-06-04	▪ All the conditions of the authorization certificate remain unchanged.
7610-08-01-70065-33	Clay borrow pit exploitation	Authorization certificate	1999-05-21	▪ Topsoil will be stored on the edge of the exploitation for use during restoration (8,550 m^3); ▪ Exploitation above the water table.
		Cession of the authorization certificate from "Mc Watters" to "Wesdome inc."	2004-06-04	▪ Exploitation of 12,000 m^3 of clay on an area of 29,000 m^2, an average thickness of 1 m and maximum of 2 m on lots 18B half East and half West, road A, Dubuisson township.
7610-08-01-70065-00	Septic system	Authorization certificate	1981-02-10	▪ For the services building with a maximum capacity of 135 persons; ▪ Installation of the septic system with a capacity of 5,744 US gallons; ▪ Installation of a pumping station with two pumps; ▪ Installation of a leaching field with an area of 3,200 ft^2.

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

5.1 Accessibility

The Project is easily accessible via paved highways from local communities, such as Val-d'Or. The Project is located 10 km west of the Val-d'Or Township (Figure 5-1) along the provincial Highway 117 and turning north on the Chemin Kienawisik gravel road. The Property can be accessed and operated on a year-round basis. The nearest airport with daily flights connecting Montreal is in Val-d'Or.

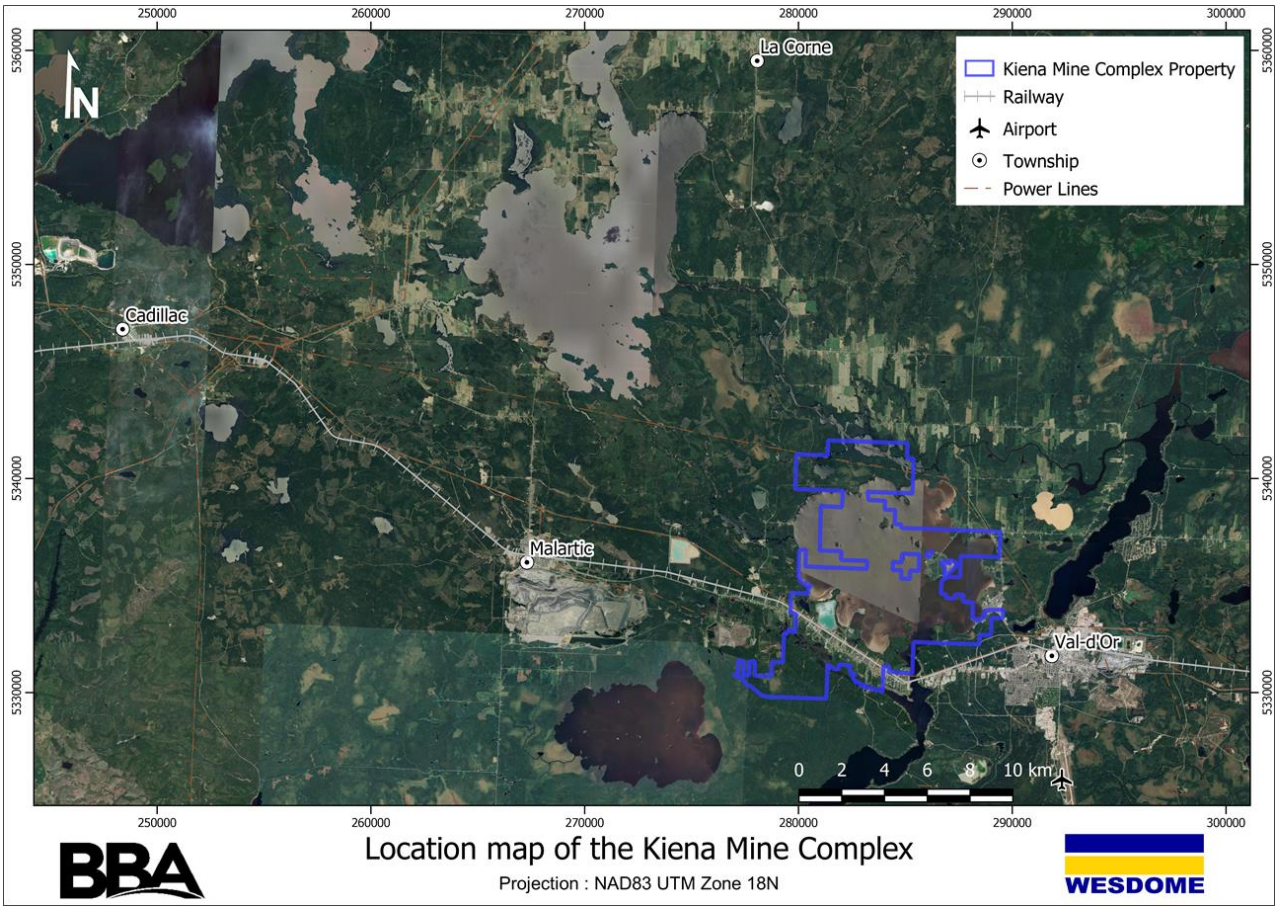


Figure 5-1: Location of the Kiena Mine Complex Property

Figure 5-2: Gravel road leading to the property with an overview of the infrastructure (© Jean-Philippe Richard, 2016)

5.2 Climate

The Val-d'Or area experiences a continental subarctic subhumid climate, characterized by short, cool summers and long, cold winters. The nearest permanent weather monitoring station maintained by Environment Canada (climat.meteo.gc.ca) is the Amos station, approximately 50 km north of the Property. According to the available data collected from 1981-2010 at this weather station, the daily average temperature for January was -17.2°C and the daily average temperature in July was 17.4°C. The record low during this period was -52.8°C, and the record high was 37.2°C.

Data collected from the Amos weather station from 1981 to 2010 indicates that the total annual precipitation was 929.0 mm, with peak rainfall occurring during July (112.1 mm average), August (98.3 mm average) and September (106.7 mm average). Snowfall is light to moderate, with an annual average of 253.3 cm. Snow typically accumulates from October to April, with a peak snowfall occurring in November (45.0 cm average), December (58.5 cm average) and January (55.6 cm average); during this period, snowpack averages 39 cm depth, with a maximum depth of approximately 142 cm. On average, the Property is frost-free for 97 days, though discontinuous permafrost exists in the area. Hours of sunlight vary from 15.5 hours at the summer solstice in June to 8.1 hours at the winter solstice in December.

The climatic conditions at the Property do not significantly impede the Project or hinder exploration or mining activities, beyond seasonal considerations for certain work. The operating season is year-round since most of the work is underground. The parts of the Project located on islands are accessible by boat during the summer season and by ice bridges in winter.

5.3 Local Resources and Infrastructure

5.3.1 Airports, Rail Terminals and Bus Services

The town of Val-d'Or, with a population of approximately 32,900 residents, is located 10 km east of the Project along the provincial Highway 117. Val-d'Or has been a mining service centre since its foundation in the 1920's. Val-d'Or is one of the largest communities in the region and has all major services including an airport with scheduled service from Montreal. CN railway line crosses the southern part of the Property, connecting east through to Montreal and west to the North American rail network. Val-d'Or is a six-hour drive from Montreal, and there are daily bus services between Montreal and the other cities in the Abitibi-Témiscamingue region.

5.3.2 Local Work Force

According to the 2016 census prepared by Statistics Canada, the population of the MRC of La Vallée-de-l'Or was 43,226 people, with 66% of the residents aged 15-64, and an average of 41 years old. Male population accounts for 51% of the population, 49% is female, and 8.5% is Aboriginal. In 2016, 64.4% of the population participated in the labour force, with 14.2% of the labour force employed in mining, quarrying, and oil and gas extraction. This portion of the workforce is experienced in mining operations, as they are currently employed at exploration and gold mines located elsewhere in the region. Local resources also include commercial laboratories, drilling companies, exploration service companies, engineering consultants, construction contractors and equipment suppliers.

5.3.3 Additional Support Services

Additional services within the town of Val-d'Or include the Val-d'Or Hospital, grocery stores, fuel stations, financial institutions and hotels. Val-d'Or has a Canada Post office and additional shipping/freight services by several providers. Landline telephone, mobile service, high-speed internet and satellite internet are available in town and the nearby vicinity.

5.4 Physiography

The Project is located in the Abitibi Greenstone Belt of the Superior Province. The topography of the area is characterized by low ridges and hills flanked by generally flat areas of glacial outwash, swamps and numerous lakes and bogs. The average elevation varies from 300 m a.s.l. to 320 m a.s.l. with some areas in the southern part of the project rising to 360 m a.s.l. Overburden varies between 0 m and 15 m and consists of stratified clays as well as glacial and glaciofluvial Pleistocene deposits (MDDELCC). Most of the project is covered by Lac De Montigny, which includes several islands.

The Project is located in the southern part of the boreal bioclimatic domain. The dominant vegetation is fir and white spruce forest, with occurrences of white birch, black spruce, jack pine, larch and aspen (MFFP website).

Two sources of water were used at the Project: surface water from Lac De Montigny and an underground source from 17 Level in the Kiena mine at a vertical depth of 170 m. Surface water is used in case of emergencies (e.g., fire). Two large pumps are located in a building behind the plant. Underground water flows naturally on 17 Level. A pumping system was installed to bring water to the surface at the pumping facilities near the No.1 shaft. This water is used as clean water for showers and toilets. A small storage tank (holding water from 17 Level) is also located on 38 Level for industrial underground use.

Electricity is available from Hydro-Québec through an above-ground power line. This source of power is used to heat the surface buildings and run the mill. An emergency diesel powered generator is also available in case of a power outage. A 10,000 L capacity diesel tank is located near the generator, linked to a tank with a capacity of 1,135 L. Four diesel tanks are located underground at mine levels 27, 33, 48 and 64. These tanks have a capacity of 4,500 L.

A Natural Gas line is also used to feed the mill, shaft, warehouse, backfill plant and heating of the fresh mine air ventilation for underground use.

5.5 Mine Infrastructure

The Project is an amalgamation of 20 former properties. Figure 4-4 shows all underground development for the Project. Table 5-1 presents a summary of the remaining infrastructure in each part (former site) of the Kiena Mine Complex.

Table 5-1: Kiena Mine Complex infrastructure

Historical mine	Underground infrastructure	Surface infrastructure
Kiena Mine	▪ Access to underground work is through the No.1 shaft to a depth of 930 m. It provides access from 12 Level (120 m below surface) to 93 Level (930 m below surface). Levels 17 to 111 are accessible by a ramp that extends from 170 m to 1,100 m below surface. A total of 49 levels was excavated. A northwest exploration drift of 947 m is present on 12 Level. A second exploration drift of about 3,500 m extends to the east on 33 Level.	▪ The shaft and head frame; ▪ A dry facility; ▪ A laboratory; ▪ A core shack; ▪ A hoisting room; ▪ Main electric substation (4 transformers); ▪ A secondary electrical substation (25 kV); ▪ Fuel reservoirs.
Wisik	▪ Exploration shaft provides access to two levels of drifting at 200 ft (61 m), 300 ft (91m)	▪ Concrete slab and old foundation walls.
Shawkey Mine Shaft No. 1 Area	▪ A shaft is connected to 7 levels of underground drifts and raises at levels 125 ft (38 m), 225 ft (69 m), 325 ft (99 m), 450 ft (137 m), 575 ft (175 m), 625 ft (191 m) and 725 ft (221 m) below surface. The drifts amount to approximately 1,125 ft (343 m); ▪ Plan views for the first five levels are available.	▪ Concrete slab and old foundation walls.
Shawkey Mine Shaft No.2 Area	▪ A 743 ft (227 m) shaft provides access to four levels of drifting at 250 ft (76 m), 400 ft (122 m), 550 ft (168 m) and 700 ft (213 m) below the surface. Drifting and cross-cutting amounted to approximately 927 ft (283 m); ▪ Total underground lateral development in the form of drives, cross-cuts and drifts amounted to 2,337 ft (712.3 m). One drive on the 4th level was extended 926 ft southeast.	▪ Concrete slab and old foundation walls; ▪ Two core racks near the intersection of Highway 117 and the road to the site.
Elmac Shaft Area	▪ A 100 ft (30 m) shaft provides access to one level of drifting. Drifting and cross-cutting amounted to approximately 460 ft (140 m)	▪ Concrete slab and old foundation walls.

Historical mine	Underground infrastructure	Surface infrastructure
Island No. 3	▪ No underground infrastructure, 23 m shaft collar.	▪ Island No. 3 is accessible by boat. The shaft was collared in 1999 at a depth of 23 m, the 10-ft hoist and surface buildings were installed in 2000-2001 to access the Wesdome Deposit and the permitting process was initiated. The acquisition of the Kiena mine in 2003 provided an alternate access and work was suspended.
Dorval-Siscoe Shaft Area	▪ A historical three-compartment shaft of 343 ft (105 m) provides access to level 300 ft (91 m), where there are 850 m of drifts and cross-cuts.	▪ The Dorval-Siscoe Shaft is located on Island No. 6, which is east of Island No. 3. The Dorval-Siscoe Shaft was built to access part of the Wesdome Deposit. All that remains of the surface infrastructure are old foundation walls, concrete slabs and pieces of broken equipment.
Siscoe Mine Area	▪ A shaft of 2,475 ft (754 m) provides access to 19 levels below surface from 150 ft (46 m) to 2,475 ft (754 m). The development work amounts to more than 15,000 m of drifts and cross-cuts.	▪ Mine site rehabilitation is complete, flooded sink holes, caused by the collapse of near-surface underground excavations, were observed. These areas are protected by fences.
Siscoe Extension Area	▪ A three-compartment shaft of 725 ft (221 m) provides access to two levels below surface at 350 ft (107 m) and 725 ft (221 m). The development work amounts to about 2,000 m of drifts and cross-cuts	▪ The area is restricted and blocked by a fence and a security camera system.

5.5.1 Crown Pillar – Kiena Mine

A crown pillar of approximately 100 m thick was left in place in the Kiena Mine. The crown pillar is located under Parker Island (S50 Zone) and the lake.

In 2010, Itasca (Andrieux, 2010) performed a preliminary empirical analysis of crown pillar stability in the North Zone. The analysis evaluated two scenarios: the first assumes that the pillar is made of poor quality rock (schist), and the second assumes good quality (hard) rock. In the first scenario, the pillar showed significant potential for instability. The second case showed a situation of near-instability.

Three recommendations were made to monitor pillar stability. Any of these three would be an adequate protocol.

1. Instrumentation:
    - Manual probe to access any rock disentanglement (weekly readings); or
    - Multi-point borehole extensometers installed toward the surface from 12 Level (weekly readings or datalogger).
2. Hydrostatic barricades;
3. Backfilling of open stopes:
    - Backfill the majority of the void: in the advent of a pillar collapse, the loose material from the pillar would not have a chance to expand and fill the rest of the opening.

The 12 Level was equipped with two extensometers at the North Zone area. During mining operations, a monthly inspection was made of 12 Level and measurements taken, and 17 Level at the North S50 area was visually inspected every 6 months: after the spring thaw and the other in fall.

Since the suspension of mining operations in June 2013, the inspections and measurements on 12 Level have been maintained on a quarterly basis. According to Wesdome, to date, drone surveys conducted on the 12 Level voids show little to no change in size. Daily visual inspections are also being done. In 2021, acoustic monitors are being installed to expand the monitoring measures being taken.

Hydrostatic barricades are currently being constructed on the six levels allowing access to the North Zone. These barricades are designed to withstand a water or backfill column. Underground tailings disposal in the North Zone is being considered as part of the PFS.

6. HISTORY

The following information was mostly borrowed from a previous NI 43-101 technical report on the property (Turcotte et al., 2015). This chapter is divided into two sections as follows: Wesdome historical involvement in the Project and historical work on the different mineralized zones of the Property.

6.1 Wesdome Principal History in the Project

1945: The origin of the Company's business can be traced back to Western Quebec Mines Inc., incorporated in 1945. Western Quebec Mines began developing the Dorval-Siscoe property and carried out various exploration works on the property until 1975.

1976: Wesdome Resources Limited ("Wesdome Resources") was created as a joint venture in 1976 for the purpose of exploring and developing the Wesdome property (formerly the Dorval-Siscoe property). The word "Wesdome" is a combination of the names Western Quebec Mines and Dome Exploration Ltd. Wesdome Resources was held 30% by Western Quebec Mines and 70% by Dome Exploration.

1984: On November 13, 1984, Western Quebec Mines agreed to purchase a 40% interest in the Joubi property from Valmag Inc.

1988-1989: The School Mine property, the Shawkey South property and a 35% interest in the Shawkey property were acquired in 1988 and 1989 by Western Quebec Mines from Valmag Inc.

1990: Production started at the Joubi mine in 1990.

1992: On October 27, 1992, Western Quebec Mines acquired the Yankee Clipper property from Goldhunter Explorations Inc.

1993: Western Quebec Mines completed its acquisition of the 100% interest in the Joubi property.

1994: Western Quebec Mines drilled 6 DDH for 2,958 m in the southwestern part of the School Mine Property. Erratic values reached 77 g/t Au.

1996: Western Quebec Mines acquired the Dubuisson West property from Republic Goldfields Inc. This property was merged with the Joubi property.

1997: On November 21, 1997, Western Quebec Mines acquired the 525,000 common shares of Wesdome Resources that were held by Dome Exploration. The result was that Wesdome Resources became wholly-owned by Western Quebec Mines. In November 1997, Western Quebec Mines also acquired the 65% interest of the Shawkey property from Placer Dome. This property hosts the past-producing Shawkey mine. The Shawkey and Shawkey South properties were merged. During the period between 1936 and 1964, the Shawkey mine produced a total of 25,637 ounces of gold from 127,737 metric tons (t) of ore grading an average 6.24 g/t Au. On December 1, 1997, Western Quebec Mines acquired the Callahan property from Placer Dome.

1998: Western Quebec Mines staked 3 claims (the Lamothe-Extension property) adjacent to the Lamothe property. On January 15, 1998, the Lamothe property was acquired by Western Quebec Mines from Robert Lamothe and Alphonse Beaudoin. On November 3, 1998, Western Quebec transferred to Wesdome Resources all its interests in the Lamothe, Lamothe-Extension (now Vassan), Yankee Clipper and Callahan properties.

1999: In October 1999, Dynacor Mines Inc. and Western Quebec Mines signed an agreement whereby ownership of the contiguous Siscoe and Siscoe-Extension (Dynacor Mines) and Wesdome, Lamothe, Lamothe-Extension, Yankee Clipper and Callahan (Wesdome Resources) properties were to be pooled into a new company in order to develop them jointly. The new company, Wesdome Gold Mines Inc., was created by Dynacor Mines, and the latter transferred its 100% interest in the Siscoe property and its 75% interest in the Siscoe-Extension property. Following this, Wesdome Gold Mines Inc. then acquired 100% of the share of Wesdome Resources from Western Quebec. During the period between 1929 and 1949, the Siscoe mine produced a total of 802,303 ounces of gold and 306,070 ounces of silver from 2,975,785 t of ore grading an average 9.22 g/t Au and 3.20 g/t Ag.

The Joubi mine was closed in 1999 after a 10-year production history. The historical production amounted to 62,283 ounces from 327,561 t of mined ore.

2003: In December 2003, Western Quebec Mines purchased the Kiena Complex and subsequently placed the property into Wesdome Gold Mines Inc., thereby completing and consolidating Wesdome's land package around Lac De Montigny. As a part of this transaction, Wesdome Gold Mines acquired a 100% interest in the Kiena, Kiena West, Lac Dubuisson, Rosenbaum, Dubuisson, Audet Block, Elmac, South Block Kiena, Option Roy and Lac De Montigny properties, and a 50% interest in the Maufort property.

Before this transaction, the Kiena mine produced a total of 1.56 million ounces of gold from 10.7 Mt of ore grading an average 4.54 g/t Au.

2006: On February 1, 2006, River Gold Mines Ltd. and Wesdome Gold Mines Inc. completed a merger to form the current company called Wesdome Gold Mines Ltd. (Wesdome).

On April 4, 2006, Wesdome staked seven claims and added them to the Vassan property. The Kiena mine was in the pre-production development stage until August 1, 2006, when commercial production commenced.

2007: On July 10, 2007, a merger was completed with parent company Western Quebec Mines on the basis of 1.45 shares of Wesdome for each share of Western Quebec Mines. Wesdome was the surviving operating entity.

2013: Wesdome continuously operated the Kiena mine until its temporary shutdown on June 30, 2013. The mine was placed under a care and maintenance program. During the period between August 2006 and June 2013, the Kiena mine produced a total of 198,708 ounces of gold from 1.826,500 t of ore averaging 3.38 g/t Au.

2014-2015: A total of 47 DDH were drilled on different areas to test extension and continuity of the zones. The number of DDH and total length (m) are as follow: Dubuisson North Zone (14 DDH; 5,634 m), S50 Zone (25 DDH; 2,809 m), Siscoe C Vein Area (1 DDH; 300 m), Presqu'ile Zone (7 DDH, 1,968 m).

2015: Wesdome released a technical report (Turcotte et al., 2015) to provide a technical summary of Wesdome's primary mining and exploration assets around the Kiena mine property, known at the time as the Quebec Wesdome Project (now collectively the Kiena Mine Complex).

Measured and Indicated Resources of 2,500,600 t at 5.59 g/t Au below the 100-m crown pillar for a total of 449,300 oz of gold, and 134,000 t at 5.48 g/t Au within 100-m crown pillar for a total of 23,600 oz of gold were estimated. Inferred Resources were estimated to be 1,563,300 t at 7.97 g/t Au below the 100-m crown pillar for a total of 400,400 oz of gold and 747,600 t at 8.22 g/t Au within the 100-m crown pillar for a total of 197,600 oz of gold.

2016-2018: Wesdome carried out an underground drilling program from June 2016 to December 2018 totalling 94,722 metres. Targets were drilled as follows: VC1 (20 DDH), VC6 (18 DDH), Kiena Deep A Zone (39 DDH), South Zone (9 DDH), S50 (45 DDH), other targets (138 DDH). In August 2017, an exploration ramp starting at 100 Level was developed to provide additional underground drilling platforms. Total development as of November 2018 was 2,200 m and allowed diamond drilling to be conducted from a better direction to intersect the steeply plunging zones and provide drilling platforms for definition drilling on the central area of Kiena Deep A Zone. In February 2018, a heliborne high-resolution magnetic survey over the entire property was conducted.

2019: In January 2019, Wesdome released a mineral resource estimate (MRE) technical report (Beausoleil et al., 2019) following its 2016-2018 drilling campaign using 269 new DDH for a total of 58,646 m. Measured and Indicated Resources of 2,957,400 t at 5.83 g/t Au below the 100-m crown pillar for a total of 554,700 oz of gold, and 162,800 t at 5.32 g/t Au within 100-m crown pillar for a total of 27,900 oz of gold were estimated. Inferred Resources were estimated to be 3,025,300 t at 7.79 g/t Au below the 100-m crown pillar for a total of 757,600 oz of gold and 1,113,200 t at 6.97 g/t Au within the 100-m crown pillar for a total of 249,600 oz of gold. In 2019, Wesdome carried out an underground drilling program totalling 59,468 m and released an updated mineral resource estimate (2019 MRE) technical report (Richard and Torrealba, 2019) using 140 new DDH for a total of 36,050 m in November 2019.

The historical resource estimates disclosed above are historical in nature.

These historical estimates have been disclosed to demonstrate the Project is not greenfield in nature and have been subject to considerable efforts by the operator.

The QP has not done sufficient work to classify the historical estimate as current mineral resources and Wesdome is not treating the historical estimates as current mineral resources.

The estimates in this technical report supersede all previous mineral resource estimates on the Project.

6.2 Detailed Historical Work by Area

6.2.1 Kiena Mine Area

The following description of historical work in the Kiena Mine Area is mostly modified and summarized from Mailhiot (1920), Cooke et al. (1931), Hawley (1931), Bell (1935; 1936; 1937), Auger (1947), Dresser and Denis (1949), Claveau et al. (1951), Salt (1960), Robinson (1961a; 1961b; 1962), Cormier (1986a; 1986b), Sauvé et al. (1993), Morasse (1998), Beauregard and Gaudreault (2005) and the annual reports of Wesdome Gold Mines Inc. and Wesdome Gold Mines Ltd. (2003–2013).

1911–1914: The first record of exploration along the shores of what was known then as Lac Kienawisik (later Lac De Montigny) dates back to 1911–1914 when prospector Barney Parker reported the discovery of native gold-bearing quartz veins in a shear zone at the northwestern end of the island on which Kiena's mill was later built.

1922–1927: Martin Gold Mines and Parker Island Gold Mines did some follow-up work on the discovery outcrop between 1922 and 1927, and tested five quartz veins with trenches and drillholes. Soon afterward, gold was also discovered in the "Wisik Vein" on the eastern shore of Moccasin Island, the larger of two nearby islands to the east of Parker Island.

1936–1940: In 1936 Kiena Gold Mines Ltd. was created. Under the control of Ventures Ltd., Kiena Gold Mines immediately initiated a major surface and exploration program on the property, which resulted in the sinking of the Parker shaft at the western end of the island. Shaft-sinking to a depth of 455 ft (-138 m) was followed by the development of exploration drifts at the 130-, 230-, 330-, and 430-ft levels in an effort to test the surface showing, but the discovery veins were found to be of limited extent. During an extensive stratigraphic winter drilling program from lake surface ice in 1937–1938, the S-21 Zone (later called the North Zone) was discovered 2,300 ft (701 m) north of the island. A crosscut was excavated toward the S-21 Zone, intersecting four mineralized veins. Mining operations ceased in 1940 due to limited ore reserves and wartime difficulties, and exploration activities were suspended for 20 years.

1948: Ventures drilled two diamond drillholes (DDH) totalling 1,313 ft (400.2 m).

1961–1965: Ventures, which controlled the 1958 joint venture between Kiena Gold Mines and Wisik Gold Mines Ltd., carried out a magnetometer and geological mapping survey on their combined claim block. A 13-hole diamond drilling program was proposed on the basis of favourable recommendations from these surveys, but R.W. Robinson cut this proposal down to only three holes. In 1961, the third and last drillhole (S50) of this exploration program, targeting the "nose of a fold" and a "magnetic low that could represent a siliceous intrusive", intersected 0.22 oz/t Au (7.54 g/t Au) over 50 ft (15.2 m) of core, at approximately 800 ft (243.8 m) below the lake's surface.

Following the discovery of the S50 Zone, Falconbridge acquired Ventures in 1962, took over the management of Kiena Gold Mines and its Kiena-Wisik property, and financed the subsequent underground exploration. Based on 79,000 ft (24,079.2 m) of drilling from the surface, original reserves of the S50 Zone were estimated at 5 million short tons (st) (4.53 Mt) averaging 0.185 oz/t Au (6.34 g/t Au; Cormier, 1986a).

In 1963, the No. 1 shaft was collared approximately 800 ft (243.8 m) east of the Parker shaft and sunk to a depth of 1,324 ft (403.55 m). Extensive underground exploration and a definition diamond drill program were carried out.

But due to adverse ground conditions found in a test stope conditions much related to the structural geology of the deposit, it was concluded that no large openings could be left unfilled and low-cost mining could not be expected. In 1965, a feasibility study on the S50 orebody showed that mining operations would be marginal at best, with gold selling for US$35.00/oz at that time. The property was placed under a care and maintenance program, leaving the S50 orebody dormant for the next 14 years.

1979–1984: In 1979, Falconbridge re-evaluated the property and recommended to bring it into production. The transition from an advanced exploration project to a mine operation was successful, and mining officially started in October of 1981. Trackless and cut-and-fill methods were employed. Custom milling at Teck's nearby Lamaque Mill was used for the first three years of production until September 1984 when Kiena's own new CIP (carbon-in-pulp) mill was operational.

1986–1994: Campbell Red Lake Mines Ltd. became Kiena's major shareholder on January 25, 1986, when Falconbridge sold 56.7% of its interest in Kiena Gold Mines. Following the amalgamation in 1987 of Placer Development Ltd., Dome Mines Ltd. and Campbell Red Lake Mines Ltd., Placer Dome Inc. of Vancouver, became the owner and operator of the Kiena mine. On January 1, 1994, Placer Dome Inc. changed its name to Placer Dome Canada Ltd.

1997–2003: Placer Dome Canada sold the mine (along with the neighbouring Sigma mine) to McWatters Mining Inc., who officially became the new owner and operator of Kiena on September 12, 1997. McWatters continuously operated the Kiena mine until its closure in September 2002.

During the period between October 1981 and September 2002, the Kiena mine produced a total of 1.56 Moz of gold from 10.7 Mt of ore grading an average 4.54 g/t Au (Table 6-1). An exploration program commenced in 2002 to further investigate five previously identified gold targets. McWatters carried out a 5,012-m underground drill exploration program, the first phase of the exploration program at the Kiena Complex to test the hanging wall mineralization of the S50 orebody. A follow-up drill program began on January 20, 2003 and was completed in March 2003. The objective of this second phase of the exploration program, corresponding to approximately 8,200 m of surface drilling program, was to extend the hanging wall mineralization, and add to or otherwise improve the resources of the Kiena Complex.

The Kiena Mine Complex was acquired by Wesdome Gold Mines Inc. in December 2003.

Table 6-1: Kiena mine production from 1981 to 2002
(Turcotte et al., 2015)

Year	Metric tonnes (t) milled	Recovered grade Au g/t	Gold production (oz)
1981	101,231	4.61	15,018
1982	287,916	6.81	63,038
1983	307,661	6.19	61,193
1984	378,014	5.48	66,658
1985	381,376	5.71	70,035
1986	453,793	4.98	72,694
1987	478,752	4.36	67,113
1988	477,947	3.79	58,219
1989	470,705	4.38	66,235
1990	473,602	4.33	65,953
1991	486,217	4.55	71,112
1992	501,827	5.03	81,195
1993	496,401	4.95	79,034
1994	504,873	5.12	83,044
1995	534,330	5.03	86,375
1996	608,701	4.32	84,609
1997	631,606	4.59	93,169
1998	594,000	4.39	83,807
1999	647,933	4.16	86,602
2000	719,363	3.74	86,610
2001	745,391	3.41	81,631
2002	415,400	2.82	37,626
Total	**10,697,039**	**4.54**	**1,560,970**

2004: Wesdome Gold Mines developed and explored its properties from underground via the Kiena shaft and underground workings. Wesdome Gold Mines drove a drift 4.5 km to the north to explore the Wesdome property on the 520 m level, and a second drift 2.0 km to the east on the 330 m level to explore the No. 22 Zone on the Shawkey property. Definition drilling and development work proceeded on known zones of gold mineralization on the Kiena property with the intent of preparing these for commercial production as soon as possible. Late in 2004, access was established to the VC Zone, located 500 m north of the shaft, and fan drilling of the zone commenced from the 520 m level.

2005: Wesdome Gold Mines continued the underground development at the Kiena Mine Complex. The North Drift (at 520 m) was used to conduct a major exploration program on the VC Zone approximately 500 m north of the shaft. Further work was completed on the North Zone and the No. 388 Zone from the 330-m level. Progress was also made on the East Drift (at 330 m). Development crews also drifted into the Martin Zone (from which Shawkey produced ore in the 1930s).

During the same year, Beauregard and Gaudreault (2005) completed an MRE on four zones (North, VC, 388 and Martin). Wesdome Gold Mines also completed a new resource estimate for the VC1 to VC3 zones.

2006–2013: The Kiena mine was in pre-production development stage until August 1, 2006, when commercial production commenced. Wesdome continuously operated the Kiena mine until its shutdown in June 30, 2013. The mine was placed under a care and maintenance program. During the period between August 2006 and June 2013, the Kiena mine produced a total of 198,708 oz of gold from 1.826,500 t of ore at an average grade of 3.38 g/t Au (Table 6-2). Another 1,959.5 oz was sold during 2016-2017 collected from the cleaning of the processing plant (Beausoleil et al., 2019).

Table 6-2: Kiena mine production from 2006 to 2013 (Turcotte et al., 2015)

Year	Metric tonnes (t) milled	Recovered grade Au g/t	Gold production (oz)
2006	94,200	3.07	9,300
2007	284,757	3.87	35,404
2008	241,641	5.19	40,344
2009	302,034	3.65	35,398
2010	285,527	3.50	32,162
2011	255,311	2.38	19,516
2012	265,872	2.20	18,814
2013	97,158	2.49	7,770
Total	**1,826,500**	**3.38**	**198,708**

6.2.2 Wisik Shaft Area

The following description of historical work in the Wisik Shaft Area is mostly modified and summarized from Bell (1935; 1936; 1937), Taschereau (1936), Denis (1937), Auger (1947), Dresser and Denis (1949), and the diamond drill logs of Wisik Gold Mines Ltd.

1934–1937: The property around the Wisik deposit was owned by H. Klee who founded Wisik Gold Mines. The property was held under option in 1935 by Teck Hughes interests, who carried out some diamond drilling. The only visible mineralized showing of substantial size was a quartz vein, exposed for only a few feet on the eastern shore of the largest Moccasin Island. In addition to the diamond drilling in the vicinity of the vein, some holes were also put down along the water stretch between the island and Shawkey Peninsula. The option was subsequently allowed to lapse.

In 1936, the overburden was removed by Wisik Gold Mines for the purpose of sinking a shaft on Moccasin Island. A three-compartment shaft was sunk below 300 ft (91.4 m). Drifts were excavated along the principal vein or shear on the 200 ft and 300 ft levels, totalling 2,100 ft (640.1 m). In 1937, metallurgical testwork was performed at the Ottawa facilities of the Department of Mines and Resources of Canada on two samples of gold ore: one from the Wisik North drift (237 lbs, 107.5 kg), and the other from Wisik South (200 lbs or 90.7 kg). The North rift sample assayed 0.125 oz/t Au (4.29 g/t Au) and 0.19 oz/t Ag (6.51 g/t Ag), whereas the South Drift sample returned 0.675 oz/t Au (23.14 g/t Au) and 0.09 oz/t Ag (3.09 g/t Ag). In September 1937, work was suspended and the mine flooded.

1958–1959: In July 1958, Ventures Ltd., who controlled the 1958 joint venture between Kiena Gold Mines and Wisik Gold Mines, drilled one hole (W-1) totalling 350 ft (106.7 m) on Moccasin Island, northwest of the Wisik shaft. In January 1959, another hole (W-2), totalling 750 ft (228.6 m), was drilled to the north of hole W-1 to test the eastern extension of the No. 1 Zone located on the Kiena mine property.

Later, Kiena Gold Mines took over management of the Wisik property and eventually formed the Kiena-Wisik property.

6.2.3 Shawkey Mine Area

The following description of historical work in the Shawkey Mine Area is mostly modified and summarized from Mailhiot (1920), Cooke et al. (1931), Hawley (1931), Bell (1935; 1936; 1937), Dresser and Denis (1949), Ingham (1950), Claveau et al. (1951), Ingham (1953), Bourret et al. (1956), Sauvé (1985), Chevalier (1989b), Sauvé et al. (1993), Beauregard and Gaudreault (2005), and the annual information forms of Western Quebec Mines Inc., Wesdome Gold Mines Inc. and Wesdome Gold Mines Ltd. (1997–2013).

1911: Fred La Palme discovered a gold-bearing vein (No. 1 Vein) on the south shore of Lac De Montigny.

1917–1919: The Martin Gold Mining Company Ltd. was incorporated to develop the new gold showing, and in 1917, underground work commenced as well as the erection of a small mill which was completed the following year. A two-compartment vertical shaft was sunk 125 ft (38.1 m) on the shore of Lac De Montigny, and a small amount of drifting and cross-cutting was done. About 600 pounds of rock were milled during the summer of 1918. Work was suspended in 1919.

1921–1923: Two years later, the property was sold to John Dalton. In 1922, an option was secured by J.J. Godfrey and underground work resumed. The shaft was deepened to 325 ft (99.1 m), and about 700 ft (213.4 m) of drifts and cross-cuts were excavated. A total of 4,500 ft (1,371.6 m) of diamond drilling was also carried out. The option lapsed in 1923.

1934–1938: The mine was purchased by Shawkey Gold Mines Ltd. who became operator. During 1935, drifting and raising on the 125 ft, 225 ft and 325 ft levels revealed continuous lengths of ore. A raise was completed to the surface, north of the shaft. Shrinkage stopes were prepared above the first and second levels.

The shaft was deepened to 725 ft (221.0 m) and new levels were established at the 450 ft, 575 ft and 625 ft horizons. A new mill was erected and started production in 1936. Several gold-bearing veins were discovered over the course of the exploration, but all the production came from the discovery vein. The vein was developed for a maximum length of 1,000 ft (304.8 m). By 1938, this had been mined out, and operations were suspended.

A total of 3,915 ft (1,193.3 m) of surface drilling, 45,885 ft (13,985.7 m) of underground drilling, and 1,125 ft (342.9 m) of drifting and crosscutting have been carried out. The total production was 25,414 oz of gold from 137,978 short tons (st) (125,174 t) of processed ore, for a recovery of 0.184 oz/t Au (6.31 g/t Au).

1945–1951: In 1945, Shawkey Mines Ltd. took over the abandoned workings and began an extensive diamond drill program. From 1945 to 1947, the company completed about 37,000 ft (11,277.6 m) of surface drilling, 28,500 ft (8,686.8 m) of which was used to explore the No. 10 Vein. Another 9,000 ft (2,743.2 m) of drilling was used in cross-sectional exploration on the east side of the Thompson River and south of the No. 10 Zone. Underground drilling from old workings amounted to 20,000 ft, used in lateral and depth tests from the 4th and 6th levels. Total underground lateral development in the form of drives, crosscuts and drifts amounted to 2,337 ft (712.3 m). Over 7,000 ft^3 (198.2 m^3) of rock was slashed. One drive on the 4th level was extended 926 ft southeast, under the Thompson River, to reach a body of diorite containing auriferous quartz veins. The No. 10 Zone was explored for 2,300 ft (701.0 m) by drilling. Underground drilling and drifting led to the discovery of new auriferous zones (No. 9, No. 11 and No. 12 zones).

The No. 2 shaft was collared in July 1950, adjacent to the north side of the No. 10 Zone, approximately 900 m south of the No. 1 shaft. Shaft-sinking to a depth of 743 ft (226.5 m) was completed in April 1951. Four level stations were established at 250 ft (76.2 m), 400 ft (121.9 m), 550 ft (167.6 m) and 700 ft (213.4 m). After carrying out 927 ft (282.5 m) of drifting and crosscutting on the 700 ft level and 605 ft (184.4 m) on the 550 ft level, as well as 2,265 ft (690.4 m) of underground drilling, the mine was closed again in September 1951. A total of 1,735 st (1,574 t) of ore grading 0.09 oz/t (3.09 g/t Au) was mined from various drifts and slashes.

1962–1964: The Project came under the control of Con-Shawkey Gold Mines Ltd. The No. 2 shaft (No. 10 Zone) was reopened, and more exploration was completed. A bulk sample of 51.3 st (46.5 t) was sent to the Ministry of Natural Resources pilot plant, and an average grade of 0.23 oz/t (7.89 g/t Au) was obtained from 53 samples. Another bulk sample of 1,039 st (942.6 t) was sent to the Malartic Goldfields Mill where an average grade of 0.053 oz/t Au (1.82 g/t Au) was obtained. In addition, 14,000 ft (4,267.2 m) of surface drilling, 1,600 ft (487.7 m) of underground drilling, 1,400 ft (426.7 m) of underground development, and Mag and EM surveys were also carried out on the Project.

1964–1966: Noranda carried out 3,710 ft (1,130.8 m) of diamond drilling that concentrated on the No. 10 Zone. Mag and EM surveys were also performed on the Project.

1972–1976: Umex acquired an interest in the Project and drilled six drillholes for a total of 2,240 ft (682.8 m). Mag and EM surveys were conducted on the Project.

1979–1989: Les Mines Sigma (Québec) Ltée started work to acquire a 65% interest in the Shawkey property from Valmag Inc. For three years, 45 km of Mag surveys and more than 11,500 m of diamond drilling were completed on the No. 22 Zone. From June to March 1984, another 24 km of Mag surveying was completed over Lac De Montigny. Twenty-three holes were drilled for a total of more than 6,000 m. From June to August 1988, nine more holes were drilled for an additional 2,224 m. The Shawkey South property and the 35% interest in the Shawkey property were acquired by Western Quebec Mines Inc. in 1988 and 1989 from Valmag Inc.

In 1988, Placer Dome Inc. acquired Les Mines Sigma (Québec) and carried out a drilling program comprising seven drillholes for a total of 1,897.5 m. Drilling concentrated on two target areas: The West Zone and the No. 22 Zone. The Shawkey Property consists of four mining concessions in Dubuisson Township.

1990–1997: The Shawkey property was under a joint venture with Placer Dome Inc. until November 1997. In 1990, Placer Dome Inc. estimated the mineral inventory of the Shawkey property (Lebel and Lafleur, 1991). The mineral inventory was contained in six lenses designated by the letters A to F. The mineral inventory was performed using the polygonal method and a specific gravity of 2.7 g/cm^3.

1997: Western Quebec Mines purchased, from Placer Dome, 70% of the outstanding shares of Wesdome Resources Ltd. (which it would later own 100%), and a 65% interest in the Shawkey property (which it would later own 100%).

2002–2003: In 2002, Western Quebec Mines drilled 11 holes totalling 1,248 m to test the No. 22 Zone. In 2003, the company added ten more holes for another 1,657 m. During the period between 1936 and 1964, the Shawkey mine produced a total of 25,637 ounces of gold from 127,737 t of ore with an average grade of 6.24 g/t Au (Table 6-3).

Table 6-3: Shawkey mine production from 1936 to 1964 (Turcotte et al., 2015)

Year	Metric tonnes (t) milled	Recovered grade Au g/t	Gold production (oz)
1936-1938 1945-1951 1962-1964	125,174 1,574 989	6.31 3.08 2.11	25,414 156 67
Total	**127,737**	**6.24**	**25,637**

6.2.4 Elmac Shaft Area

The following description of historical work in the Elmac Shaft Area is mostly modified and summarized from Mailhiot (1920), Cooke et al. (1931), Hawley (1931), Bell (1935; 1936), Koulomzine and Brossard (1946), Dresser and Denis (1949), Chevalier (1989a), Beauregard and Gaudreault (2005), and the annual information forms of Western Quebec Mines Inc., Wesdome Gold Mines Inc. and Wesdome Gold Mines Ltd. (1997–2013).

1919–1935: The property was originally known as Fosie-Kengrow, these being the names of the men who staked the claims in 1919. From 1919 to 1935, the property was successively controlled by Union Mining Corporation, Unison Gold Mines Ltd., Lorette Mines Ltd., Minorand Co-operative Company Ltd., Minrand Gold Ltd., and finally by Crossroads Gold Mines Ltd. The deposit was discovered in 1919. The discovery vein, which was exposed over a distance of roughly 200 ft (61.0 m), displayed a contorted pattern with numerous offshoots and stringers. Some diamond drilling was done in 1922, establishing sufficient ore for mining purposes, as long as ore grades were high enough. Mining machinery was installed, and in 1925, a shaft was sunk to a depth of 100 ft (30.5 m) where some underground work was carried out. The mine was again operated in 1932–33, and there has been sporadic development work, including a small amount of underground work and diamond drilling, as well as the assembling of a small mill. In 1935, the mine workings were flooded.

1940–1946: Elmac Malartic Mines Ltd. carried out work around the shaft and in underground workings. A new mineralized zone was discovered (the “Carbonated Zone”). All known zones were resampled. A drilling program was also conducted in the northern part of the property (Claims Potter-Kee). A magnetic survey was carried out over the entire property in 1945.

1963–1965: Elmac Malartic Mines conducted an 18-hole drilling program for a total of 8,800 ft (2,682.2 m). Seven holes were drilled in the northern part of the property (Potter-Kee Claims) on Lac De Montigny. Nine holes were drilled in the centre of the property, in the shaft and Carbonated Zone areas. The drilling program cut the Carbonated Zone, which assayed 23.78 g/t Au over 2.74 m. Two holes were drilled near the Piché River in the southern part of the property. In 1965, another drilling program (1531.0 m) was conducted on Lac De Montigny, resulting to the discovery of a new mineralized zone associated with feldspar porphyries. The new zone assayed 4.14 g/t Au over 6.24 m.

1978–1983: Les Mines Sigma (Québec) Ltée optioned the property. A total of seven holes were drilled on the Carbonated Zone totalling 1,905 m. Between 1981 and 1983, the company carried out 22.3 km of magnetic surveying and 3.6 km of induced-polarization surveying. In addition, a total of 53,823 ft was also drilled on the property.

1989: Les Mines Sigma (Québec) conducted an 8-hole drilling program in the northern half of the property during the winter of 1989 for a total of 2,117.75 m.

1997–2002: Placer Dome Canada Ltd. sold the Elmac property to McWatters Mining Inc., who officially became the new owner and operator of the property until 2003.

2003: In December of 2003, Western Quebec Mines purchased the Kiena Complex, including the Elmac property.

6.2.5 Joubi Mine Area

The following description of historical work in the Joubi Mine Area is mostly modified and summarized from Mailhiot (1920), Bell (1935), Ingham (1944b), Hinse (1975), Lavery (1983), Laforest (1987) Castonguay (1995), and the annual information forms of Western Quebec Mines Inc. (1996–2006).

1919: Before 1919, prospecting work on the Clowse claim led to the discovery of a narrow gold-bearing vein. The vein was traced at surface over a length of 100 ft (30.5 m). A 27-ft-deep (8.2-m) exploration shaft was sunk on the vein.

1934: During spring and summer 1934, Amity Gold Mines Ltd. carried out surface trenching, surface sampling and diamond drilling, chiefly in the vicinity of the old shaft sunk by Clowse Claim. Four holes were reportedly drilled to test this old occurrence.

1941–1942: A diamond drilling program was started in July 1941 by Seventh Malartic Mines Ltd., and it continued until April 1942, when 28 holes had been completed for a total of 13,550 ft. Only four holes were located on the Joubi property, one of which constituted the first indication of the presence of the Range Line Zone.

1943–1944: In 1943, Perron Gold Mines optioned the property, and diamond drilling was resumed in the spring of 1944. Eighteen holes were drilled during the program.

1960: Iso-Newlund Mines covered part of the property with a Mag survey, followed by five drillholes.

1964: One hole was drilled by Amerel Mining Company, followed by a Mag survey.

1973: The Joubi Mining Corporation acquired the property in 1973, and drilled 24 holes on the Range Line Zone.

1979: Massey-Gauthier Ltd. drilled six holes on the Range Line Zone.

1981: A joint venture was made up of U. F. Venture Associates (70% participating interest) and Messeguay Mines Inc. (30% participating interest). The Joubi JV continued the assessment of the Range Line Zone with 1,341.4 m of core drilling in 11 holes during 1981, bringing the total drilling in the zone to 6,838 m in 46 holes.

1982: The participation in the JV was modified in 1982. The result was a 51.11% participating interest for U. F. Venture Associates, a 30% participating interest for Messeguay Mines, and a 18.89% non-participating interest for V. Audet. In 1982, based on all drilling results to date, W.N. Ingham completed a mineral resource estimate (Lavery, 1983).

Between 1982 and 1983, 12 holes, totalling 2,402.7 m, were added on the Range Line Zone. In late 1982, detailed total field magnetic, vertical gradient magnetic and VLF surveys had been completed by the Joubi JV.

1984–1986: Western Quebec Mines carried out an exploration program including different geophysical methods, followed by some 3,178 m of diamond drilling distributed in 11 holes.

1987–1989: Western Quebec Mines agreed to grant Messeguay Mines and Oasis Resources Inc. working rights and an option to acquire a 49% interest in the Joubi property. Western Quebec Mines had Minexpert Inc. complete an evaluation review, which recommended an underground exploration program to define the essential parameters required for a better understanding of the nature of the gold mineralization. Surface installations were constructed, a three-compartment shaft was sunk to a depth of 253 m with four shaft stations, 2,780 m of crosscuts and drifts were developed, 1,094 m of raises were completed, and a diamond drilling program totalling 47,812 m was carried out. During this period, Western Quebec Mines developed the zones defined by the previous underground exploration program, conducted detailed drilling of the zones (3,645 m) and began mining operations.

1990: In 1990, Western Quebec Mines completed development work and mining of the zones defined in 1988–89, conducted a drilling program of 15,084 m (zone definition, surface exploration, and exploration below the last level). In 1991, exploration work discovered the F Zone at the border of the Dubuisson West property. Western Quebec Mines drilled 12,614 m of core on the Joubi property. Since the mineable reserves on the Joubi property were almost exhausted, Western Quebec Mines decided to deepen the shaft by 71 m to reach the lower A Zone.

1992: In early 1992, Western Quebec Mines was granted an option from Republic Goldfields Inc. on the Dubuisson East property. Western Quebec Mines also purchased the Dubuisson West property from Republic Goldfields Inc. Both properties were adjacent to the Joubi property.

In spite of the shaft deepening, reserves were insufficient to maintain the operations. One kilometre (1 km) of drifts was developed at the 2nd level to access the Dubuisson East Zone. Western Quebec Mines drilled 268 m of core on the Dubuisson West property, 3,610 m on the Dubuisson East property (exploration and definition), and 2,409 m on the Joubi property. By the end of the year, the reserves from the known economic zones on the Joubi property had been exhausted.

1993: The Dubuisson East Zones were developed and mined from the 2nd level to the surface. Since reserves were rapidly running out, it was decided that the drift on the 5th level would be extended under the Dubuisson East workings to access additional reserves. Definition drilling (3,633 m of core) was completed on the Dubuisson East property. Mining of the Dubuisson East zones was slowed by wall instabilities, which resulted in an excessive dilution problem.

1994: Western Quebec Mines continued to mine the Dubuisson East upper zones and to develop the lower zone. The company also extracted a few pockets of ore on the Joubi property. Also carried out were definition drilling and exploration on the Dubuisson East property (3,171 m), and drilling of the A Zone below the 5th level (659 m) on the Joubi property.

1995: Western Quebec Mines commenced a two-year underground exploration and development program on the Joubi property. The first phase of this program involved 200 m of shaft sinking to a depth of 524 m, upgrading of production infrastructure, development of a loading system, and establishment of levels 360 and 440. Due to the shaft sinking program, production was interrupted for more than six months. Almost all production came from the Dubuisson East property where the bulk of the reserves were located.

1996: The emphasis continued to be on exploration following the shaft-deepening program and the establishment of new drifts at a depth of 440 m. Western Quebec Mines completed a total of 15,000 m in 110 drillholes along the 1.1-km strike length of the Joubi shear zone. This drilling identified a small, high-grade zone (DE-F) above the 440 m level. In addition, preliminary drilling on the 440 m level identified the down-dip extension of zones DE-B/C over a strike length of 80 m, which remained open to the east and at depth.

1997–1998: In 1997, 90 holes were drilled for a total of 8,600 m, resulting in the discovery of the DE-F Zone below the 440 m level. In 1998, the company conducted an extensive underground exploration effort involving 480 m of drifting and 14,200 m of drilling in 81 holes. The goal of the program was to test the depth and strike potential of the Joubi shear zone in an effort to identify significant new reserve blocks. Results were insufficiently encouraging to justify further development.

1999: The Joubi mine was closed in 1999 after a 10-year production history. Total production amounted to 62,283 ounces from 327,561 t of mined ore. Ore was custom-milled at facilities in the Val-d'Or area when excess capacity was available, with gold recoveries of 98%. The headframe was dismantled and the shaft sealed. The three-compartment shaft reached a depth of 524 m with levels developed at 70 m, 120 m, 220 m, 280 m, 360 m and 440 m. Ore chutes were installed at levels 120 m, 220 m, 280 m and 440 m.

Table 6-4: Joubi mine production from 1990 to 1999
(Turcotte et al., 2015)

Year	Metric tonnes (t) milled	Recovered grade Au g/t	Gold production (oz)
1990	30,146	10.94	10,601
1991	36,998	6.99	8,319
1992	36,572	7.64	8,983
1993	28,419	4.49	4,104
1994	57,115	4.55	8,351
1995	20,166	5.37	3,484
1996	35,270	4.73	5,364
1997	39,191	4.72	5,947
1998	27,470	4.17	3,679
1999	16,214	6.62	3,451
Total	**327,561**	**5.91**	**62,283**

2016: Agnico Eagle Mines purchased some claims from Wesdome to acquire a portion of the property. Agnico Eagle Mine granted a 3% NSR on the Joubi Property.

6.2.6 Dorval-Siscoe/Wesdome Deposit Area

The following description of historical work in the Dorval-Siscoe Area is mostly modified and summarized from Beckman et al. (1933), Bell (1937), Ross et al. (1938), Ross and Asbury (1939), Koulomzine (1941; 1942; 1943), Ingham (1944a; 1947), D'Aragon (1947), Auger (1947), Dresser and Denis (1949), Young (1955), Salamis (1970), Prud'homme (1970), Audet (1975a; 1975b), Matheson (1976), Audet (1979), Gardiner (1987; 1988), Sauvé et al. (1993), Beauregard and Gaudreault (1999; 2005), Turcotte and Pelletier (2009), and the annual information forms of Western Quebec Mines Inc., Wesdome Gold Mines Inc. and Wesdome Gold Mines Ltd. (1996- 2013).

1933–1937: Dorval-Siscoe Gold Mines drilled 37 DDH for a total of 7,050 m, and established the presence of a wide and intensely sheared zone (Dorval-Siscoe Main Break). Snowshoe Mines Ltd. drilled five holes totalling 844 m on the Snowshoe intrusion. A diamond drillhole cut the Dorval Siscoe Main Break. The material in the zone consisted of schistose and talcose granodiorite with laminated quartz-carbonate stringers up to 10 in (25 cm) wide, and a feldspar porphyry dyke about 10 ft (3 m) in section width that was injected and replaced by vein quartz. The mineralization consists of pyrite, chalcopyrite, galena and tourmaline.

Good results obtained by Dorval-Siscoe Mines prompted the company to sink a three- compartment shaft on Island No. 6 in 1937 and 1938, to a depth of 343 ft (104.5 m). Other work included the development of about 850 m of drifts and crosscuts on the 300 ft level, and a drilling program of 14 underground DDH totalling 686 m. A strong vein was exposed for a length of 780 ft (237.7 m) in a drift on the 300 ft level.

Particular emphasis was placed on extending the K Zone in the Siscoe gold mine onto the Dorval-Siscoe property. At the time, the K Zone was considered spatially related to most of the mineralization on the Dorval-Siscoe property. The primary focus of the work was to outline a broad zone of weak to moderate mineralization, as described in the report by Koulomzine (1941).

1941–1943: Camp Bird Gold Mines Ltd. took an option on the Project held by Dorval-Siscoe Gold Mines. A dip needle magnetic survey was carried out over 590 ha on the Project, outlining many magnetic anomalies. Twenty-four holes totalling 5,400 m were drilled from the surface, and 16 holes totalling 1,467 m were drilled underground from the earlier Dorval-Siscoe workings.

1945–1948: Another dip needle magnetic survey was conducted by Snowshoe Gold Mines Ltd. over an additional 191 ha. In addition, 14 surface DDH, totalling 2,671 m, were drilled on the periphery of a large circular magnetic depression (granodiorite plug) located in the centre of the property. In 1946 and 1947, Western Quebec Mines (incorporated in 1945) began developing the property held by Camp Bird Mines Ltd. Twelve surface DDH, totalling 3,394 m, were drilled on magnetic anomalies, providing encouraging results. 1955: Snowshoe Gold Mines Ltd. carried out an EM survey over the western part of the Snowshoe intrusion.

1963–1965: Western Quebec Mines drilled four holes (holes 87 to 90), totalling 1,559 m, on the A Zone, and in 1964, completed a Mag survey over the western and southwestern part of the property.

1965–1970: Kerr Addison Mines and Western Quebec Mines conducted a geophysical survey over the Project, including 209 km of Mag and 12.9 km of EM surveys, and drilled four holes totalling 1,613 m (65-1 to 65-4). In 1970, Western Quebec Mines initiated other geophysical surveys on the western part of the Project, including 132.9 km of Mag, and 182 km of Max-Min (EM) and induced polarization (IP). The goal of these surveys was to delineate massive sulphides associated with peridotites. In addition, nine holes (W70-1 to W70-9) were drilled, totalling 1,373 m.

1975–1983: Wesdome Resources Ltd., a company owned by Dome Exploration Co. Ltd. and Western Quebec Mines, drilled 136 holes totalling 37,999 m. This phase can be split into four major periods:

- 1975–1976: A 78-km VLF survey was carried out on the Project, and 38 holes (82-1 to 82-38) were drilled for a total of 10,584 m. The goal of this program was to evaluate the general potential of the Project;
- Winter 1979: Four holes (82-39 to 82-42) were drilled, for a total of 1,273 m. The goal was to complete a transverse section north of the A and B Zones (Audet, 1979);
- 1980–1981: A drilling program was carried out to delineate the mineralization and evaluate gold reserves in the A and B Zones and associated “flat” quartz veins. In total, 19,740 m (67 holes) were drilled;
- Winter 1983: Drilling was conducted to extend the A Zone mineralization eastwards. A first reconnaissance field program was carried out to identify mineralization in an altered (albitized) monzodiorite on the western periphery of the property, adjacent to a project held by Falconbridge Nickel Ltd. A total of 26 holes (82-105 to 82-130) were drilled for a total of 6,348 m (Duhaime, 1983).

1984: Wesdome Resources established a grid of 116.5 km on the Project, and a total field and gradient Mag survey was undertaken. During the same year, 51 holes, totalling 18,656 m, were drilled on the Project, mainly in two sectors: 21 holes in the intrusive dyke complex (the Falconbridge Zone) for a total of 10,935 m; and 30 holes in the A and E Zone extensions for an additional 7,721 m. During this program, a new zone, the F Zone, returned encouraging results.

1987: Wesdome Resources completed a geophysical survey (seismic refraction) followed by a 30-hole drilling program (holes 82-177 to 206) for a total of 12,180 m.

1988: Another geophysical program consisting of seismic refraction, IP, Mag, EM and bathymetry was carried out by Sigma Mines Ltd. After that, Wesdome Resources completed a 13-hole winter drilling program (holes 82-207 to 219) for a total of 5,318 m, followed by another 13-hole drilling program in the fall of 1988 (82-220 to 82-232), for an additional 4,524 m.

During the 1980s, geologists of Placer Dome Inc. and subsidiary Les Mines Sigma (Québec) Ltée separately estimated the total contained mineralization for the A, B, C, D, E and E3 Zones (Beauregard and Gaudreault, 2005).

1998–1999: Wesdome Gold Mines built a 5.5-km access road on the northern peninsula, and completed rock mechanic tests and seismic surveys for overburden and rock quality on the former site of the decline portal. Results revealed the presence of a deep trough (over 60 m) over an east-west striking, multi-metre wide, sheared and altered ultramafic unit. These results indicated a ramp project would not be feasible. One hundred (100) holes (37,278 m) were drilled, including six holes on the Yankee Clipper property to the north. Three distinct zones of the Wesdome property were tested: the sector north of the K Zone, the E3 and E Zones, and the A and B Zones. Wesdome Gold Mines was planning to sink an exploration shaft on Island No. 3. A 600-m pilot hole was completed on the island for rock mechanic purposes prior to shaft sinking, with positive results. Preparation work, which consisted of levelling the site (Island No. 3), was conducted during the winter of 1998–99. In 1999, permitting and engineering work continued, and limited mining equipment was purchased.

2000–2001: During 2000, work concentrated on the construction of the surface infrastructure required to commence shaft sinking and underground development. The shaft was collared at a depth of 23 m, the hoist and surface buildings were installed, the wharf-barge access system was made fully functional, and was made using a specialty submarine electrical cable. In addition, two exploration holes were drilled to test the western limits of the A Zone.

6.2.7 Siscoe Mine Area

The following description of historical work in the Siscoe Mine Area is mostly modified and summarized from Mailhiot (1920), Cooke et al. (1931), Hawley (1931), Bell (1937), Auger (1940), Auger (1947), James (1949), Dresser and Denis (1949), Gill (1981), Timmins and Wing, (1981), Trudel (1985), Allard (1988) and Sauvé et al. (1993).

1912–1919: Siscoe Island was staked in 1912 by Siscoe Mining Syndicate. Gold was discovered on the northernmost part of the property during the initial prospecting in 1911 and 1912, but gold showings on Siscoe Island were not reported until three or four years later. The first gold discovery was made in 1913.

Between 1913 and 1919, exploration work was conducted on four principal veins: A, B, C, and D. A 45-ft exploration vertical shaft was sunk on the A Vein. The vein was exposed over a length of 90 ft (27.4 m). The vertical exploration shaft of the B Vein was 32 ft (9.8 m) deep. On the C Vein, an exploration shaft was sunk to a depth of 100 ft (30.5 m), inclined at an angle of 35°, following the dip of the vein. The zone of the D Vein comprised a number of scattered veins and quartz lenses measuring several feet in length. Several quartz veins contained nests or pockets rich in native gold, in which wonderful specimens were sometimes found. An exploration shaft 88 ft (26.8 m) deep was sunk on the vein.

1923–1927: The property was bought by British Mineral Corporation, and in 1923, Siscoe Gold Mines Ltd. was created. In June 1926, the underground workings included a shaft sunk to a vertical depth of 105 ft (32 m) on the D Vein, and about 1,500 ft (457.2 m) of drifting and crosscutting. Most of this work has been of an exploratory nature, directed to finding single veins large enough to mine. The long crosscut that runs southwest from the shaft to the granodiorite-basalt boundary cut a vein, known as the K Vein, which seemed to follow that boundary. In 1927, active development on the C shaft and C Vein was recommenced. Since renewing operations, the shaft was continued to a depth of 500 ft (152.4 m), and much lateral work was completed on the 170 ft, 300 ft, 400 ft and 500 ft levels.

1929–1930: Milling equipment was installed, and in January 1929, the mine came into production. Later in 1929, following the discovery of a vein nearly parallel to the cross-cut driven south of the D-shaft, an extensive diamond drilling program was carried out that proved the continuity of the vein, which became known as the New Vein. A new three-compartment shaft (the Central shaft) was completed in spring 1930 to a depth of 472 ft (143.9 m). From this, cross-cuts were driven northwest on the 300 ft and 450 ft levels at distances of 50 (15.2 m) and 100 ft (30.5 m) beyond the New Vein. From the Central shaft, the workings on the 300 ft level were driven east to connect with the 5th level of the inclined C shaft by an 85-ft raise from the latter. At the 450 ft level, a cross-cut was driven east to intersect the C Vein on this level.

1930–1946: By the end of 1936, the mine had been developed on levels to a vertical depth of 1,350 ft (411.5 m). For the first 600 ft (182.9 m), levels are at intervals of 150 ft (45.7 m); below this they were spaced at 125-ft intervals. The Central shaft serviced the entire mine. It was deepened to 1,900 ft (579.1 m) to permit the opening of the newly opened 11th and 12th levels. By the end of 1939, the Central shaft had reached the 19th level at a depth of 2,475 ft (754.4). At this time, the mine had been continuously in production since 1929.

1946–1951: Starting in 1940, and despite an intense exploration program, no new ore sources were found. In 1949, the mine was closed and all reserves were mined out. In 1951, the mill and all equipment were sold. Exploration activities were suspended until 1981.

1981: Canzona Mineral Inc. carried out an EM (VLF) survey covering all of Siscoe Island. In the spring, a 19-hole diamond drilling program was completed on Siscoe Island. Due to the significant mineralized zones intersected, Phase 2 of the drilling program was completed in June, consisting of four DDH, drilled northward into a VLF EM conductor associated with the K Zone.

1984–1990: From 1984 to 1990, Maufort Resources Inc. completed exploration and underground work in joint venture partnership with Teck Corporation (1984–1987) and Cambior Inc. (1987–1989). A total of 20,693 m of diamond drilling was carried out and the mine pumped dry.

1993–1997: In 1993, Maufort Resources Inc. changed its name to Dynacor Mines Inc. Dynacor Mines completed mapping, outcrop stripping, diamond drilling and sampling. In 1997, nine DDH were sunk on the property for a total of 3,170 m.

1999: As part of its Reorganization Agreement, Dynacor Mines transferred all its interests in the Siscoe property (100%) to Wesdome Gold Mines on November 9, 1999.

During the period between 1929 and 1949, the Siscoe mine had produced a total of 882,303 ounces of gold and 306,070 ounces of silver from 2,975,785 t of ore grading an average of 9.22 g/t Au and 3.20 g/t Ag (Table 6-5).

Table 6-5: Siscoe mine yearly production from 1929 to 1949

Year	Metric tonnes (t) milled	Recovered grade Au g/t	Gold production (oz)	Recovered grade Ag g/t	Silver production (oz)
1929	27,067	17.07	14,853	1.38	1,200
1930	30,613	18.02	17,740	1.44	1,420
1931	50,508	22.10	35,883	0.99	1,614
1932	58,059	26.06	48,651	2.04	3,810
1933	87,407	19.48	54,729	3.14	8,826
1934	112,630	16.93	61,291	2.83	10,247
1935	135,236	14.82	64,446	3.70	16,089
1936	164,364	13.08	69,138	3.23	17,090
1937	181,895	12.89	75,383	3.61	21,095
1938	170,342	12.19	66,783	3.63	19,906
1939	171,965	9.76	53,982	3.76	20,765
1940	176,251	8.15	46,159	3.17	17,973
1941	208,710	6.63	44,461	3.79	25,457
1942	288,668	5.13	47,630	4.20	38,961
1943	290,407	4.35	40,656	3.51	32,808
1944	294,366	4.16	39,384	3.30	31,196
1945	241,466	4.33	33,610	2.30	17,864
1946	143,148	4.95	22,799	2.24	10,304
1947	64,649	10.37	21,556	2.33	4,847
1948	60,984	8.27	16,212	2.15	4,206
1949	17,049	12.69	6,957	0.72	392
Total	**2,975,785**	**9.22**	**882,303**	**3.2**	**306,070**

6.2.8 Siscoe Extension Area

The following description of historical work in the Siscoe Extension Area is mostly modified and summarized from Hawley (1931), Bell (1937), Denis (1937), Ross et al. (1938), Auger (1940), Auger (1947), Dresser and Denis (1949), Sullivan Consolidated Mines Ltd. (1957) and Dussault (1988).

1929–1937: During the winter of 1929–1930, some 10,000 ft (3,048 m) of diamond drilling was undertaken, by Siscoe Extension Gold Mines Ltd., on the property. Drilling was undertaken in the hope of picking up the eastern extension of the Siscoe granodiorite and possible veins. Following the drilling program, a three-compartment shaft was sunk on the lake's shore, and underground work was carried out on the 350 ft level. In 1936, diamond drilling was carried out from both underground and surface (from ice on Lac De Montigny). The latter was chiefly concerned with exploration for the presumed continuation onto the property of the northeastern branch or split of the K Zone of the Siscoe mine. But the drilling program was unsuccessful in locating the Siscoe K Zone beneath the lake. The overburden was penetrated to depths of up to 202 ft (61.6 m), but bedrock was not definitively reached. In 1937, the shaft reached a depth of 750 ft and the 750 ft level was established. Drifting was carried out on the 750 ft level.

1938: Operations were suspended in the fall of 1938, and the mine was flooded.

1946: Siscoe Gold Mines optioned the property and dewatered the underground workings. Geological surveying and re-sampling were carried out underground. Following this work, the option lapsed and the underground workings were flooded again.

1957: Sullivan Consolidated Mines Ltd. optioned the property and carried out an EM survey on the property. Surface diamond drilling was done from the ice south of the main underground workings.

1986: Maufort Resources Inc. acquired the property from Extension Holdings Inc. A total of 12 holes were drilled on the property for a total of 10,400 ft (3,169 m).

1987: Maufort Resources carried out a 56-hole drilling program on the property for a total of 81,361 ft (24,798.8 m). IP, Mag and seismic surveys were also carried out. The underground workings were dewatered.

1993: Maufort Resources Inc. changed its name to Dynacor Mines Inc.

1999: As part of its Reorganization Agreement, Dynacor Mines transferred all its interests in the Siscoe-Extension property (75%) to Wesdome Gold Mines Inc. on November 9, 1999.

7. GEOLOGICAL SETTING AND MINERALIZATION

The Kiena Mine Complex area is located in northwestern Quebec (Figure 7-1), straddling the southern part of the Abitibi Greenstone Belt (AGB) and the northern part of the Pontiac Subprovince.

7.1 Regional Geology

7.1.1 Abitibi Greenstone Belt

The following description of the AGB is mostly modified and summarized from Monecke et al. (2017) and references therein.

The Neoarchean AGB forms the northeastern portion of the Abitibi-Wawa Subprovince in the southeastern portion of the Superior Province (Figure 7-1). The southern Superior Province consists of a collage of E-trending Mesoarchean to Neoarchean terranes that underwent a complex history of aggregation between 2720 and 2680 Ma (Percival, 2007). To the north, the AGB is bounded by the Opatica Subprovince (Figure 7-1), a high-grade metamorphic terrain that consists of tonalite, granodiorite, and granite intrusions, with minor outcrop areas of volcanic and sedimentary rocks (Benn et al., 1992; Sawyer and Benn, 1993; Davis et al., 1994). Geophysical constraints indicate that rocks of the Opatica Subprovince structurally underlie the supracrustal rocks of the AGB (Benn and Moyen, 2008).

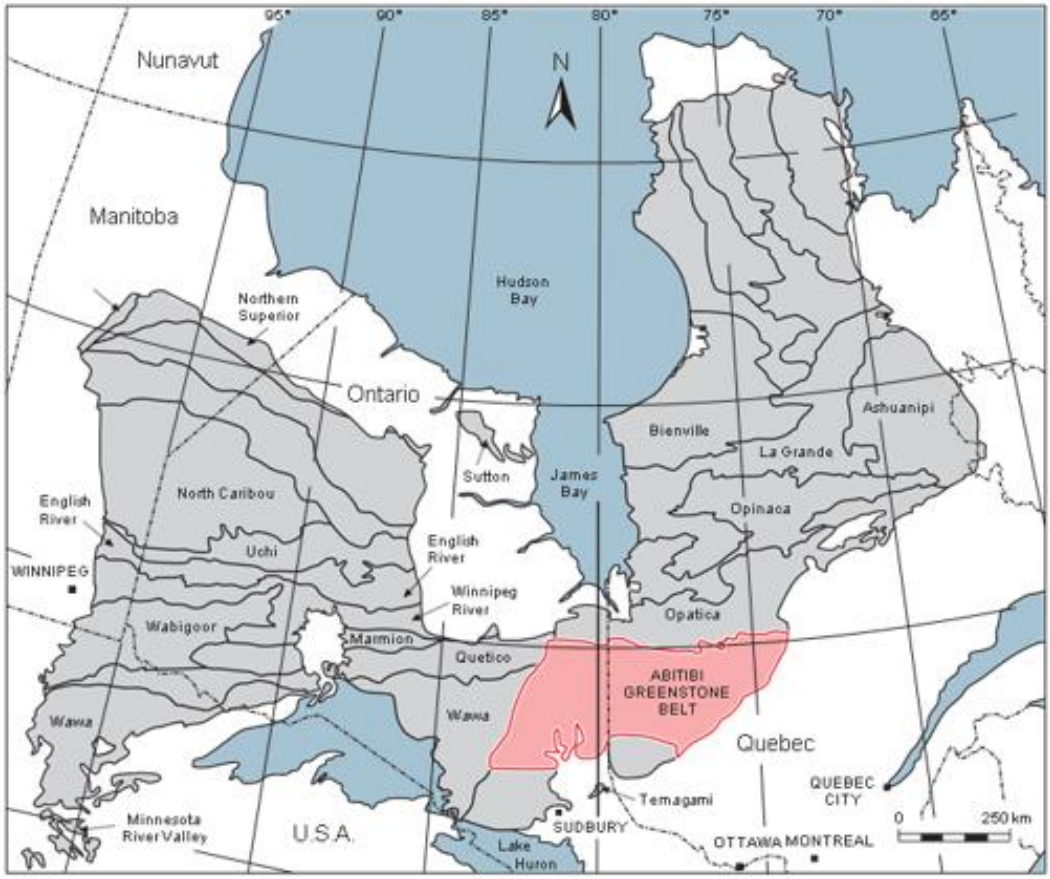


Figure 7-1: Location of the Abitibi Greenstone Belt within the Superior Province Subdivision of Superior Province from Thurston et al. (2008) and Stott et al. (2010); modified from Monecke et al. (2017)

To the east and southeast, the AGB is truncated by the Mesoproterozoic Grenville front tectonic zone, which is a southeasterly dipping zone of thrusts that juxtapose granulite facies metamorphic rocks with low-grade of the AGB (Indares and Martignole, 1989; Daigneault et al., 1990; Culshaw et al., 1997; Ludden and Hynes, 2000). To the southeast, the AGB is bounded by the Pontiac Subprovince (Figure 7-2). Structural studies along the Abitibi-Pontiac contact indicate that the AGB was thrust over the Pontiac Subprovince from the north (Camiré and Burg, 1993; Benn et al., 1994; Daigneault et al., 2002; Bedeaux et al., 2017). To the west, the AGB is interrupted by the 500-km-long NNE-trending Kapuskasing structural zone that exposes granulite facies metamorphic rocks (Percival and West, 1994). The Kapuskasing structural zone is a W-dipping thrust of Paleoproterozoic age along which Archean lower continental crust was upthrust (Percival et al., 1989). The uppermost part of the stratigraphy of the Wawa Greenstone Belt (Williams et al., 1991) to the west of the Kapuskasing structural zone is correlative with the AGB to the east (Percival and Card, 1983; Ayer et al., 2010).

The AGB comprises E-trending successions of folded and faulted volcanic and sedimentary rocks and intervening domes of intrusive rocks (Daigneault et al., 2004; Goutier and Melançon, 2007; Thurston et al., 2008; Ayer and Chartrand, 2011). The volcanic successions in the AGB typically have a steep dip and commonly young away from major intervening domes of intrusive rock (Thurston et al., 2008). Submarine mafic volcanic rocks prevail, forming approximately 90% of the outcrop area. Felsic volcanic rocks account for most of the remainder (Goodwin and Ridler, 1970; Goodwin, 1977; Hannington et al., 1999), with komatiites forming a small but important part of many of the volcanic successions (Imreh, 1984; Sproule et al., 2002; Houlé and Lesher, 2011; Dostal and Mueller, 2013).

An important geologic feature of the AGB is the occurrence of major, E-trending ductile-brittle fault zones. These zones cut across the entire belt from the Kapuskasing structural zone in the west to the Grenville front in the east, dividing the supracrustal rocks and intervening domes into distinct lozenge-shaped domains. The two most important fault zones in the southern AGB are Destor-Porcupine Fault Zone (DPFZ) in the north and Larder Lake-Cadillac Fault Zone (LLCFZ) in the south (Figure 7-2). These faults are subvertical (70°-90°) and dip either to the north or the south. They have highly variable widths, ranging from tens to hundreds of metres (Poulsen, 2017), and are generally marked by intense ductile-brittle deformation and penetrative fabric development. Most geologists agree that the fault zones are long-lived structures that controlled sedimentation and volcanism in the AGB since at least 2679 Ma. (Dimroth et al., 1982; Mueller et al., 1991, 1994; Cameron 1993; Mueller and Corcoran, 1998; Daigneault et al. 2002; Bleeker, 2012).

Based on recent geochronological information (Ayer et al., 2002b, 2005; Thurston et al., 2008), six volcanic assemblages are distinguished that formed by submarine volcanic activity between ca. 2750 and 2695 Ma (Figure 7-2). These assemblages spanned over 50 Ma years and 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 (2719-2711 Ma);
- Tisdale Assemblage (2710-2704 Ma);
- Blake River Assemblage (2704-2695 Ma).

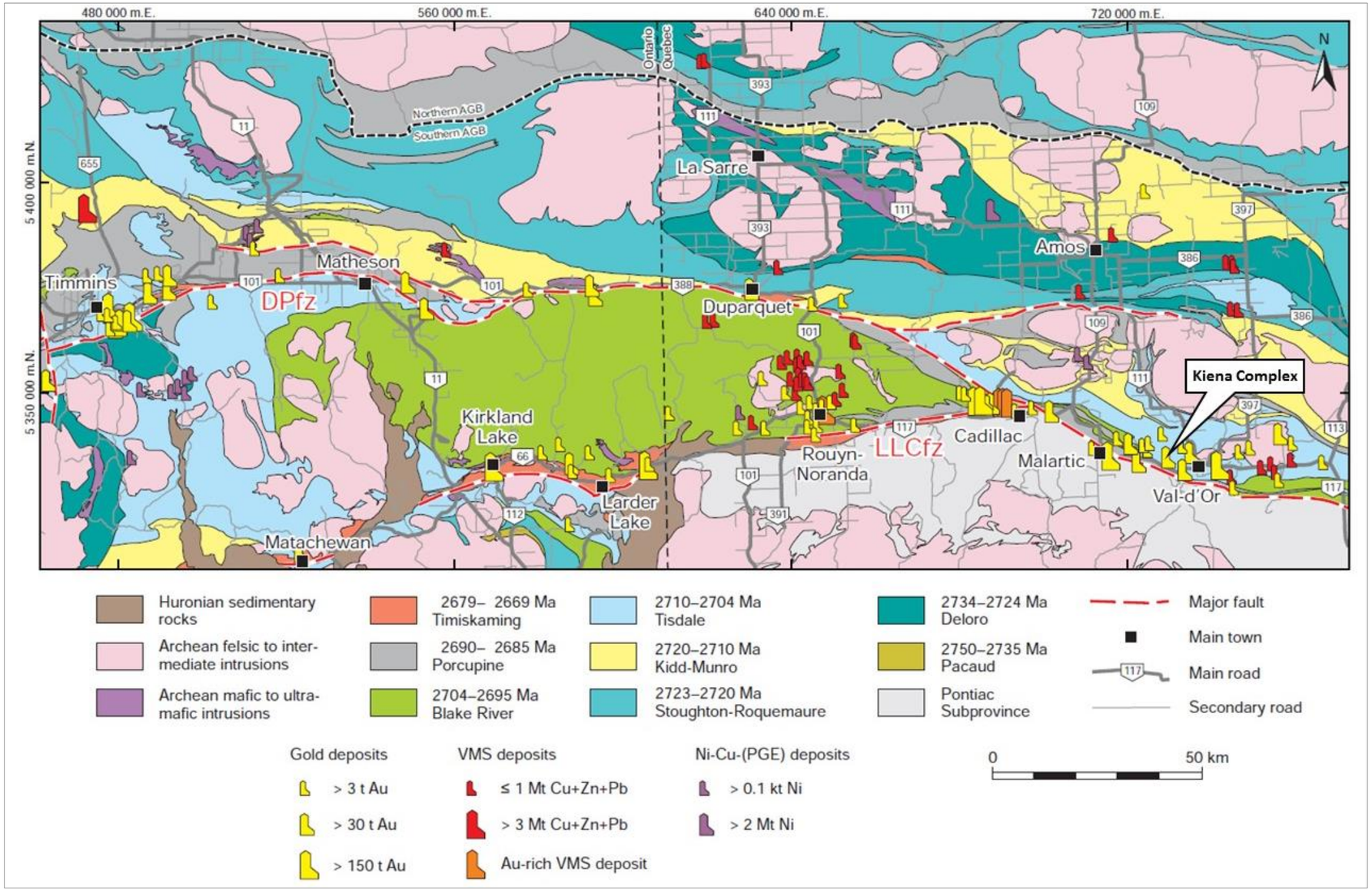


Figure 7-2: Geologic map of the southern Abitibi Greenstone Belt and location of the Kiena Complex
LLCFZ = Larder Lake-Cadillac Fault Zone; DPFZ = Destor-Porcupine Fault Zone (modified from Thurston et al. (2008) and Poulsen (2017))

Volcanic rocks older than 2750 Ma locally are found in the AGB, as indicated by recent studies southwest of Chibougamau, where 2795 to 2759 Ma volcanic rocks were mapped (Mortensen, 1993; Bandyayera et al., 2004; Davis and Dion, 2010; Leclerc et al., 2011, 2012)

In the southern Abitibi Greenstone Belt, the ≤2690 to ≤2685 Ma Porcupine assemblage consists of flysch-like deposits that records sedimentation in a deep submarine environment (Rocheleau, 1980; Lajoie and Ludden, 1984; Stone, 1990; Born, 1995; Frieman et al., 2017). The Porcupine assemblage locally includes minor calc-alkaline volcanic rocks (i.e., the Krist Formation in the Timmins area; Ayer et al., 2002b). The ≤2679 to ≤2669 Ma Timiskaming assemblage in the southern AGB is characterized by molasse-like clastic rocks deposited in a terrestrial setting (Hewitt, 1963; Dimroth and Rocheleau, 1979; Hyde, 1980; Rocheleau, 1980; Mueller et al., 1991, 1994; Legault and Hattori, 1994; Born, 1995). The clastic deposits of this assemblage are locally intercalated with predominately alkaline volcanic rocks (Cooke and Moorhouse, 1969; Mueller et al., 1994; Wilkinson et al., 1999; Ispolatov et al., 2008).

The supracrustal rocks of the AGB are intruded by plutons of variable compositions and sizes. Depending on emplacement age, several groups of plutons can be distinguished (Rive et al., 1990; Feng and Kerrich, 1992a, 1992b; Sutcliffe et al., 1993; Mueller et al., 1995; Chown et al., 2002; Beakhouse, 2011). Pre-2695 Ma intrusions in the southern AGB are commonly tonalitic to granodioritic composition and their ages overlap with those of supracrustal rocks. The intrusions are weakly to well foliated and complexly deformed, suggesting that they were folded together with volcanic host-rock successions (Beakhouse, 2011). Pre-2695 Ma mafic to ultramafic intrusions are found throughout the southern AGB. Many of these intrusions form sills or lenticular units that crosscut stratigraphy at a low angle. Compositionally, the intrusions range from peridotite to gabbro and diorite. A large number of intrusions in the southern AGB range in age 2695 to 2660 Ma, which broadly corresponds to the timing of sedimentation in the Porcupine and Timiskaming successor basins. Intrusions of this age range are typically of grandioritic to granitic and dioritic to quartz monzodioritic composition. Post 2660 Ma intrusions of granitic or granodioritic compositions are rare in the southern AGB and mostly form part of large, multiphase batholithic complexes (Beakhouse, 2011).

The greenstone belt is affected by a widespread greenschist facies metamorphism (Jolly, 1978; Powell et al., 1993; Dimroth et al., 1983; Benn et al., 1994). The grade of metamorphism increases to amphibolite at the fringes of some plutons and approaching the Pontiac and Opatica Subprovinces or the Grenville front tectonic zone.

7.1.2 Pontiac Subprovince

The Pontiac Subprovince (PS) (Figure 7-2 and Figure 7-3) consists principally of a turbiditic succession composed of graywacke and mudstone with minor intercalated conglomerate and basalt (Gunning and Ambrose, 1940; Imreh, 1976; Sansfaçon and Hubert, 1990; Fallara et al., 2000; Pilote, 2013; De Souza et al., 2017). The metamorphic grade increases rapidly southward in the Pontiac Group (Imreh, 1976; Card, 1990; Camiré and Burg, 1993; Benn et al., 1994), from the biotite-chlorite zone along the southern contact of the Larder Lake-Cadillac Fault Zone, to garnet and staurolite zones within about 2 km to the south (Sansfaçon and Hubert, 1990; Fallara et al., 2005; Grant et al., 2005; Piette-Lauzière et al., 2014).

The minimum depositional age of the Pontiac Group is constrained by the crosscutting 2682 ±1 Ma Lac Fournière Pluton (Fallara et al., 2000), whereas the maximum age is given by the youngest detrital zircons from greywacke dated at about 2685 Ma (Davis, 2002; De Souza et al., 2017).

7.2 Geology of the Kiena Mine Complex Area

The southern Abitibi Greenstone Belt in the Kiena Complex area consists of 2714–2700 Ma volcano-plutonic assemblages, including the Malartic and Louvicourt groups, intruded by calc-alkaline plutonic rocks (Figure 7-3). The Malartic Group comprises mainly komatiitic and tholeiitic basalt flows and sills, with minor sedimentary rocks, which are interpreted as an oceanic floor in an extensional environment related tomantle plumes, whereas the Louvicourt Group is mainly composed of mafic to felsic volcanic rocks that formed in a subduction-related arc setting (Desrochers et al. 1993; Daigneault et al. 2002; Scott et al. 2002). From south to north, the Kiena Complex area is underlain by the lithologies of the Pontiac Group (PO), the Piché Group (PG), the Cadillac Group (CG), and formations belonging to the Louvicourt Group and the Malartic Group.

The following description of the Kiena Complex area geology is mostly modified and summarized from Champagne et al. (2002), Champagne (2004), Scott et al. (2002), Olivo and Williams-Jones, (2002), Scott (2005), Pilote et al. (1998a, 1998b, 1999, 2014a, 2014b, 2015a, 2015b, 2015c), Pilote (2015a, 2015b), Monecke et al., 2017, and references therein.

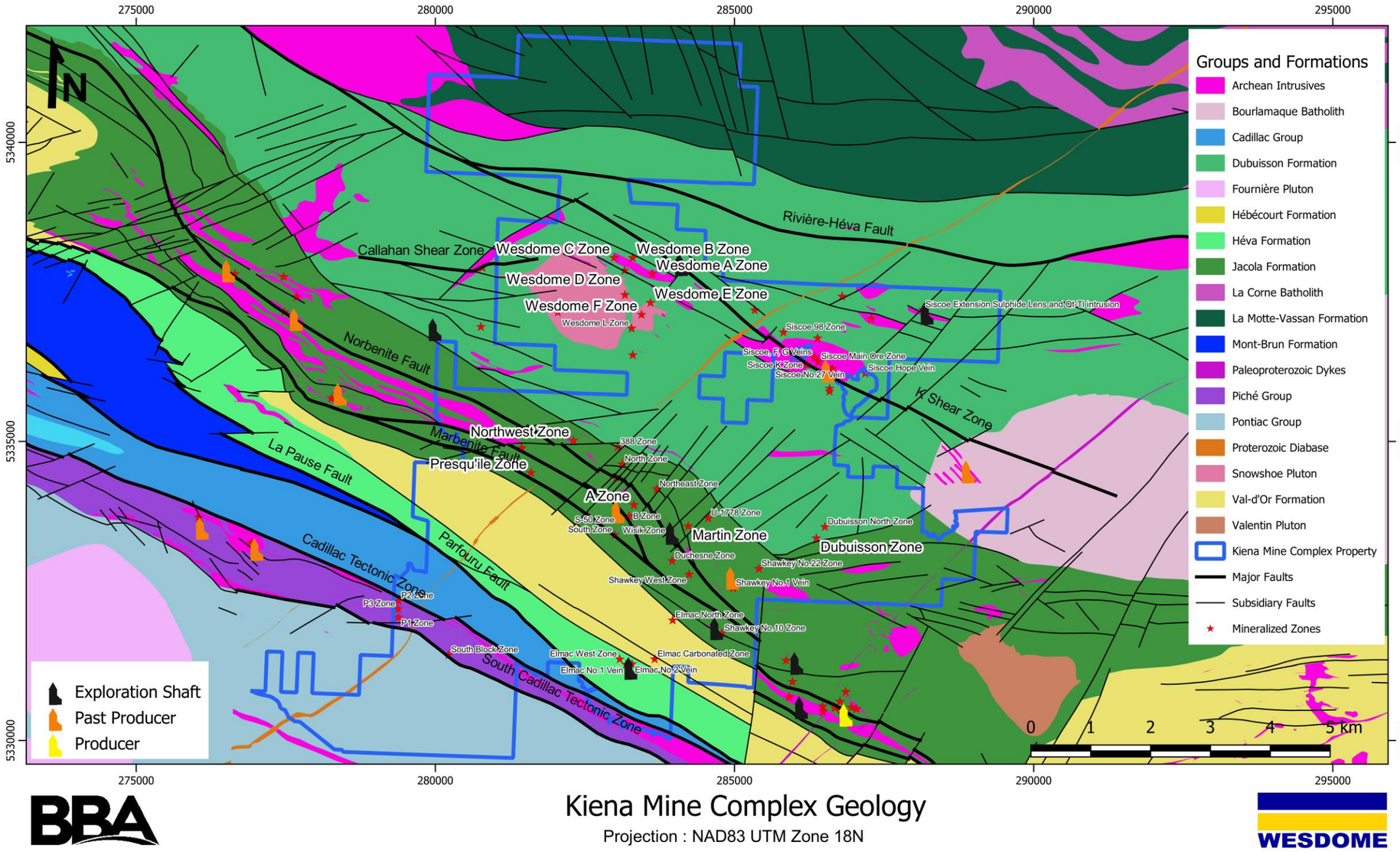


Figure 7-3: Kiena Mine Complex property geology with historic and active mines and mineralized zones (Adapted and modified from Pilote (2013, 2015a, 2015b))

7.2.1 Stratigraphy

Stratigraphic concepts were used to subdivide the E-Trending successions of folded and faulted volcanic rocks from the AGB into formally named rock units (Figure 7-4). Stratigraphic criteria have been successfully applied in mapping individual mining camp from the AGB. However, stratigraphic correlation at the larger scale across and between mining camps is hampered by the fact that boundaries between lithostratigraphic units of higher rank are often structural in nature. In addition, the glacial cover is extensive in some areas, further complicating stratigraphic correlation.

The AGB in the Val-d'Or–Malartic region has been divided into two stratigraphic groups based on regional tectonics and volcano-sedimentary stratigraphy: the basal Malartic Group comprising the La Motte-Vassan (LVF), Dubuisson (DF) and Jacola Formations (JF), and the upper Louvicourt Group comprising the Val-d'Or (VDF) and Héva Formations (HF) (Figure 7-3).

Originally, the volcanic rocks in the Val-d'Or–Malartic region were assigned to the Malartic Group (Gunning and Ambrose, 1940). Latulippe (1976) revised the stratigraphic nomenclature and distinguished a Lower and Upper Malartic Group. According this stratigraphic nomenclature, the La Motte-Vassan and Dubuisson Formations form the Lower Malartic Group, whereas the Jacola, Héva, and Val-d'Or Formations represent the Upper Malartic Group. Based on a subsequent revision of the stratigraphic nomenclature by Scott et al. (2002), the Malartic Group today encompasses the La Motte-Vassan, Dubuisson, and Jacola Formations. Scott et al. (2002) added the Louvicourt Group in the recent stratigraphic nomenclature, which was further divided into the Héva and Val-d'Or formations.

7.2.1.1 Pontiac Group (PO)

In the Kiena Complex area, the Pontiac Group (PO) covers the sector to the south of the LLCFZ. The PO, located only in the Pontiac Subprovince, consists of turbiditic units (mostly greywacke and siltstone) with rare monomict conglomerate intercalations intruded by thin ultramafic units (Dimroth et al., 1982; Goulet, 1978, Ludden and Hubert, 1986). It is interpreted as an accretionary prism created by the subduction of the Pontiac Subprovince under the Abitibi Subprovince (Camiré and Burg, 1993; Card, 1990; Davis, 2002). It is contemporaneous with the Cadillac Group with a maximum age between 2682 Ma and 2685 Ma (Davis, 2002; Mortensen, 1993). The sedimentary strata of the Pontiac and Cadillac groups are interpreted as synorogenic flysch-type assemblages (Mueller et al., 1996; Daigneault et al., 2002), whereas Dimroth et al. (1982) considered the Pontiac Group to have been deposited in a foreland belt.

7.2.1.2 Piché Group (PG)

The Piché Group (PG), at the contact between the Cadillac and Pontiac sedimentary successions, forms a relatively narrow (<2 km) band that marks the trace of the LLCFZ (Latulippe 1976; Dimroth et al. 1982; Doucet and Lafrance 2005). The LLCFZ was then delimited (Ambrose, 1944; Gunning, 1937; Gunning and Ambrose, 1940) based on the presence of talc-schist and chlorite-schist correlated to the PG. The PG consists mainly of highly strained and metamorphosed volcanic rocks (tholeiitic basalts, porphyritic andesites, calc-alkaline tuffs, and komatiites) crosscut by felsic to mafic dykes and sills (Latulippe 1976; Dimroth et al. 1982; Imreh 1984; Landry 1991; Beaudoin and Trudel 1989). However, it has since been shown that the rocks of the LLCFZ were not uniformly affected due to heterogeneities in the distribution and intensity of deformation. Primary textures, such as spinifex and cumulates, have been preserved in areas where deformation is less intense. These less deformed rocks are typically discontinuous and encompassed by bands of schists. Latulippe (1976) proposed that the Piché Group be considered as a lithostratigraphic unit, whereas Imreh (1984) proposed that the Piché Group be considered as a discordant unit.

The minimal age for the PG is constrained by a U-Pb age of 2708.8 ±1.0 Ma obtained from a tonalite dyke that cuts the ultramafic units of the Buckshot pit near the Canadian Malartic deposit (David et al., 2018).

7.2.1.3 Cadillac Group (CG)

The Cadillac Group (CG) (Figure 7-3) crops out extensively immediately to the north of the LLCFZ, stretching from Rouyn Noranda in the west (Dimroth et al., 1982; Rocheleau, 1980) to Val-d'Or (Figure 7-4) in the east (Pilote, 2015a). The CG (≤2687 Ma; Davis, 2002) consists of sedimentary rocks including greywacke, pelitic schists, polymictic conglomerates, and iron formations (Trudel et al. 1992). Daigneault et al. (2002) proposed that the sedimentary rocks of the Cadillac Group represent an extensive volcano-sedimentary apron sequence that straddles the LLCFZ.

The CG is a flysch-type sedimentary basin that rests unconformably over volcanic assemblages (Mueller and Donaldson, 1992). The CG is a 150 km by 5 km basin located along the LLCFZ to the north. The group is interpreted to be a lateral equivalent of the Porcupine Group in Ontario (Ayer et al., 2002a; Thurston et al., 2008).

The sedimentary basin pinches out to a few hundred metres thick southeast of the Kiena Complex (Figure 7-4). It is mostly composed of turbiditic sedimentary rocks with rare local interlayering of polymictic conglomerates. The CG is identified based on its distinctive banded iron formations (Dimroth et al., 1982). Deposition ages for sediments are estimated at 2687 Ma ±3 (Davis, 2002).

7.2.1.4 Louvicourt Group (LG)

The Louvicourt Group (LG) is divided into the Héva (HF) and Val-d'Or Formations (VDF) (Scott et al., 2002).

Héva Formation (HF)

The Héva Formation (HF) can be as thick as 2.5 km and is composed of laterally extensive massive to pillowed basalt flows, gabbroic sills and dykes, and minor felsic to mafic volcaniclastic deposits (Scott et al., 2002). Volcaniclastic units are characterized by coarse or fine tuff horizons with millimetre-scale laminations, intruded by gabbro. The volcanic rocks of the HF have a U-Pb age of 2702 ±2 Ma (Scott et al., 2002). The HF belongs to the Blake River assemblage.

Val-d'Or Formation (VDF)

The Val-d'Or Formation (VDF) has a stratigraphic thickness of 3 km to 5 km and is dominated by massive to pillowed andesitic to rhyolitic lavas and associated volcaniclastic deposits. Diorite intrusions represent a minor component. Andesites form up to 1-km-thick and 9-km-long amalgamed flow units. Individual flow can be as thick as 80 m. The flows are intercalated with amalgamated volcaniclastic beds 5 m to 40 m thick. The pillows exhibit a variety of forms, from strongly amoeboid to lobed. Felsic lavas are laterally more restricted, but can be traced for 1 km to 3 km along strike. Volcaniclastic units reach up to 40 m in thickness. The volcaniclastic beds are composed of lapilli tuff, lapilli and blocks tuffs, and, to a lesser extent, fine to coarse tuffs. The VDF yielded a U-Pb age of 2704 ±2 Ma (Scott et al., 2002). The VDF belongs to the Blake River assemblage.

7.2.1.5 Malartic Group (MG)

The Malartic Group (MG) encompasses the La Motte-Vassan (LVF), Dubuisson (DF), and Jacola Formations (JF) (Scott et al.,2002).

Jacola Formation (JF)

The Jacola Formation (JF) has an apparent stratigraphic thickness of 1 km to 2 km and consists of basalts and komatiites. Basaltic flows are typically non-vesicular, ranging from massive to pillowed, and sometimes are in the form of flow breccias. Mafic volcaniclastic deposits, primarily formed by quench fragmentation, are locally abundant. Individual basalt units can be traced for distances of up to 5 km along strike. Individual komatiite units reach thicknesses of 100 to 200 m and range from massive to pillowed. Occasionally, spinifex-textured komatiite flows can be observed. The top of the JF has a U-Pb age of 2703.8 ±1.3 Ma (Scott et al., 2002). The JF belongs to the Tisdale assemblage.

Dubuisson Formation (DF)

The Dubuisson Formation (DF) (2708 ±2 Ma) consists mainly of pillowed and massive basalt with various interbedded komatiitic flows (Imreh, 1980; Pilote et al. 1999). Ultramafic and mafic flows are similar to those described in the LVF but in different proportions. The DF belongs to the Tisdale assemblage.

La Motte-Vassan Formation (LVF)

The La Motte-Vassan Formation (LVF) crops out on the north side of Lac De Montigny. Komatiites occur as sheet and tube-shaped flows that are intercalated with pillowed or massive basalt flows (Latulippe, 1976; Imreh, 1984, Pilote et al., 2009). Komatiites are more abundant than basalts (Imreh, 1980). A well-exposed example of spinifex-textured komatiite flow of the Kidd-Munro assemblage is located at Spinifex Ridge north of Rivière-Héva (Champagne et al., 2002; Champagne, 2004; Houlé et al., 2017). The komatiite succession at Spinifex Ridge is interpreted to be age-equivalent to the 2714 ±2 Ma komatiite succession of the LVF at Marbridge Ni deposit to the southwest (Pilote et al., 2009; Houlé et al., 2017).

The age of the LVF (2714 ±2 Ma) suggests it may be contemporaneous with the upper part of the Kidd-Munro Assemblage (Figure 7-1). The LVF consists of komatiites, tholeiitic basalts and magnesian basalts metamorphosed to amphibolite facies. The base of the sequence is mostly represented by komatiites with some minor intercalated basalt. However, a decrease in the proportion of komatiites is observed towards the top of the sequence (Imreh, 1984). Komatiites are mainly found as two morphofacies: classic sheet flows with spinifex textures or tube-shaped flows, or mega-pillows. The basalt flows are usually massive or pillowed (Imreh 1980).

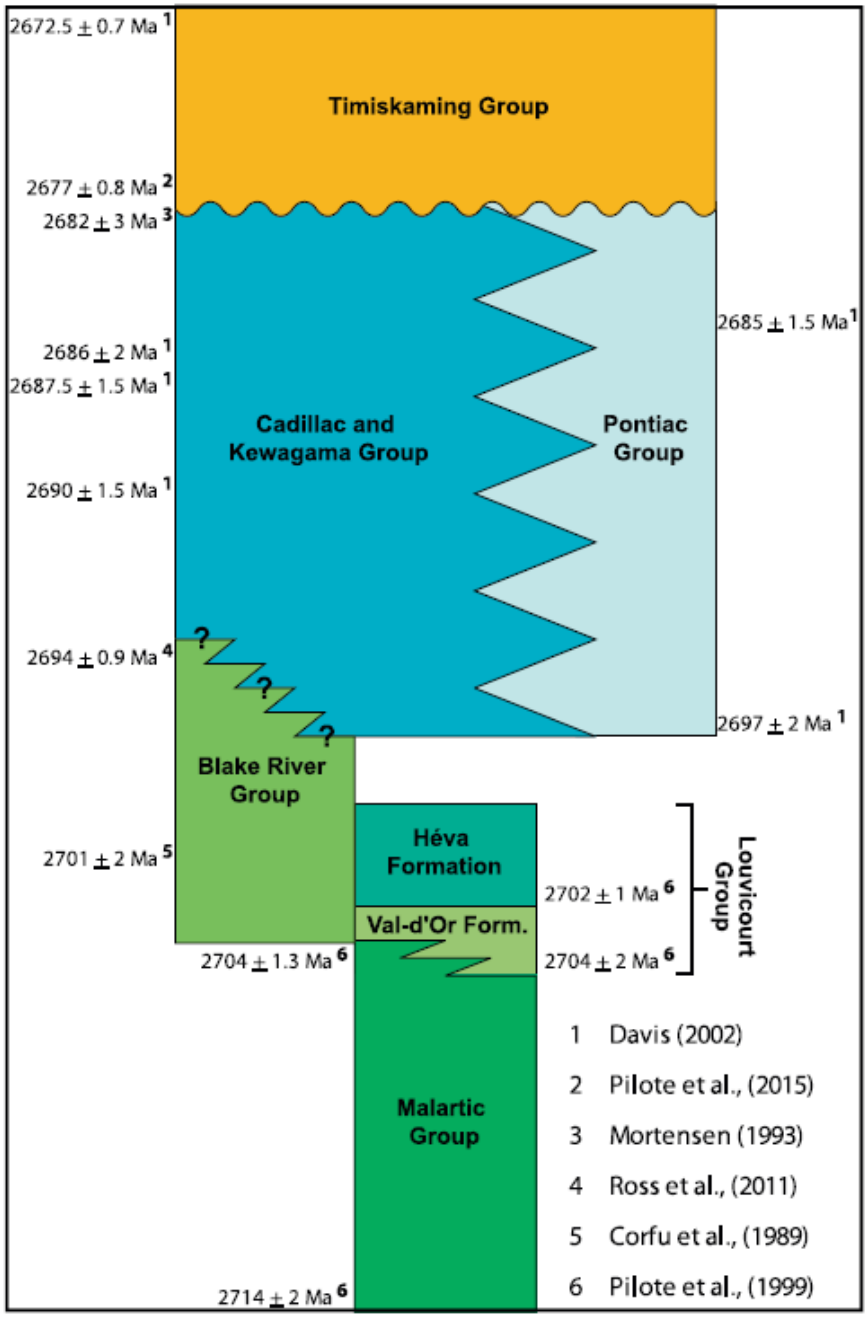


Figure 7-4: Stratigraphic chart of the Kiena Complex region with relevant isotopic ages Stratigraphic relationships based on Dimroth et al., (1982); Mueller et al., (1996); Scott et al., (2002); Tourigny et al., (1988). Figure from Bedeaux et al. (2017)

7.2.2 Intrusive Rocks

Pre-2695 Ma mafic to ultramafic intrusions are found throughout the southern AGB. Many of these intrusions form sills or lenticular units that crosscut stratigraphy at a low angle. Compositionally, the intrusions range from peridotite to gabbro and diorite. In the Kiena Complex area, the pre-2695 Ma Bourlamaque batholith (2699 ±1 Ma; Wong et al., 1991) is a good example of synvolcanic intrusive rocks.

A large number of intrusions in the southern AGB range in age from 2695 to 2660 Ma, which broadly corresponds to the timing of sedimentation in the Porcupine and Timiskaming successor basins. Intrusions of this age range are typically of granodioritic to granitic and dioritic to quartz monzodiorite composition.

In the Kiena Complex area, intrusions with ages between 2695–2685 Ma were emplaced immediately prior to the deposition of the Temiscaming assemblage. A number of small granodiorite, tonalite and monzonite range in age 2694 to 2685 Ma. This includes Camflo stock (Jemielita et al, 1990), Kiena porphyry dykes (Morasse et al., 1995), Lamaque stock (Jemielita et al., 1989), Norlartic dykes (Pilote et al., 1993), Sigma-Lamaque feldspar porphyry dykes (Wong et al., 1991), and Snowshoe stock (Morasse et al., 1995).

Intrusions emplaced at 2679 to 2660 Ma formed during and immediately after the deposition of the Timiskaming assemblage. Several small monzonite intrusions from the Malartic area yielded U-Pb zircon and titanite ages of ~2678 Ma (De Souza et al., 2017).

Post-2660 Ma intrusions of granitic or granodioritic compositions are rare in the southern AGB and mostly form part of large, multiphase batholithic complexes (Beakhouse, 2011). The La Corne batholith is a good example and represents a large intrusion post-2660 Ma (Feng and Kerrich, 1991; Powell et al., 1995; Ducharme et al., 1997).

The youngest igneous activity in the Kiena Complex area corresponds to the emplacement of Proterozoic diabase dykes that cut across the Superior Province along a NE trend.

7.2.3 Structural Elements from Kiena Complex

Although the structural geology of individual mining camps is well established, there is no widely agreed upon model for the structural evolution and generation of structures in the southern Abitibi Greenstone Belt (Monecke et al., 2017). This is in part due to the highly variable quality of exposures, strain heterogeneity, the lack of temporal constraints and, most importantly, variable preservation of early structures in the different mining camps. Actually, the structural and tectonic evolution of the southern AGB is not yet widely accepted.

Pilote et al. (2015c) established the nomenclature for the various structural elements in the Kiena Complex area, as described below.

The oldest regional schistosity is S1. It is systematically subparallel to bedding, S0. Within the formations of the Malartic Group, the overall S1 trend is NW-SE. Both fabrics, S0 and S1, are coplanar and show a moderate to steep dip to the north. S1 contains the primary stretching lineation L1. In the southern and central parts of the Property, S0 and S1 are jointly folded into Z-folds, with an average axial plane of N095°/85° and generally E-W axially planar cleavage (S2). The axes of F1 and F2 folds are parallel to the plunges of the L1 stretching lineation contained in S1.

A late S3 cleavage is the product of kinking and chevron folds in highly altered units showing a strong pre-existing anisotropy. Dykes, mainly tonalite and monzonite, are deformed and affected by S2. They trend to the SE, subparallel to the trace of the La Pause Fault. In places, they exhibit a stretching lineation with a shallow westward plunge.

7.2.4 Large-scale Fault Zones

The Kiena Complex area has a series of large-scale shear zones and related subsidiary faults trending ESE-WNW to SE-NW, subparallel to stratigraphy and dipping steeply to the north (Figure 7-3). They are, from south to north: Larder Lake-Cadillac Fault Zone (LLCFZ); Parfouru Fault (PF); Marbenite Fault (MF), Norbenite Fault (NF); Callahan Fault (CF); K Shear Zone (KSZ); and Rivière Héva Fault (RHF). The Kiena Complex area is cut by all of them. Most faults at surface have been interpreted by geophysics and traced on high-resolution aeromagnetic maps. Some faults correspond to a break in metamorphic grade.

The shear zones contain dykes or stocks of monzonitic or tonalitic composition that vary widely in age (pre-, syn- or post-tectonic) and are spatially associated with gold mines (Norlartic, Marban, Kiena, Sullivan, Goldex, Siscoe, Joubi, Sigma and Lamaque). The observed diversity in the styles and ages of gold mineralization related to these large-scale shear zones demonstrates that several distinct episodes of mineralization occurred.

7.2.4.1 Larder Lake-Cadillac Fault Zone (LLCFZ)

The LLCFZ in the AGB is a first order gold-bearing structure on a province-wide scale, accounting for half of the gold production and reserves in Abitibi and more than 25% in Canada (based on data by Dubé and Gosselin, 2007). The LLCFZ is a 250-km-long, moderately to steeply dipping structure with a curvilinear trace (Figure 7-2 and Figure 7-3). It dips northward or southward depending on location along its strike. A remarkable characteristic of the LLCFZ is that over much of its 250-km length it marks the contact of a persistent band of volcanic rocks, which in places is only 1 km to 100 m thick. Composed mainly of ultramafic komatiite and tholeiitic basalt, the volcanic rocks provide sharp lithologic contrasts with adjacent sedimentary units.

The LLCFZ in the Kiena Complex area is sharply defined as the contact between the southward-facing volcanic successions of the Malartic Group and the younger folded, but dominantly northward-facing, graywacke-mudstone successions of the Pontiac Group to the south (Poulsen, 2017). The LLCFZ is confined to a 200-m-wide high-strain zone containing thin, strongly deformed, but distinctive lithologic units. From north to south, these are porphyritic diorite that intrudes mafic rocks of the Malartic Group, graywacke and mudstone with lenses of conglomerate (assigned by most workers to the Cadillac Group), and felsic to ultramafic volcanic rocks of the Piché Group (Robert, 1989). The ultramafic talc-chlorite-carbonate schist of the Piché Group is compositionally a komatiitic to komatiitic basalt. Spinifex textures are locally preserved (Robert et al., 1990). The anomalous high strain is manifested by shape fabrics in primary lithic clasts and phenocrysts, intense schistosity, and common minor folds that result in strong transposition of layers (Robert, 1989).

Located south of the Kiena Complex, the LLCFZ is generally oriented N110° and dips steeply to the NNE.

7.2.4.2 Parfouru Fault (PF)

The PF is an ESE-WNW shear zone that dips steeply (75°) to the north or northeast and is interpreted as an early synvolcanic structure (Daigneault, 1996; Bedeaux et al., 2017). The shear zone can reach 300 m wide and has been traced for tens of kilometres.

7.2.4.3 Marbenite Fault (MF)

This major second order gold-bearing fault trends ESE-WNW to SE-NW and dips steeply to the northeast. The MF was first observed at the Marban mine (Trudel and Sauvé, 1992; Sauvé et al., 1993; and Beaucamp, 2010). Similar to the LLCFZ, the MF is characterized by the presence of ultramafic talc-chlorite-carbonate schist that came from strongly deformed and metamorphosed komatiitic basalt and/or komatiite. The MF is represented by an anastomosed schist that sometimes is folded in some places. Thin mylonitic fabrics are also present within the schist in some places.

In the Kiena Complex area, the MF is more than 100 m thick. It hosts the South Zone from the 27 Level to 32 Level and the S50 Zone from the 57 Level to 100 Level. A subsidiary folded fault, located 400 m north of the MF, merges with the MF master fault at a depth of 500 m. This subsidiary fault hosts a large part of the S50 Zone between the 12 Level and 54 Level.

7.2.4.4 Norbenite Fault (NF)

The Norbenite Fault (NF) is a major second-order gold-bearing fault that strikes WNW and dips 40-60° to the northeast in the Norlartic Mine area where the fault was first observed (Trudel and Sauvé, 1992; Sauvé et al., 1993). This fault also hosts the gold-bearing Kierens Zone that was accessible by underground working from Norlartic mine.

The NF is 15 m to 110 m wide and has been traced for more than 10 km. The VC Zone at Kiena occurs within this shear zone. The fault affects mainly the komatiitic units and occasionally the basaltic units of the JF. It can be divided into two or three branches in some places. Similar to the LLCFZ and the MF, the NF is characterized by the presence of ultramafic talc-chlorite-carbonate schist that came from strongly deformed and metamorphosed komatiitic basalt and/or komatiite.

7.2.4.5 Callahan Shear Zone (CSZ)

The CSZ strikes N090° and dips 60-80° to the north. The CSZ can reach up to 200 m in width (Beaudoin et al., 1987).

7.2.4.6 K Shear Zone (KSZ)

The KSZ is a shear zone 300 m to 600 m wide that has been traced for more than 3 km. It strikes N295° and dips 80° northeast. It is composed of talc and chlorite schists, actinolite schists and minor sericite schists, and bodies of pure talc and massive actinolite (Olivo and Williams-Jones, 2002; Olivo et al., 2007). The shear hosts the K Zone at the former Siscoe mine and bounds the Main Zone to the south. The volcanic domain north of the KSZ is at upper greenschist to amphibolite facies.

7.2.4.7 Rivière-Héva Fault (RHF)

The RHF is an 18-km-long ESE-WNW shear zone that dips steeply (80°) to the north or northeast (Daigneault, 1996). The shear zone can reach 300 m wide and has been traced over many kilometres. This structure corresponds to a change in metamorphic grade, from greenschist facies in the south to amphibolite in the north.

7.2.5 Mineralization Types

The information of this section was taken from Turcotte et al. (2015) and Beausoleil et al. (2019). Gold mineralization in the Property occurs in all rock types except the Proterozoic dykes but is more common in intrusive bodies and basalt as these acted as competent rock units that promoted fracturing during deformation. Gold mineralization is concentrated where there is a marked competency contrast between these competent units and the adjacent deformed komatiite and/or chlorite-talc schists.

According to Couture et al. (1994), there are at least two main gold mineralizing events in the region: young deposits in which the gold mineralization did not experience much deformation after its emplacement; and early mineralization in which mineralized bodies are commonly affected by D1 asymmetric folds, are highly strained and are locally dismembered. In a few deposits, both generations are present. Precise U-Pb zircon dating of an inter-mineral granodiorite dyke assigns a minimum age of 2686 ±2 Ma to the gold mineralization at the Kiena Mine (Morasse et al., 1995). This age reveals that gold mineralization postdates volcanism and the Snowshoe plutonism but predates regional syn-metamorphic deformation (ca. 2677-2645 Ma).

Gold-bearing veins in the region exhibit a great variety of orientations, mineralogy and crosscutting relationships. For the purpose of this report, they are classified into the following three main types:

- Type 1: early quartz-carbonate veins cut by various dykes;
- Type 2: deformed veins within a shear zone;
- Type 3: relatively weak deformed late quartz±tourmaline veins cutting all intrusive types and previous gold-bearing vein systems.

On occasion, all three types may occur together.

At least 63 mineralized zones have been observed on the Property and are described in Turcotte et al. (2015). The veins of these zones have been categorized as Type 1, 2 or 3 based on their principal characteristics.

In general, mineralized zones on the Property occur near a large-scale fault. They are often associated with a subsidiary shear zone that may be proximal, adjacent or host to the mineralization. Alteration minerals are dominantly albite, carbonates and pyrite with lesser chlorite and silica.

The gold occurrences found in shear zone settings are mainly restricted to competent units, and thus the size and shape of the mineralized zones often depend on the size, shape and concentration of the competent intrusive or basalt. Generally, it is possible to observe a flexure in the shear zone.

In zones of structural dislocation, three settings for gold mineralization have been recognized:

- Shattered intrusive bodies, such as diorite or feldspar porphyry dykes, enclosed in talc-chlorite schist;
- Zones of fracturing and brecciation in large bodies, such as basalt;
- Visible gold in quartz veins localized in schist and komatiite or in basalt and flow breccia near the contact with the schist and komatiite.

In large bodies of basalt, fracturing was generally restricted to narrow zones, and subsequent mineralization resulted in narrow and often closely spaced mineralized zones. In narrower dykes, the whole body is affected by fracturing, and subsequent mineralization was able to spread throughout the dyke, forming large mineralized zones. Three factors control the size and shape of mineralized zones associated with dykes in shear zone settings: the size of individual dykes, the density of the swarms, and the shear zone folding intensity.

Quartz veins with visible gold are usually found in a folded shear zone (schist). Generally, the veins seem to fill the available space created during the shear zone fracturing. The quartz veins density clearly increased going toward the nose of the fold. In some areas, two or three zones are stacked in the nose of the fold.

7.3 Mineralized Zones

Past gold production on the Property came from the Kiena, Siscoe and Shawkey mines. A summary of the geological setting and mineralization is presented for each of these past producers. The following sections are modified from Turcotte et al. (2015), Beausoleil et al. (2019), Richard and Torrealba (2019), and Corkal et al. (2020).

7.3.1 Kiena Mine

A total of six zones were mined at Kiena. Five are aligned in an N-S corridor (from south to north): South, S50, VC, North and 388. Those zones are situated in a sector were the Marbenite and Norbenite faults are changing direction. Regionally, those faults are N290º but change direction to N320º near those five zones. The sixth zone, Martin, is about 1.2 km east of the Kiena shaft.

7.3.1.1 S50 Zone

The S50 Zone was the main orebody at Kiena (Figure 7-5). It consists of a multistage carbonate-albite-pyrite stockwork, breccia and replacement vein system (Morasse et al., 1995; Morasse, 1998). It is located on the N-S limb of a local fold adjacent to the MF and is more or less concordant with the upper contact of a moderately west-dipping tholeiitic flow with a variably altered and schistose basaltic komatiite assigned to the JF. The mineralized zone measures between 225 m and 600 m long, between 10 m and 50 m wide, and has a vertical extent of 250 m to 1,000 m.

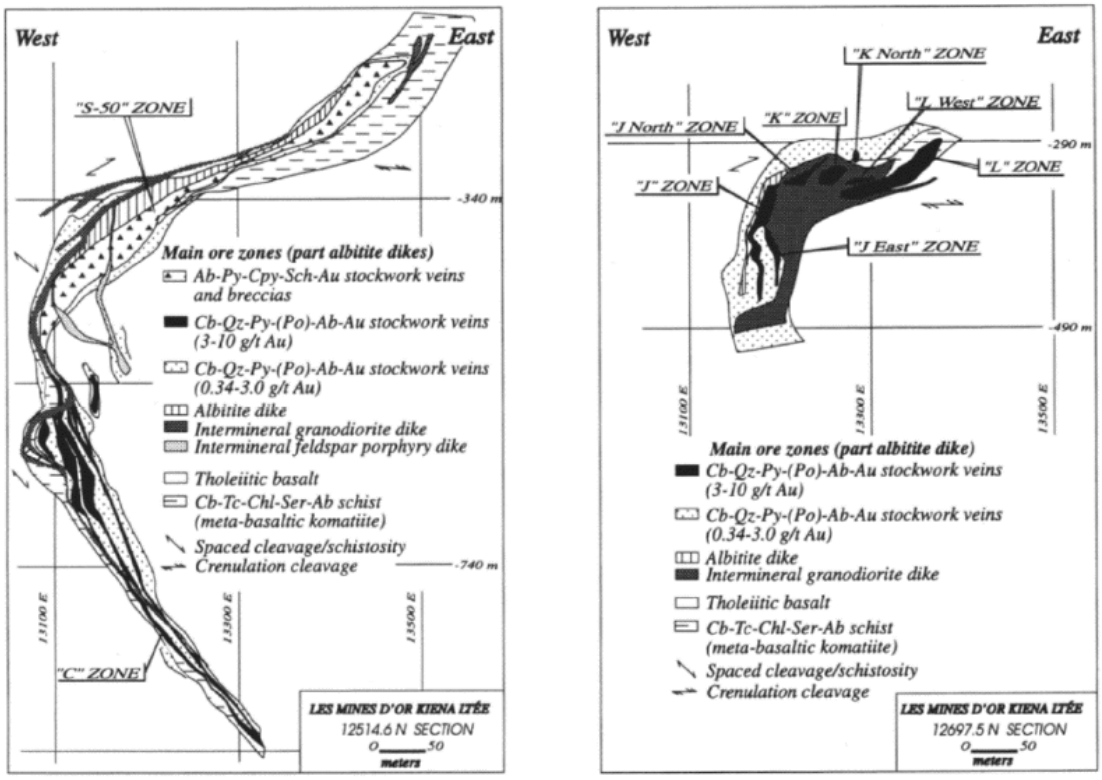


Figure 7-5: Geological cross-section of S50 Zone (Morasse et al., 1995)

The S50 Zone is associated with an intermediate to felsic dyke complex composed of a pre-mineralization albitized diorite (albitite) dyke swarm and post-mineralization granodiorite and feldspar porphyry dykes. It comprises Type 1 veins consisting of two mineralized types (from oldest to youngest):

1. Carbonate (ankerite)-pyrite-Au replacement veins, also known as the “Breccia 1” mineralization type.
2. Albite stockwork veins and breccias with disseminated pyrite, chalcopyrite, scheelite and gold, also known as the “Breccia 2” mineralization type.

These Type 1 veins are cut by various dykes, including porphyry dykes that are weakly mineralized by Type 3 calcite-quartz-pyrite-Au stockwork veins. The S50 Zone was divided into seven individual mineralized zones (A, B, C, D, J, K, L) based on the attitude and composition of the mineralization and the presence or absence of a granodiorite dyke. These mineralized zones can be regrouped into a high-grade core consisting of the A, B and D Zones and the lower part of the C, and a lower-grade mineralized shell comprised of the J, K and L Zones and the upper part of C (Figure 7-4). The C Zone is transposed within the MF.

Almost all zones in the MF are thin and elongated lenses and have a pitch angle of 60° to the east, making an angle of around 30° with the down-plug axis of the fault plane. This probably represents the stretching lineation in the fault zone.

7.3.1.2 S50 Deep Extension

Five mineralized lenses have been defined in the deep extension of the S50 Zone (Figure 7-6). These lenses correspond to the extension at depth of the S50 Zone below the 96 Level. They are named (from south to north) S50_104, S50_100, S50_101, S50_102 and S50_103. The lenses are sheeted and parallel to, or contained within, the MF and are all part of the S50 mineralized system. The five lenses are separated from each other by 5 m to 15 m of less mineralized material. S50_102 and S50_103 are associated with a mylonite that defines the hanging wall of the MF or its splay, near the contact between basalt and talc-chlorite schist. S50_102 is in direct continuity at depth with the C Zone and is similar in composition. The upper part of the S50_102 was clipped on the mined-out area starting below the 94th level (940 m vertical) and has lateral and vertical extents of 250 m and 110 m, respectively. The S50_103 has lateral and vertical extents of 110 m and 150 m, respectively.

S50_100, S50_101 and S50_104 occur at the eastern end of S50_102 in an area where the MF widens and branches out. They are also spatially associated with the mylonite zone. They have vertical extents between 430 m and 230 m and a horizontal extent of less than 150 m. Their thicknesses vary from 3 m to 15 m. Host rocks are talc-chlorite-carbonate schists with mineralized breccias typical of the S50 Zone. These breccias contains about 1-2% disseminated pyrite and pyrrhotite. Albite and ankerite alteration are present in the S50_100 and S50_102 lenses. Sometimes visible gold in quartz veins is observed inside mineralized breccia. Quartz veins with visible gold crosscut the mineralized and albitized breccia. Those veins are similar to those observed in the Kiena Deep A Zone.

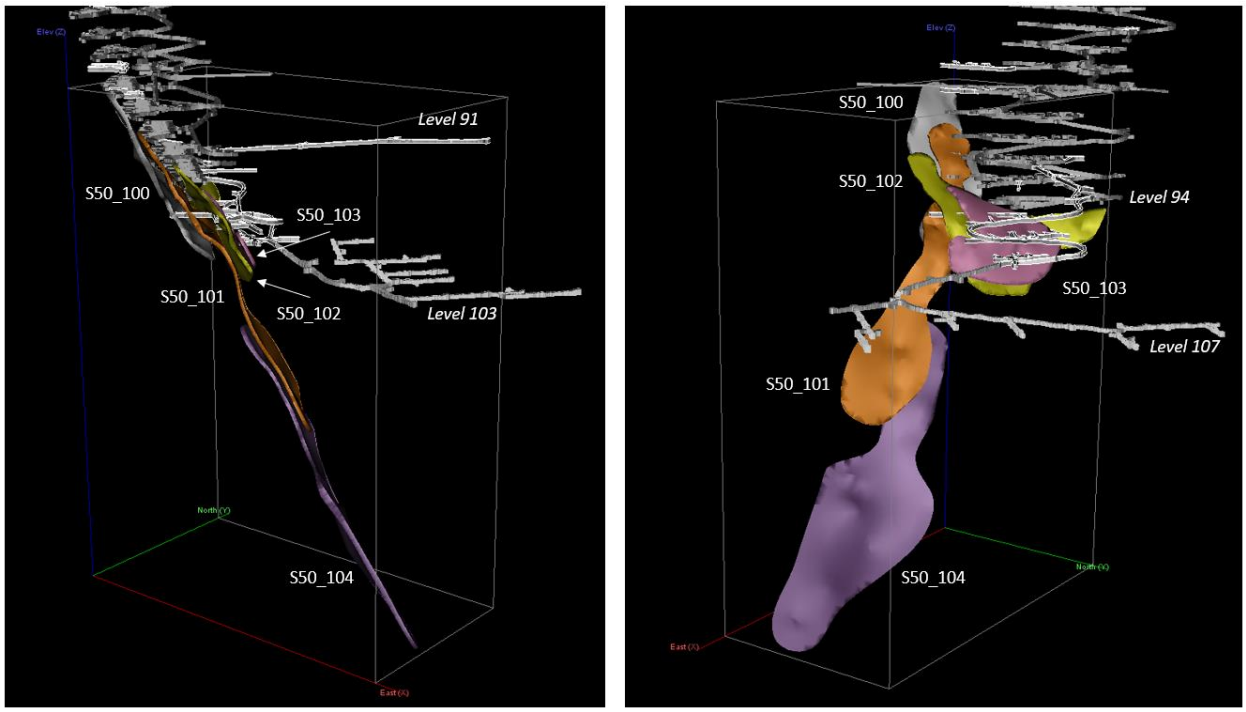


Figure 7-6: S50 deep extension mineralized zones
(left: looking northwest; right: looking southwest)

7.3.1.3 VC Zone

The VC Zone is approximately 700 m north of the S50 Zone and is bounded to the north by the Norbenite Fault. The following description of the VC Zone is mostly modified and summarized from Beauregard and Gaudreault (2005). The VC Zone was mined by Wesdome between 2006 and 2013. A total of 841,625 t of material with an average grade of 4.16 g/t Au was mined from this zone.

The mineralized zone occurs as four vertical E-W pinch-and-swell lenses (VC1, VC2, VC3 and VC4) in locally brecciated albitized basalt. The four lenses were mined over a vertical extent of 475 m. In cross-section, the lenses occur in a pseudo-en echelon pattern within a steep south-plunging corridor 100 m across. In plan view, the individual lenses are 100 m to 180 m long and average 8 m to 10 m wide, with a maximum width of 30 m. Gold mineralization is associated with quartz veins and carbonate-quartz-pyrite veins (Turcotte et al., 2015), presenting a similar mode of occurrence as seen in the S50 Zone.

In 2017, VC6 Zone was discovered when DDH 6187 intercepted a mineralized basalt and a non-mineralized granodiorite dyke. DDH 6187 was drilled down-dip of the zone and returned a value of 5.24 g/t on 48.7 m.

7.3.1.4 VC1 and VC6

Four mineralized lenses were defined at depth in the extension of the VC1. The VC1_114 lens is the continuity at depth of the VC1 lens and is interpreted to a depth of 315 m below the last level mined (i.e., 67 Level). In plan view, the VC1_114 lens is 65 m long and averages 4 m to 6 m across. The VC1_114 lens is generally associated with a diorite dyke located within the talc-chlorite schist. The dyke contains 1-5% pyrite and quartz-chlorite-pyrite veinlets ranging from 2 mm to 5 cm. In places, quartz veins are present within the dyke or at its contact. Visible gold is usually observed within the quartz veins and veinlet. The three other lenses are located about 10 m north of the VC1_114 lens. VC1_111, VC1_112 and VC1_113 are subparallel to the VC1_114. The best results from VC1 (VC1_113 lens) were obtained in DDH 6205 where a diorite dyke hosting quartz veins with visible gold returned a 7.5 m interval grading 262.09 g/t Au or 36.54 g/t Au cut at 90 g/t. Another significant intersection within VC1 (VC1_111 lens) comes from hole 6531. This hole returned an average grade of 36.44 g/t Au (20.48 g/t Au cut at 90 g/t) over 6.1 m. This gold intersection, located at the elevation of the 107 Level, is characterized by the presence of visible gold within white to grey quartz veinlets. Stockwork is hosted by an albitized and pyritized basalt.

Two mineralized lenses were defined in the VC6 area. The VC6_123 lens is located 150 m southeast of VC1_114. The mineralization is typical of the S50 Zone. The lens is elongated in shape, plunging to the south-west and measuring 185 m high by 80 m wide with thicknesses ranging from 5 m to 12 m. The lens is oriented NE, parallel to a 20 m thick chlorite-carbonate shear zone that bounds the mineralization to the northwest. It is hosted in a sheared basalt and in minor porphyritic or dioritic dykes. The main alteration minerals are chlorite and carbonates, with minor patches of albite, silica and/or amphibole alteration. Pyrite and pyrrhotite account for less than 3% of the rock. Locally, visible gold was observed in quartz-carbonate veinlets. This zone is crosscut by a massive, slightly sheared and barren granodiorite dyke. The dyke was possibly deformed at the same time as the volcanic host rocks. The VC6_124 lens was defined with the drillholes completed during the 2020 Program. This lens is subparallel to the VC6_123 and located northwest of it, in a similar environment on the other side of the chlorite-carbonate shear zone. Figure 7-7 presents the VC1 and de VC6 mineralized lenses.

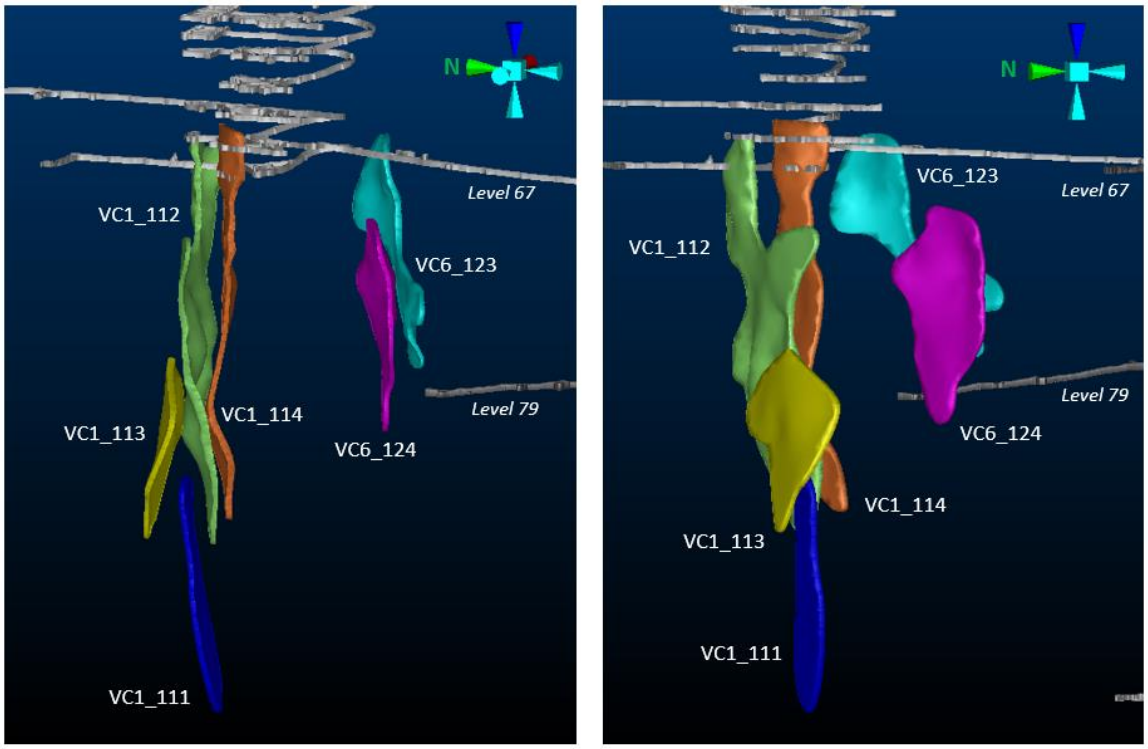


Figure 7-7: VC1 and VC6 mineralized zones
(left: looking west-southwest; right: looking south-southwest)

7.3.1.5 North Zone

The North Zone (formerly the No. 1 Zone) was first encountered by surface drilling in 1938. The zone is 900 m north of the S50 Zone and 200 m north of the NF (Figure 7-3). It consists of two parallel lenses shaped like flattened tubes, each measuring 10 m by 60 m to 90 m in plan view and plunging 70° north, which can be traced over a length of at least 520 m. From 2007 to 2013, Wesdome extracted 458,860 t of material from the North Zone with an average grade of 2.44 g/t Au.

The main structural control for the gold mineralization is a sodium-rich diorite intrusion in basalt host rock. Mineralization occurs as a quartz-carbonate-sulphide matrix-filling in brecciated and albite-altered diorite, and as quartz-carbonate-pyrite veins and veinlets in basalt wall rock (Turcotte et al., 2015). These veins and veinlets are predominantly developed in a steeply dipping orientation that is suggestive of a sheeted or ribbon-vein system. There are subsidiary veinlets at oblique and sub-random orientations, which may be described as a pseudo-stockwork. Well-developed albitized envelopes with disseminated pyrite encompass the quartz-carbonate veins, and it is common to observe 5-10% disseminated pyrite where intense and closely spaced veining is developed. The North Zone is cut by unmineralized granodiorite and feldspar porphyry dykes.

7.3.1.6 388 Zone

The following description of the 388 Zone is mostly modified and summarized from Laplante (2000). The 388 Zone has been defined as a small deposit parallel to the North Zone at a distance of 300 m to the north (Figure 7-3). It was mined by Wesdome from 2007 to 2013. A total of 146,268 t of material was mined from the zone at an average grade of 3.01 g/t Au.

The 388 Zone has a vertical extend of 100 m. Gold mineralization is associated primarily with a single major quartz vein 2 m wide that developed in albitized basalt alongside a diorite dyke. The vein is rather erratic and structurally meanders within an irregular zone of altered basalt 3 m to 6 m wide. Small scattered quartz-carbonate veinlets with minor pyrite occur in the wall rock.

7.3.1.7 South Zone

The South Zone lies 100 m to the south of the S50 Zone in the MF deformation corridor. It strikes northwest and dips 65° northeast. Discovered in 1981 (hole S-129), the South Zone includes the hanging wall and footwall lenses, which are 18 m apart. They were mined on three levels only (between levels 27 and 32) over a vertical extent of 55 m. They are parallel and concordant to the enclosing volcanic host rocks, predominately basalts with minor diorite dykes and komatiitic ultramafic flows. The basalts are strongly chloritized and carbonatized with local silicification and brecciation (quartz-carbonate veining) accompanied by low concentrations of pyrite (5%). The mineralization is similar to that observed in the S50 Zone. The South Zone is generally strongly fractured by the fault zone. Wesdome tried to mine this zone in 2012 and 2013, but after producing 37,076 t with an average grade of 1.74 g/t Au, production was abandoned due to dilution problems.

7.3.1.8 ZS-130, ZS-131, ZS-132, ZS-133 and ZS-135

Five mineralized lenses have been defined in the South Zone area (Figure 7-8). Wesdome extended the footwall and hanging wall lenses up-plunge to near-surface and 200 m down-plunge, and these extensions are now named the ZS_131 and ZS_132 lenses, respectively. ZS_130 is located less than 20 m south of ZS_131 and is subparallel to it. The maximum lateral extension of the ZS_130 is 500 m, but both ends are poorly drilled up to date. ZS_135 is a small lens intercalated between ZS_130 and ZS_131, with lateral extents of 150 m and vertical extents of 175 m. The thickness of the alteration and mineralization envelope that encloses the four lenses ranges from 25 m to 60 m. A fifth lens (ZS_133) was defined near the contact with the S50 Zone, in a distinct band of basalt. The ZS_133 is located 25 m north of ZS_132 and has a vertical extent of 285 m with thicknesses ranging from 3 m to 6.5 m.

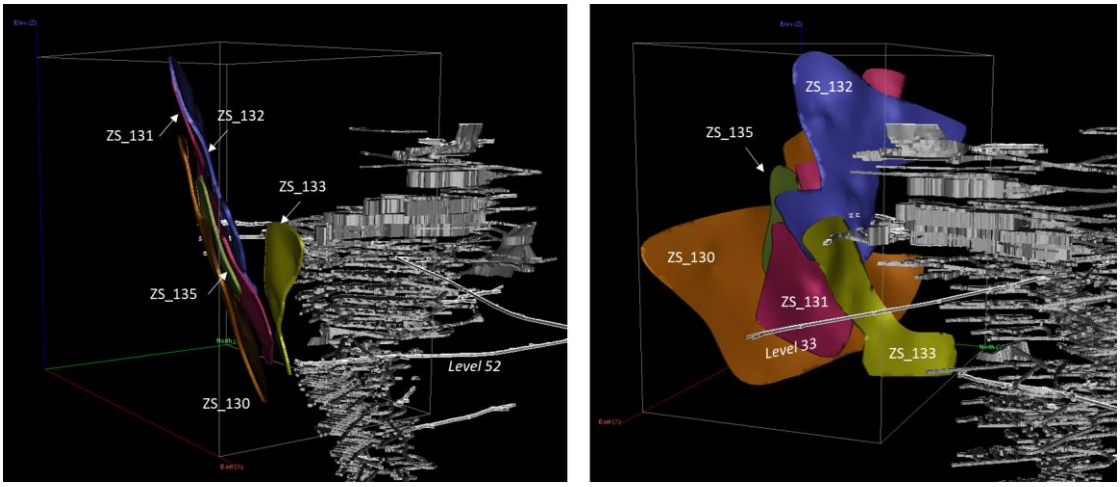


Figure 7-8: Zone South mineralized zones
(left: looking northwest; right: looking west-southwest)

7.3.1.9 Kiena Deep A Zone

The first hole that intersected the Kiena Deep A Zone was 4320 collared from the 91 Level in December 2007. This hole cut a quartz vein containing about 10 specks of visible gold within a talc-chlorite schist. A value of 28.23 g/t Au over 1 m was obtained. In January 2010, two more holes (4928 and 4929) were drilled in the Kiena Deep A Zone from the 91 Level. Hole 4928 returned an average grade of 12.31 g/t Au (8.42 g/t Au cut) over 12.8 m. Hole 4929 cut an average grade of 25.42 g/t Au (14.73 g/t Au cut) over 10.5 m. In 2012, four more holes were drilled from the same level with the following best results: hole 5965B with 66.77 g/t Au (11.04 g/t Au cut) over 3 m, hole 5966 with 51.86 g/t Au (21.49 g/t Au cut) over 7.5 m, hole 5967 with 10.17 g/t Au (8.20 g/t Au cut) over 5.4 m, and hole 5974 with 42.49 g/t Au (8.80 Au g/t cut) over 14.9 m. The six subsequent holes also intersected the Kiena Deep A Zone and cut gold-bearing quartz veins and veinlets hosted by a talc-chlorite schist, like in hole 4320. In June 2016, Wesdome tried again to explore the area of the Kiena Deep A Zone.

The Kiena Deep A Zone is localized within the Marbenite Fault deformation corridor. Contrary to previous beliefs, the Kiena Deep Zone is not the extension of the S50 Zone. The structures, mineralization type, host rocks and grades are completely different. The MF plane at that depth (1,350 m below surface) strikes NNW and dips shallower (55°) to the east compared to the WNW direction near the surface and the dip of 70°. The geology and structures were interpreted in 3D by Ravenelle (2018) based on drillhole data and reviewed by Beausoleil et al. (2019), Richard and Torrealba (2019) with the collaboration of Wesdome's geologists.

The Kiena Deep A Zone is divided into three main lenses: ZA, ZA1 and ZA2. A fourth smaller lens (H1ZA) is located in the hanging wall of the ZA at the contact between the basalt and the flow breccia units (Figure 7-9). The lenses occur along an isoclinal fold associated with the MF and a subsidiary fault. All lenses in the Kiena Deep A Zone are variably altered to chlorite, albite, carbonate and amphibole. Pyrite content is less than 2% with traces of pyrrhotite and chalcopyrite.

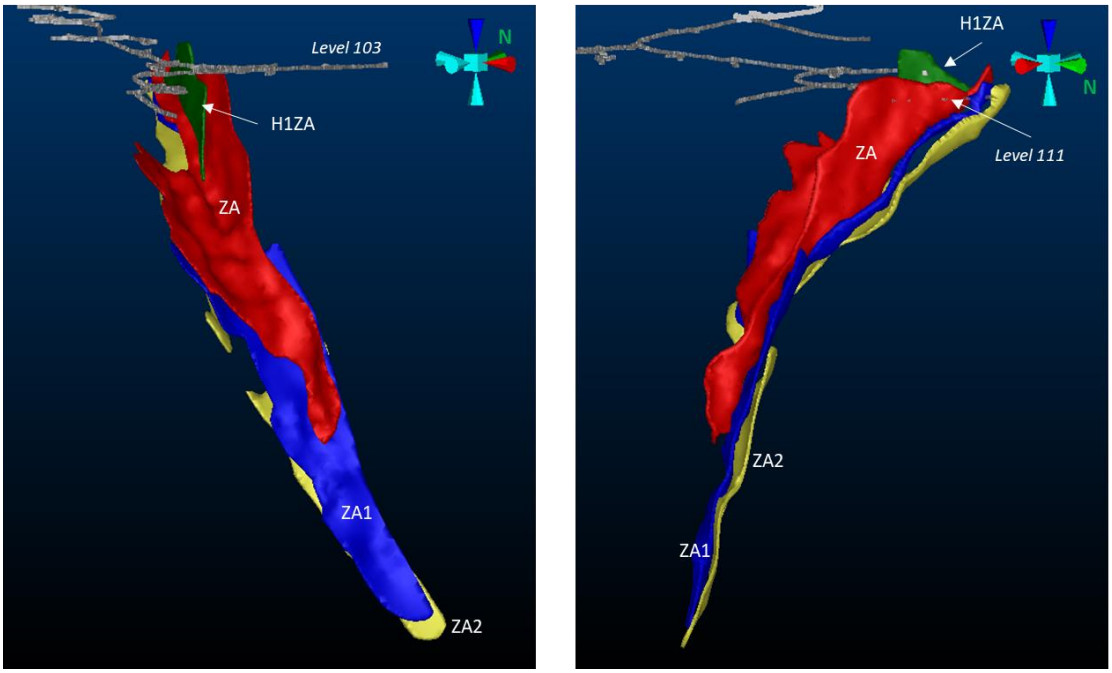


Figure 7-9: Kiena Deep A Zones folded within the schist
(left: looking northwest; right: looking south-southwest)

According to Ravenelle (2018), two styles of gold mineralization occur in the zone; the first style largely dominates Kiena Deep and the second style is seen occasionally:

1. Laminated veins (shear veins) hosted within sheared ultramafic rocks (grade is up to several ounces of gold per ton). Folding is observed in shear zones. Veins are composed of vitreous to milky to grey quartz with carbonates. There is more than one gold bearing vein generation. Sulphides range from traces to 1% locally (pyrite, pyrrhotite, chalcopyrite, sphalerite and galena).
2. Stockworks of quartz-carbonate veins and veinlets hosted in basalt (like the S50 style of gold mineralization) typically yielding less than 10 g/t Au.

The 2020 underground development in the ZA at the 111 Level, allowed to observe the geological complexity associated to ZA gold mineralization. A metric deformed corridor containing gold mineralization characterized the ZA. Intense folding of quartz veins is observed within a sheared corridor. An important amphibolization seen to be closely related to the folded laminated veins (Figure 7-10). It is probable that the protolith of the amphibolitized rock is an ultamafic rock. Mineralization also seems to have taken place during multiple veining events. Although visible gold is frequently seen in deformed quartz veins, it has also been possible to observe visible gold several times with the amphiboles corresponding to the amphibolitized corridor. Despite the geological complexity and the intense deformation observed, the ZA corridor shows good geological continuity over several tens of metres.

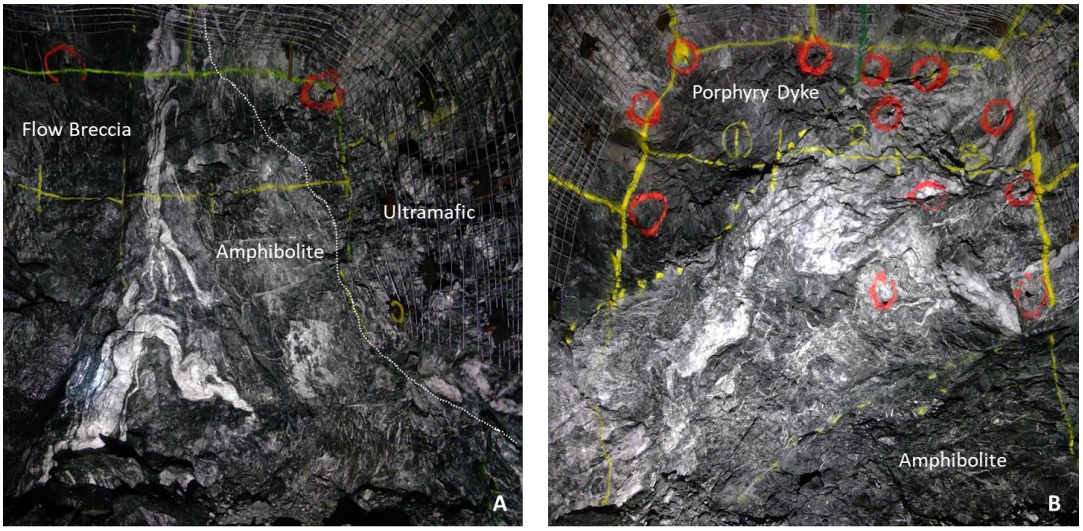


Figure 7-10: ZA Development at the 111 Level; A) NE Subvertical area; B) Fold hinge area

The main lenses ZA, ZA1 and ZA2 are coplanar to the MF to the SW, conforms to the fold hinge in the middle part and is subvertical to the NE. The axial plan of the isoclinal fold undulates and plunges abruptly to the ESE (Figure 7-11). Lens ZA is located at the contact of the basalt domain and the chlorite-carbonite schist. We generally observe a feldspar porphyry dyke in the hanging wall of the ZA, which is a guide for the interpretation. This dyke seems to have undergone the same deformation as the host schist. The ZA has a horizontal footprint of 150 m and a vertical extent of 460 m with thicknesses ranging from 3 m to 18 m.

ZA1 and ZA2 lenses are subparallel to the ZA and are entirely in the chlorite-carbonate schist of the fault along with minor intermediate dykes. ZA1 shows a vertical extent of 550 m and a horizontal footprint of 120 m with thicknesses ranging from 3 m to 8 m. ZA2 is less drilled than ZA and ZA1 and is actually separated in two domains by a low-grade area. The lower part of the ZA2 has a much higher grade than the upper part and is also thicker with thicknesses of up to 15 m in the lower part. The overall vertical extent of the ZA2 is 590 m. The lenses ZA1 and ZA2 still have to be defined on the subvertical contact. Definition drilling is ongoing in this area.

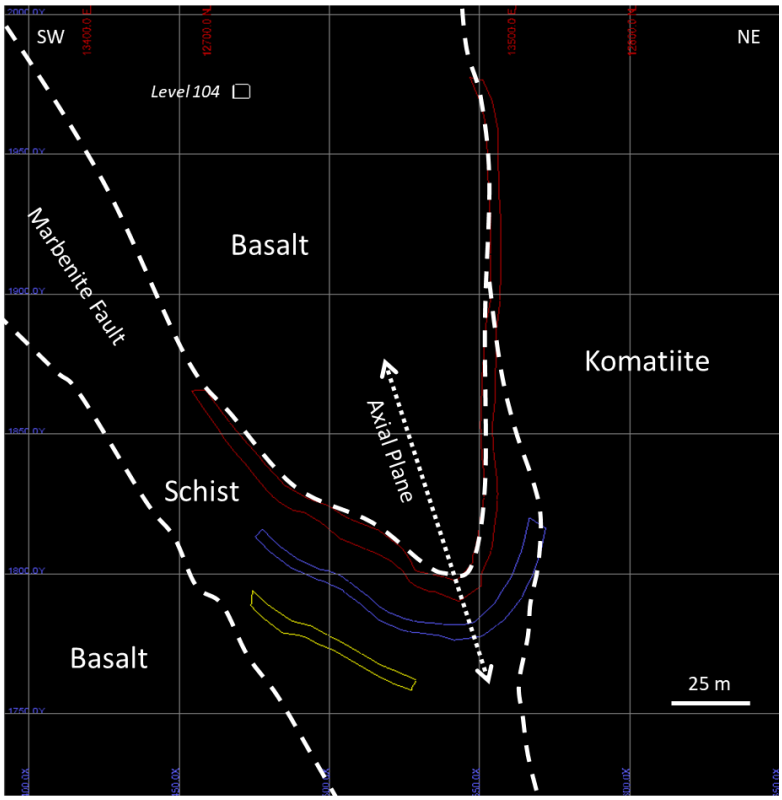


Figure 7-11: Typical cross-section of the Kiena Deep A Zones (1615NE)

7.3.1.10 Kiena Deep B Zone

The Kiena Deep B Zone consists of the ZB_140 lens (Figure 7-12). It is the only mineralization at Kiena hosted in the footwall of the MF. The shape, elongation and plunge of the zone are similar to that of S50 Zones at depth in the MF. The lens is stacked 25 m below the S50_100 lens in basalt. Alteration consists of chlorite, amphibole and carbonate, with local biotite and silica. The pyrite content is less than 2%. High-grade intervals are associated with quartz-carbonate-albite-tourmaline extension veins and veinlets.

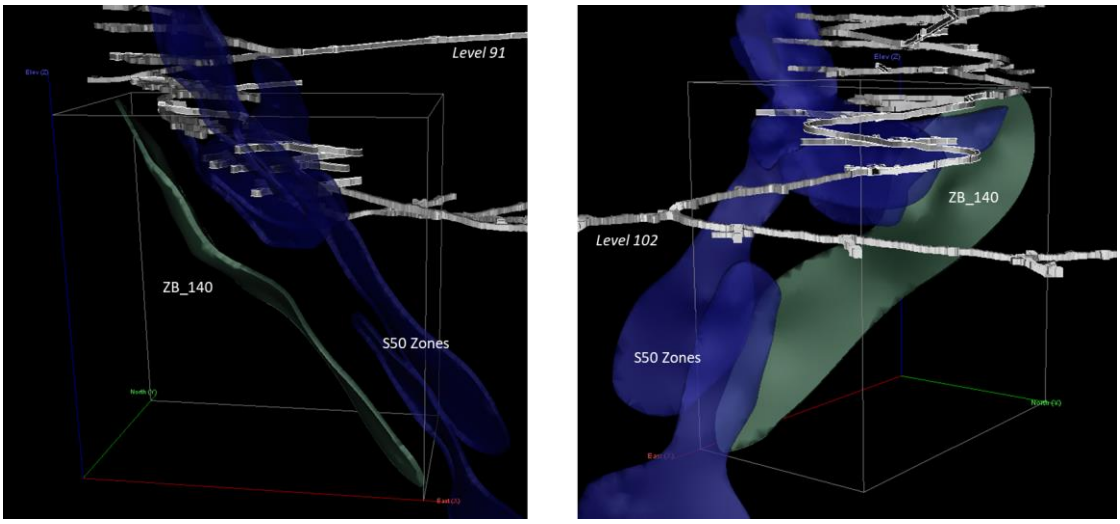


Figure 7-12: Kiena Deep B Zone
(left: looking north; right: looking southwest)

7.3.2 Siscoe Mine

The geology of the Siscoe mine is predominantly felsic intrusive rock of the Siscoe Stock and tholeiitic basalts with minor intercalations of ultramafic lavas belonging to the DF. The contact between the Siscoe Stock and the DF is marked by the K Shear Zone. Most of the producing veins are located in the stock (85-90% of total production), although minor amounts of gold were also extracted from the shear zone and from small veins hosted in volcanic rocks of the DF. Type 1, 2 and 3 veins were documented in the most productive gold vein systems of the mine. Type 1 veins correspond to the Siscoe Main Zone that produced almost half of the total production from the mine, and to the Siscoe, Hope, F and G veins. Type 2 veins correspond to the dismembered quartz veins of the K Zone. Type 3 veins correspond to the Siscoe C Vein, the first producer at the mine that yielded some 40,000 oz of gold, and to the Siscoe No. 27 Vein.

7.3.3 Shawkey Mine

The rocks that underlie the Shawkey mine area belong to the Jacola Formation. The characteristic lithologies found on the Property consist of ultramafic to mafic flows at the base of the sequence, followed by massive and minor pillowed basalts, basaltic flow and pillow breccia, tuff breccia and basaltic tuffs. The Shawkey No. 1 Vein is located about 250 m northeast of the NF in a secondary shear called the Martin Shear Zone. This shear zone is filled by a quartz vein (the Main Vein) accompanied by quartz stringers (Type 2 veins) in the walls. The Main Vein is hosted by altered basalts cut by dioritic dykes.

7.3.4 Other Mineralized Zones on the Property with Current Mineral Resources

In 2019-2020 a total of 30 zones were reviewed and modelled in 3D in order to convert previous polygonal resources to block model resources (Athurion et al., 2021). The following descriptions of the mineralized zones are modified from Turcotte et al. (2015), and Beausoleil et al. (2019), in accordance with the updated model.

7.3.4.1 Presqu'ile Zones

Discovered in 2012, the Presqu'ile Zone is a gold occurrence located 1.4 km northwest of the Kiena Mine (see Figure 7-3). It lies approximately 800 m southwest of the Northwest Zone, and 100 m northeast of the Marbenite Fault, in the Jacola Formation. Mineralization in the Presqu'ile area consists of a series of steeply dipping, subparallel, smokey quartz-carbonate-chlorite veins mineralized with pyrite, chalcopyrite, minor sphalerite, galena and visible native gold. The veins are hosted in mafic volcanic tuffs and flows, along a sheared contact with komatiite flows. Mineralization is associated with a moderate carbonation and chloritization, Ducharme (2013).

The Presqu'ile Zones comprise five subparallel mineralized lenses. This gold mineralized system spans approximately 300 m in length and continues to at least 300 m below surface. The mineralized lenses are generally oriented at N300° with a dip of 55°–60° northeast.

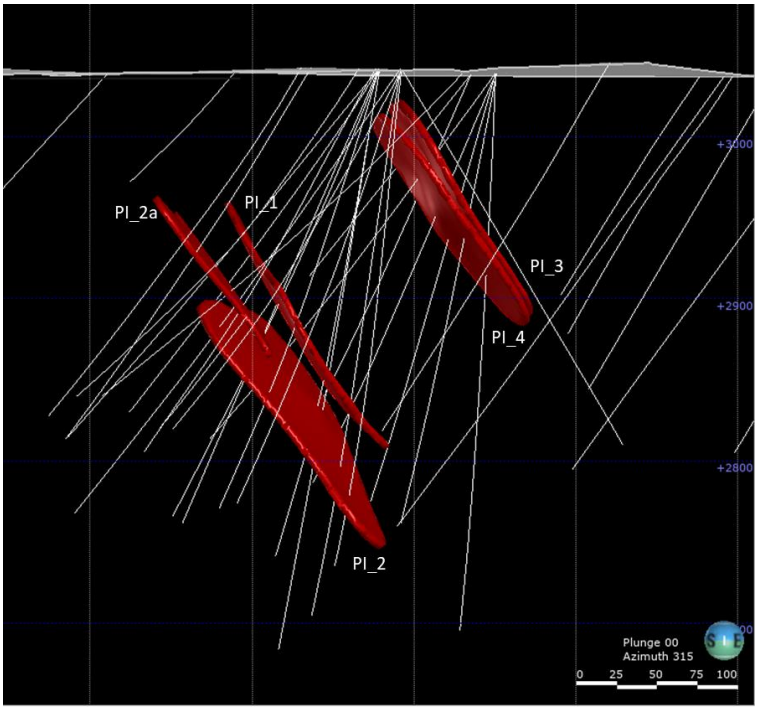


Figure 7-13: Presqu'ile Zones

7.3.4.2 Northwest Zone

The following description of the Northwest Zone is mostly modified and summarized from Robinson (1962), Lebel (1989) and Donovan (1994).

Diamond drillhole No. 2 of Martin Gold Mines Ltd. intersected a wide zone of quartz stringer - schist to the northwest of Parker Island. The Northwest Zone (see Figure 7-3 and Figure 7-14) lies 750 m northwest of the North Zone, and is located about 100 m north of the Norbenite Fault.

The local geology is represented by moderately to strongly altered mafic to ultramafic volcanic flows intruded by diorite to granodiorite dykes. It appears that the most significant gold mineralizing event is directly associated with a highly altered (silicified, pyritized, carbonated and albitized) diorite dyke. The thickness of the gold-bearing diorite ranges from 7 m to 70 m. The diorite dyke is enclosed in weakly sheared chlorite-talc schist. The mineralized zone is associated with 5% quartz-carbonate veinlets, accompanied by pyrite.

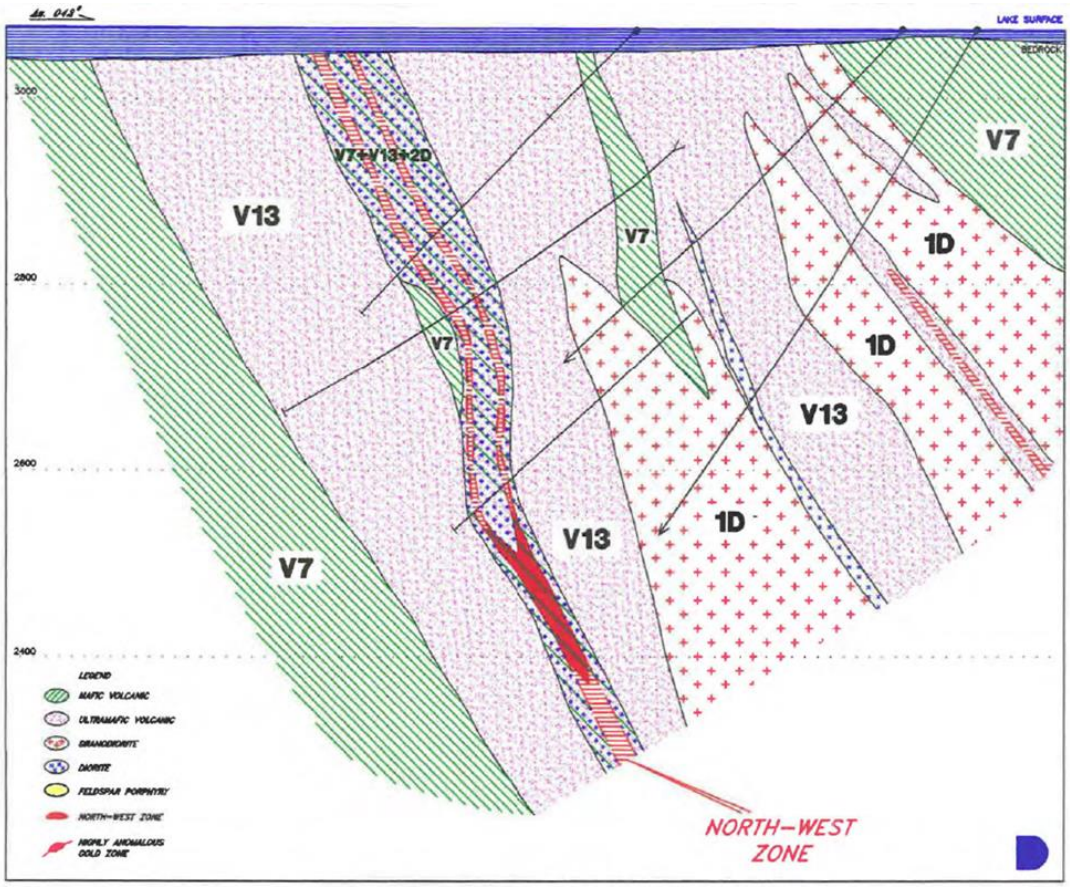


Figure 7-14: Simplified geology cross section of the Northwest Zone (from Donovan, 1994)

This Northwest Zone was modelled in 3D and strikes N330° and dips 50°–70° ENE (Figure 7-15). The mineralized zone has a lateral extent of 350 m and a vertical extent of 250 m.

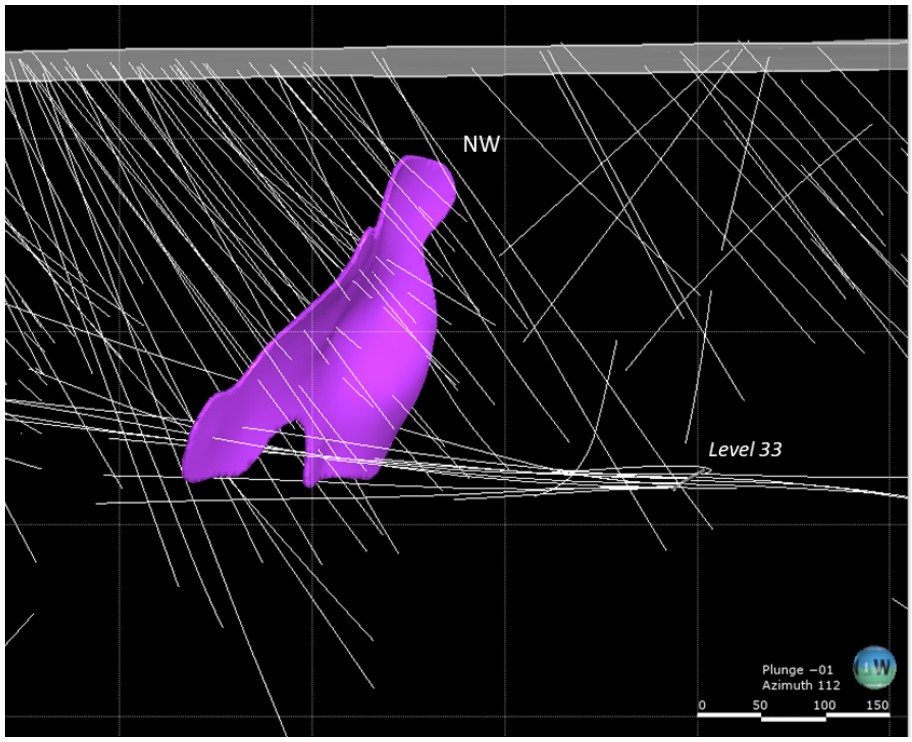


Figure 7-15: Northwest Zone

7.3.4.3 Dubuisson Zones

The section of the Dubuisson Formation encountered to date consists of a sequence of massive or foliated to schistose komatiites with intercalations of talc-chlorite schist, basalt, albitized diorite dykes, and feldspar porphyry dykes.

Dubuisson Zone

Discovered in 2008, the Dubuisson Zone is located approximately 3,500 m east of the Kiena shaft and its central infrastructure, and about 1,000 m east of the end of development heading 3314 on the 33rd level. It lies 1.7 km northeast of the Norbenite Fault.

The Dubuisson Zone is known to contain at least three lenses of gold mineralization, oriented E-W. Gold is hosted by brittle, fractured, albitized, carbonated and chloritized diorite and chloritized feldspar porphyry dykes within a broad section of the deformed ultramafic sequence. Sheared talc-chlorite schist is located near or adjacent to the dykes hosting the gold mineralization.

Mineralization consists of a stockwork of quartz-albite±tourmaline-pyrite veinlets commonly carrying free gold within diorite dykes, and a stockwork of quartz-tourmaline-chlorite-pyrite veinlets commonly carrying free gold within feldspar porphyry dykes. In some cases, the mineralization is also in the talc-chlorite schist adjacent to the feldspar porphyry or diorite dykes. The host rocks display alteration containing disseminated pyrite.

Dubuisson North Zone

The Dubuisson North Zone was discovered in 2012 by underground drilling from drift 33-14 (level 330 m) about 200 m to the south of the Dubuisson Zone. The Dubuisson North Zone lies 1.9 km northeast of the Norbenite Fault. At least two steeply-dipping E-W lenses have been partially delineated and defined above a depth of 300 m.

The geological setting and mineralization of the zone is very similar to the Dubuisson Zone. Mineralization consists of stockwork quartz-carbonate±albite±chlorite±tourmaline veining carrying pyrite and native gold hosted by feldspar porphyry and/or albitized diorite dykes encased in komatiites and talc-chlorite schist. Gold mineralization is often close to a shear zone (talc-chlorite schist). In some cases, mineralization is also present in the talc-chlorite schist adjacent to the feldspar porphyry or diorite dykes.

One lens of the Dubuisson Zone was modelled in 3D (the most continuous) and two lenses were modelled in the Dubuisson North Zone, up to date (Figure 7-16). The Dubuisson lens is dipping 85° north and has lateral and vertical extents of 200 m and 275 m, respectively. The Dubuisson N1 and N2 lenses are dipping 80° SW and 65° NE, respectively and have similar extents as the Dubuisson lens.

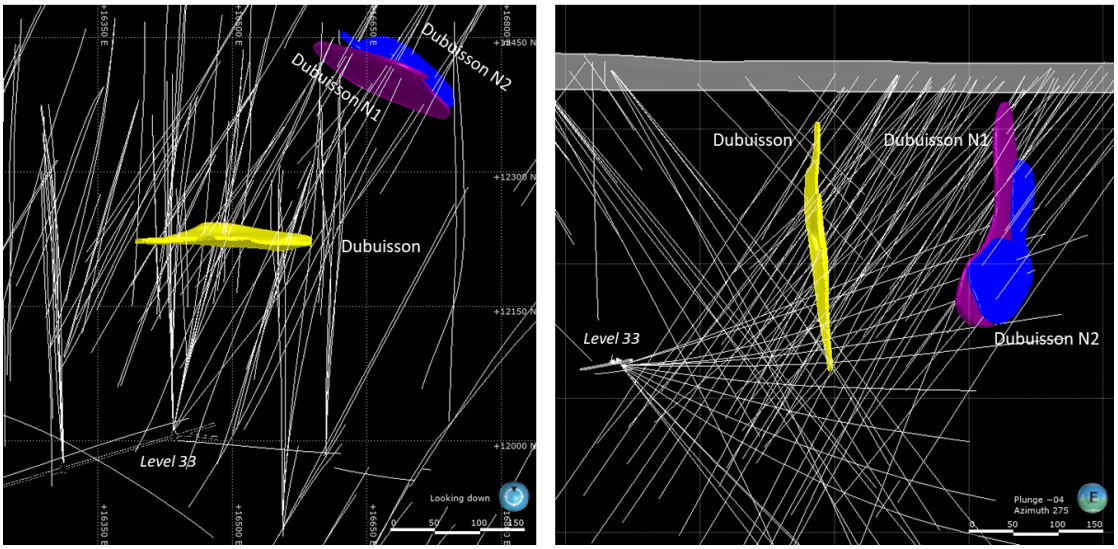


Figure 7-16: Dubuisson and Dubuisson North Zones

7.3.4.4 Martin Zone

The following description of the Martin Zone is mostly modified and summarized from Lebel (1991), Lebel (1992) and Donovan (1994).

The Martin Zone lies 300 m northeast of the Norbenite Fault and is approximately 900 m to the east of the S50 Zone (Figure 7-3). The zone strikes approximately N340°. This zone is hosted by silicified, brecciated, pyritized and albitized basalts. The basalt is cut by numerous quartz veins and granodiorite dykes. The zone is observed close to a major faulted/sheared komatiite (talc-chlorite schist) and brecciated basalts related to the Martin Shear Zone (Figure 7-17). This shear strikes northwest and dips steeply to the southwest. Gold mineralization is present as quartz-carbonate±albite veins with pyrite-chalcopyrite±native gold.

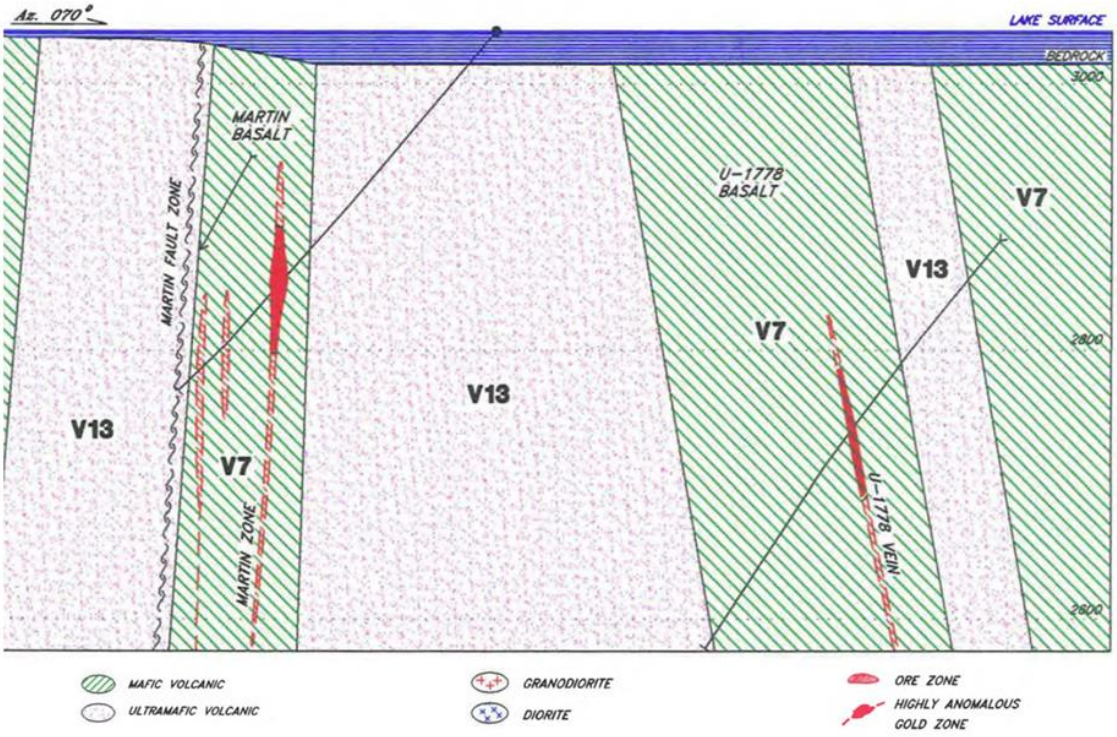


Figure 7-17: Simplified geology cross section of the Martin and U-1778 Zones (from Donovan, 1994)

In the past years, some exploration development was conducted in the Martin Zone. Between 2007 and 2013, one drift and one sublevel were subsequently developed in the zone at the 33 Level. A total of 12,538 t of material was extracted from the zone at an average grade of 2.76 g/t Au. The Martin Zone was modelled in 3D for the purpose of the herein Mineral Resources update. The mineralized lens has a vertical extent of 300 m and a lateral extent of 270 m. Figure 7-18 presents de 3D model of the zone as well as the exploration development completed in the past years.

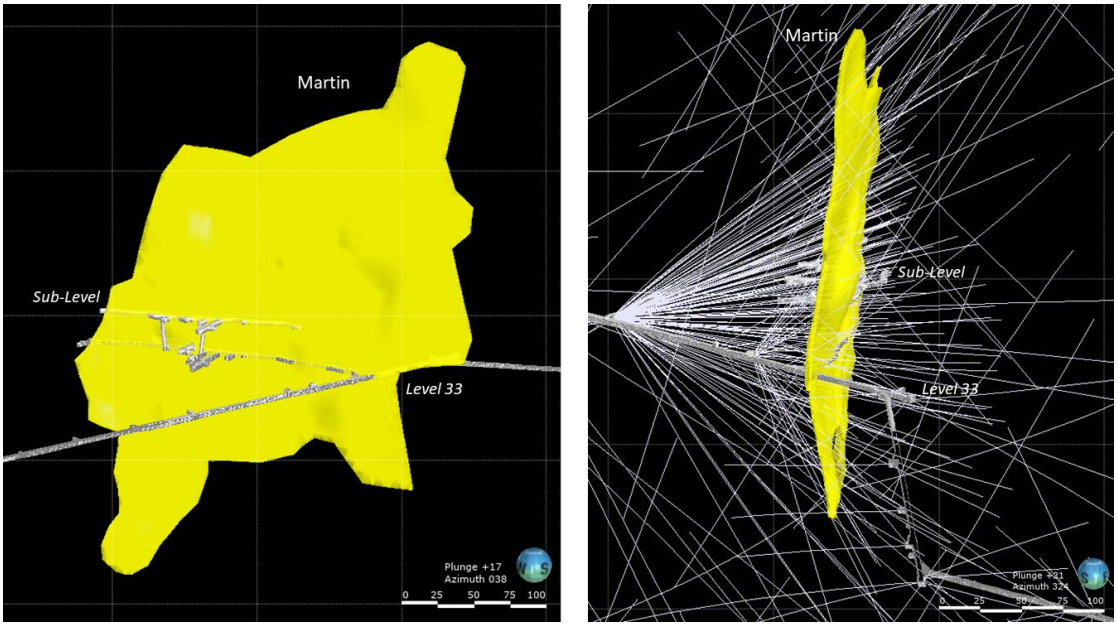


Figure 7-18: Martin Zone

7.3.4.5 Wesdome Deposit

The following description of the Dorval-Siscoe Shaft and Wesdome Deposit area is mostly modified and summarized from Ingham (1947), Sauvé (1987), Gardiner (1987; 1988), Sauvé et al. (1993), and Turcotte and Pelletier (2009), and references therein.

The area lies in the northwest extension of the K Zone of the Siscoe mine. Virtually all of the property geology is known from diamond drillholes. The geologic units that crop out on the Wesdome property are as follows: 1) dominant tholeiitic basalts and andesites with intercalations of ultramafic lavas belonging to the Dubuisson Formation; 2) intrusive rocks, referred to as the Snowshoe Batholith; and 3) dyke swarms commonly observed throughout the property, but particularly abundant in basaltic rocks in the area east of the Snowshoe Batholith.

Volcanic rocks are split into two major groups: ultramafics and basaltic rocks. The ultramafic rocks are dark green to black in colour and very soft, often talcose, in nature. These rocks are heavily altered and rich in serpentinite and magnetite. Local concentrations of biotite, actinolite and quartz are commonly observed in thin sections. The ultramafic flows are readily identifiable on magnetic maps, and typically generate topographic depressions filled with overburden material. A wide variety of basaltic rocks occur throughout the property, including pillowed and massive flows (tabular types), and minor porphyritic and variolitic flows. The basaltic rocks range in colour from dark green to jet black and are typically fine grained. These rocks retain little of their original texture, having been recrystallized. Textures range from massive to locally schistose.

All of the volcanic flows strike WNW and dip vertically to steeply northwards. Tops indicators in pillowed flows indicate the rocks are younger to the south. The rocks can be divided into two major domains: those to the south and southeast of the Snowshoe Batholith, and those to the north and northeast of the plug. The former domain is characterized by flows that curve southwards along the margin of the plug, and strike at N120° in the southeast portion of the area. The latter domain is characterized by the mirror image, or flows that curve northwards along the northwest periphery of the plug to emerge on the northwest edge trending N300°. This pattern is particularly evident on gradiometer aeromagnetic maps of the region, which clearly show a large sausage-like area of low magnetic contrast enclosed by an envelope of highly magnetic ultramafic rocks. The Snowshoe Batholith forms the western limits of the structure, while Siscoe Island (the Siscoe mine) forms the eastern end of the “sausage”.

The K Shear Zone of the Siscoe mine cross-cuts the northeast corner of the area. Several drillholes intersected the zone but failed to return significant gold values. Ingham (1947) reported significant gold values in actinolite schist, interpreted as an extension of the K Shear Zone in this sector.

The Snowshoe Batholith is a circular intrusive mass in the centre of the property that slightly exceeds 1 km in diameter. The limits of the stock are fairly well defined from drilling and a magnetic survey but are poorly understood in the northwest corner of the Property. Rocks of the Snowshoe Batholith are light grey to white in colour and equigranular. The composition is variable, locally ranging from granodiorite to quartz-diorite and monzodiorite.

Dyke swarms are common throughout the Project but are particularly abundant in basaltic rocks in the area east of the stock. The absence of dykes in the plug suggests that the dykes predate plug emplacement. Locally, dyke rocks may account for up to 50% of the volcanics. The dykes range in thickness from 1 m to 200 m, with an average of less than 10 m. The orientation of these dykes is poorly understood, but exposures on several islands indicate that most of them strike E-W and dip subvertically. Many of the shear zones on the Project host dyke-like material. Typically, the dykes are mottled grey-green and often display a porphyritic texture. The dykes are easily distinguished from the main intrusive plug by an increase in mafic minerals.

The most widespread mineralization is found in volcanic rocks intruded by subvolcanic porphyritic quartz diorite dykes to the northeast and east of the Snowshoe Batholith. But, gold intersections in diamond drillholes have also been observed within the Snowshoe Batholith. This may reflect a lack of drilling in this area. Up to now, the best mineralization on the Wesdome deposit occurs in basaltic rocks that have been heavily intruded by porphyritic quartz diorite dykes. The structures hosting mineralization (quartz veins and shear zones) generally strike at N120° and dip 45°–75° southwest. Locally, the dip can be less than 45°. The veins cut the basaltic rocks and felsic dykes.

Mineralization can be split into three distinct types: 1) pyritiferous quartz veins; 2) albite alteration zones; and 3) pyritiferous shear zones. A large number of zones are found on the Wesdome deposit. The zones are named using letters; the most northerly is the “A” Zone, and the others were assigned sequential letters moving southward.

Wesdome A Zone

The Wesdome A Zone is located 300 m southwest of the K Shear Zone (see Figure 7-3 and Figure 7-19). It strikes N120° and dips 50° south. The zone has been traced for 900 m in strike length and extends more than 500 m in known depth. Typically, the zone is between 1–2 m thick and consists of quartz ribbons 15–80 cm thick, enclosed in heavily silicified and carbonatized basalt and/or dioritic dyke. Minerals commonly include 1% to 5% pyrite, minor tourmaline and rare breccia zones. The mineralized material shoots are aligned downdip. These areas are typified by an increase in the amount of pyrite and a corresponding increase in intensity of alteration (albitization) in wall rocks. An old drift 600 m southwest of the Dorval-Siscoe shaft explored the zone for 200 m. Drift mapping has demonstrated the ribbon-like nature of the veins, and the lenticular nature of the gold mineralization. The Wesdome A Zone therefore consists of a number of en echelon vein systems enclosed in a weak shear zone that dips south at 50°. The Wesdome A Zone is split into two distinct zones in one sector.

A total of seven mineralized lenses were modelled in 3D in the Wesdome A Zone (Figure 7-20) as a result of the conversion of Polygonal resources. These lenses are named, Wesdome A, Wesdome AF, Wesdome A1, Wesdome AH and Wesdome AH1, Wesdome AH2 and Wesdome AH3.

Wesdome B Zone

The Wesdome B Zone is located 300 m southwest of the K Shear Zone (Figure 7-3 and Figure 7-19). The Wesdome B Zone consists of a strong shear zone oriented N110° and dipping subvertically (70°–85°) to the south and is approximately 10 m thick. Mineralization is discontinuous, and consists of numerous quartz-tourmaline-pyrite veins oriented parallel to the contacts of the shear zone. Continuation eastwards from the Wesdome B Zone coincides with a shear zone explored by underground drifting roughly 200 m south of the Dorval-Siscoe shaft. This zone was modelled in 3D (Figure 7-20).

According to Gardiner (1988), this shear zone could be the equivalent of the Siscoe K Shear Zone as it continues onto the Wesdome deposit area; however, Wesdome believes this premise must be tested with more drilling, especially since it is not supported by the recent interpretation by Pilote et al. (2015c).

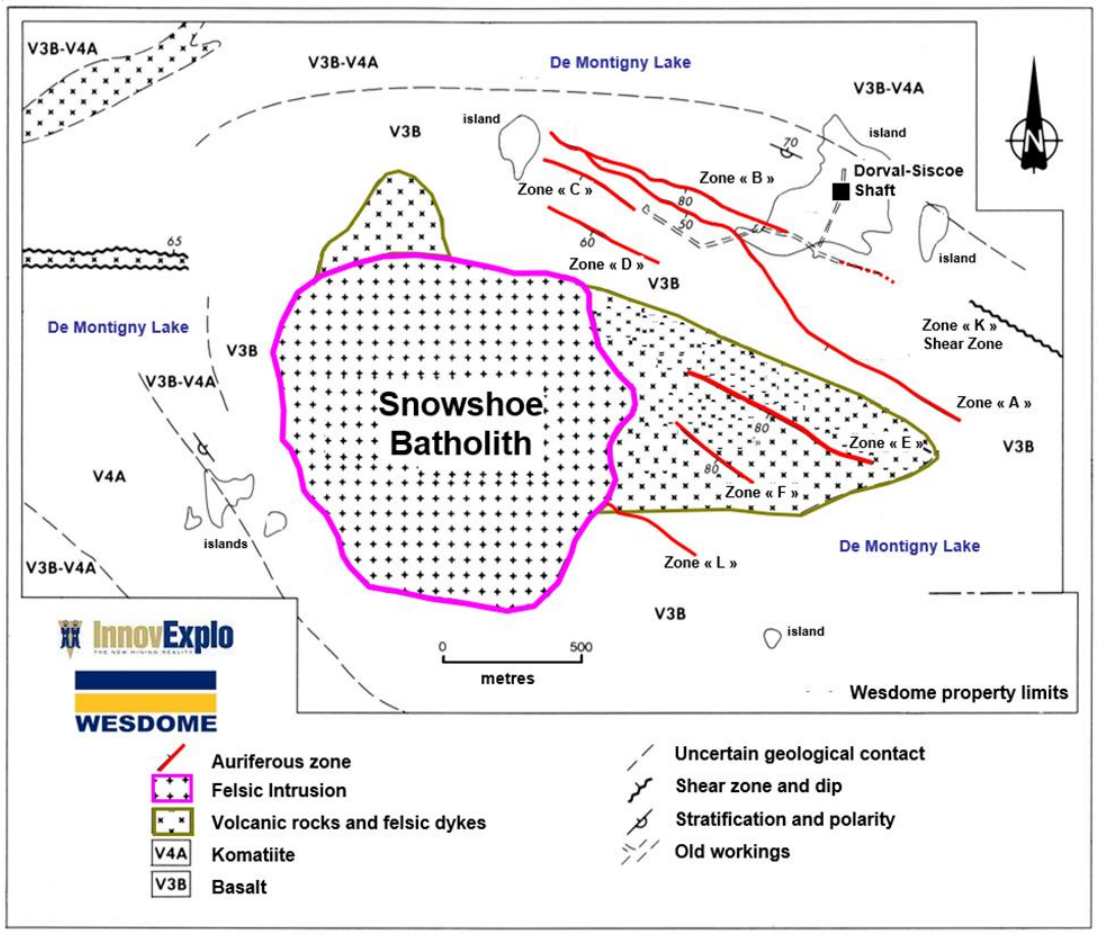


Figure 7-19: Distribution of main mineralized zones on the Wesdome property (Adapted and modified from Gardiner, 1988; Sauvé at al., 1993; Turcotte and Pelletier, 2009)

Wesdome C and D Zones

The Wesdome C and D Zones are located 400 m southwest of the K Shear Zone (Figure 7-3 and Figure 7-19). They consist of weak shear zones striking N120° and dipping about 60°–70° south. The zones are hosted by basalt. Mineralization consists of quartz-carbonate veins and veinlets, accompanied by pyrite and other minor sulphide minerals. These zones were weakly sheared and/or schistose.

These two zones were modelled in 3D, see Figure 7-20.

Wesdome E Zone

The Wesdome E Zone is located 800 m southwest of the K Shear Zone (see Figure 7-3 and Figure 7-19). This zone is a large quartz-carbonate vein ranging in thickness from 0.7 m to 8.0 m, confined within a large shear zone striking N120° and dipping about 50°–80° south. The E Zone contains abundant wall rock fragments (up to 15%).

A total of five mineralized lenses were modelled in 3D in the Wesdome E Zone (Figure 7-20) as a result of the conversion of Polygonal resources. These lenses are named, Wesdome E, Wesdome E0, Wesdome E1, Wesdome E3 and Wesdome E4.

Wesdome F Zone

The Wesdome F Zone is located 1 km southwest of the K Shear Zone (see Figure 7-3 and Figure 7-19). The Wesdome F Zone is a complex series of small pyritiferous quartz-carbonate veins and alteration zones. The zone is confined within a weak shear zone striking N120° and dipping approximately 55°–65° south. The thickness is typically 2–3 m. The zone is complexly splayed.

A total of five mineralized lenses were modelled in 3D in the Wesdome F Zone (Figure 7-20) as a result of the conversion of Polygonal resources. These lenses are named, Wesdome F, Wesdome F1, Wesdome F2, Wesdome F4 and Wesdome F6.

Wesdome L Zone

The Wesdome L Zone is located 1.3 km southwest of the K Shear Zone (see Figure 7-3 and Figure 7-19). The Wesdome L Zone constitutes an albitized quartzitic diorite injected by a mineral assemblage of quartz±carbonate±tourmaline±albite accompanied by pyrite± chalcopyrite±galena±native gold. A weak shear zone is associated with the zone. This zone was modelled in 3D, see Figure 7-20.

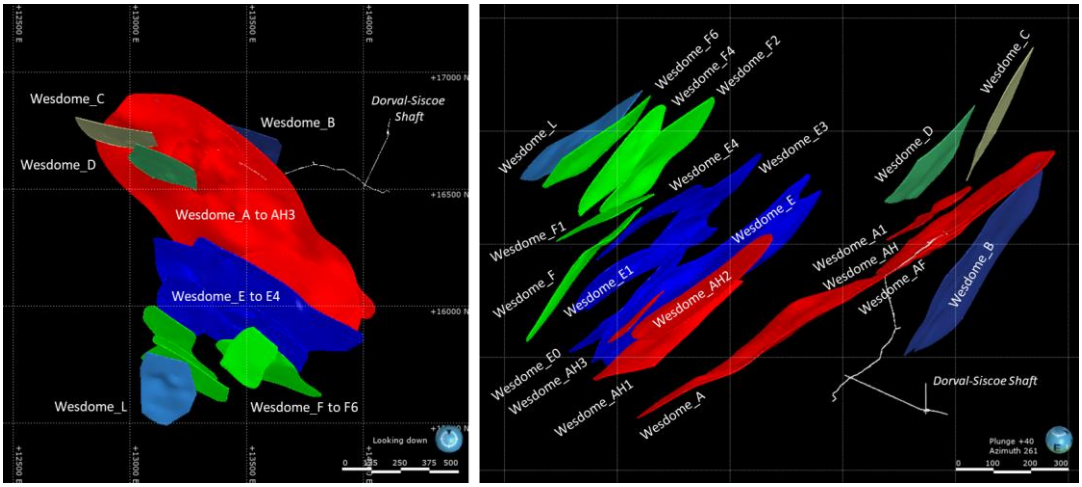


Figure 7-20: Wesdome Deposit 3D Model

8. DEPOSIT TYPES

Gold mineralization on the Kiena Mine Complex shares many geological attributes with other vein-type gold deposits of the Val-d'Or district and with orogenic gold deposits (also known as lode gold, greenstone-hosted quartz-carbonate vein, or mesothermal deposits) in terms of host rock composition, mineralogy and hydrothermal alteration.

The degree of fracturing was the primary control on the formation of mineralized zones on the Property. Thus, gold mineralization is mainly hosted in fractured competent units that acted as fluid conduits and precipitation sinks, both during and after deformation. Alteration, notably albitization, likely played a key role in host unit competency. The competency contrast between intrusive bodies, or basalts, and the talc-chlorite schists may be responsible for strain localization at the rheological boundary, and it induced a secondary permeability that provided greater access to hydrothermal gold-bearing fluids during episodic shear zone movements.

The presence in the same area of more than three types of gold-bearing veins exhibiting a wide range of orientations, mineralogy and crosscutting relationships, and the fact that several generations of dykes and veins are involved, suggests that gold mineralization was the product of multiple mineralizing phases (Beausoleil et al., 2019).

8.1 Archean Greenstone-Hosted Orogenic Gold Deposits

The Kiena Mine Complex mineralization presents characteristics of typical Archean greenstone-hosted orogenic gold deposits. The following description is taken from Simard et al. (2013) unless specified otherwise.

Greenstone-hosted quartz carbonate vein deposits occur in deformed greenstone belts of all ages elsewhere in the world; especially those with variolitic tholeiitic basalts and ultramafic flows intruded by intermediate to felsic porphyry intrusions, and sometimes with swarms of albitite or lamprophyre dykes (Dubé and Gosselin, 2007).

Archean greenstone-hosted orogenic gold deposits are typically distributed along first-order compressional to transpressional crustal-scale fault zones (Figure 8-1), characterized by several strain increments (e.g., Cadillac–Larder Lake Fault Zone) that mark the convergent margins between major lithological boundaries, such as volcano-plutonic and sedimentary domains. Large-scale carbonate alteration is also commonly distributed along those major fault zones and associated subsidiary structures (Dubé and Gosselin, 2007). This gold deposit type is, however, seldom located within these first-order structures. Major, or first-order faults are interpreted as primary hydrothermal pathways to higher crustal levels (Eisenlohr et al., 1989; Colvine, 1989; McCuaig and Kerrich, 1998; Kerrich et al., 2000; Neumayr and Hagemann, 2002; Kolb et al., 2004; Dubé and Gosselin, 2007); however, only a few significant gold deposits are hosted in major faults such as the Ajjanahalli mine, Dharwar Craton, South India (Kolb et al., 2004), and the McWatters mine and the Orenada deposit, Abitibi Subprovince, Canada (Robert, 1989; Morin et al., 1993; Neumayr et al., 2000; 2007). Significant mineralized quartz veins are commonly hosted in second-

and third-order shear zones (Eisenlohr et al., 1989). Structurally, these shear zones vary from brittle–ductile to ductile, depending on their depth of formation (Hodgson 1993; Robert and Poulsen, 2001). They are formed at intermediate depths ranging from 5 km to 10 km (Dubé and Gosselin, 2007). At depths greater than 10 km, quartz veins are seldom located within shear zones whereas gold mineralization is mostly associated with disseminated sulfides (Witt and Vanderhor, 1998).

At the deposit scale, the nature, distribution and intensity of the wall-rock alteration is largely controlled by the composition and competence of the host rocks and their metamorphic grade. Typically, the alteration haloes are zoned and characterized at greenschist facies by iron-carbonatization and sericitization, with sulphidation of the immediate vein selvages (mainly pyrite, less commonly arsenopyrite) (Dubé and Gosselin, 2007).

The main gangue minerals are quartz and carbonate with variable amounts of white micas, chlorite, scheelite and tourmaline. The sulphide minerals typically constitute less than 10% of the mineralized material. The main minerals are native gold with pyrite, pyrrhotite and chalcopyrite without significant vertical zoning. The mineralization is syn- to late-deformation and typically post-peak greenschist-facies or syn-peak amphibolite-facies metamorphism (Dubé and Gosselin, 2007).

There is a general consensus that the greenstone-hosted quartz-carbonate vein deposits are related to metamorphic fluids from accretionary processes and generated by prograde metamorphism and thermal re-equilibration of subducted volcano-sedimentary terranes. The deep-seated gold transporting metamorphic fluid has been channelled to higher crustal levels through major crustal faults or deformation zones. Along its pathway, the fluid has dissolved various components, notably gold, from volcano-sedimentary packages, including a potential gold-rich precursor. The fluid is then precipitated as vein material or wall rock replacement in second and third order structures at higher crustal levels through fluid pressure cycling processes and temperature, pH and other physico-chemical variations (Dubé and Gosselin, 2007).

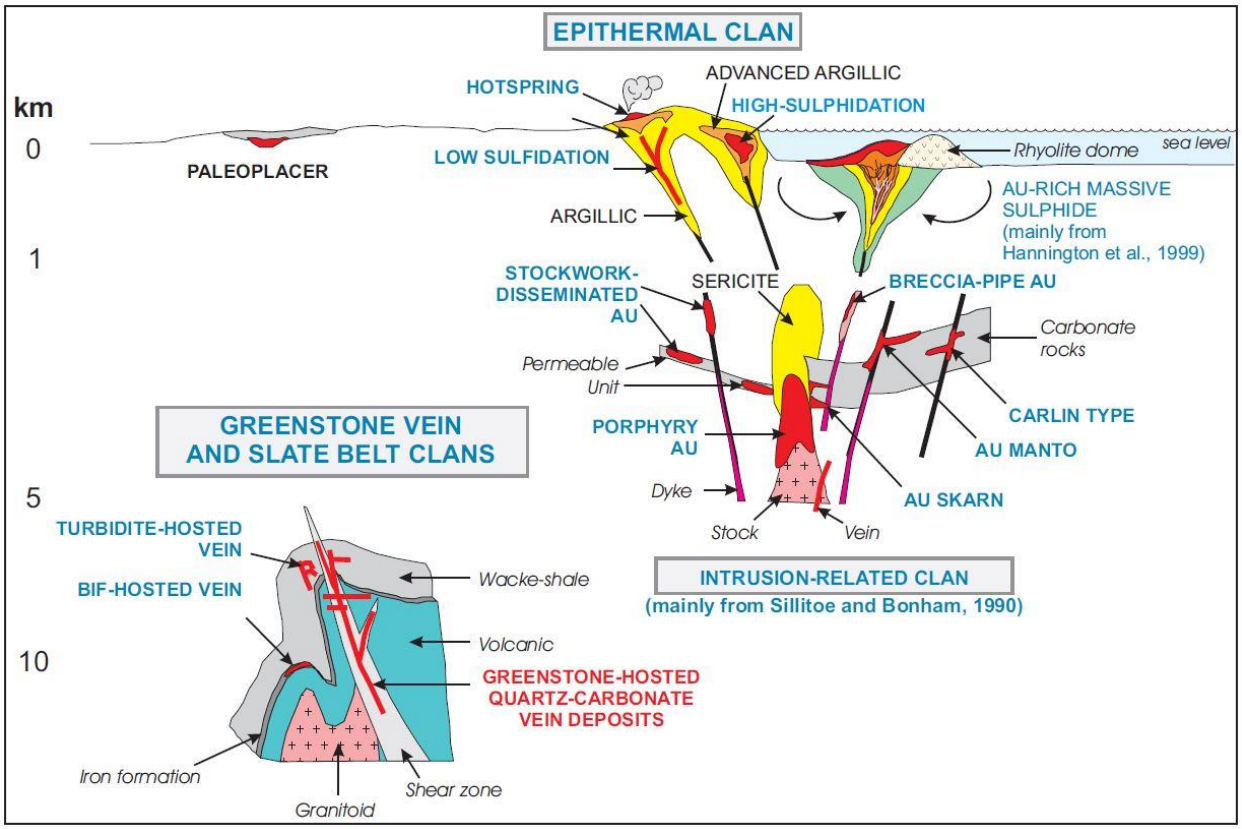


Figure 8-1: Inferred crustal levels of gold deposition showing the different types of gold deposits and the inferred deposit clan (from Dubé et al., 2001; modified from Poulsen et al., 2000)

8.2 Gold Mineralization in the Val-d'Or District

The following is taken from Couture et al., (1994), who published a detailed description and chronology of the Archean greenstone-hosted quartz carbonate vein of the Val-d'Or district.

Gold mineralization occurs in all rock types but is more commonly located within intrusive bodies that acted as competent rock units promoting fracture during deformation. In the Val-d'Or district, there are two main generations of gold quartz veins: young deposits in which the gold mineralization did not experience much deformation after its emplacement; and early mineralization in which mineralized bodies are commonly affected by D_2 asymmetric folds, are highly strained and locally dismembered. In a few deposits both generations are present.

At least two major gold mineralizing events have been recognized in the Val-d'Or district on the basis of morphological and structural features, mineralization and alteration mineral assemblages, and crosscutting relationships with dated intrusive rocks (Robert, 1990, 1994; Sauvé et al., 1993; Couture et al., 1994). The older mineralizing event is manifested by veins and breccias (e.g., Norlartic, Marban, Kiena mines, and Main mineralized zone at Siscoe mine) that are mainly associated with second-order shear zones and commonly folded or boudinaged by D1 deformation. These veins and breccias are cut by diorite and tonalite dykes, which have U-Pb zircon ages of 2692 ±2 (Pilote et al., 1993) and 2686 ±2 Ma (Morasse et al., 1995). The younger gold event, which produced the Sigma, Lamaque, Perron-Beaufor, Shawkey, Wesdome and Camflo deposits, as well as the C Vein (quartz-tourmaline) at the Siscoe mine, is represented by veins commonly associated with third-order shear zones. These veins clearly crosscut plutonic rocks intruded between 2694 ±2 Ma (Wong et al., 1991) and 2680 ±6 Ma (Jemielita et al., 1990), and they may have formed during the last stages of D1 deformation.

Young gold mineralization is characterized by networks of shear-hosted quartz-carbonate±tourmaline±scheelite veins and associated subhorizontal extension veins. This is well documented at the Sigma mine and also occurs in other deposits east of Val-d'Or, namely Lamaque, Perron, and L.C. Béliveau. Mineralized veins and associated structures crosscut all rock types except Proterozoic dykes. In the Sigma deposit (Robert and Brown, 1986) the gold-bearing quartz-tourmaline vein system is hosted by andesite of the Val-d'Or domain (2705 ±1 Ma, Wong et al., 1991; 2706 ±3 Ma, Machado et al., 1991), porphyritic diorite, and feldspar porphyries (2704 ±3 Ma and 2694 ±2 Ma, respectively, Wong et al., 1991) metamorphosed to greenschist facies. The porphyritic diorite is deformed, but feldspar porphyry dykes cut D_2 folds and thus postdate regional D_2 folding (Robert and Brown, 1986). The vein network consists of coeval and cogenetic steeply dipping shear-hosted veins and sub-horizontal extensional veins (Robert and Brown, 1986; Figure 8-2). Preserved delicate vein-filling textures and crosscutting relationship indicate that gold mineralization postdates the youngest intrusion as well as metamorphism and much of the deformation (Robert and Brown, 1986). Rutile associated with the mineralization has been dated by U-Pb at 2599 ±9 Ma (Wong et al., 1991). Similar vein geometry and morphology were also described in the Lamaque mine (Daigneault, 1983), where most of the mineralization is hosted by small circular tonalite plugs crosscutting porphyry intrusion similar to that of Sigma. Jemielita et al. (1990) reported U -Pb ages of 2685 ±3 and 2682 ±2 Ma for the Lamaque Main tonalite plug and 2593 ±5 Ma for rutile associated with gold mineralization. Similar age relationships can be inferred from structural studies at the Perron and Béliveau mines (Tessier, 1990; Gaumond, 1986, respectively). West of Val-d'Or, significant gold was extracted from the post-D_2 Camflo quartz monzonite dated at 2680 ±6 Ma (Jemielita et al., 1990) and 2685 ±10 Ma (Zweng et al., 1993), whereas titanite and rutile, associated with the gold mineralization, yield ages of 2625 ±7 Ma (Jemielita et al., 1990) and 2621 Ma (Zweng et al., 1993).

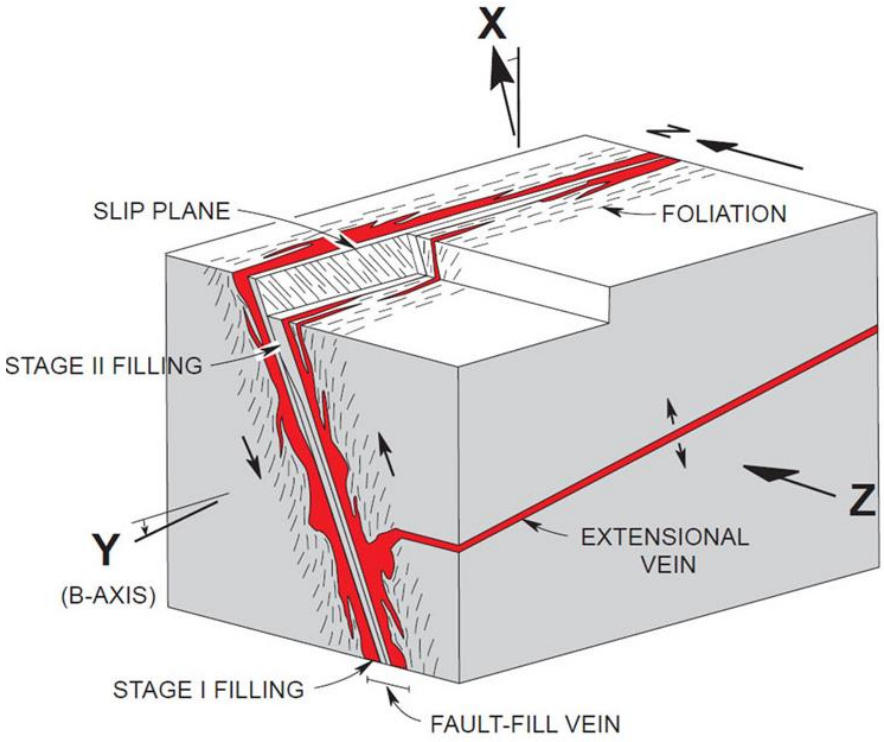


Figure 8-2: Schematic diagram of the geometric relationships between the structural elements of veins and shear zones and the deposit-scale strain axes (Robert, 1990; Modified after Dubé and Gosselin, 2007)

9. EXPLORATION

This chapter presents the exploration work performed by Wesdome on the Project since the 2019 MRE (August 6, 2019).

9.1 Surface Exploration

9.1.1 Geophysical survey

From February 2, 2020 to March 3, 2020, Wesdome hired Quantec Geosciences Ltd. to perform a TITAN magnetotelluric (MT) survey over de Kiena Mine Complex. The following information was taken from the Quantec report (Tournerie, 2020) unless specified otherwise. A total of 543 sites distributed along 21 profiles oriented at N041° were surveyed. Survey points were mostly on top of the Lac de Montigny ice and surveying was performed using a snowmobile (Figure 9-1).

The MT data were analyzed using 3D inversion and a total of 13 main features over the area were exposed. The features generally presented a subvertical structure with a depth extension close to 1 km.

According to Tournerie (2020), the more resistive and less resistive zones seem to identify the principal units, faults or contact over the area. Based on the geologic information available, they observed that the more resistive zones seemed to map the basalt units while the less resistive zones seemed to be associated with the ultramafic units. Quantec did the survey perpendicular to the inferred lithological units, which are interpreted as a sequence of mafic and ultramafic units (Figure 9-1).

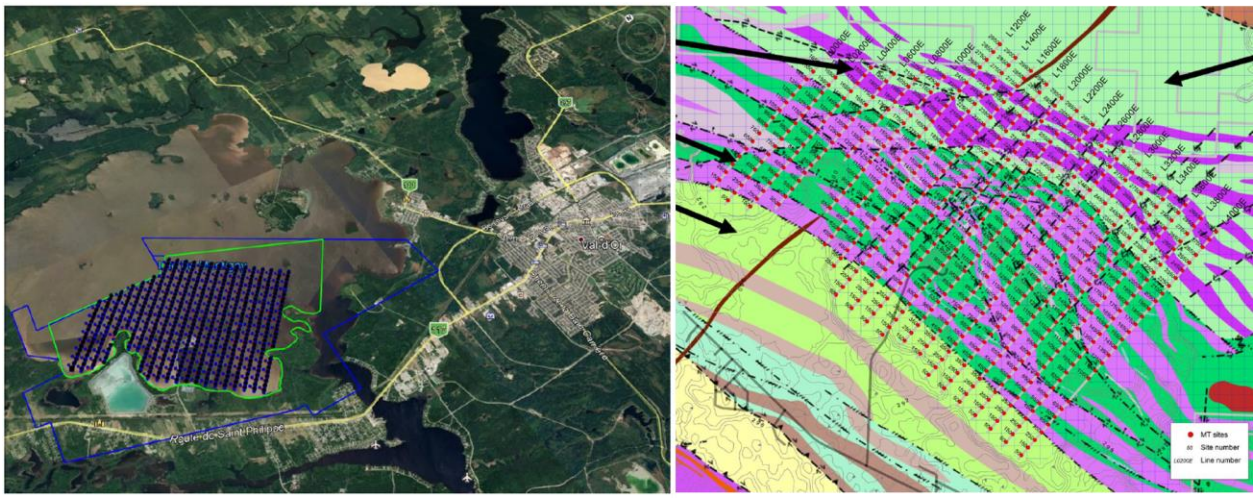


Figure 9-1: General location map of the survey (left) and TITAN-MT survey points over the SIGEOM geological map (right) (Tournerie, 2020)

9.1.1.1 Procedure

The MT method is a natural source utilizing electromagnetic (EM) waves that measures the variation of both the electric (E) and magnetic (H) fields on the surface of the earth to determine the distribution at depth of the resistivity of the underlying rocks.

The objective of the inversion of MT data is to compute a distribution of the resistivity of the surface that explains the variations of the MT parameters, i.e., the response of the model that fits the observed data. The solution, however, is not unique and different inversions must be performed (different programs, different conditions) to test and compare solutions for artefacts versus a target anomaly.

The depth of the investigation is determined primarily by the frequency content of the measurement. Depth estimates from any individual soundings may easily exceed 20 km. However, the data can only be confidently interpreted when the aperture of the array is comparable to the depth of investigation.

The inversion model is dependent on the data, but also on the associated data errors and the model norm. The inversion models are not unique; they may contain artefacts of the inversion process and therefore may not accurately reflect all the information apparent in the actual data. Inversion models need to be reviewed in context with the observed data and model fit. The user must understand the model norm used and evaluate whether the model is geologically plausible.

2D inversion parameters

2D inversions were performed on the data for two selected profiles, L0200E and L2400E.

The 2D inversions presented in the Report were carried out along each profile using the CGG2D inversion algorithm. For each profile, the strike direction is perpendicular to the profile for all sites.

The 2D inversions were performed using resistivity and phase data interpolated at six frequencies per decade, assuming 10% and 5% error for the resistivity and phase, respectively, which is equivalent to 5% error on the impedance component Z. For each profile, static shift was included as a parameter of the inversion.

The topography was included in the inversions of each profile. To accommodate topographic and near surface variation (i.e., the lake), the vertical mesh was set with 10 m thick cells for the first 50 m, and then the thickness of the cells increased logarithmically (factor 1.06) with depth. Each 2D inversion started from a half space model of 1,000 Ω·m.

3D inversion parameters

The 3D inversion was carried out using the CGG RLM-3D inversion code. The 3D inversions of the MT data were completed over an area of approximately 4 km x 3 km. All MT sites (i.e., 543 stations) from this current survey were used for the 3D inversion.

A summary of the 3D inversion parameters is provided in Table 9-1.

Table 9-1: 3D MT inversion parameters

Number of sites	543 MT sites
Mesh cell size (core mesh) *(see Figure 3-1)*	X-Grid (EW): 121 x 33.33m (~4km) Y- Grid (NS): 88 x 33.33m (~3km) Z-Grid: 75 cells covering ~4.5km
Padding coverage	Over 40km in each X, Y, and Z direction
Data used	Complex Impedances Zxx, Zxy, Zyx, Zyy at 24 frequencies from 10kHz to 0.01Hz sampled at 5 frequencies per decade (gap between 4 and 0.1Hz); location of the magnetic sensor provided for each site; data rotation to X (i.e. Ex) at 309deg.

9.1.1.2 Results and Recommendations

Two dimensional (2D) sections were prepared for each line of sampling (21 lines) and the results of the data in 3D were presented as cross-sections of the resistivity along the profiles and as elevation plan maps. Figure 9-2 presents the magnetic 1st vertical derivative map with inferred structural features from the MT3D model. The 13 main features are labelled A to M in the figure. The more and less resistive zones seem to identify the principal units, faults or contacts over the area. Quantec suggested to review the resistivity model with the known geology and drilling information.

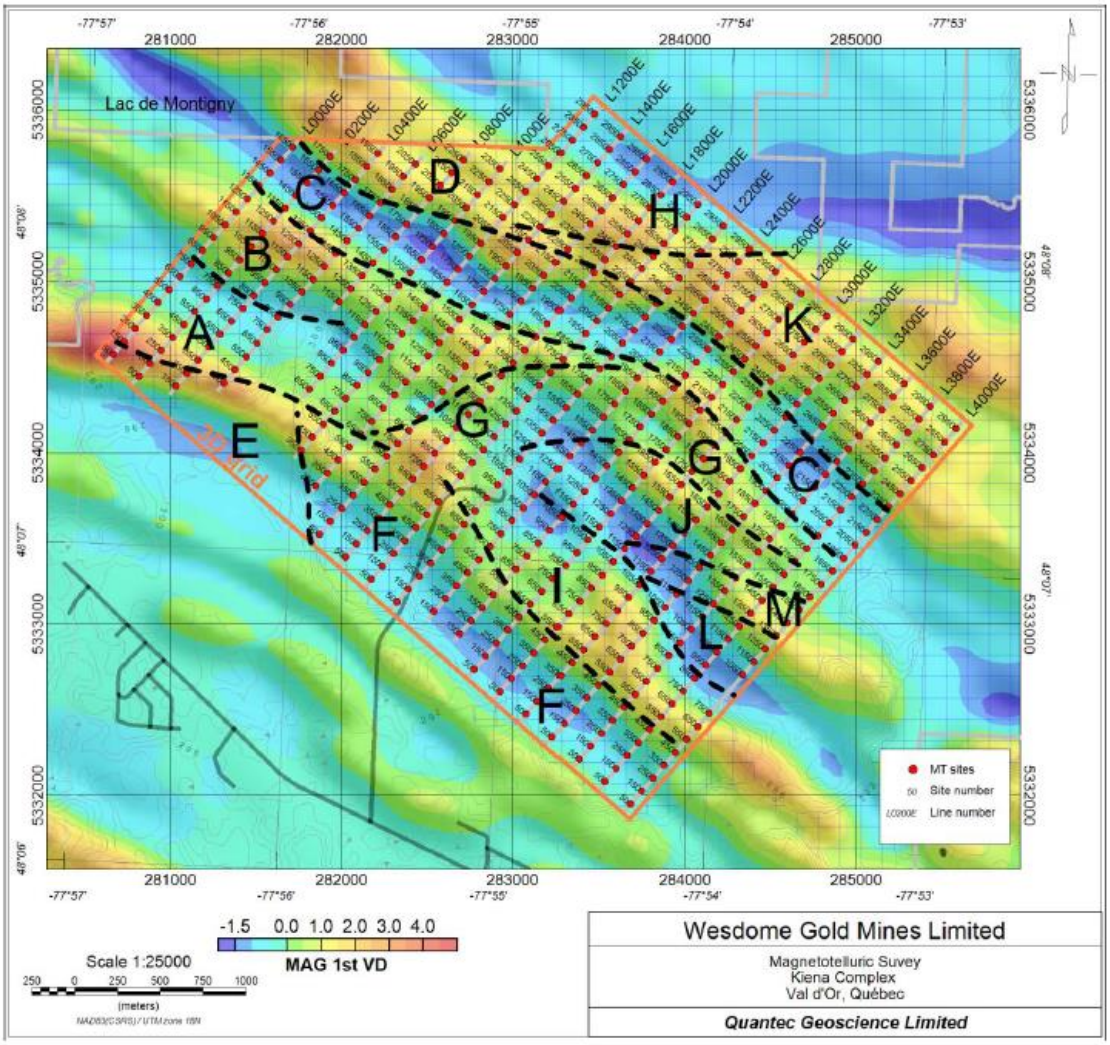


Figure 9-2 : Magnetic 1st vertical derivative map with inferred structural features from the MT3D model (Tournerie, 2020)

9.2 Underground Exploration

In August 2017, Wesdome developed an exploration ramp to provide additional underground drilling platforms. The exploration ramp allowed shorter holes to be drilled with better angles and accelerated access to the Kiena Deep lenses.

Ramp development started at the 100 Level. The CMAC-THYSSEN Mining Group Inc. was contracted to perform the underground development. The first drilling bay was completed in October 2017 and diamond drilling began in the Kiena Deep Zone.

From August 2017 to November 2018, a total development of 2.2 linear kilometres allowed diamond drilling to be done from a more optimal direction (i.e., to the northwest) to intersect the steeply plunging lens to the southeast and provide drilling platforms that allowed definition drilling in the central area of the Kiena Deep A Zone. The enhanced drill platforms also made it easier to drill several step-out exploration holes.

In 2019, development has continued with a new exploration ramp on the 79 Level. Totalling 575.5 m of development, the 79 Level ramp has been completed in early 2020. It provides optimal drill platforms for testing the possible up-plunge extension of the Kiena Deep A Zone and extensions of VC Zones between the 670-m level and the 1,070-m level. Prior to the COVID-19 suspension on March 25, 2020, a total of 172 m of new development was achieved in the main ramp, below the 1,070 m elevation.

Activities at Kiena restarted on May 11, 2020. Down ramp development continued to enhance and increase drill platforms and provide access to the lower areas of the A Zone. Access development from the 109 Level (main ramp) was completed towards the A Zone on the 111 Level (secondary ramp) to position Wesdome to perform underground development of the A Zone. From May to December 2020, a total of 240 m of new development was achieved in the main ramp below 1,090 m elevation.

As of January 2021, the ramp has advanced below the 111 Level (1,110 m) and progression is ongoing towards the 112 Level.

9.3 Underground Bulk Sample Program

The 2020 bulk sample program consisted of:

- Underground access and level development;
- Geological truthing with mapping, chip sampling and muck sampling;
- Milling and processing.

The main objectives of the 2020 bulk sample program were to gain further knowledge on the geological settings, to better understand the mineralized zones, shears and mineralization controlling structures, and to improve the understanding of the gold content and distribution. The bulk sample was also a good way to find the right combination for the ground support and excavation size.

Milling and processing of the material was performed on-site with the refurbished existing infrastructure. The infrastructure was kept under care and maintenance since 2013 and was refurbished in 2019-2020.

The bulk sample consisted of 3.7 m wide x 4.0 m high lateral development on the 111 Level along the A Zone and A1 Zone (Figure 9-3; Figure 9-4). A total of approximately 66 drift rounds were blasted, which allowed for thorough geotechnical assessment and geological mapping, photographing and sampling from the faces of each development round. Chip and muck sampling was completed systematically to determine average grade for each drift round. A total of 984 muck and 238 chip samples were collected in addition to quality control samples. During the development of this portion of the A Zone, visible gold was observed in approximately 75% of the faces.

The initial lateral development has confirmed the continuity of the A Zone's high-grade gold mineralization along strike. Visible gold is associated with folded quartz veins that are located within an overall zone of strong amphibole alteration within a sheared corridor. The amphibolite is located at the contact between basalt/feldspar porphyry dyke and ultramafic rocks. Detailed muck and chip samples (uncapped) presented in Figure 9-5, show the grade and geological continuity.

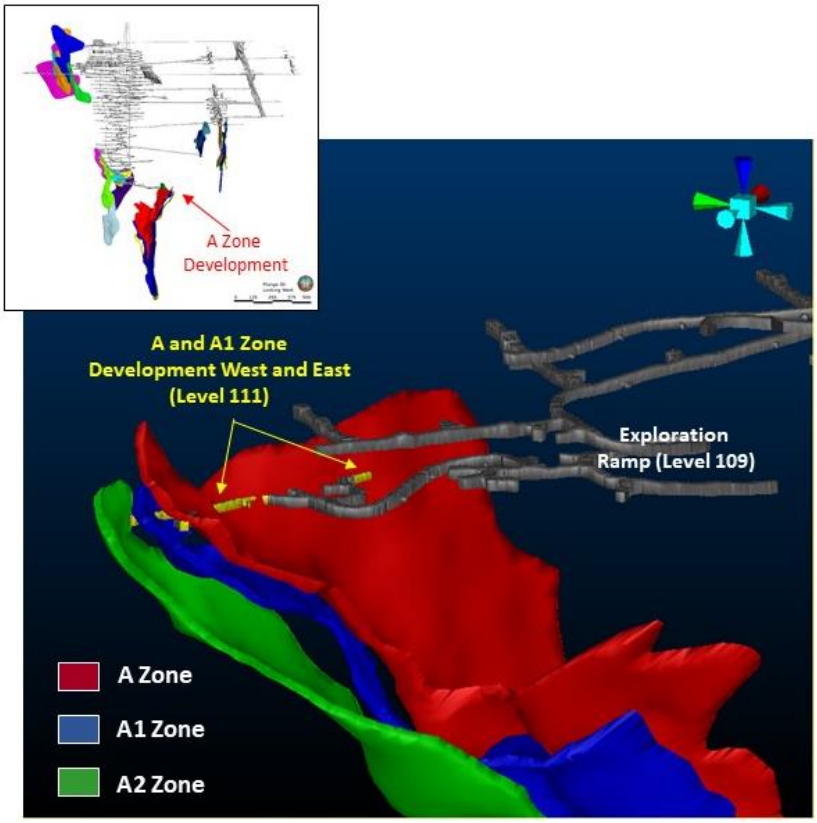


Figure 9-3: Bulk sample location
(Turcotte and Lapointe, 2021)

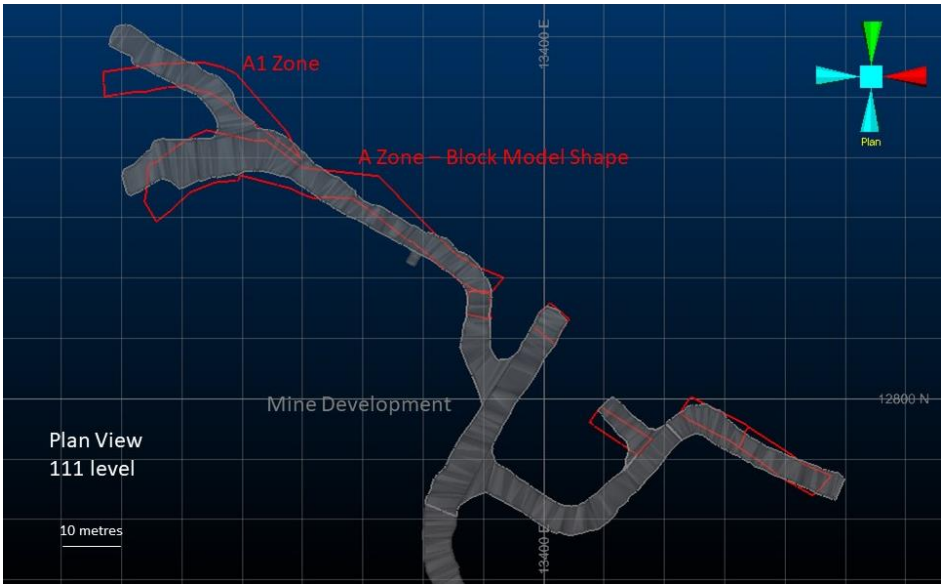


Figure 9-4: Plan view of the block model A Zone and the mine development (Turcotte and Lapointe, 2021)

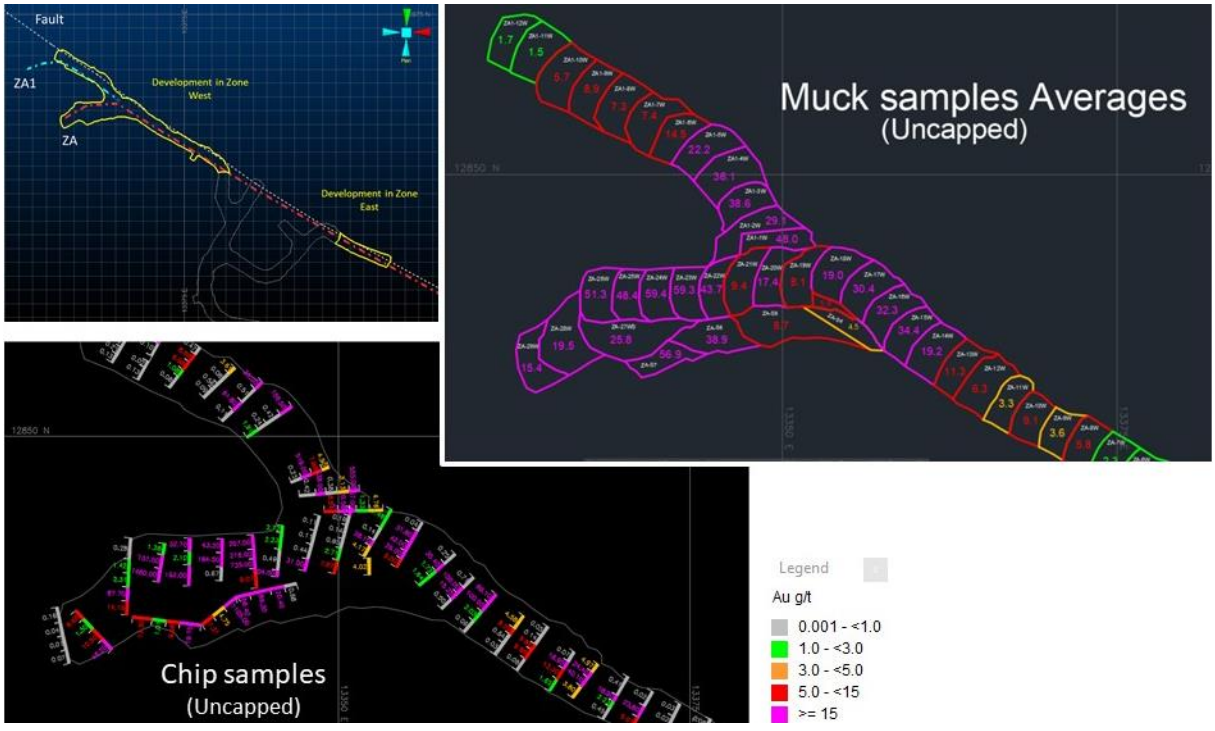


Figure 9-5: Muck and chip preliminary results (Turcotte and Lapointe, 2021)

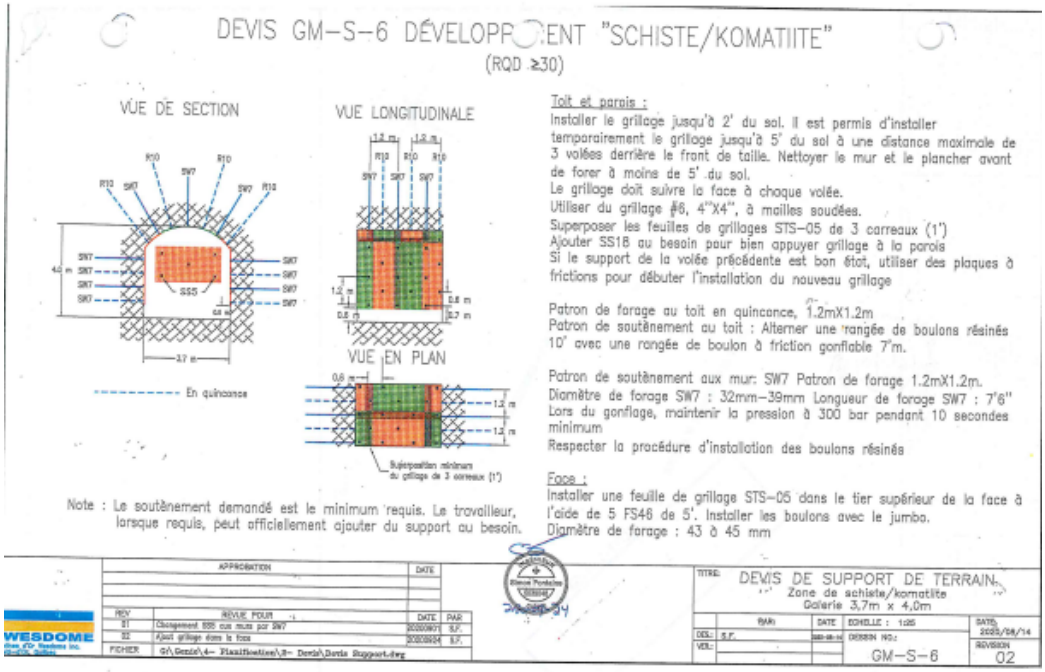
9.3.1 Underground Access and Level Development

The down ramp in the Kiena Deep A Zone was advanced beyond the location of the level access to the 111 Level. The first round of the 2020 bulk sample program on the 111 Level was taken on August 20, 2020. The last round was taken on November 14, 2020.

9.3.1.1 Excavation Sizes and Ground Control Measures

The initial development of the access to the 111 Level was designed to accommodate the various mine services (compressed air, water, ventilation). The size of the development was originally 4.5 m wide by 4.8 m high using a back support with a staggered pattern of 10-foot rebar, 7-foot Swellex on a 1.2-m x 1.2-m grid, and a wall support with 5-foot split set on a 1.2-m x 1.2-m grid. Due to ground conditions, the excavation size was reduced to 3.7 m wide by 4.0 m high with a drift advance of 2.4 m (Figure 9-6). Back support used a staggered pattern of 10-foot rebar and 7-foot Swellex on a 1.2-m x 1.2-m grid. Wall support used a 7-foot Swellex on a 1.2-m x 1.2-m grid. Weldmesh screen was also added on the working face.

Prior to lateral development, pre-support was installed to prevent overbreak and improve the overall stability of the opening. The pre-support consisted of a 12-foot split set installed in the direction of the drift before the blast. A minimum of 12 split sets were installed with the drilling jumbo before each drift round.



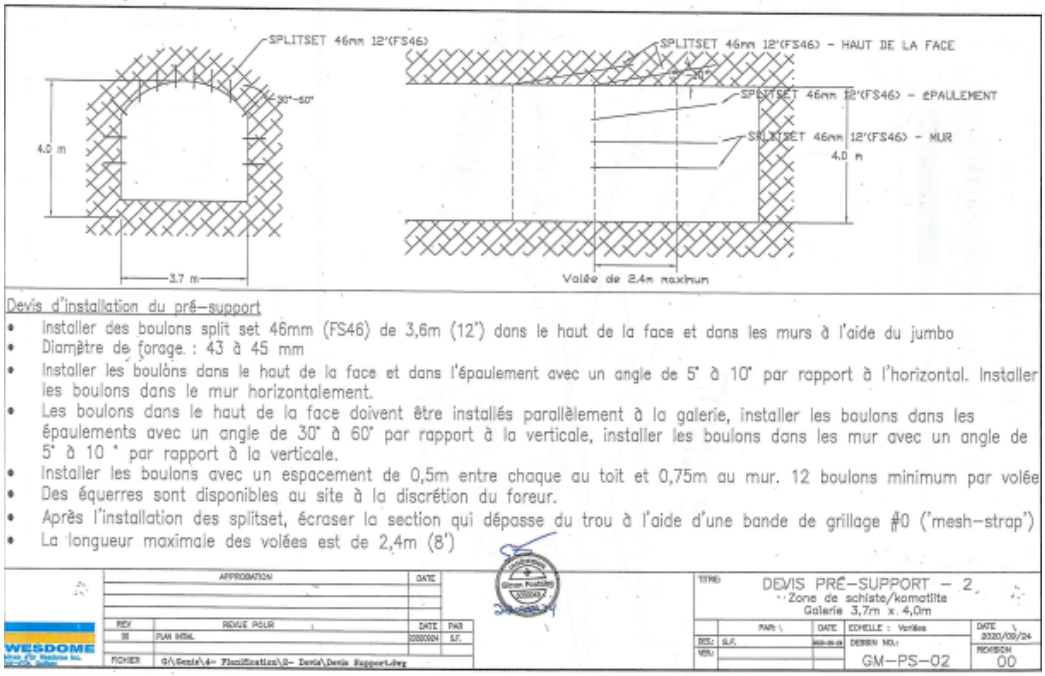


Figure 9-6: Development plan for the 111 Level mineralized zone

9.3.1.2 Productivity and Methods

The following mobile equipment was used to move waste rock and mineralized material:

- Haul Truck 426 MT with a 26-t capacity;
- Haul Truck MT 420-10 with a 20-t capacity;
- LHD Atlas Copco 1030 with a 6-yd^3 bucket;
- LHD Wagner with a 3.5-yd^3 bucket.

The material was hauled to the mineralized material pass on the 79 Level, and waste rock was hauled to the waste pass on the 79 Level. After skipping all the high-grade mineralized material, low-grade mineralized material was dumped in the mineralized material pass to be skipped to surface.

9.3.2 Geological Mapping

Geological mapping was conducted for each round during the 2020 bulk sample program. Lithologies, veining, mineralization and principal structures were mapped using Datamine's Studio Mapper software. All the mapping was done on a Microsoft Surface tablet. Using this method, digitalization of the map was not necessary, and all the data is regrouped within the Studio Mapper software. Mapping of both walls, the face and the back was carried out during this process. A portable battery-powered LED lamp was used during the mapping and to take pictures of the heading (Figure 9-7; Figure 9-8; Figure 9-9).

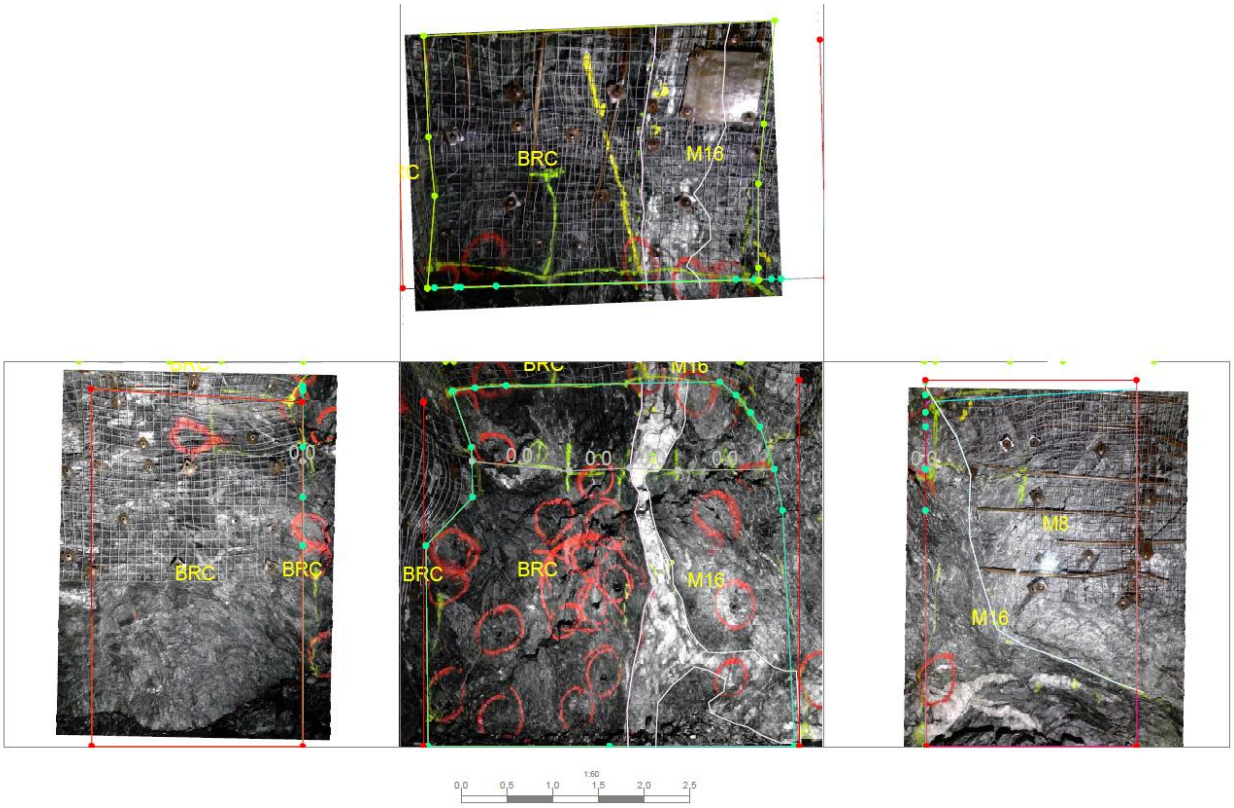


Figure 9-7: Heading map of round ZA-15W

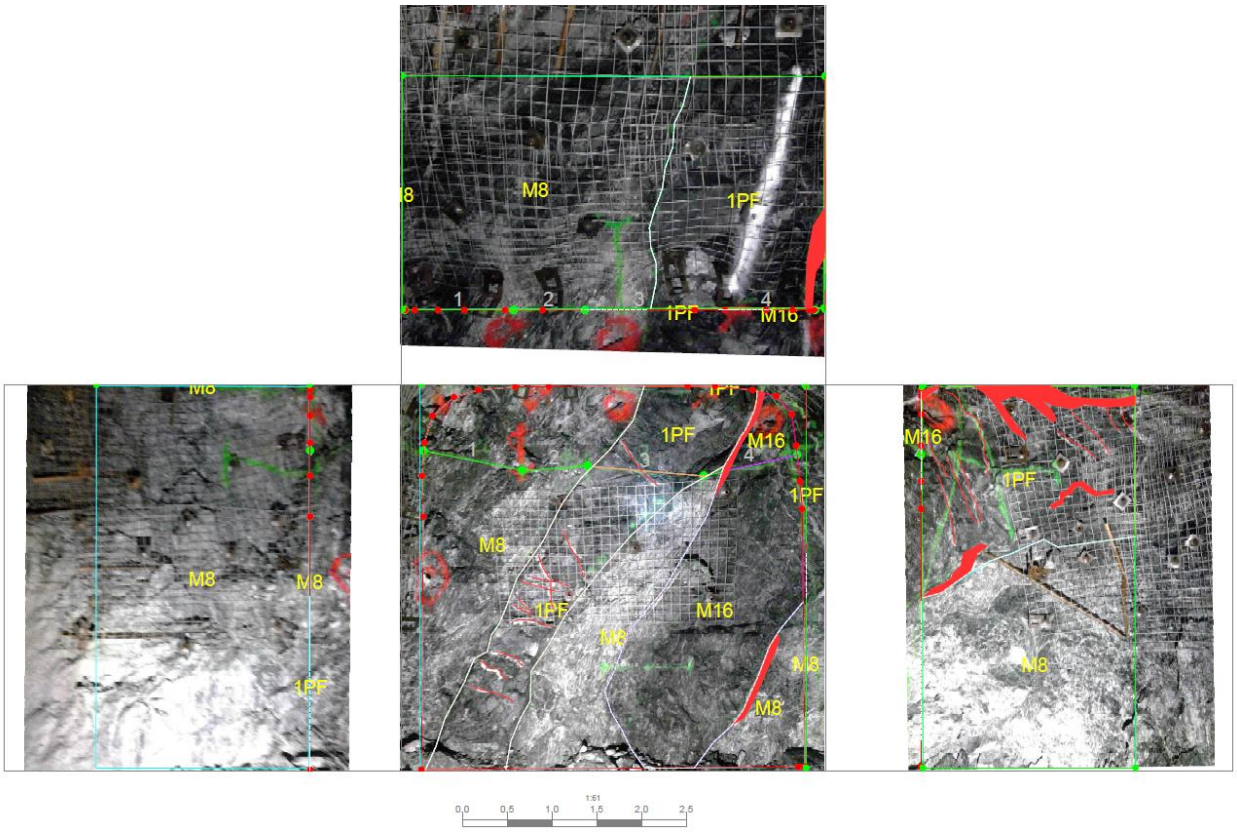


Figure 9-8: Heading map of round ZA1-4W

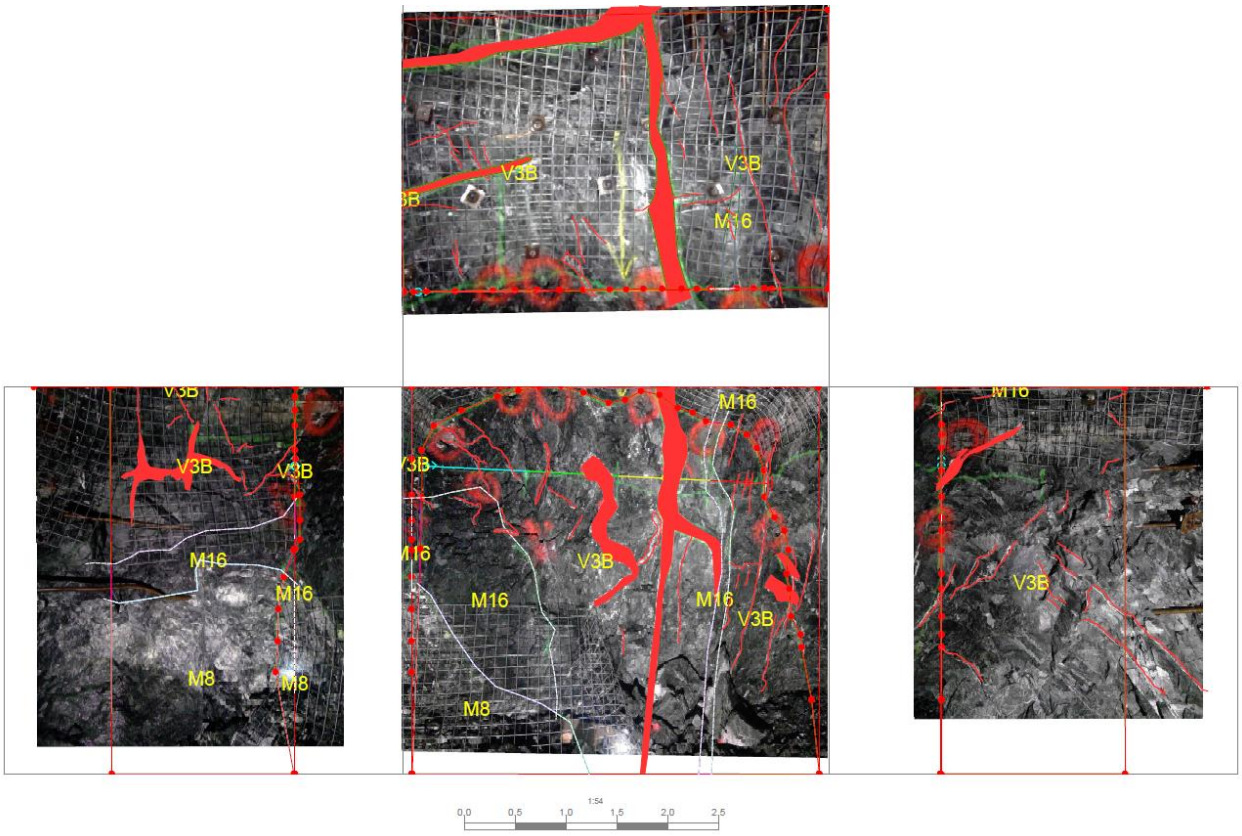


Figure 9-9: Heading map of round ZA-10E

9.3.3 Chip Channel Sampling

Chip channel samples were taken by the geologist following Kiena Mine sampling procedures.

The chip channel sampling was completed using a battery-operated percussion hammer. A Makita 36v battery operated percussion drill/hammer was purchased and utilized for this purpose. The battery-operated instrument effectively sampled the soft ultramafic unit as well the sheared and/or foliated rock units.

To avoid any bias, the sample location was almost always taken at the same height in the face (-1 m from the back). Rarely, when the scissor lift was not available, samples had to be taken at shoulder height. The location and length of the samples were defined using lithological contacts and varying characteristics in the rock (sulfides percentage, vein proportions, etc.). Samples were taken along a painted horizontal line to remain consistent in each drift round. When developing in the mineralized zone, the face was systematically chipped on its full width. During the cross-cutting, perpendicular to the mineralized zone, both drift walls were sampled.

9.3.4 Muck Sampling

Muck samples were taken for each drift round by the geologist during the re-mucking process. Sampling was done following the Kiena Mine underground sampling procedures.

9.3.4.1 Procedures

Each drift round was systematically sampled during the program. Samples were taken directly from the scoop bucket with a designated hand-scoop to assure a relatively similar weight for each sample. During the process, the geologist alternates sides in each scoop bucket to mitigate the risk of sample location within the scoop bucket. About halfway into the program, mining operations changed from a 6-cubic yard scoop bucket to a 3.5-cubic yard scoop bucket due to ventilation constraints. When using the 3.5 cubic yard scoop bucket, sampling was performed every second scoop bucket to correlate with one sample taken every 6 cubic yard scoop bucket. Samples weighed on average 2.3 kg and a sampling rate of about 1 sample was taken for every 6 tonnes of material.

9.3.5 Processing

The mill that is part of the Kiena Mine Complex was restarted to process the 2020 bulk sample from the Kiena Deep A Zone. The material resulting from the lateral development came from two mineralized zones, namely the A and A1 Zones. The milled tonnage was measured using a conveyor scale. Daily throughput reached up to 41 tonnes per hour (“tph”) with no issues experienced during the extraction of the bulk sample. The grade and recovery values provided by the mill were determined using multiple sampling points, per cent solids measurements, tailings grade sampling, laboratory analysis and refined ounces (during the milling), and the ounces recovered during the cleaning of the SAG and ball mills after milling was completed.

All calculations and sampling methods have been validated using an independent mineral processing consultant. The material recovered during the cleaning of the SAG and ball mills was sent to a third-party refiner, Sipi Metals Corp (a precious metal refinery based in Chicago, USA) and was included in the overall bulk sample reconciliation (Turcotte and Lapointe, 2021).

9.3.6 Highlights and Results of the Bulk Sample Program

According to the April 12, 2021 Wesdome press release from Turcotte and Lapointe (2021), the key highlights of the of the 2020 bulk sample program are as follows:

- A total of 7,032 tonnes generated from the planned area were processed at the Kiena mill. Additional lower grade material outside of the planned area of the bulk sample was stockpiled on surface;

- Successfully tested an area within the 2020 Mineral Resource Estimate dated December 15, 2020 and confirmed the continuity of the high-grade gold mineralization within the A and A1 Zones;
- The bulk sample recovered 6% more gold than the 2020 MRE with a feed grade of 15.7 g/t versus model grade of 14.7 g/t;
- Reconciliation with capped muck samples (165 g/t Au) was 8% above mill feed;
- Geotechnical conditions were sound overall and allowed Wesdome to assess and optimize development protocols;
- Total gold produced was 3,479 oz with gold recovery in the Kiena mill at 98.2%.

10. DRILLING

This chapter presents the drilling program carried out by Wesdome between August 7, 2019 and September 18, 2020 (the "2019-2020 Program") on the Kiena Mine Complex. The drill database close-out date for the 2020 MRE is September 18, 2020.

10.1 Drilling Methodology

The surface drilling program was performed by Forage Hébert Inc. and used two diesel drills, built on tracks. The drills are B-20 (hp 225 and hp 275) that can reach a maximum depth of 1,200 m on NQ caliber (47.6 mm core diameter). The surface drilling was completed on land up to January 2021.

The underground drilling program was performed by Forage Orbit Garant Inc. (Orbit-Garant). Drilling was conducted with NQ caliber (47.6 mm core diameter) and telescoped with BQ caliber when rock quality was poor (i.e., faults, shears, schist). A total of five electric diamond drill rigs were used in 2019 and increased to a total of seven rigs in January 2020. The drill types and models were as follows:

- Two YU1800 (125 hp): max 1,200 m (NQ); max 1,600 m (BQ);
- Two YU1200 (100 hp): max 750 m (NQ); max 1,000 m (BQ);
- One YU615 (100 hp): max 400 m (NQ); max 600 m (BQ);
- Two B15 rigs (100 hp): max 450 m (NQ); max 650 m (BQ).

Every hole was drilled with maximum stabilization using two hexagonal core barrels and a 36" shell until reaching the schistose unit.

10.1.1 Drillhole Location/Set-up

Diamond drillholes (DDH) for the 2019-2020 Program were planned using vertical cross-sections and plan views with the aim of improving the accuracy of the interception angle in mineralized zones.

The coordinate system in use was a Local Mine Grid.

The software used were Geotic Mine, GeoticGraph, AutoCad and Promine. Drillhole collars were established by a surveyor who drew a line between the front sight spad and a back sight spad. The drillers aligned themselves according to the surveyed line and collared the hole at the most suitable place. After drilling, drillhole collars were initially surveyed by contract surveyors, and later by a Wesdome technician. Collar azimuth and dip were measured when possible.

10.1.2 Drillhole Orientation during Operation

The drillhole orientation is checked and monitored using a downhole surveying device as follows:

- REFLEX EZ-Trac instrument is used to conduct deviation surveys. Single-shot measurements are taken every 50 m until reaching the schistose unit, at which point the stabilization tools are removed;
- Multi-shot measurements are taken at every 3 m while the rods are pulled out;
- If ground conditions in the schistose area are deemed acceptable, single-shot measurements are resumed;
- At the end of the hole, a final multi-shot survey is performed with measurements every 3 m while the rods are pulled out until reaching the end of the first multi-shot survey or the beginning of the schist.

The REFLEX instrument is handled by the drilling contractor who transfers the multi-shot data to the Wesdome geology department. The single-shot data is transcribed and provided in paper format to Wesdome geologists.

10.1.3 Drilling

Recovered diamond drillhole cores by the drilling contractor are in NQ and BQ size. Drillholes are generally started in NQ caliber in mafic rocks. When the drillhole reaches the komatiites, schists or a major fault, BQ caliber is used, otherwise NQ rods get stuck at the bottom of the drillhole. The core is collected in a standard drilling tube and the driller's helper carefully places the core into wooden core boxes at the drill rig. He/she also marks the depth (m) after each 3 m run with wooden blocks and wraps the boxes with tape once it is full. Core trays are numbered with a permanent marker indicating the drillhole number and the sequential box number.

Generally, the drillhole is abandoned in the komatiites or schists. Occasionally, it is possible to pierce and stop the drillhole in the mafic units; if this occurs, the drillhole is terminated by the geologist.

Once the drillhole is terminated and the final downhole survey reading is collected, the drill crew pulls the rods for mobilization to the next drill site. Every hole is systematically grouted with cement. The collars of underground holes are identified by a conical plug with a metal tag displaying the hole number. Sometimes it is not possible to complete the multi-shot survey due to the rock and structures encountered (faults, high fracturing, etc.).

10.1.4 Core Logging and Measurement

In the core shack, Wesdome employees remove the tape and place the boxes on the logging tables. The geologists rotate the core so that all pieces are fitted together, showing a cross-sectional view. They verify that distances are correctly indicated on the wooden blocks placed every 3 m. The core is then measured and the boxes are labelled.

Wesdome geologists log and record the data using GeoticLog software. Lithologies (principal and secondary), alteration, mineralization, veins, structures, magnetism, samples and assay results are compiled in the database.

10.1.4.1 Core Recovery

The core recovery is calculated by measurement in centimetres of core in the core tray divided by the centimetres claimed to be drilled on the meterage blocks. This number multiplied by 100 is recorded as percent recovery. Core recovery is recorded for each drill run. Specific areas of loss are noted, if possible, and marked by placement of a wooden marker and the estimated loss. The ideal core recovery is 100%; however, it is not always possible due to ground conditions or sometimes loss of drill core during the coring process, e.g., grinding, etc. For the DDH included in the 2020 MRE, the average core recovery was 98%.

10.1.4.2 Rock Quality Designation

The rock quality designation (RQD) is designed to give qualitative and quantitative information on the stability of rock surrounding and included in mineralized material. This information is used to determine the mineability and rock control procedures that will be required to extract the mineralized material.

RQD is a quantitative index of rock quality based on a core recovery procedure in which the core recovery is determined by incorporating only those pieces of hard, solid core longer than twice the diameter of the core. For NQ core, the nominal diameter is 5 cm, so the length index is 10 cm; shorter lengths of core are ignored. RQD is determined for each core run as these are the only definitively known distance markers. RQD is determined using the following formula for each core run:

RQD (%) = 100 x the sum of the length of the core pieces equal to or longer than 10 cm / Core run length

It is important to distinguish between mechanical breaks and natural breaks identified in the core.

RQD is valid for solid core only and should not be used for very poorly disaggregated materials such as highly weathered rock, clays or un-cemented aggregates.

The average RQD is 86% based on 20,309 measurements for the DDH included in the 2020 MRE.

10.1.4.3 Core Photography

Once logged by the geologist, all drill core is photographed wet, four boxes at a time. The objective of core photos is to have a digital image recorded with sufficient detail to clearly see core features prior to destructive sampling procedures. This record can be used later to qualify rock quality features and to examine core images against geological logging if the core is unavailable for examination. The photos are also used, as required, during the construction of geological sections.

Once the core is photographed, it is assigned to the core saw operator for splitting and sampling.

10.1.5 Core Storage

After the sampling process, the core boxes are stored under roofed racks in the outdoor core storage area. Every box is labelled with an aluminum tag displaying the hole number, the box number and the From-To meterage. An Excel spreadsheet serves as an inventory of box location in the core storage area.

10.2 Recent Diamond Drilling

As of September 18, 2020 (close-out date of the drill database), Wesdome had completed 213 new DDH during the 2019-2020 campaign on the Property for a total of 60,865 m (Table 10-1). Of these 213 new drillholes (Appendix B), 122 are in Kiena Deep Zones, resulting in the inclusion of an additional 35,280 m of drill data (Figure 10-1 and Figure 10-2).

Table 10-1: Summary of the drilling completed on the Property during the 2019-2020 Program (included in the 2020 MRE)

Location	Drillhole count	Total length (m)	Used in 2020 MRE
Surface	12	6,425	No
Underground	213	60,865	Yes

10.2.1 Post 2020 MRE Diamond Drilling

10.2.1.1 Underground Diamond Drilling Program

Between September 18, 2020 and April 12, 2021, Wesdome has completed an additional 143 underground DDH for 41,780.8 m that are not included in the 2020 MRE. Seven underground drill rigs have continued to operate to test the extensions of VC1 and VC6 Zones and perform infill drilling in the Kiena Deep A Zones as well as testing its down dip extension until the end of 2020. In 2021, underground drilling once again focused on expansion and exploration drilling. Seven underground drill rigs have continued to operate to test: 1) northwest extension of Martin Zone at 38 Level; 2) extensions of VC1 Zone at 79 Level; and 3) extensions of Kiena Deep A Zone.

10.2.1.2 Surface Diamond Drilling Program

Wesdome started a surface drilling program on August 17, 2020 with one drill on the Shawkey West Zone located at about 1,700 m southeast of the mine shaft. A second drill was added on October 1, 2020 to investigate the Presqu'ile Zone located some 1,500 m northwest of the mine shaft. A third drill arrived on site on January 19, 2021 and left on March 23, 2021. This drill investigated the Shawkey Zone 22 with five holes and was also sent on an ice bridge built between the Presqu'ile Zone and the Parker island. Two holes were drilled on the ice before the weather got warm and the ice started melting. As of April 12, 2021, a total of 15,733 m were drilled and are divided in 28 completed holes. Nine holes investigated the Shawkey West Zone, five holes were drilled on the Shawkey Zone 22 and 14 holes were tested on the Presqu'ile Zone. Two holes are still ongoing.

The main objective of the Presqu'ile Zone is to verify the down-plunge extension of Zones 1 to 4. In the Shawkey West area, the main targets are several gold grades located on either side of a breccia flow. For Shawkey Zone 22, the holes were aiming at gold anomalies within a vertical dyke at the contact between a basalt and a komatiite. Targets on ice bridge were the east extension of the Presqu'ile Zones and the Marbenite Fault.

The drilling contractor is Forage Hébert. The diesel drills are built on tracks. The drills are B-20 and can reach a maximum depth of 1,200 m on NQ. On an ice bridge, a heliported drill (hp 171) was used and could drill to a maximum depth of 600 m on NQ.

Those holes are not included in the 2020 MRE.

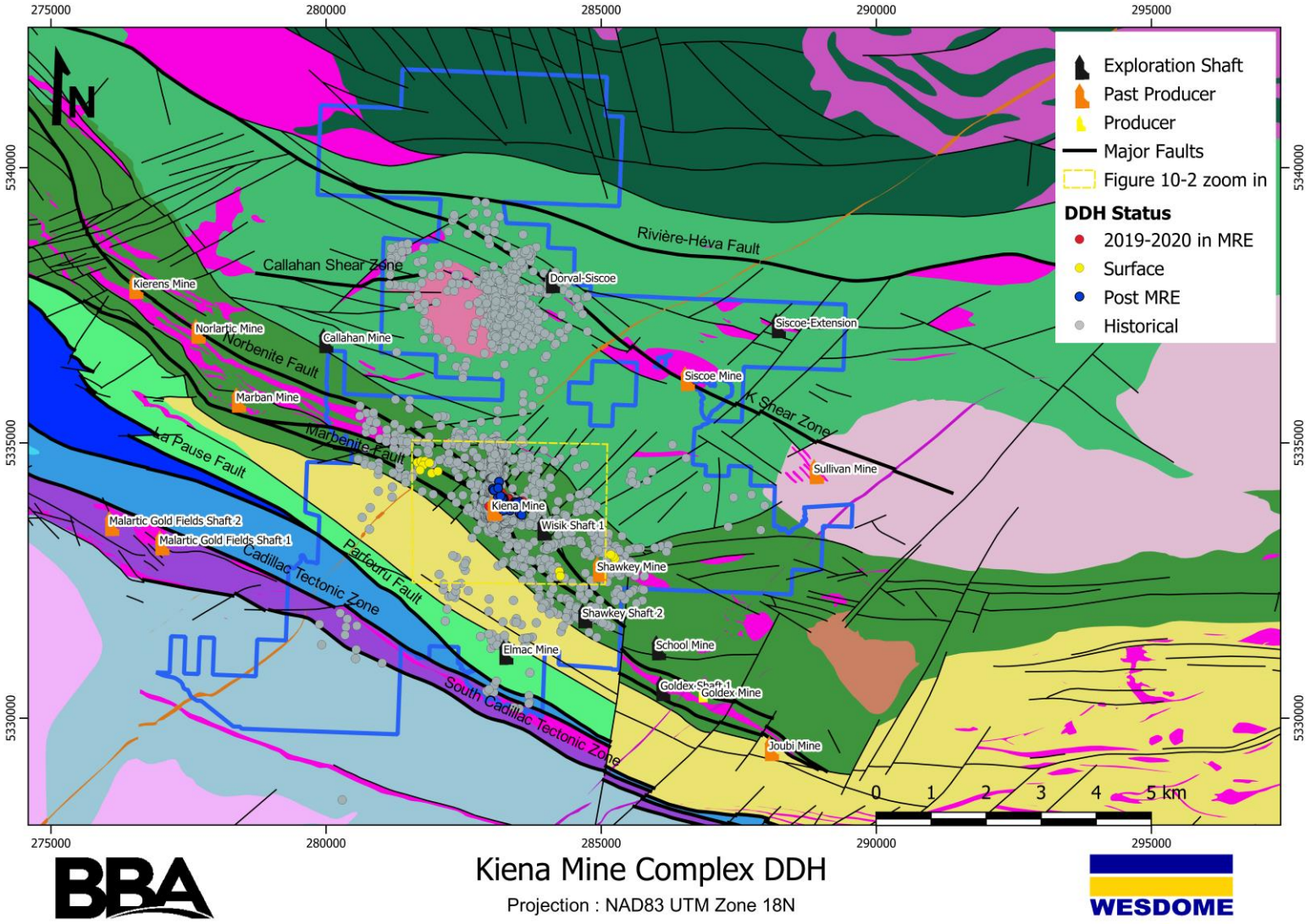


Figure 10-1: Location of drillholes throughout the Property with their status

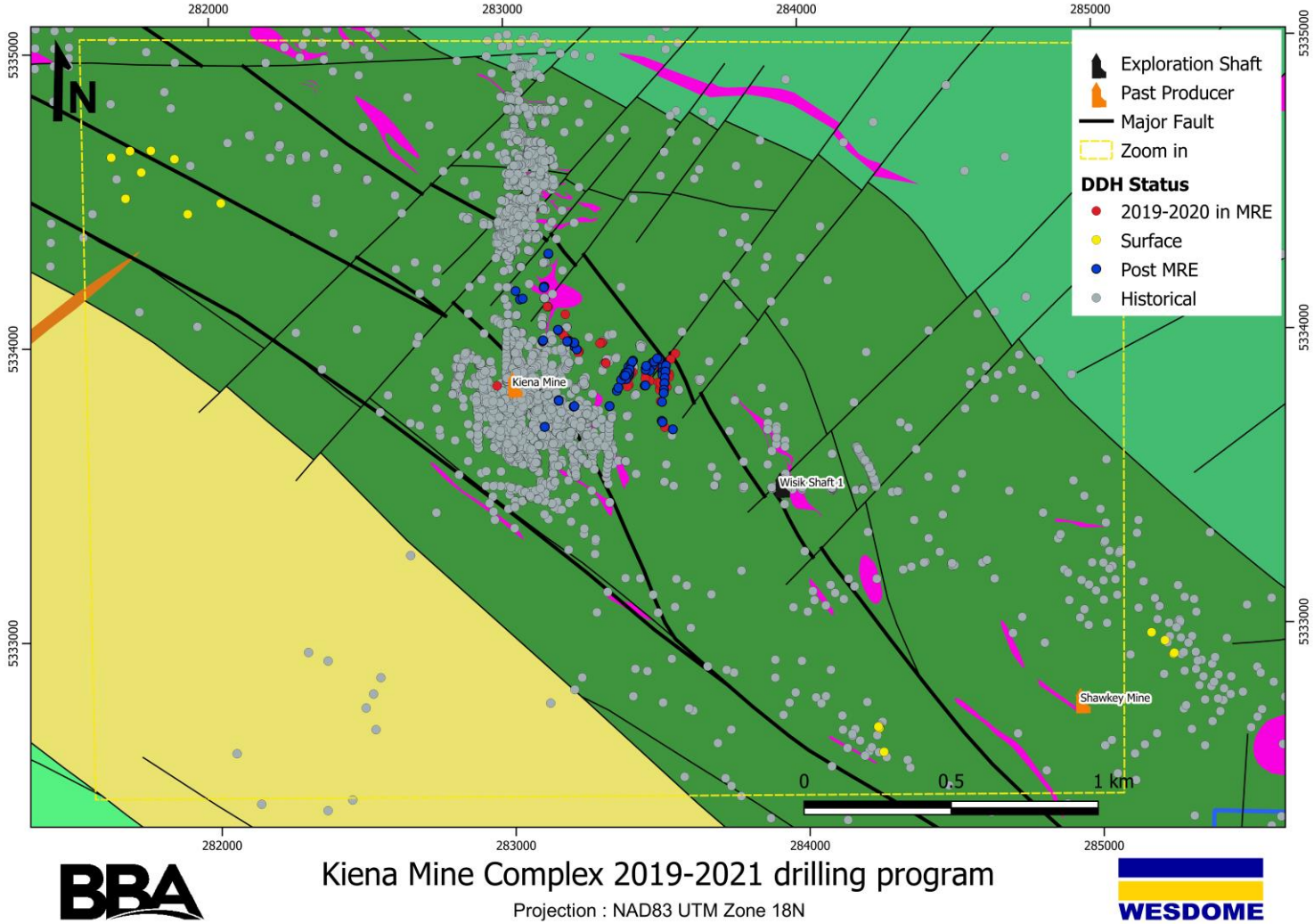


Figure 10-2: Kiena Mine Complex diamond drillhole locations, close-up view

11. SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Wesdome Data

Since February 9, 2019, Wesdome has been sending diamond drill core samples to ALS Global in Val-d'Or, Quebec.

11.1.1 Core Handling, Sampling and Security

Diamond drill core is boxed and sealed at the drill rigs and transported by the drillers to the underground station and brought to surface via the service cage at the end of each shift. A technician from Wesdome takes over the core handling and brings it to the core shack. After being logged and sampled, individual sample bags are placed in rice bags along with the list of samples. QA/QC samples are prepared and bagged ahead of time by Wesdome personnel and are batched at the core shack following the geologist's instructions. Batches are shipped daily to the ALS Global facility in Val-d'Or, Quebec. Batches and shipments contain variable numbers of samples.

11.1.1.1 Gold Assays Samples

To create representative and homogenous samples, sampling honours lithological contacts, i.e., no sample crossed a major lithological boundary, alteration boundary or mineralization boundary.

Sampling intervals are determined by the geologist during logging and marked on the core boxes or on the core itself using coloured lumber pencils with a line drawn at right angles to the core axis. Sample lengths typically range from 0.5 m to 1.50 m with a preferred length of 1.0 m for the mineralized zones. The sampled core is considered representative. Two shoulder samples, each having a sample length of approximately 1.0 m to 1.5 m, are collected from the non-mineralized core above and below the mineralized intervals.

Samples are numbered in consecutive order utilizing sample tag books containing numerical sequences of 50 pre-labeled triplicate water durable sample tags (three tags per sheet). The first of the tags remained with the sample tag book as an archival record of the samples' parameters. The second tag is used to indicate the position of the sample in the core box. This is a permanent sample reference that will remain on the wooden core box. The third and last tag is inserted inside the sample bag. From each sample sheet, the last two tags are separated from the page and tucked under the core at the beginning of each sample by the geologist.

The sample sequence includes blank samples, duplicate samples and Certified Reference Materials (CRMs) that are inserted into the sample stream using sample numbers that are in sequence with the core samples. A CRMs sample, consisting of material of known metal content and internationally recognized and verified, is included in the sample sequence by the trained core sampler. A "blank" sample is material technically devoid of any metals. Blanks and CRMs

are stored in a designated secure area in the core shack. There is never any written reference to the location of any control samples on sample bags, sample tags or dispatch documentation for the assay lab.

Once logged and labelled, the core of each selected interval is sawed in half using a typical table-feed circular rock saw. The core saw operator, trained in core cutting procedures, executes the core cutting at the Wesdome core shack. The logging geologist has already clearly marked out all pertinent cores for cutting and sampling. The core is sawn in half, along its length, with a diamond saw. One half (consistently from the same half of the split core) is put into the plastic sample bag and the other half is retained and kept in the core box for later reference. The paired sample tags are then torn with one tag stapled to the core box at the start of its sample interval and the other tag placed into the sample bag with the core sample. When cutting, the core saw operator looks for visible gold inside the veins and reports it to the geologist when positively observed.

The sample tag number is also written on the outside of the sample bag using a permanent marker. The bag will then be closed using a zip tie and stored in sequence prior to sample dispatch preparation. Sample bags are packed in large 'rice' bags sealed with a zip tie that is 'broken' or opened at the assay laboratories only.

The range of sample numbers inside the bag is written on the rice bag. The sealed rice bags are stored in the core shack in Val-d'Or until shipping to the laboratories. For the 2019-2020 drilling campaign, the samples were transported by a Wesdome employee to ALS Global labs in Val-d'Or, where the samples were prepared and processed.

11.1.1.2 Core Density Samples

New specific gravity (SG) tests were conducted on 39 samples. SG was measured by water displacement method at the ALS laboratory using the OA-GRA08 ALS method (see method below).

About 0.10 m to 0.20 m of core was selected for each density measurement. The dry mass was measured followed by the submerged mass. Both measurements were recorded and the density was measured using the following formula:

$$Specific\ Gravity = {Mass_{Dry\ in\ air}} \Big/ \left(Mass_{Dry\ in\ air} - Mass_{Submerged\ in\ water}\right)$$

11.1.2 Methods of Preparation, Processing and Analysis

11.1.2.1 Lab Accreditation and Certification

ALS has the ISO/IEC 17025:2005 accreditation through the ALA (Canadian Association for Laboratory Accreditation Inc). It is an independent commercial laboratory.

11.1.2.2 ALS Sample Analysis Procedure

At ALS laboratories, samples are sorted, bar-coded and logged into the ALS Webtrieve program. Damaged samples are documented and Wesdome personnel are informed. Samples are dried to constant weight and weighted (WEI-21). The sample is then crushed to P_{70} 2,000 µm (CRU-31). A split is collected using a riffle splitter (SPL-21) and a reject duplicate split is prepared from that original sample (SPL-21d). A pulverization split of 250 g is then prepared for both the original and duplicate split (PUL-31; PUL-31d) at P_{85} 75 µm. A pulp duplicate is also prepared from the original sample (SPL-34). When a metallic sieve analysis is conducted (Au-SCR21), a pulverization of 1,000 g P_{95} 106 µm is done (PUL-35a).

Samples are then analyzed by fire assay (FA) with atomic absorption (AA) spectroscopy from 30 g pulps (Au-AA23). The lower detection limit is 0.005 g/t. When assay results are higher than 3 g/t, the sample is re-assayed with a gravimetric finish (Au-GRAV21) on a 30 g pulp. If results are higher than 10 g/t, gravimetric finish is done and the Metallic sieve method (Au-SCR21) is also conducted. In this case, 1,000 g is pulverized and screened to 100 µm. Duplicate assay is done on screen undersize and the entire oversize fraction is assayed.

Results are provided through a secure server and downloaded, by the geologist in charge of the project, in Excel format and the official certificate (sealed and signed) in PDF format.

As part of ALS internal quality control program, four QA/QC samples are inserted by ALS per batch of 24 samples (one blank, two standards and one pulp duplicate). A method blank and certified reference material is applied and reported for each furnace load to monitor the fire assay process. A duplicate crushed sample is drawn at random and assayed for each work order to monitor precision.

11.1.3 Sample Shipping and Security

The following procedures are applied to ensure a safe and secure management of materials and data as it pertains to core samples at the Kiena Mine Complex:

- All core samples submitted for preparation and analysis to the laboratories are secured in rice bags with zip ties and collected directly at the core shack by the laboratory under the supervision of a member of Kiena's team;

- The lab is notified by email that the samples are en route and is instructed to notify Kiena's geologists when the samples arrive at the prep lab;
- The sample shipment contains a sample submittal form as well as a sample dispatch list detailing the security tag number, rice bag number and the number of samples contained in each rice bag;
- The sample submittal form and sample dispatch list are electronically transmitted to the laboratories once the shipment has left the core shack;
- Samples are sent to:

  ALS
  1324, rue Turcotte
  Val-d'Or, Qc, J9P 3X6

- Results are downloaded by Bruno Turcotte, Senior Project Geologist for Wesdome, via a secure server, as Excel files;
- QA/QC data is evaluated before the samples are integrated into a master database;
- The core boxes are stored under roofed racks in the outdoor core storage area enclosed by secure fencing. The exact location of each hole in the outdoor core library is recorded in an Excel spreadsheet for future reference;
- The sample pulps and rejects are stored at the Kiena Mine Complex.

11.2 Quality Assurance and Quality Control (QA/QC)

Canadian National Instrument 43-101 (NI 43-101) Standards of Disclosure for Mineral Projects requires mining companies reporting results in Canada to comply with the CIM Best Practice Guidelines. The guidelines describe the elements required in the reports, but do not provide guidance for Quality Assurance and Quality Control (QA/QC) programs.

QA/QC programs have two components: Quality Assurance (QA) deals with the prevention of problems using established procedures, while Quality Control (QC) aims to detect problems, assess them and take corrective actions. QA/QC programs are implemented, overseen and reported on by a Qualified Person (QP) as defined by NI 43-101.

QA programs should be rigorous, applied to all types and stages of data acquisition and include written protocols for: sample location, logging and core handling; sampling procedures; laboratories and analysis; data management; and reporting.

QC programs are designed to assess the quality of analytical results for accuracy, precision and bias.

The materials conventionally used in mineral exploration QC programs include standards, blanks, duplicates, and check assays. Definitions of these materials are presented hereunder:

- **Standards** are samples of known composition that are inserted into sample batches to independently test the accuracy of an analytical procedure. They are acquired from a known and trusted commercial source. Standards are selected to fit the grade distribution identified in the Kiena mineralization;
- **Blanks** consist of material that is predetermined to be free of elements of economic interest to monitor for potential sample contamination during analytical procedures at the laboratory;
- **Duplicates** are samples submitted to assess both assay precision (repeatability) and to assess the homogeneity of mineralization. Duplicates can be submitted from all stages of sample preparation with the expectation that better precision is demonstrated by duplicates further along in the preparation process;
- **Check Assays** consist of a selection of original pulps that are submitted to a second analytical laboratory for the same analysis as at the primary laboratory. The purpose is to assess the assay accuracy of the primary laboratory relative to the secondary laboratory.

As per NI 43-101, quality control samples were inserted into the sample batches sent to the laboratory. Inserts included pulp duplicate samples, blank samples and standards. Check assays at a secondary laboratory were also performed during the 2019-2020 QA/QC programs.

Table 11-1 summarizes the QA/QC samples submitted to the laboratories along with routine drill core samples.

Table 11-1: Samples submitted to the laboratories for analysis from August 7, 2019 to September 18, 2020

Type of sample	Quantity	%
Primary drill core samples	30,084	79.6
Field blanks	1,685	4.6
CRMs	1,766	4.7
Coarse duplicates	1,535	4.0
Pulp duplicates	1,541	4.0
Check assays	1,185	3.1
Total	**37,796**	**100%**

11.2.1 Duplicates

Duplicate samples are submitted to assess both assay precision (repeatability) and to assess the homogeneity of mineralization.

Coarse duplicates consist of second splits of crushed material. This material will then need to be pulverized.

Pulp duplicates consist of second splits of prepared samples ready to be analyzed and are indicators of analytical precision, which may also be affected by the quality of pulverization and homogenization.

As part of the Wesdome QA/QC program, the laboratory assayed one coarse duplicate for every 20 samples. Coarse duplicate started within the first batch of samples sent to ALS. The QA/QC program also included one pulp duplicate for every 20 samples. Figure 11-1, Figure 11-2 and Figure 11-3 show the scatterplots of the various duplicates for each laboratory. The correlation coefficient varies from 90.9% to 91.0%. The results show relatively good repeatability.

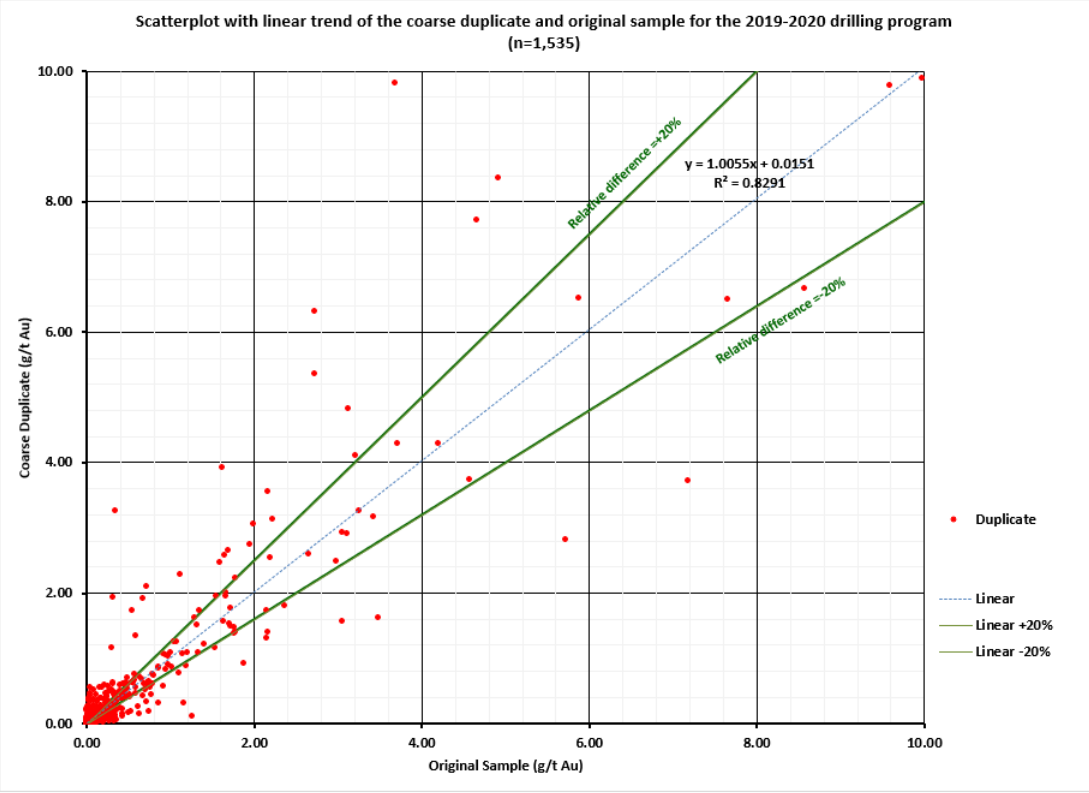


Figure 11-1: Scatterplot with linear trend of the coarse duplicates and original samples results for the 2019-2020 drilling program (n=1,535)

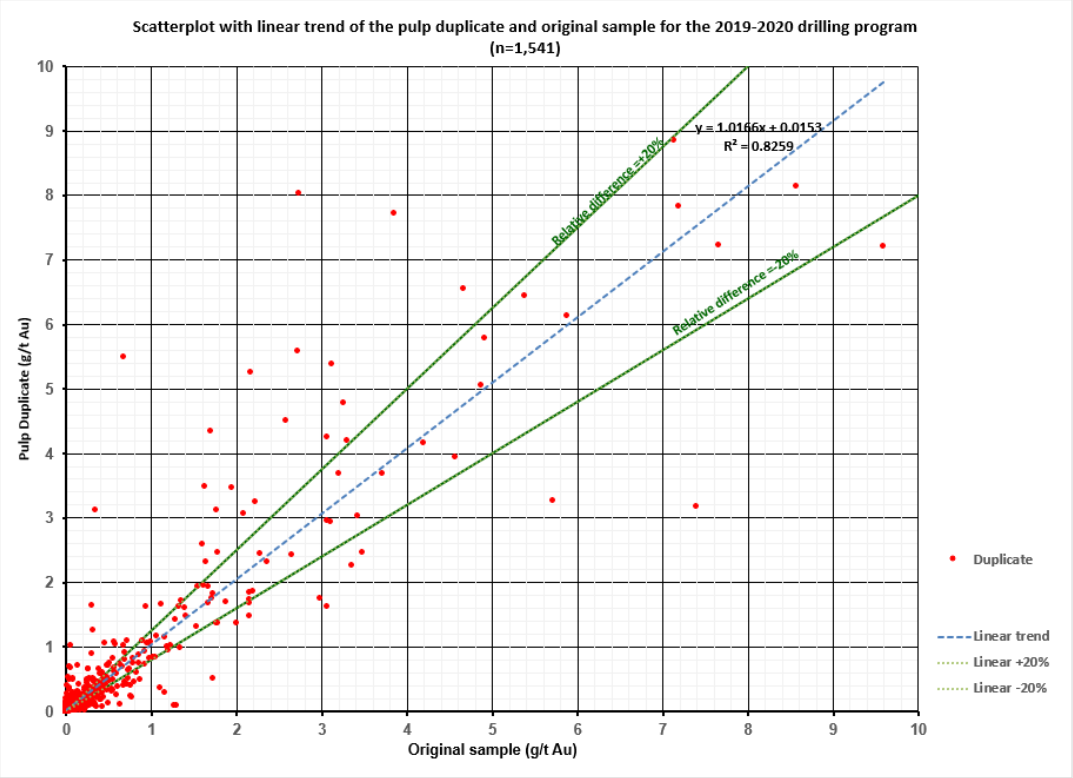


Figure 11-2: Scatterplot with linear trend of the pulp duplicates and original samples results ALS for the 2019-2020 drilling program (n=1,541)

11.2.2 Blanks

Blanks are used to monitor for potential sample contamination that may take place during sample preparation and/or assaying procedures at the laboratory. Sample of barren crushed white marble (blank) were used by Wesdome.

One blank sample was inserted for every 20 samples.

According to Wesdome's QA/QC protocol, if any blank yields a gold value above 0.1 g/t Au, all samples from the 20 samples batch should be reanalyzed. Of the 1,685 blanks analyzed, seven blank samples failed the protocol, which represents 0.4%. Figure 11-3 shows the results of the blank material used during the 2019-2020 Program on the Project.

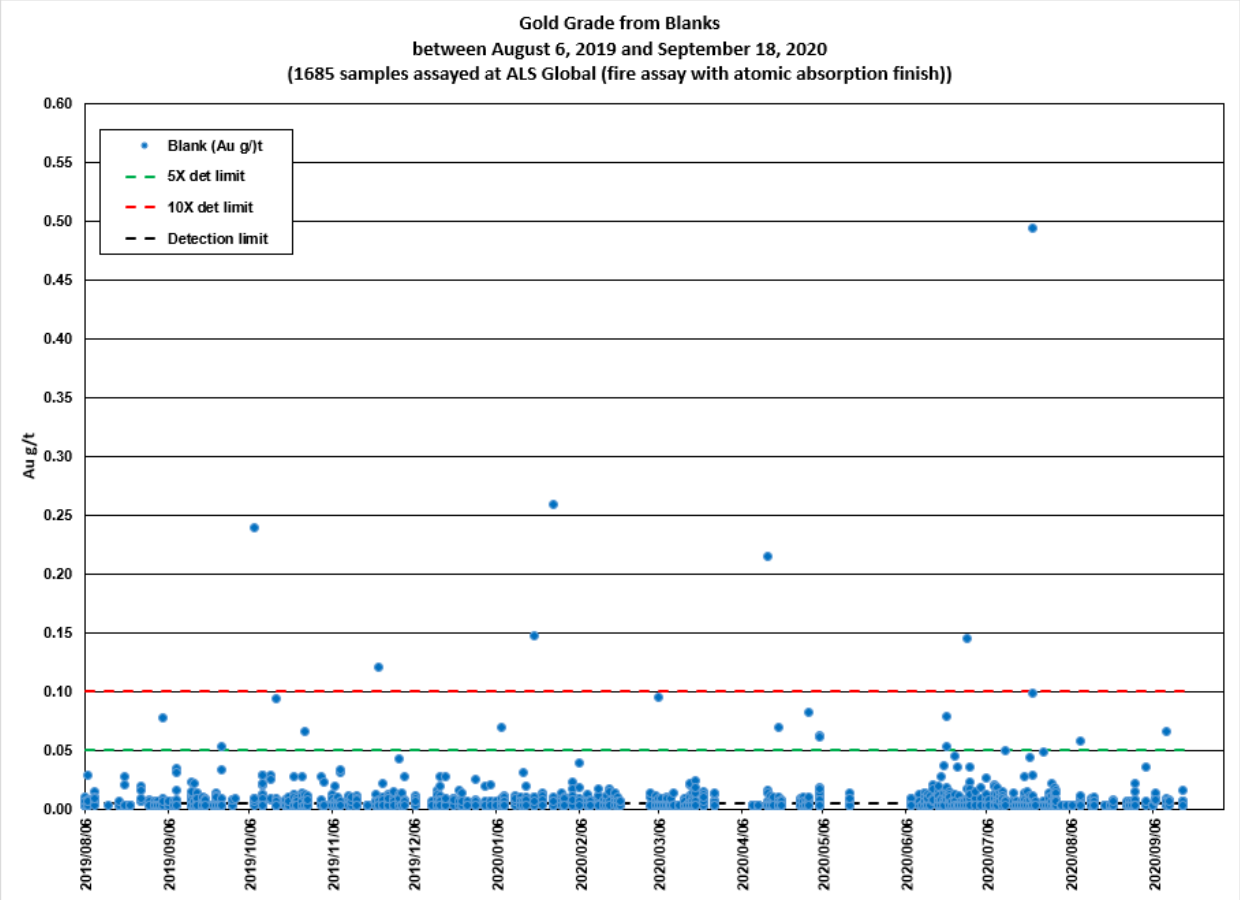


Figure 11-3: Results for blanks used by Wesdome during the 2019-2020 drilling program Detection limit of 0.005 (g/t). (1,685 samples assayed by fire assay with atomic absorption finish)

Generally, the blank indicates little contamination at the laboratory. The re-assay of one batch was requested following a failure for sample A0348615. The ALS laboratory was called and asked for an explanation. Following the re-assay, it was concluded that probable contamination had occurred during the sample pulverization and that the sample analysis sequence may not have been followed.

11.2.3 Certified Reference Materials (Standards)

Accuracy and precision are monitored by the insertion of Certified Reference Materials (CRMs). A suite of commercially available CRMs is used on the Project. One sample was inserted for every 20 samples. Table 11-2 shows the CRMs used for the 2019-2020 Program.

Table 11-2: Standard certified reference materials used at the Kiena Mine Complex during the 2019-2020 drilling campaign

Standard (CRMs)	Method	Laboratory	Certified gold value (g/t)	Quantity inserted	Mean grade (Au g/t) w/o gross outliers	Lower process limit (mean - 3SD)	Upper process limit (mean + 3SD)	Failed (outliers)	Gross outliers	(%) passing QC
SK112	GRAV	ALS	4.113	112	4.147	3.801	4.425	4	0	96.4
SK112	AA	ALS	4.113	122	4.072	3.801	4.425	4	0	96.7
SL108	GRAV	ALS	5.744	69	5.699	5.330	6.158	1	5	98.6
SL108	AA	ALS	5.744	158	5.619	5.330	6.158	5	6	96.8
SJ95	AA	ALS	2.789	559	2.813	2.627	2.951	21	2	96.2
SK109	GRAV	ALS	4.102	82	4.137	3.808	4.396	1	0	98.8
SK109	AA	ALS	4.102	481	4.017	3.808	4.396	18	3	96.3
SN91	GRAV	ALS	8.679	54	8.509	8.097	9.261	1	0	98.1
SN91	AA	ALS	8.679	69	8.525	8.097	9.261	1	1	98.6
SQ87	GRAV	ALS	30.868	60	29.614	28.717	33.019	3	1	95.0
Total				**1,766**				**59**	**18**	**96.7**

The selection of the CRMs was based on anticipated cut-off grade and the 10 g/t re-analysis metallic sieve grade.

CRMs were considered failed by Wesdome when a gold result exceeded three standard deviations (±3 SD) beyond the expected value. If the analytical value exceeded the ±3 SD control limits, systematic re-assaying was not always requested. For each failed standard, Wesdome QC protocol stipulates that only samples included with mineralized zone material should be reanalyzed. If there is no significant gold result within the batch of 20 samples, no re-assay is ordered. During the 2019-2020 drilling programs, 59 CRMs representing 3.3% failed. Considering the low failure rate, the location of these failed CRMs, and the actions taken when such failures occurred, the QP is of the opinion that the failed CRMs are not material for the purpose of this MRE and show the natural statistical spread in the data.

11.2.4 Check Assays

As part of the Wesdome QA/QC program, rejects from a selection of samples were submitted to a second laboratory to assess both the assay precision (repeatability) and the homogeneity of mineralization. Rejects Check Assays consists of the resulting crushed material from the original assay procedure being sent to a different laboratory. This material will then need to be pulverized and assayed by the second laboratory.

In this case, ALS was the original laboratory and SGS was chosen as the secondary laboratory. Figure 11-4, Figure 11-5 and Figure 11-6 show the scatterplots of the various assays methodology for each laboratory. The overall correlation coefficient is good, even if it is from a high nugget effect mineralization. As previously mentioned in Corkal et al. (2020), it is noteworthy to observe that metallic sieve shows a better reproducibility than gravimetric finish and that the latter shows a significantly better reproducibility than atomic absorption finish. Based on these observations, metallic sieve should continue to be used and perhaps even used for lower grades; currently samples above 10 g/t Au.

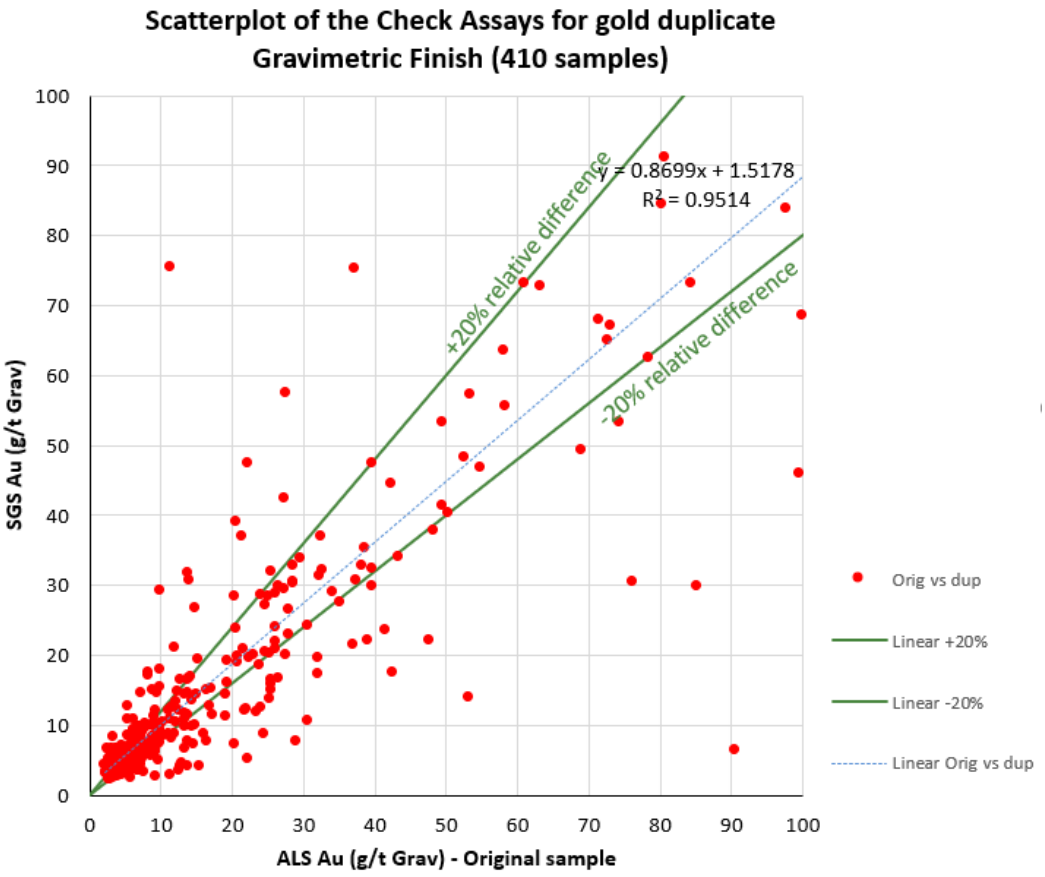


Figure 11-4: Scatterplot showing results from the check assay program for the gravimetric finish methodology
Note that Wesdome's protocol ensures that samples returning grades above the 10 g/t Au threshold using the atomic absorption or gravimetric finish methodology are re-assayed using the metallic sieve methodology.

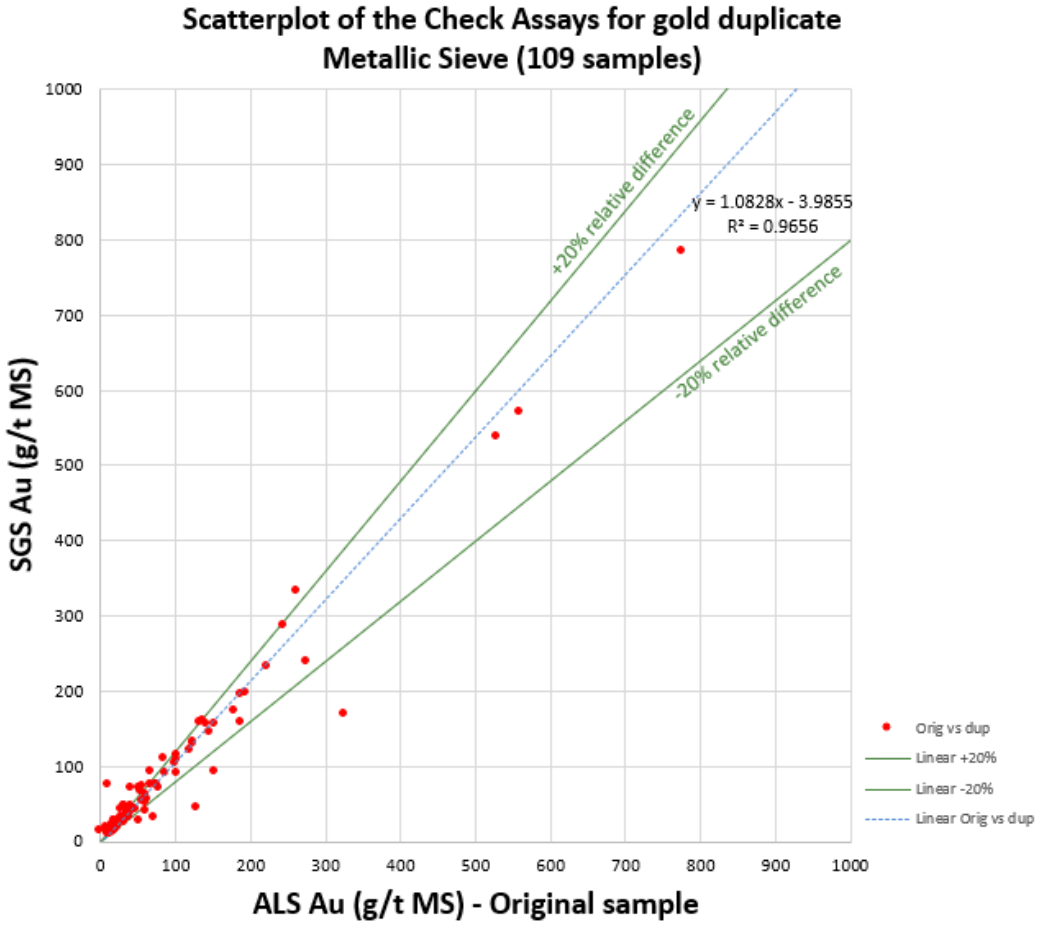


Figure 11-5: Scatterplot showing results from the check assay program for the metallic sieve methodology

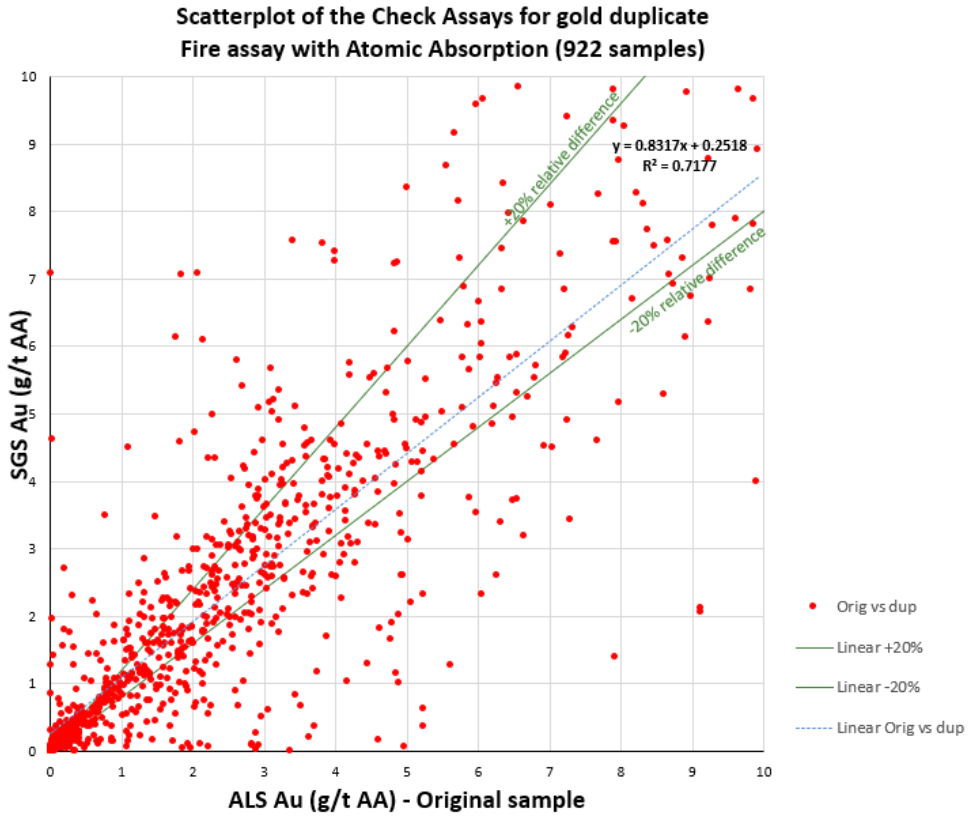


Figure 11-6: Scatterplot showing results from the check assay program for the atomic absorption methodology
Note that Wesdome's protocol ensures that samples returning grades above the 3 g/t Au threshold using the atomic absorption methodology are re-assayed using the gravimetric finish methodology.

11.3 Conclusions

In 2020, Wesdome mandated Qualitica Consulting Inc.to assess the Quality Control procedures applied on the Kiena Mine Complex. They concluded, in their report by Jolette and Bloom (2020), that "Kiena maintains a quality control program that meets or exceeds industry standards. Sample preparation, security and analytical procedures are all industry-standard and produce analytical results for gold with accuracy and precision that is suitable for Mineral Resource estimation". Qualitica Consulting also made several recommendations to improve and streamline the current Quality Control and assay best practices.

The QP reviewed this report (Jolette and Bloom, 2020) and its recommendations as well as the sample preparation, analytical and security procedures, as well as insertion rates and performance of blanks, standards, duplicates, and check assays for the 2019-2020 drilling programs and concluded that the observed failure rates are within expected ranges and that no significant assay biases are present. According to the QP's opinion, the procedure and quality of the data are adequate to industry standards and support the 2020 Mineral Resource Estimate.

12. DATA VERIFICATION

The 2020 Mineral Resource Estimate (2020 MRE) in this Report is based on drill data from several eras of drilling on the Kiena Mine Complex that includes holes completed in the 1940s.

For the purpose of the 2020 MRE, BBA performed a basic verification on the resource database. All data was provided by Wesdome in a local mine coordinate system. The drill database close-out date for the resource estimate is September 18, 2020; data from 6,823 DDH (951,602.62 m) was incorporated in the resource estimate block model area. The last drillhole included in the database was #6704.

12.1 Site Visit

Charlotte Athurion, P. Geo., and Qualified Person of BBA, visited the Kiena Mine Complex on October 16, 2020 and November 17, 2020. The site visit included a visual inspection of diamond drill core, a tour of the core storage facility and discussions of the current geological interpretations with Wesdome geologists.

The site visit also included a review of sampling and assay procedures, QA/QC program, downhole survey methodologies, and the descriptions of lithologies, alteration and structures (Figure 12-1 and Figure 12-2). Charlotte Athurion, P. Geo., visited the Wesdome office several times during the last two years to support Wesdome's geologists with the geological interpretation, modelling and was part of the August to November 2020 bulk sampling program to support the Wesdome team.

Pierre-Luc Richard, P. Geo., and Qualified Person of BBA, visited the property from August 6 to 8, 2019 as part of the 2019 MRE mandate, and visited the site on a few other occasions during the course of the previous mandate to exchange ideas with the on-site geologists.

12.2 Sample Preparation, Analytical, QA/QC and Security Procedures

Wesdome procedures are described in Chapters 10 and 11 of the current Report. Discussions held with on-site geologists and technicians confirmed that the procedures were adequately applied.

Charlotte Athurion reviewed sections of the mineralized diamond drill core while visiting the Project (Figure 12-1). All diamond drill core boxes were labelled and properly stored. Sample tags were present in the boxes and it was possible to validate sample numbers and confirm the presence of mineralization in witness half-core samples from the mineralized zones.

The historical data used in the 2020 MRE was established before the implementation of the National Instrument NI 43-101. Little information is available about sample preparation, analytical, QA/QC or security procedures. Although it is reasonable to assume that the companies conducted their exploration activities in accordance with prevailing industry standards at the time, the QPs conducted validation and comparison between previous and current populations and concluded that the historical drillholes could be used in the 2020 mineral resource estimate.

Figure 12-1: Core review in the core logging facility during the site visits
A) Core logging facility with Wesdome's employee describing the core;
B) Core box close-up showing a mineralized interval with associated sample tag

Figure 12-2: Sampling procedures review during the site visits
A) Core saw used to sample the core; B) Standards used for QA/QC;
C) Sample batches ready to be sent to the laboratory; D) Core boxes after being sawn and sampled

12.2.1 Drillhole Location

For drilling conducted between 2019 and 2020, drill collars have been surveyed using a Leica TS16. The collar data was verified for location discrepancies. For the historical data, approximately 10% of the drillholes intersecting mineralized zones were previously checked by BBA against the historical paper logs (Corkal et al., 2020).

12.2.2 Downhole Survey

Downhole survey data for the 2019-2020 drilling programs were checked for discrepancies. Spurious measurements are tagged by the Wesdome geologist as “false” in the database and are not considered by the software for the modelling. For the historical data, surveys of approximately 10% of the drillholes intersecting mineralized zones were previously checked by BBA against the historical paper logs at the Kiena Mine Complex (Corkal et al., 2020).

12.2.3 Assays

Clovis Auger, P. Geo., from BBA, was granted access to the original assay certificates directly from ALS for all holes drilled by Wesdome (2019-2020 Program) on the Project. The original logs and assay certificates for the historical DDH were also available at the Kiena Mine Complex office. All assays from ALS were compiled and verified for the 2019-2020 Program. Assays for approximately 10% of the historical DDH intersecting the mineralized zones were also verified previously by BBA against historical logs (Corkal et al., 2020). The assays recorded in the database were compared to the original certificates from the different laboratories. Minor discrepancies were noted and modified for the historical DDH.

In the assay table, the gravimetric finish result always replaces a value obtained by AA finish and when a sample was assayed using the metallic sieve procedure, the value recorded as “Au (g/t AVG)” always corresponds to the gold value obtained by metallic sieve method. Values lower than the detection limits were set to zero (0). With AA and gravimetric finish, when a sample has a reject duplicate, the average of the assay results is used in the Wesdome database.

12.3 Underground Voids

Underground workings were imported from the Wesdome working files. Previous MRE voids were validated in the 2018 MRE and the robustness of the 3D shapes were also validated. The underground voids include shafts, drifts, raises, stopes and the exploration ramp developed since 2017.

The QPs consider that the precision and details from the voids are acceptable and reliable for the current estimate. Uncertainties remain, especially concerning the stopes. In order to address the uncertainty, all block grades were initialized to zero within a 5-m buffer around existing stopes, using a clipping boundary. Uncertainties do not have a significant effect on the 2020 MRE.

12.4 Conclusion

The QPs are of the opinion that the drilling, sampling and assaying protocols in place are adequate. The database for the Kiena Mine Complex is of good overall quality. In the QPs opinion, the Project database is suitable for use in the estimation of mineral resources.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

This chapter summarizes metallurgical testwork carried out for the Project from 2018 to 2020. Section 13-1 presents the testwork performed for the preliminary economic assessment (PEA) in 2018 and 2019. In 2020, a new metallurgical testwork program, based on the PEA observations and the former operation, was designed and managed by Soutex (Section 13-2). The testwork was performed by Base Metallurgical Laboratories located in Kamloops, British-Columbia, between September and December 2020.

13.1 Historical Testwork

Metallurgical testwork was carried out for the Project in 2018 and 2019 by the Centre Technologique des Résidus Industriels (CTRI) in 2018 and by SGS Lakefield in 2019.

13.1.1 CTRI 2018 Metallurgical Tests

In 2018, Wesdome mandated CTRI (Noël, 2019) to perform preliminary gold leaching tests on mineralized material from the Kiena Deep A and S50 Zones. The objective of the testwork was to benchmark the leaching performance of the Kiena Deep A Zone to the previously mined S50 Zone. Fifteen 48-hour cyanidation tests in 4L bottles on gold mineralized material were performed. The mineralized material received included 57 sub-samples for a total of 92 kg that were divided to produce five composites: four for the Kiena Deep A Zone and one for the S50 Zone.

The 57 sub-samples were assayed for gold individually and a blended sample recipe was prepared. The resulting composite charges were assayed for gold and used for direct cyanidation for each composite.

Cyanidation tests were conducted for each composite in 4L bottles for 48 hours. The cyanidation parameters used were:

- Grind: 80% passing 75 µm;
- Cyanide: 500 g/t NaCN;
- Lead Nitrate: 75 g/t;
- pH: 11.3 (maintained with lime).

No air or oxygen was added during the tests and the lead nitrate was added in the bottle one hour prior to the addition of cyanide to simulate the real circuit. At the end of each test, the solid residue and the solution were collected and sent for gold assaying.

The gold recoveries for all cyanidation tests are shown in Table 13-1. The 48-hour recoveries for the Kiena Deep A Zone cyanidation tests ranged from 98.4% to 99.7%. The 48-hour recoveries for the S50 Zone gave a value of 95.7%, which is in the range of the historical data from the Kiena Mill.

Table 13-1: CTRI Cyanidation test results

Composites	NaCN	CaO	Au Tails	Recovery	Re-calc Head
	(kg/t)	(kg/t)	(g/t)	(%)	(g/t)
Kiena Deep A Zone – 1	0.15	1.22	0.19	98.8	16.4
Kiena Deep A Zone – 2	0.18	1.23	0.23	98.4	13.88
Kiena Deep A Zone – 3	0.11	1.14	0.02	99.7	7.12
Kiena Deep A Zone – 4	0.11	1.3	0.16	99.3	22.21
S50 Zone	0.21	1.56	0.12	95.7	2.83

Lime consumption for all cyanide tests ranged from 1.1 kg/t to 1.6 kg/t. The consumption of NaCN varied between 0.11 kg/t and 0.21 kg/t. The S50 Zone reagent consumption was higher than the Kiena Deep A Zone probably due to the presence of sulphur (pyrite) in the S50 Zone.

13.2 BBA 2019 Metallurgical Testwork

In July 2019, BBA was selected by Wesdome to design and manage the 2019 testwork program for the Kiena Mine Complex PEA project. The testwork started in August 2019 and ended in November 2019. The metallurgical testwork was performed at SGS Lakefield (SGS, 2020). Sample selection and sample preparation planning were performed by BBA in order to prepare composites for the Kiena Deep A Zone, the A1 Zone and the A2 Zone.

13.2.1 Comminution Tests

The comminution test results are summarized in Table 13-2. The SMC and BWi values showed that the three zone samples are moderately soft and the BWi test suggested that the samples are moderately soft even at finer sizes. According to SGS, the Bond Abrasion Test results are in the range of slightly abrasive to moderately abrasive.

Table 13-2: Comminution test results

Sample Name	Relative Density	SMC Parameters			Work Indices (kWh/t)			Ai
		A x b	t_a	SCSE	CWi	RWi	BWi	(g)
A	2.82	54	0.5	8.8	9.5	13.2	12.5	0.181
A1	2.89	55.2	0.49	8.9	N/A	12.2	12	0.207
A2	2.84	52.3	0.48	9	N/A	13	12.7	0.213

13.2.2 Gravity and Cyanidation Tests

Whole ore leaching (WOL) tests were conducted on the three master composites: A, A1 and A2. Common conditions for leaching testwork were:

- Grind: 80% passing 75 µm;
- Cyanide: 2.4 kg/t NaCN (1.5 g/L maintained);
- pH: 11.3 (maintained with lime);
- DO: 8-10 mg/L (using air);
- Leaching time: 48 hours;
- Pulp density: 40%.

Cyanidation tests were conducted in 2L bottles for each composite. Due to higher gold grade and the presence of coarse gold in the composites, it was decided to use a higher cyanide concentration (2.4 kg/t NaCN) than the one used historically at the Kiena Mill (0.5 kg/t NaCN). Slurry was aerated with air at the beginning of the testwork including preconditioning and lead nitrate was added at the beginning of the preconditioning. The cyanide and pH were monitored at regular intervals to maintain the proper conditions.

The WOL test results summary is shown in Table 13-3. WOL recoveries for Composites A and A2 are 99% with and without the preconditioning step. The WOL recovery for Composite A1 without preconditioning is 95% and with preconditioning is 100% (approximately). All samples consumed 20-25% of the added NaCN.

Table 13-3: WOL Test results of all three master composites

Test ID	Lead nitrate addition (g/t)	Reagent consumption		Au extraction, including mill clean out, %					Head average (g/t) (direct and calculated)
		NaCN (kg/t)	CaO (kg/t)	2h	6h	24h	48h	Normalized % Au extraction	
A-FWOL.A	250	0.49	0.93	48	89	96	99	99	19.6
A-FWOL.B	0	0.51	0.78	25	54	90	99	99	
A1-FWOL.A	250	0.47	0.84	40	69	100	100	100	11.8
A1-FWOL.B	0	0.6	0.72	23	45	82	95	95	
A2-FWOL.A	250	0.45	0.89	59	86	99	99	99	16.9
A2-FWOL.B	0	0.5	0.76	27	61	88	99	99	

Gold recoveries for samples A and A2 with preconditioning reached 85-90% in approximately six hours. More than 80% of the gold was recovered in less than 25 hours. Pre-conditioned samples reached recoveries above 95% in 25 hours while non-pre-conditioned samples reached recoveries between 80% and 90% in 25 hours.

Knelson/Mozley gravity test results are shown in Table 13-4. The tests resulted in 58.9%-66.7% gold recovery for the three samples.

Table 13-4: Knelson/Mozley gravity test results

Sample name	Grind size, P_{80}, µm	Gravity concentrate			Tailings assay Au, g/t	Head (calc) Au, g/t	Head (direct) Au, g/t
		Wt %	Assay Au, g/t	Au Rec. %			
A	251	0.1	16,044	60.7	7.8	19.9	21.3
A1	254	0.1	9,374	66.7	2.9	10.7	10.8
A2	246	0.1	11,717	58.9	6.7	16.4	17.8

Gravity tails leaching (GTL) was conducted on the three master composites: A, A1 and A2. The leach testwork conditions were the same as the WOL tests except for the cyanide addition which was 1.5 kg/t. The cyanidation tests were conducted in 2L bottles. The gravity tails leaching test results summary is shown in Table 13-5.

Table 13-5: Gravity tails leaching test results

Zone	Test ID	P_{80} (µm)	Reagent consumption		Au extraction, including mill clean out, %					Head average (g/t) direct and calculated
			NaCN (kg/t)	CaO (kg/t)	2h	6h	24h	48h	Normalized % Au extraction	
A	FGT.101.A	73	0.20	0.89	57	90	~99	98.2	98.3	7.82
	CGT.201.A	98	0.22	0.84	55	87	98	96.8	97.1	
	FGT.102.B	73	0.21	0.85	38	77	~99	97.9	98.1	
	CGT.202.B	98	0.18	0.71	35	75	98	96.5	96.7	
	FGT.103.A	73	0.31	0.88	63	96	~99	97.5	97.8	
	CGT.203.A	98	0.33	0.91	56	91	~99	96.6	96.9	
	FGT.104.A	73	0.23	0.93	53	89	~99	97.8	98.1	
	CGT.204.A	98	0.18	0.95	44	79	98	96	96.2	

Zone	Test ID	P_{80} (µm)	Reagent consumption		Au extraction, including mill clean out, %					Head average (g/t) direct and calculated
			NaCN (kg/t)	CaO (kg/t)	2h	6h	24h	48h	Normalized % Au extraction	
A1	FGT.101.A	75	0.2	0.84	55	85	98	96.5	97.1	3.8
	CGT.201.A	98	0.35	0.82	52	83	98	95.3	95.8	
	FGT.102.B	75	0.37	0.81	45	78	~99	96.2	96.8	
	CGT.202.B	98	0.32	0.79	41	71	98	94.9	95.5	
	FGT.103.A	75	0.32	0.89	57	87	~99	96.5	97.1	
	CGT.203.A	98	0.31	0.8	51	78	91	95.3	95.8	
	FGT.104.A	75	0.19	0.92	50	82	98	96.6	97.4	
	CGT.204.A	98	0.28	0.86	41	75	98	95.4	96.1	
A2	FGT.101.A	70	0.44	0.92	76	94	~99	97.6	97.6	6.63
	CGT.201.A	106	0.33	0.88	72	88	95	95.7	95.5	
	FGT.102.B	70	0.45	0.87	39	67	97	97.8	97.7	
	CGT.202.B	106	0.39	0.83	37	67	95	95.5	95.2	
	FGT.103.A	70	0.35	0.79	84	96	99	97.4	97.4	
	CGT.203.A	106	0.36	0.88	77	92	~99	96.7	96.5	
	FGT.104.A	70	0.24	1.03	35	66	96	95.1	96.2	
	CGT.204.A	106	0.24	0.96	62	88	97	96.3	96.2	

The results indicated that gravity concentration followed by GTL could yield extremely high gold recoveries. It also indicated that recoveries higher than 95% could be achieved with WOL.

WOL samples consumed more NaCN than GTL samples and, lime consumption was average for this type of mineralized material. However, a deeper analysis of the data shows that most of the cyanide consumption for the WOL tests occurred during the second half of the cyanidation while less than 10% of the gold was dissolved and no more cyanide was added. An example of this observation is shown on Figure 13-1.

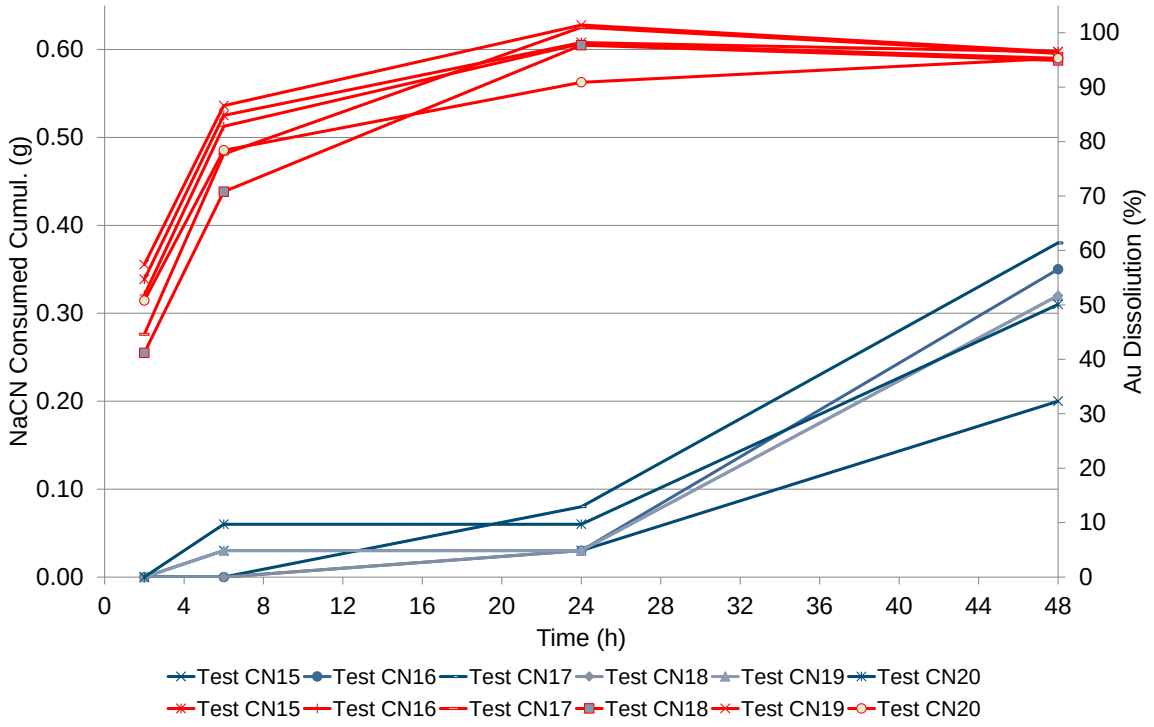


Figure 13-1: NaCN Consumption (blue) and au dissolution (red) for A1 Zone tests at 1.0 g/l NaCN

During this testing phase, the cyanidation of gravity tails was not conducted immediately after the gravity process so the samples were subjected to many hours of natural oxidation in the slurry which can explain a lower cyanide consumption. Furthermore, the WOL tests were performed at a higher cyanide concentration and air addition that could have promoted cyanide evaporation. As a result, it cannot be concluded that the absence of a gravity circuit would result in an increase in cyanide consumption.

13.2.3 Solid and Liquid Separation Tests

Tailings thickening tests required a dosage of 9 g/t and 12 g/t of flocculent for composite A and composites A1 and A2 respectively. These dosages were found to be relatively low when they were benchmarked with similar mineralized material types.

13.2.4 Conclusions

The main conclusions drawn from the PEA metallurgical testwork were:

- The comminution test results have shown that all samples were moderately soft and in the range of slightly abrasive to moderately abrasive;
- The WOL tests resulted in gold recoveries of between 95.7% and 99.7% while GTL tests resulted in gold recoveries of between 99% and 99.3%;
- Pre-conditioning lead nitrate addition was found to have a minimal effect on the overall gold recoveries, but it improved the leach kinetics in both the WOL and GTL tests;
- The majority of WOL test samples reached their maximum gold recoveries in approximately 48 hours whereas the majority of the GTL test samples reached their maximum gold recoveries in approximately 24 hours;
- It cannot be concluded that the absence of a gravity circuit would result in an increase in cyanide consumption since WOL and GTL tests were not performed at the same cyanide concentrations and GTL samples were subjected to many hours of natural oxidation in the slurry after gravity separation;
- Gold recoveries of between 59% and 67% were achieved in Knelson/Mozley gravity tests indicating that a gravity circuit would not significantly increase the overall gold recovery but would increase the kinetics and reduce the variability in the leaching circuit;
- Slightly increased gold recovery was achieved by finer grinding (75 microns) and a higher amount of NaCN addition;
- NaCN consumptions in WOL and GTL tests were less than 0.6 kg/t, which is considered low. However, in GTL, samples consumed 26% to 54% less cyanide than the amounts consumed in WOL. However, the initial CN addition was not the same for WOL and GTL tests and the majority of the CN was consumed in the last 24 hours, when most of the gold was already leached;
- SNF Flomin 913 VHM was found to be the most effective and efficient flocculant among the flocculants tested. The dosage range of 9-12 g/t was found to be relatively low.

13.3 Current Testwork

Following the PEA, a metallurgical testwork program was developed to further characterize the ore from the Kiena Deep deposit and VC Zone in a context of a prefeasibility study (PFS). The objectives of the PFS are to develop the process design criteria and assess the possibility of processing the material from the Kiena Deep deposit in the Kiena concentrator. S50 and Martin zones were added to the resources late in the PFS so they could not be tested. However, S50 Zone was substantially mined in the past therefore its metallurgical characteristics are well known. S50 and Martin will represent 11.5% of the total run of mine and 3.9% of the total gold ounces. It is therefore not considered as a significant risk.

13.3.1 Composite and Sample Selection

The testwork was realized on composite samples made from drill cores. The number of composite samples was determined according to the homogeneity of the material within the deposit.

The material was located in four zones, which were constituting the mine plan at that time. The drill core sections were selected to have material dispersed evenly in the whole mass of each zone and to have a total mass sufficient for metallurgical testing requirements. Once the core sections were chosen, a length from the section located in the zone was selected making sure that the grades remained within the expected range.

Four composite samples were used for the metallurgical testwork program called A Zone, A1 Zone, A2 Zone and VC Zone along with five variability samples (A Zone 6388, A Zone 6428A, A1 Zone 6502, A2 Zone 6383 and A2 Zone 6580 BW2). Considering the volume of ore and the short period of the mine life, the number of composites is considered sufficient to evaluate the metallurgical performance. The different core sections' cut grade (cut value at 90 g/t) and length used to form the composite samples for the study are listed below in Table 13-6.

Table 13-6: Sample selection for the PFS

Zone	Hole ID	From	To	Au grade	Composite samples	Variability samples
		m	m	g/t	kg	kg
A	6559	67.6	76.3	56.24	10.1	0
	6388	110.8	122.3	12.02	13.0	10
	6550	302.1	309.6	20.88	8.7	0
	6660	285.15	290.3	52	6.0	0
	6291	320.5	327	17.5	7.5	0
	6428A	173.4	185.7	5.39	14.6	10
A1	6465	78.4	83.9	64.17	5.2	0
	6606	128.9	139.8	30.45	10.4	0
	6431	192.9	200.5	9.8	7.3	0
	6502	221.8	229.7	24.9	5.8	10
	6540	244.1	249.3	10.48	5.0	0
	6509	426	434	22.41	7.6	0
	6581	632	639.1	25.85	6.8	0
A2	6450	93.5	97.8	128.25	4.2	0
	6555	703.1	712.5	17.89	9.1	0
	6601	486.6	496.7	24.06	9.8	0
	6383	553	577	28.01	13.6	10
	6580BW2	332.7	342.4	86.44	9.4	10
VC	6201	265.1	271.2	6.61	12.2	0
	6523	423.9	428.6	23.99	9.4	0
	6586	451.6	457.6	12.72	12.0	0
	6556	484.7	490.7	3.67	12.0	0

Each composite sample was analyzed and characterized with multiple tests listed below:

- Head assays and mineralogical characterization;
- Comminution tests;
- Gravity tests;
- Leaching tests;
- Cyanide destruction tests;
- Sedimentation and filtration tests.

The following sections of this report present a more in-depth view of the different types of tests and their respective results.

13.3.2 Head Assays and Mineralogical Characterization

In order to determine the major mineral components of the samples and the possible deleterious elements present in them, mineralogical characterization and analytical assays were done.

The following mineralogical characterization was done:

- Bulk Mineralogical Analysis (BMA) at 106 µm.

The following analytical assays were done:

- Full ICP-MS scan;
- Total Organic Carbon, Sulphur (S2-) assay by Leco;
- Mercury (Hg) assay by Cold Vapour Atomic Absorption Spectrometry (CVAAS);
- Satmagan for magnetic susceptibility;
- Metallic screens for gold grade;
- Silver grade.

13.3.2.1 Mineralogical Characterization

Bulk Mineralogical Analysis (BMA) at 106 µm was done to define the mineral composition of the composite samples. The results are presented in Table 13-7.

Table 13-7: Mineral composition (%) of A, A1, A2 and VC composite and variability samples

Minerals	Mineral compositions (wt. %)			
	A Zone	A1 Zone	A2 Zone	VC Zone
Chalcopyrite	<0.1	<0.1	<0.1	<0.1
Pentlandite (1)	<0.1	<0.1	0.1	<0.1
Sphalerite/Galena	<0.1	<0.1	<0.1	<0.1
Pyrrhotite	0.1	0.3	0.4	0.2
Pyrite	0.2	0.1	0.1	0.4
Iron Oxides (2)	0.3	0.1	0.3	0.5
Ilmenite	0.6	0.8	0.7	0.6
Quartz	14.9	14.7	25.8	12.7
Chlorite	26.7	23.7	19.7	28.5
Amphibole (Actinolite)	3.9	20.2	16.8	5.2
Plagioclase Feldspar	7.7	7.3	12.6	14.1
Talc	16.7	14.7	9.2	10.8
Dolomite/Ankerite	25.1	14.2	10.3	21.3
Calcite	2.2	2.1	3.0	4.4

Minerals	Mineral compositions (wt. %)			
	A Zone	A1 Zone	A2 Zone	VC Zone
Andradite	0.6	0.2	0.1	0.2
Biotite/Phlogopite	0.3	0.1	0.4	0.2
Akimotoite	0.2	0.3	0.2	0.3
Pyroaurite/Sjogrenite	<0.1	0.1	0.1	0.1
Pyroxene (Augite/Diopside)	0.1	0.2	0.1	<0.1
Muscovite/Illite	0.1	<0.1	<0.1	<0.1
Epidote	<0.1	<0.1	<0.1	<0.1
Ti Minerals	0.1	0.1	<0.1	0.1
Apatite	0.1	0.3	0.1	0.2
Zircon	<0.1	<0.1	<0.1	<0.1
Others (3)	0.3	0.4	0.2	0.3

Notes:

(1) "Pentlandite" includes trace amounts of Millerite and Polydimite;

(2) "Iron Oxides" includes Hematite, Magnetite, Steel/Pure Iron, Goethite and Limonite;

(3) "Others" includes trace amounts of Barite, Diaspore/Corundum, Sb Mineral and unresolved mineral species.

The sulphur repartition according to the sulphur-bearing mineral of the samples are presented in Table 13-8.

Table 13-8: % Sulphur bearing mineral of total sulphur of the composite samples

Mineral	Sulphur of total sulphur (%)			
	A Zone	A1 Zone	A2 Zone	VC Zone
Chalcopyrite	2.2	7.6	3.0	1.0
Pentlandite	3.1	3.4	10.2	0.9
Sphalerite/Galena	1.2	0.2	0.2	0.2
Pyrrhotite	21.5	66.5	64.3	30.3
Pyrite	71.9	22.2	22.3	67.7
Barite	<0.1	0.1	<0.1	<0.1

Pyrite and pyrrhotite were identified as the predominant sulphur bearing minerals. The analysis on sulphides composition indicates the presence of pentlandite and chalcopyrite. Quartz, chlorite, amphibole, talc, feldspars, and dolomite were the predominant non-sulphide gangue minerals and varied quite significantly between samples.

13.3.2.2 Analytical Assays

A full ICP-MS scan was conducted to measure the elements composition of the composite and plant samples. The elements composition determined from QEMSCAN and the chemical assay are summarized in Table 13-9.

Table 13-9: Chemical composition (%) of the A, A1, A2 and VC composite and variability samples

Sample ID	Assay						
	Au - SFA	Au	Ag	S	Ni	Cu	Zn
	g/t	g/t	g/t	%	g/t	g/t	g/t
A Zone	15.1	15.30	1.5	0.15	314	87	40
A1 Zone	40.0	20.30	1.9	0.15	283	69	35
A2 Zone	11.6	14.80	1.5	0.22	223	48	28
VC Zone	19.8	9.18	1.5	0.25	246	67	39
A Zone 6388	2.99	3.47	1.2	0.79	77	199	52
A Zone 6428A	1.42	4.09	0.6	0.18	542	39	39
A1 Zone 6502	10.9	11.00	1.7	0.14	349	41	38
A2 Zone 6383	30.0	33.10	1.7	0.19	304	53	34
A2 Zone 6580BW2	41.9	37.10	5.8	0.3	129	48	25

The Total Organic Carbon (TOC), Sulphur (S2-) assay by Leco and Gold grade are presented in Table 13-10.

Table 13-10: Total organic carbon, sulphur (S2-) and gold grade of A, A1, A2 and VC composite and variability samples

Sample ID	Element			
	Au - FA	S	C	TOC
	g/t	%	%	%
A Zone	15.30	0.149	2.74	0.0366
A1 Zone	20.30	0.148	1.69	0.0279
A2 Zone	14.80	0.223	1.50	0.0256
VC Zone	9.18	0.253	2.60	0.0252
A Zone 6388	3.47	0.786	0.78	0.0278
A Zone 6428A	4.09	0.179	3.21	0.0202
A1 Zone 6502	11.00	0.139	3.13	0.0328
A2 Zone 6383	33.10	0.189	1.55	0.0285
A2 Zone 6580BW2	37.10	0.295	1.13	0.0262

Metallic screens for gold grade measurements were conducted on the composite and variability samples. The results are presented in Table 13-11.

Table 13-11: Metallic screen for gold grade of the A, A1, A2 and VC composite and variability samples

Sample	Head portion	Au assay (g/t)	Dist. Au (%)
A Zone	+100 mesh	56.10	17.6
	-100 mesh	13.05	82.4
	Total	**15.10**	**100.0**
A1 Zone	+100 mesh	335.00	39.7
	-100 mesh	25.35	60.3
	Total	**40.00**	**100.0**
A2 Zone	+100 mesh	36.20	14.8
	-100 mesh	10.40	85.2
	Total	**11.60**	**100.0**
VC Zone	+100 mesh	158.00	32.0
	-100 mesh	14.00	68.0
	Total	**19.80**	**100.0**
A Zone 6388	+100 mesh	14.00	19.1
	-100 mesh	2.52	80.9
	Total	**2.99**	**100.0**
A Zone 6428A	+100 mesh	0.85	2.8
	-100 mesh	1.45	97.2
	Total	**1.42**	**100.0**
A1 Zone 6502	+100 mesh	137.00	47.7
	-100 mesh	5.91	52.3
	Total	**10.90**	**100.0**
A2 Zone 6383	+100 mesh	209.00	29.1
	-100 mesh	22.25	70.9
	Total	**30.00**	**100.0**
A2 Zone 6580BW2	+100 mesh	262.00	28.3
	-100 mesh	31.55	71.7
	Total	**41.90**	**100.0**

Clay XRD measurements were conducted on the A, A1 and A2 composites. The results are presented in Table 13-12. These tests were performed as part of the sedimentation and filtration testwork. VC Zone was not included as it is marginal compared to Kiena Deep.

Table 13-12: Semi-quantitative X-ray diffraction results for the A, A1 and A2 composites

Sample	Fraction	Crystalline mineral assembly (relative proportions based on peak height)			
		Major	Moderate	Minor	Trace (1)
		(>30%)	(10% -30%)	(2% -10%)	(<2%)
A Zone	Bulk	-	chlorite, talc, dolomite, quartz, plagioclase	amphibole	calcite, ilmenite
	Clay	talc	chlorite	-	-
A1 Zone	Bulk	talc	chlorite, amphibole	dolomite, quartz, plagioclase	calcite, ilmenite
	Clay	talc	chlorite	-	-
A2 Zone	Bulk	-	chlorite, quartz, talc, amphibole, plagioclase	dolomite	calcite, ilmenite
	Clay	-	talc	chlorite	-

Note

(1) Tentative identification due to low concentrations, diffraction line overlap or poor crystallinity

Silver levels between 0.6 g/t and 5.8 g/t were measured in the composites. Low levels of total organic carbon were also measured. Screened metallic assays for gold indicated the presence of coarse “free” gold particles in the composites, and gold content measured between 1.4 g/t to 41.9 g/t. Nickel, copper, and zinc were present in varying minor amounts and could impact cyanide consumption and downstream processes, such as cyanide destruction. No other deleterious element, mineral or potential issue was identified according to these tests.

13.3.3 Comminution Testwork

Comminution tests were conducted to the composite samples (A, A1, A2 and VC) to determine ore hardness. The tests performed were SMC, and Bond ball mill work index. A relative work index was also estimated for the variability samples, using the grind calibration for each sample, and the work indices for the known samples. See Table 13-13 below for the results.

SMC tests resulted in A x b values of 52 to 81, which is on the soft end of the JK Tech database, between the 14th and the 33rd hardness percentile. Likewise, Bond Ball mill work indexes varied between 10.7 kWh/t and 12.8 kWh/t at a closing screen size of 106 µm, which would be considered soft in ball mill grinding duty. Relative work index estimates on the other samples indicate there is a variation in hardness, with one sample estimated at 22 kWh/t.

Table 13-13: Comminution test results of the A, A1, A2 and VC composite and variability samples

Sample ID	DWi	DWi	SCSE	A x b	BWi	Relative Wi (estimate)
	kWh/m³	%	kWh/t		kWh/t	kWh/t
A Zone Composite	5.18	31	8.79	54.327	10.68	-
A1 Zone Composite	4.41	22	8.20	64.020	10.82	-
A2 Zone Composite	5.33	33	8.92	51.903	12.76	-
VC Composite	3.46	14	7.44	80.801	-	9.9
A Zone 6388	-	-	-	-	-	10.0
A Zone 6428A	-	-	-	-	-	8.8
A1 Zone 6502	-	-	-	-	-	12.5
A2 Zone 6383	-	-	-	-	-	12.9
A2 Zone 6580BW2	-	-	-	-	-	21.8

The composites are characterized as relatively soft from SAG milling and ball milling perspectives, with A x b values of between 51.9 to 80.8, and Bond work indexes of 10.68 kWh/t to 12.76 kWh/t, at a closing screen size of 106 µm. Comparative work indexes indicated a wide range of hardness for the variability samples.

13.3.4 Gravity and Leaching Testwork

Leaching tests were carried out for the four composite samples (A, A1, A2 and VC) and the variability samples to determine the gold recovery from the leaching process. Two grind sizes were tested (80 µm and 120 µm) as well as other different operating conditions to study the effects on the gold recoveries. Both the WOL and GTL tests were performed. From the initial stage of optimization testing, the WOL at a grind of 80 µm with 75 g/t lead nitrate addition was chosen as the final conditions. These conditions were used on the variability samples.

There was significant coarse, visible gold recovered to the gravity concentrate at low mass recovery. Gold recovery varied from 20 to 71 percent in gravity concentrate. No significant final gold recovery difference between WOL and GTL was observed on the Kiena Deep composite samples after 48 hours of leaching. This comparison is presented on Figure 13-2.

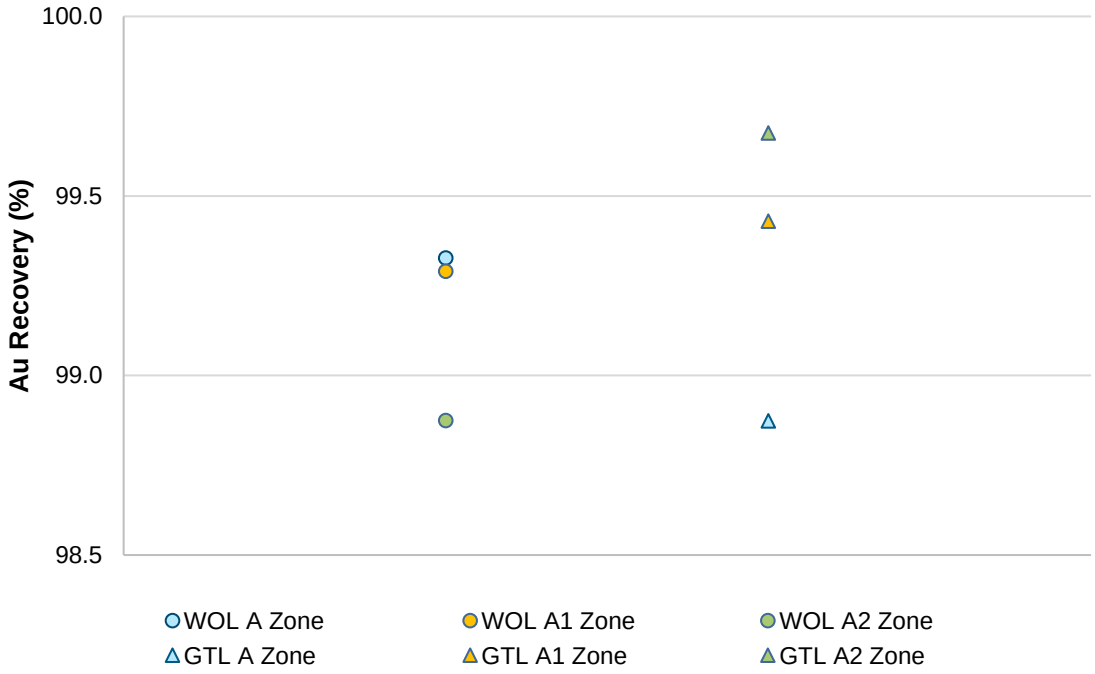


Figure 13-2: WOL and GTL gold recovery results for the Kiena Deep composites at 80 µm, 75 g/t $Pb(NO_3)_2$ addition

Lead nitrate was critical to improving leach kinetics, especially for the WOL tests, and in assisting leaching of any free coarse gold in the samples.

Lower cyanide consumption was obtained during the GTL tests and could be the result of sulphides' oxidation during the gravity recovery process and storage of the gravity tails. For the test performed at 80 µm, the gravity tails were reground while the tests at 120 µm were done directly on the material coming from the gravity tests. The cyanide consumption comparison is presented on Figure 13-3. The cyanide consumption observed is in line with the expected effect of oxidation of gravity tails material as the gravity concentrate that has been reground shows a cyanide consumption similar to WOL.

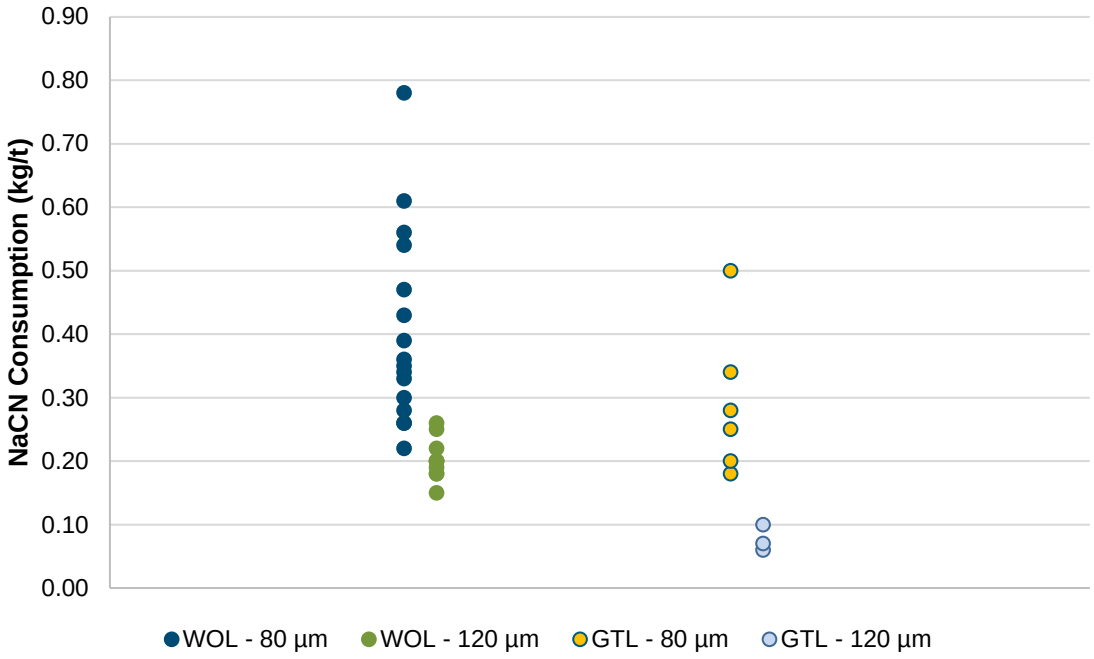


Figure 13-3: Cyanide consumption for the WOL and GTL tests at 80 and 120 µm

The sodium cyanide consumption ranged from 0.22 kg/t to 0.78 kg/t for the composite samples.

Overall, the majority of the gold was extracted after 24 hours, but an additional 1.2% to 29.7% gold extraction was measured between 24 and 48 hours. Tests without lead nitrate had significantly slower kinetics, as well as significant variance in triplicate tailings assays most likely due to the presence of large nuggets of native gold with low surface area not fully leaching due to slow kinetics. Tests at the finer primary grind of 80 µm had increased gold extraction of one to three percent for over 120 µm for the three composites. The effect of the lead nitrate addition is shown on Figure 13-4. The dotted lines show the range of the various tests. The effect of the grind size is shown on Figure 13-5.

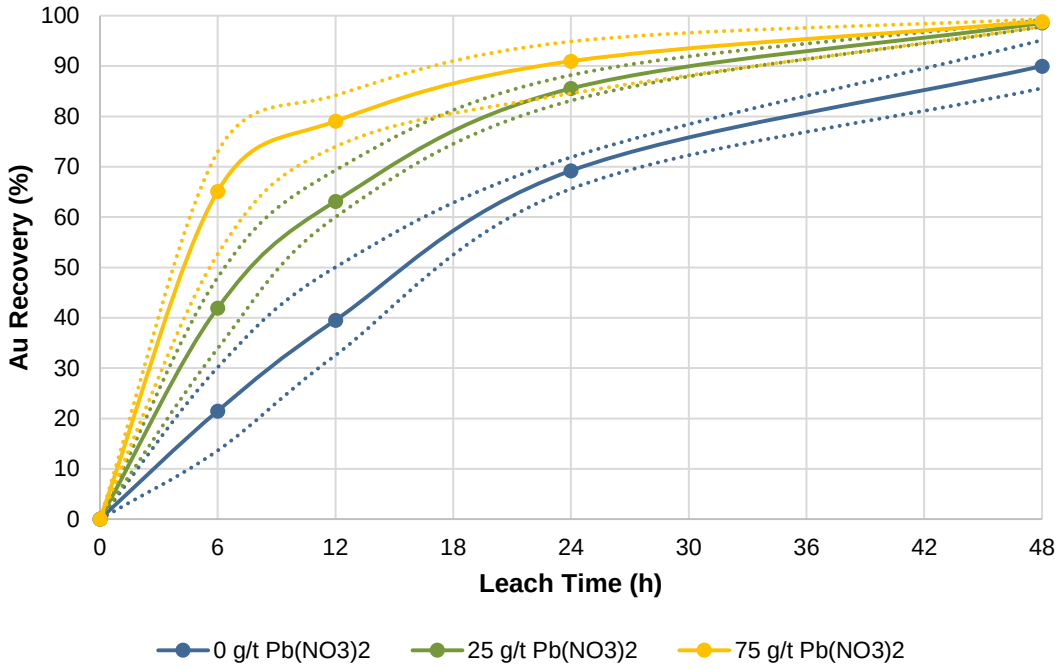


Figure 13-4: Effect of the lead nitrate addition on the leaching Kinetics for WOL tests on the Kiena Deep composite samples at 80 µm

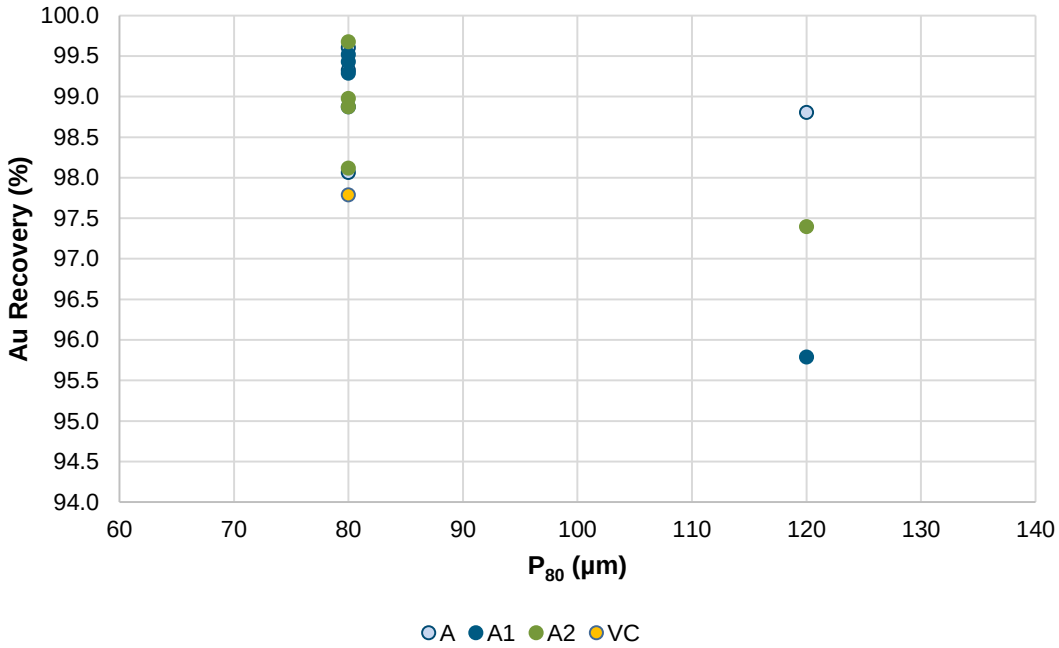


Figure 13-5: Effect of the grind size on the final recovery for the Kiena Deep and VC composites and variability samples at 75 g/t lead nitrate addition

Figure 13-6 shows the gold recovery results for the WOL tests in the conditions of an estimated grind size of 80 µm, 500 ppm for the NaCN concentration, 75 g/t $Pb(NO_3)_2$ addition, and air addition. As a comparison, the GTL performed on A Zone composite for the same conditions is presented. These results were collected during the test numbers 22, 24, 26 and 39. The three composite samples from Kiena Deep (A, A1, A2) have very similar gold recovery trends. Their final gold recovery percentage varies from 98.9 to 99.3% after 48 hours. The VC Zone composite has a lower recovery percentage than the three Kiena Deep composites at 97.8%.

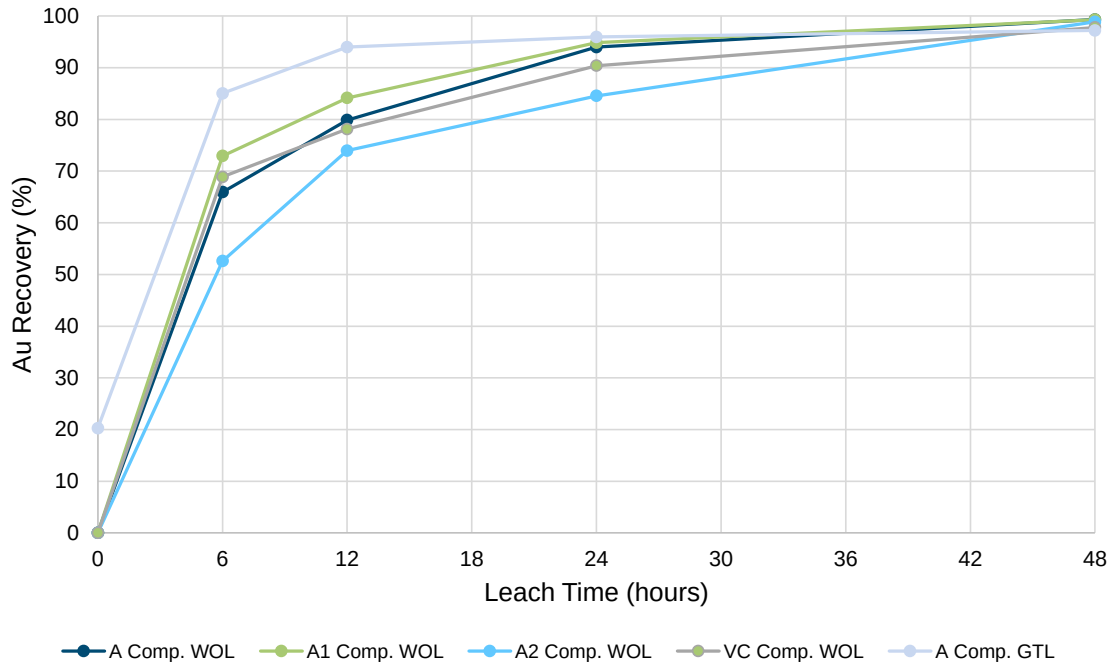


Figure 13-6: Gold recovery results for the Kiena Deep composites and VC composite at 80 µm, 75 g/t $Pb(NO_3)_2$ addition without gravity separation

13.3.5 Cyanide Destruction

The SO_2-air tests were performed on the Kiena Deep super composite of around 60 kg made of zones A, A1 and A2 material in similar proportions (1/3 each) to establish the reagents consumption. Table 13-14 shows the cyanide destruction feed solution composition and Table 13-15 shows the cyanide destruction test conditions and final CN_{WAD} concentration.

Table 13-14: Cyanide destruction feed solution composition

CN_{WAD}	CN total	Cu	Fe	Ni	Zn
ppm	ppm	ppm	ppm	ppm	ppm
205.23	218.23	3.81	4.65	2.05	1.285

Table 13-15: Cyanide destruction test conditions and final CN_{WAD} concentration

Test	SO_2:CN ratio	Cu addition	Retention time	Final CN_{WAD} concentration	Final metals concentration (ppm)			
	-	ppm	min	ppm	Cu	Fe	Ni	Zn
C1	5	50	60	2.0				
C2	6	25	60	2.7	0.32	0.03	1.03	<0.01
C3	6	50	60	2.1	0.50	0.01	0.90	<0.01
C4	7	50	60	1.8	0.52	0.14	0.75	<0.01
C5	7	50	90	1.6	0.29	0.01	0.68	<0.01
C6	5	10	60	2.9	0.02	0.07	1.24	<0.01

The detoxified solutions measured between 1.8 CN_{WAD} and 2.9 CN_{WAD} after 60 minutes. Even with conditions of 7:1 SO_2:CN ratio, 50 ppm Cu addition, and 90 min retention time, the final CN_{WAD} did not meet the 1 ppm target. The presence of nickel in the solutions influences the final solution CN_{WAD} levels. The majority of nickel in the effluent is in the form of cyanide complexes thus explaining the CN_{WAD} concentrations. However, the SO_2-air process is well known to have the capacity to reduce all CN_{WAD} species. Additional testwork is recommended to find the optimal reagents dosage that will eliminate the nickel cyanide. It is nonetheless expected that the reagents dosage will be in the same order of magnitude as the ones used for this testing thus the impact on operation costs will be low.

13.3.6 Sedimentation and Filtration Testwork

Dewatering tests were conducted on the cyanide destruction tails of the Kiena Deep super-composite. Two scenarios were tested, including thickening and pressure filtration on unclassified tailings, as well as on the fine fraction of tailings screened at 53 µm, simulating a cycloning step.

13.3.6.1 Sedimentation Testwork

Sedimentation tests were carried out on the Kiena Deep super-composite (A, A1, A2) with the presence of a flocculant. Tests were performed on the leach feed, the whole cyanide destruction tails and on classified cyanide destruction tails which were screened to -53 µm to simulate a cycloning step before thickening.

Table 13-16 below shows the results for pre-leach thickening tests. Table 13-17 shows the results for the whole cyanide destruction tails thickening tests and Table 13-18 shows the results for the classified cyanide destruction tails. The pre-leach test performed at a pH of 11 and flocculant dosage of 20 g/t gave the best results in terms of turbidity and underflow density. The whole cyanide destruction tails tests performed at a pH of 8.4 gave similar results for floc dosages of 20, 30 and 40. Finally, the <53 µm material had slower settling rates, requiring higher flocculant dosages, and lower final settled densities.

Table 13-16: Settling results for the Kiena Deep pre-leach super-composite

Test	pH	Flocculant dosage	Feed density	Settling rate	Final density	Turbidity
	-	g/t	%	mm/s	%	NTU
S1	10	0	13.2	0.17	59.9	383
S2	10	10	13.1	3.44	59.6	105
S3	10	20	13.4	4.31	56.8	49
S4	8.2	20	13.5	6.38	58.3	64
S5	11	20	13.3	3.94	57.7	4
S6	10	30	13.4	5.89	60.6	43
S7	11	30	13.3	4.53	48.2	6

Table 13-17: Settling results for the Kiena Deep whole cyanide destruction tails super-composite

Test	pH	Flocculant dosage	Feed density	Settling rate	Final density	Turbidity
	-	g/t	%	mm/s	%	NTU
S10	8.4	20	13.5	2.96	52.7	0
S11	8.4	30	13.7	3.15	50.6	0
S12	8.4	40	13.9	3.17	51.1	0
S13	8.4	40	14.0	3.29	54	0
S14	10	40	13.8	3.8	54.6	0
S15	11	40	14.2	3.55	53.7	0
S16	8.4	40	10.8	6.73	52.8	14
S17	8.4	40	15.0	4.07	51.6	9
S18	8.4	40	20.2	1.88	53.3	4

Table 13-18: Settling results for the Kiena Deep classified cyanide destruction tails super-composite

Test	pH	Flocculant dosage	Feed density	Settling rate	Final density	Turbidity
	-	g/t	%	mm/s	%	NTU
S19	8.4	20	14.0	0.95	48.1	19
S20	8.4	40	14.2	1.32	45.9	10
S21	8.4	60	14.0	2.49	44.7	7
S22	8.4	60	18.1	0.49	46.4	0
S23	8.4	60	9.6	2.93	45.4	7
S24	8.4	80	13.7	2.29	46	7

A flocculant dosage of 20 g/t gave the best results on the pre-leach sample while similar results were obtained for floc dosages between 20 and 40 g/t on the whole cyanide destruction tails. Higher flocculant dosages were required for the classified cyanide destruction tails resulting in lower final settled densities. The settling rates obtained from testwork are significantly higher than the expected rise rate within the thickeners.

13.3.6.2 Filtration Testwork

Filtration tests were carried out for the Kiena Deep super-composite in a Micronics MicroPress plate pressure filtration unit with a chamber diameter of 150 mm and varying chamber widths to adjust the cake thickness. One series of tests was done on the thickened whole cyanide destruction tails, one on the whole un-thickened cyanide destruction tails and the last series on the classified (-53 µm) un-thickened cyanide destruction tails. Figure 13-7 below shows the cake moisture evolution for the three series of tests.

The tests consisted of three phases:

- Cake formation, which consists in filling the unit with slurry using a diaphragm pump between 60 and 90 psi;
- Cake pressing, which consists in squeezing the chamber for 20-30 seconds at a pressure of 150 psi;
- Cake blowing, which consists in blowing compressed air through the chamber for various time periods.

The filtrate was collected and measured for each stage. The cake moisture could then be calculated.

The first cake moisture measurement was done after the cake formation. The thickened sample produced the lowest moisture content during cake formation followed by the un-thickened whole tailings and thickened classified tailings. Both the thickened whole tailings and the thickened classified tailings showed similar initial filtration rates but the classified tailings sample final moisture was higher. Well formed filter cake was generated for all tests even at a high cake thickness and showed good filtering properties. The final moisture content for the unclassified tailings was between 10% and 12% water for thickened cyanide destruction tails (P2), and between 12% and 16% water for un-thickened cyanide destruction tails (P1). Pressure filtration on the classified tailings produced final cakes of between 14% and 19% water (P3).

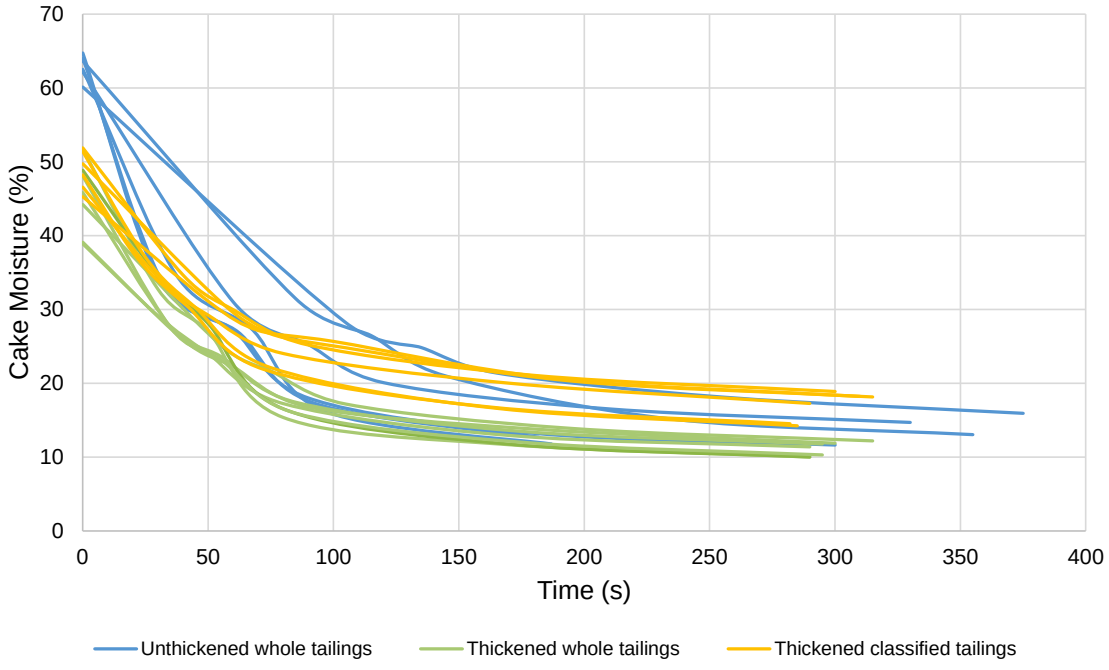


Figure 13-7: Filtration results for the Kiena Deep super-composite cyanide destruction tails

13.4 Bulk Sample Processing

A bulk sample of 7032 tonnes from the Kiena Deep Zone was processed in the Kiena mill under the supervision of Soutex in November and December 2020 over a period of 12 days, in order to validate the feed grades and metallurgical performances of the Kiena Deep ore using the equipment already in place in the mill. The metallurgical balance was not completed at the time of this study so the Au feed grade and recovery could not be confirmed. However, the following observations were made:

- At a feed rate of around 950 tpd and a P_{80} of around 67 µm, the combined SAG and ball mill power draw was in the 925 kW to 950 kW range, or 20 kWh/t to 24 kWh/t, which indicates a higher power consumption than what could be expected from the comminution testwork results. This is normal considering that the mill was running at a lower throughput than the design value;
- Final Au tails were between 0.15 g/t and 0.25 g/t most of the time with higher values from time to time, which are consistent with the cyanidation testwork;
- The final recovery has not yet been obtained since the analysis is in progress;
- The cyanide concentration was maintained at 500 ppm (NaCN) and the pH was controlled by lime titration that was between 500 ppm and 700 ppm (CaO). Lead nitrate addition was set at 75 g/t;
- The cyanide consumption was at 0.85 kg/t (NaCN), the lime consumption was at 1.28 kg/t (CaO), for cyanidation circuit only as there was no cyanide destruction, and the lead nitrate consumption was at 0.078 kg/t, which is coherent with metallurgical testwork;
- The grinding media consumption was at 0.23 kg/t for 125 mm balls (SAG) and 0.95 kg/t for 50 mm balls (Ball Mill);
- The 50 mm ball consumption could be considered accurate, but the 125 mm ball is lower than what could be expected since the ball charge of the SAG mill was higher than needed at the beginning of processing period.

13.5 Extraction of Metallurgical Data for Process Design Criteria

Since the Kiena concentrator is to process the material from the Kiena Deep deposit along with material from the S50 and Martin zones, the metallurgical data obtained from the metallurgical testwork program was used to develop the process design criteria and evaluate the Kiena process performance. The historical performances of S50 Zone were used to develop the process design criteria for S50 and Martin zones. This allowed to be conservative on Martin Zone's performances as S50 shows lower recoveries than Kiena Deep. The tests performed on the bulk sample were used to confirm the testwork performance in the plant.

The grindability test results and leaching test results were used to evaluate the performance of the grinding circuit and leaching circuit and the reagents consumption. The WOL and GTL test results were used to determine the need for a gravity circuit. The cyanide destruction tests were used to evaluate the reagent consumption. The sedimentation test results were used to evaluate the performance of the thickeners. The filtration test results were used to size the filtration plant and evaluate its performance.

The Kiena process design criteria and process are detailed in Chapter 17.

13.6 Testwork Interpretation

13.6.1 Head Assays and Mineralogical Characterization

Silver levels between 0.6 g/t and 5.8 g/t were measured in the composites, which did not represent issues in the leaching process. Low levels of total organic carbon were also measured, which did not cause preg-robbing of gold in the leaching process. Screened metallic assays for gold indicated the presence of coarse “free” gold particles in the composites, and gold content measured between 1.4 g/t to 41.9 g/t. A full ICP-MS scan, a Sulphur (S^{2-}) assay by Leco, as well as a Clay XRD were also completed on each composite. Nickel, copper, and zinc were also present in varying minor amounts and could impact cyanide consumption and downstream processes, such as cyanide destruction as it was observed with nickel in the cyanide destruction testwork. No other deleterious element, mineral or potential issue was identified according to these tests.

Bulk Mineralogical Analysis (BMA) at 106 µm using QEMSCAN were completed on the composites. Pyrite and Pyrrhotite were identified as the predominant sulphur bearing minerals. Pyrrhotite can be more reactive in cyanide leaching, leading to an increase in reagent consumption but at these levels, only a minor impact would likely be measured, and no impact was observed during the tests. The analysis on sulphides composition indicates the presence of pentlandite and chalcopyrite, which explains the presence of nickel and copper in the cyanidation effluent. No other potential issue was identified according to these test results.

Quartz, chlorite, amphibole, talc, feldspars, and dolomite were the predominant non-sulphide gangue minerals and varied quite significantly between samples.

13.6.2 Comminution Testwork

Comminution testing consisting of SMC and bond ball mill work index tests were completed on the composites. The composites are characterized as relatively soft from SAG milling and ball milling perspectives, with A x b values of between 52 to 81, and Bond work indexes of 10.7 kWh/t to 12.8 kWh/t, at a closing screen size of 106 µm. Comparative work indexes indicated a wide range of hardness for the variability samples. The comminution testwork results, along with the bulk sample processing indicate that the grinding installed power (1,000 hp for the SAG mill and 900 hp

for the ball mill) is sufficient for the hardness range identified in the testwork at the design throughput. Based on the specific power consumed by the grinding circuit being between 20 kWh/t and 24 kWh/t, the P_{80} of 67 µm and assuming a 90% power transmission to the pinion, the operating Bond work index value is between 15 kWh/t and 18 kWh/t.

13.6.3 Gravity and Leaching Testwork

Leaching tests were carried out for the four composite samples (A, A1, A2 and VC) and the variability samples. From the initial stage of optimization testing, WOL at a grind of 80µm with 75 g/t of lead nitrate addition was chosen as the final conditions. Under these conditions, the three composite samples from Kiena Deep (A, A1, A2) had very similar gold recovery trends with their final gold recovery percentage ranging from 98.9% to 99.3% after 48 hours. The VC Zone composite had a lower recovery percentage than the three Kiena Deep composites at 97.8%.

- Although gravity testing on the three main composites indicated coarse, free gold, WOL and GTL tests indicated there was no significant difference to final recovery with or without inclusion of gravity for a leach time of 48h but with higher kinetics that make dissolution almost complete after 24h for the GTL. In operation, coarse gold remains in the grinding circuit until it is small enough to report the cyclones overflow adding substantial leaching time that is not reflected in the testwork. As a result, gravity separation was not included in the process design;
- Lead nitrate was critical to improving leach kinetics, especially for the WOL tests, and in assisting leaching of any free coarse gold in the samples;
- Finer primary grinding from 120 µm to 80 µm increased gold extraction by about 1% to 3% for the three main composites;
- The sodium cyanide consumption ranged from 0.22 kg/t to 0.78 kg/t for the composite samples. The lime consumption ranged from 0.42 kg/t to 0.75 kg/t for the composite samples.

13.6.4 Cyanide Destruction

Cyanide Detox and dewatering tests were conducted on a Master composite constructed from equal parts of the A, A1, and A2 zones composites. Detox testing using SO_2/air consistently reached a residual CN_{WAD} of about 2 ppm to 3 ppm, higher than the target of 1 ppm. The cyanide destruction tests showed good copper elimination, but nickel concentrations remained high. The nickel in solution is complexed with cyanide thus is the CN_{WAD} species that has not been completely eliminated. Reagents optimization will be necessary to lower the nickel and CN_{WAD} in the cyanide destruction effluent. Additional testing is recommended, using SO_2:CN ratios around 8:1, to further refine the conditions required, as well as assess the variability in cyanide destruction requirements on various samples.

13.6.5 Sedimentation Testwork

Sedimentation tests were carried out on the Kiena Deep super-composite with the presence of a flocculant. Tests were performed on the leach feed, the whole cyanide destruction tails (P_{80} around 80 µm) and on classified cyanide destruction tails (P_{80} around 37 µm).

A flocculant dosage of 20 g/t gave the best results on the pre-leach sample while similar results were obtained for floc dosages between 20 g/t and 40 g/t on the whole cyanide destruction tails. Higher flocculant dosages were required for the classified cyanide destruction tails resulting in lower final settled densities. The settling rates obtained from testwork are significantly higher than the expected rise rate within the thickeners.

13.6.6 Filtration Testwork

All tests showed good filtering properties. The final moisture content for the unclassified tailings was between 10% and 12% water for thickened cyanide destruction tails, and between 12% and 16% water for un-thickened cyanide destruction tails. Pressure filtration on the classified tailings produced final cakes of between 14% and 19% moisture. Tailings filtration is not envisioned at the moment but should dry stacking be necessary in the future, tailings filtration is expected to perform well but air blowing duration will need to be optimized to meet the final moisture requirements.

13.7 Conclusions and Recommendations

The PFS metallurgical testwork was conducted for the Kiena Mine Complex Base Metallurgical Laboratories in Kamloops. The test program was performed using composite samples representing zones A, A1, A2 and VC. The following conclusions and recommendations can be drawn:

- The comminution test results have shown that all samples are moderately soft, which is supported by the observations made during the bulk sample processing;
- Since the grinding circuit has some inefficiencies at lower throughputs than its design capacity, the higher-power consumption values observed during the bulk sample processing must be used for the grinding power evaluation as the throughput of the grinding circuit will be in the same range as it was during the bulk sample processing;
- No significant difference in gold recovery was observed between GTL and WOL after 48 hours of leaching time. Furthermore, only minor cyanide consumption reductions, if any, could be achieved. By considering a leaching of 48 hours, the installation of a gravity circuit in the plant is not recommended;
- Higher gold recoveries were obtained at a finer grind size (80 µm vs 120 µm). For achieving the highest gold recoveries, it is suggested to keep the final grind size at around 80 µm although a finer grind size, as what was obtained during the bulk sample processing, is appropriate;

- The addition of lead nitrate has a significantly positive effect on leach kinetics; therefore, a dosage of 75 g/t is recommended;
- All the leach tests were performed at a NaCN addition of 500 ppm. It is suggested that the NaCN addition be maintained at around 500 ppm;
- The three composite samples from Kiena Deep (A, A1, A2) showed final gold recovery percentage ranging from 98.9% to 99.3% after 48 hours. The VC Zone composite had a lower recovery percentage than the three Kiena Deep composites at 97.8%;
- The cyanide destruction tests did not reach the target of 1 ppm CN_{WAD} in the tails, event at a SO_2 ratio of 7 SO_2:CN, most likely due to the presence of nickel. Additional testwork is recommended to determine the proper reagents addition requirements. A SO_2 ratio of 8 SO_2:CN and the addition of 50 ppm of copper as copper sulfate is recommended for the PFS along with additional testwork at these conditions;
- A flocculant dosage of 20 g/t gave the best results on the pre-leach sample while similar results were obtained for floc dosages between 20 and 40 g/t on the whole cyanide destruction tails. A dosage of 20 g/t to 25 g/t for each thickener is recommended;
- Martin Zone has not been subjected to testwork nor been processed in the past. It is possible that the recovery associated with this zone is lower than expected or that it contains certain minerals or has certain characteristics that are detrimental to the process. Exploitation of that zone is scheduled to start in 2024 so it is recommended, in the meantime, to perform mineralogical characterization and preliminary leaching tests on material from that zone. The results will require analysis by a metallurgist to determine if processing in the existing plant will give appropriate metallurgical performances and if further testwork is required;
- All tests showed good filtering properties. Should dry stacking be necessary in the future, tailings filtration is expected to perform well but air blowing duration should be the subject of future tests to meet the final moisture requirements.

14. MINERAL RESOURCE ESTIMATE

14.1 Introduction

In 2020, BBA was retained by Wesdome Gold Mines to review and audit the updated Mineral Resource Estimate (MRE) deemed the 2020 MRE for the Kiena Mine Complex project (the "Project"), which incorporates historical diamond drilling data and recent diamond drilling programs. The 2020 MRE presented herein was initially presented in a Technical Report entitled "Update of the Mineral Resource Estimate for the Kiena Mine Complex Project" dated January 28, 2021 (Athurion et al., 2021). It includes diamond drill data as of September 18, 2020.

14.2 Methodology

The herein MRE covers the whole Kiena Mine Complex project (Figure 14-1).

The geological wireframes were constructed in Leapfrog Geo™ by Charlotte Athurion of BBA, Clovis Auger of BBA, and Karine Brousseau of Wesdome with the help of Bruno Turcotte of Wesdome. Leapfrog Geo v 5.1.1 was also used to generate mineralized intercepts. Datamine Studio RM version 1.6.87 was used for the compositing, 3D block modelling, interpolation and classification. Statistical studies were undertaken using Excel and Snowden Supervisor v. 8.11.

The methodology for the estimation of the mineral resources involved the following steps:

- Database verification and validation;
- Drillhole intercept;
- Basic statistics and composite generation for each unit;
- Capping;
- Geostatistical analysis including variography;
- Block modelling and grade interpolation;
- Block model validation;
- Resource classification;
- Cut-off grade calculation;
- Reasonable prospects of eventual economic extraction assessment;
- Preparation of the mineral resource statement.

In order to be consistent across the Kiena Mine Complex, all previous polygonal mineral resource estimates for the zones outside of the Kiena Mine area prepared by Turcotte et al. (2015) were converted into block model mineral resource estimates for this 2020 MRE using the same methodology as the block model mineral resource estimate for the zones in the former Kiena Mine area.

Pierre-Luc Richard (QP) was involved in each of these steps and an extensive review of the Datamine Studio RM project was carried out by BBA's geologist, Charlotte Athurion (QP). Some key steps were conducted again for comparison and validation purposes. During these reviews, compositing and capping, block model coding, interpolation, classification and reporting process were validated.

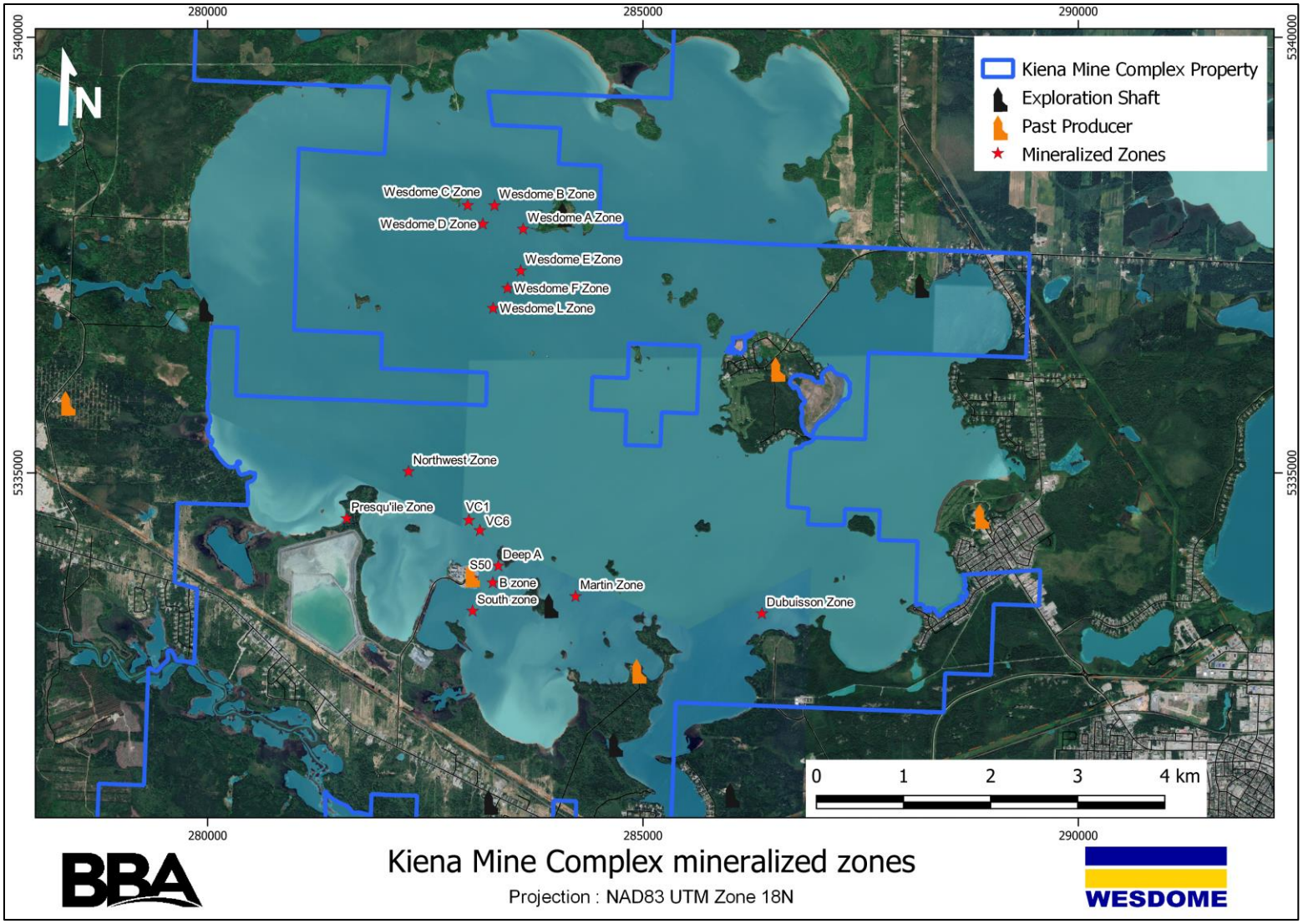


Figure 14-1: 2020 MRE block model resources location

14.3 Resource Database

The Project resource database, as of September 18, 2020, consisted of 350 surface collared and 6,473 underground collared diamond drillholes with a cumulative length of 951,602.62 m (Figure 14-2). The average length of a diamond drillhole (DDH) is 139.5 m. Of these 6,823 diamond drillholes, a subset of 1,145 diamond drillholes cut across the mineralized lenses.

Since the 2019 MRE (September 25, 2019), 213 diamond drillholes for a total of 60,865 m were added to the Project. Of which, 122 new diamond drillholes targeted the Kiena Deep Zone for a total of 35,280 m.

The resource estimation for the Project relies on historical and recent drilling programs. BBA included the historical drillhole information into the resource estimation for the following reasons: 1) historical information was validated as part of the mandate and no discrepancies were found; and 2) recent drillholes were drilled in the vicinity of historical drillholes and the results show comparable geology and mineralization outlines.

The resource database was validated before proceeding to the resource estimation. The validation steps are detailed in Chapter 12 of this Report. Minor variations have been noted during the validation process but have no material impact on the 2020 MRE.

The QPs are of the opinion that the database is appropriate for the purposes of the mineral resource estimation and that the sample density allows a reliable estimate to be made of the size, tonnage and grade of the mineralization in accordance with the level of confidence established by the mineral resource categories as set forth in the CIM Standards.

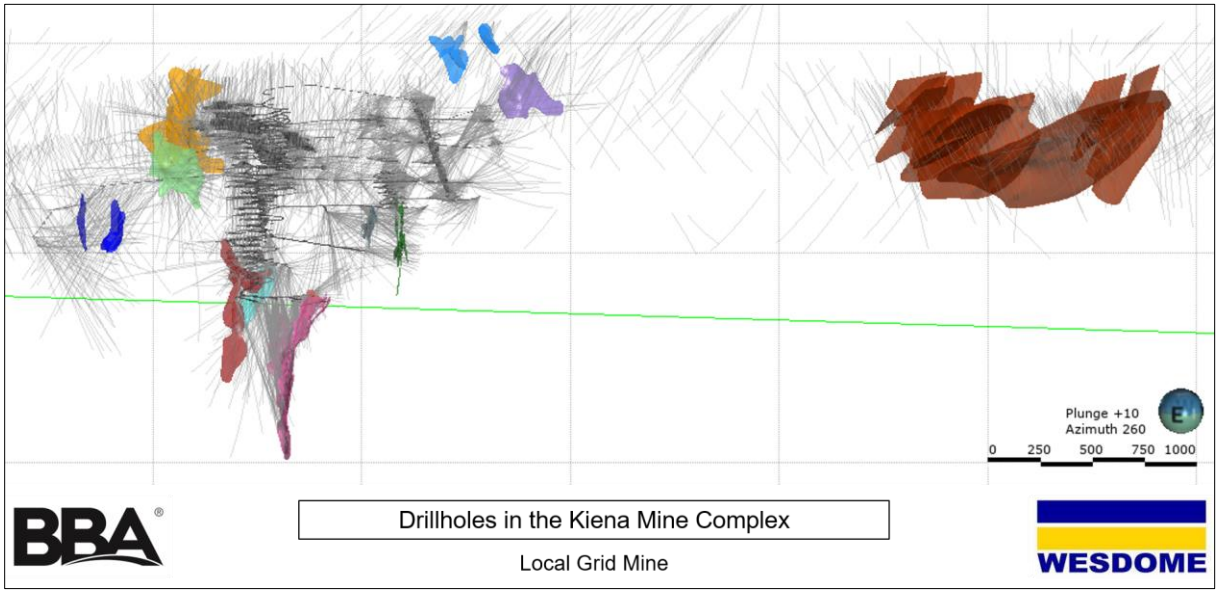


Figure 14-2: 3D view looking west of the 3D model and of the drillholes included in this resource estimate

Between April 12, 2021 and the drill database close-out date of September 18, 2020 used as basis for the 2020 MRE, Wesdome has completed an additional 143 DDH for 41,780.8 m that are not included in the 2020 MRE. Seven underground drill rigs have continued to operate to test the extensions of VC1 and VC6 Zones and perform infill drilling in the Kiena Deep A Zones as well as testing its down dip extension until the end of 2020. Since the beginning of 2021, underground drilling has refocused on expansion and exploration drilling. Seven underground drill rigs have continued to operate to test: 1) northwest extension of Martin Zone at 38 Level; 2) extensions of VC1 Zone at 79 Level; and 3) extensions of Kiena Deep A Zone.

The QPs are of the opinion that while the addition of these new holes would increase knowledge and confidence on the Project and have a positive impact on the resource estimate, it does not have a material impact on the current study presented in this Report.

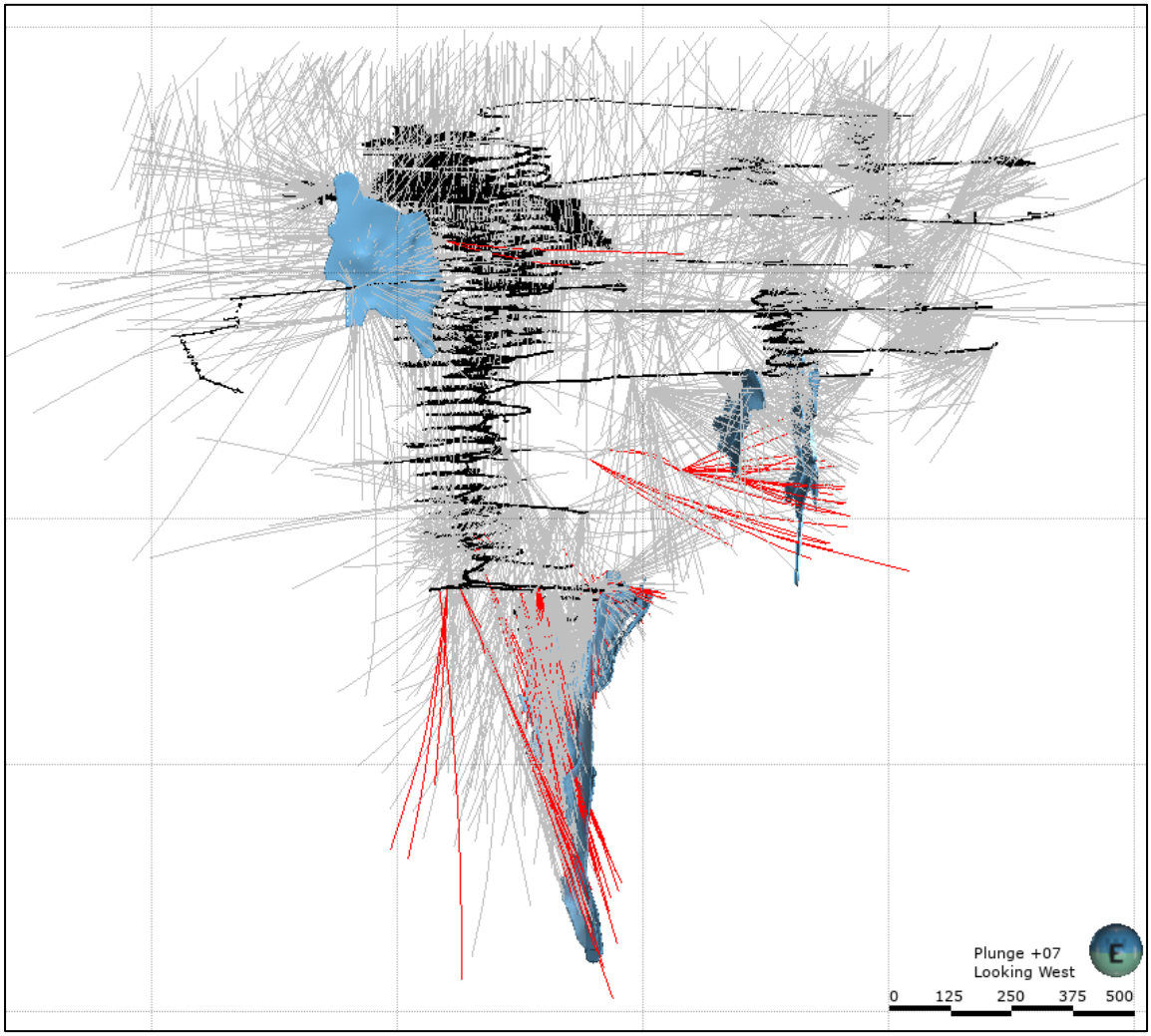


Figure 14-3: 3D view of the DDH drilled after the close-out date of the Resource Database not included in this resource estimate (in red) - as of April 7, 2021

14.4 Geological Interpretation and Modelling

A total of 52 mineralized lenses were interpreted for the purpose of the 2020 MRE (Table 14-1).

Table 14-1: Mineralized lenses of the 2020 MRE

2020 Mineralized zones							
Zone	**Wireframe name**	**Blockcode**	**Source**	**Zone**	**Wireframe name**	**Blockcode**	**Source**
Kiena Deep A	ZA_200	200	Updated	Dubuisson	DUB_1210	1210	New 3D Model
	ZA1_210	210			DUB_N1_1220	1220	
	ZA2_220	220			DUB_N2_1230	1230	
	H1ZA_300	300		Martin	MART_1310	1310	New 3D Model
VC1	VC1_111	111	Updated	NorthWest	NW_1410	1410	New 3D Model
	VC1_112	112		Wesdome	WES_A_2010	2010	New 3D Model
	VC1_113	113			WES_A1_2020	2020	
	VC1_114	114			WES_AF_2030	2030	
VC6	VC6_123	123	Updated		WES_AH_2040	2040	
	VC6_124	124			WES_AH1_2050	2050	
S50	S50_100	100	Carried out		WES_AH2_2060	2060	
	S50_101	101			WES_AH3_2070	2070	
	S50_102	102			WES_B_2110	2110	
	S50_103	103			WES_C_2120	2120	
	S50_104	104			WES_D_2130	2130	
South	ZS_130	130	Carried out		WES_E_2210	2210	
	ZS_131	131			WES_E0_2220	2220	
	ZS_132	132			WES_E1_2230	2230	
	ZS_133	133			WES_E3_2240	2240	
	ZS_135	135			WES_E4_2250	2250	
B	ZB_140	140	Carried out		WES_F2_2260	2260	
Presqu'ile	PI_1	1110	New 3D Model		WES_F4_2270	2270	
	PI_2	1120			WES_F6_2280	2280	
	PI_2A	1125			WES_L_2310	2310	
	PI_3	1130			WES_F_2320	2320	
	PI_4	1140			WES_F1_2330	2330	

14.4.1 Geological Model

The herein Block Model comprises 52 mineralized lenses, out of which 10 mineralized lenses were updated since the 2019 MRE using new drilling information and 11 mineralized lenses were carried over from the 2019 MRE (Table 14-1). The remaining 31 lenses were modelled in 3D for the purpose of the 2020 MRE in order to convert previous Polygonal resources to Block Model.

The switch from Geovia GEMS software to Datamine Studio RM software in 2020 can explain some discrepancies in the mineralized lenses carried over from the 2019 MRE, as well as the manual reproduction of the clippings used for classification.

The geological wireframes were constructed in Leapfrog Geo™ by Charlotte Athurion of BBA, Clovis Auger of BBA and Karine Brousseau of Wesdome with the help of Bruno Turcotte of Wesdome. All the mineralized lenses have a minimum thickness of 3 m, except for the Wesdome Zone for which the minimum thickness was fixed to 2.4 m based on their low dipping angle and an expected shrinkage mining method (Figure 14-4 and Figure 14-5). They were modelled using geological knowledge of the deposit, grade continuity and geological information provided in the DDH logs (i.e., lithology, alteration and structure) and in the historical underground mappings. Geological interpretation of the basalt and schists units was also carried out for the purpose of better constraining the mineralized lenses. These zones are further described in Section 7.3 of this report.

As the Kiena Deep A lenses are folded, they were subdivided in three subdomains representing the two limbs and the hinge for each mineralized lenses (Figure 14-6).

The QPs reviewed the geological model in 3D view, plan view and cross-section and is of the opinion that the level of detail to which the geology model was constructed represents adequately the Complexity of the deposit. In the QPs opinion, the geological model is appropriate for the size, grade distribution and geometry of the mineralized lenses and is suitable for the resource estimation of the Kiena Mine Complex.

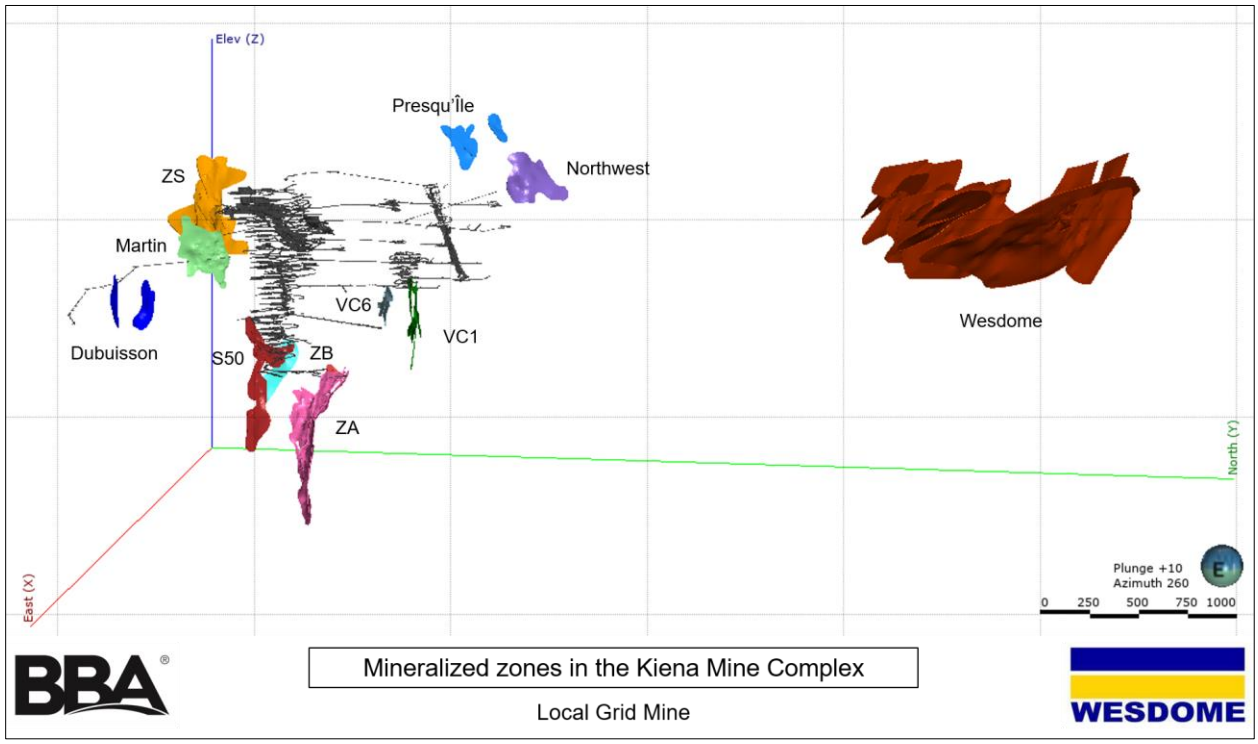


Figure 14-4: 3D view showing the mineralized zones and undergrounds voids in the Kiena Mine Complex area looking west

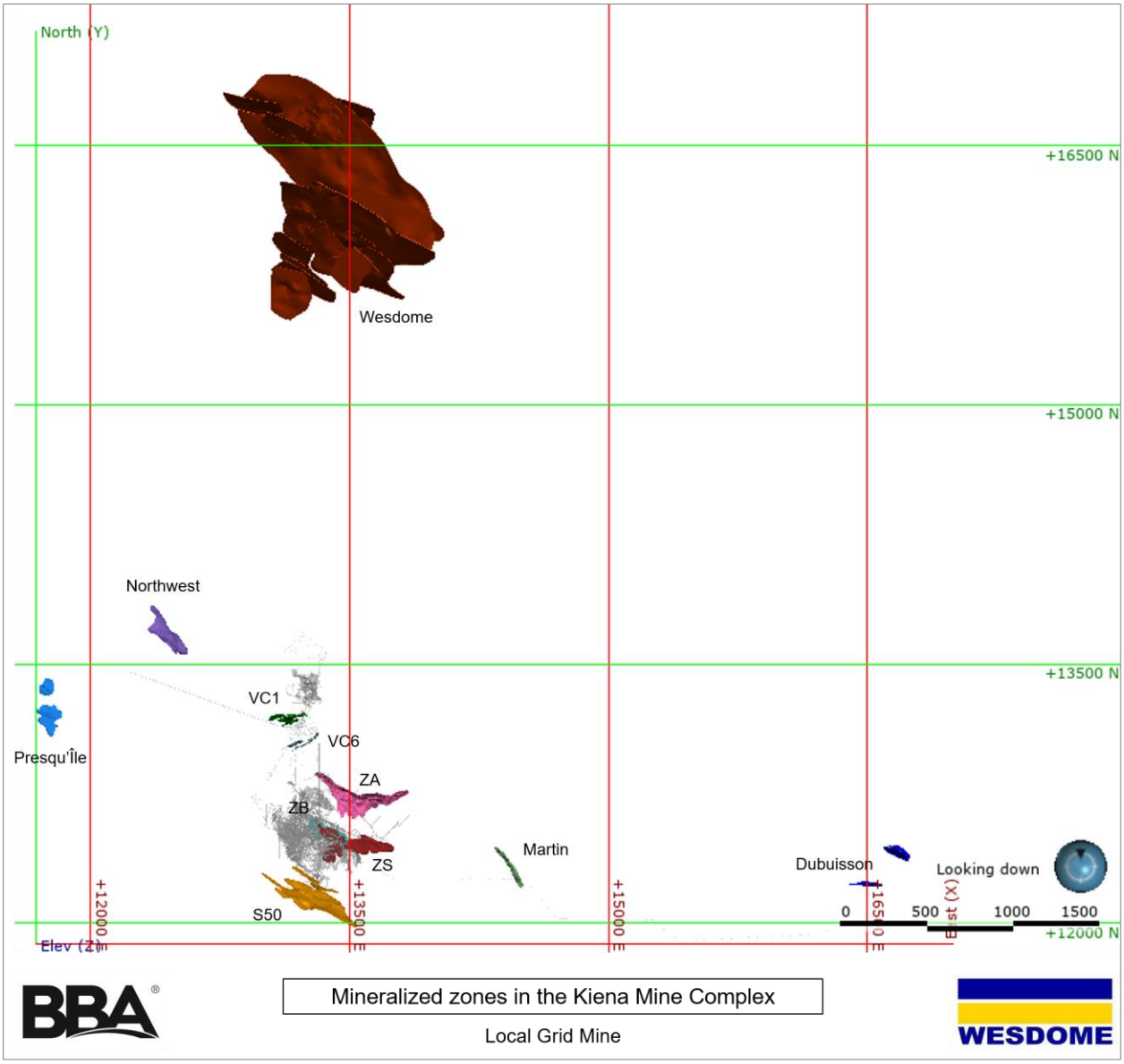


Figure 14-5: Plan view showing the mineralized zones and undergrounds drifts in the Kiena Mine Complex area looking down

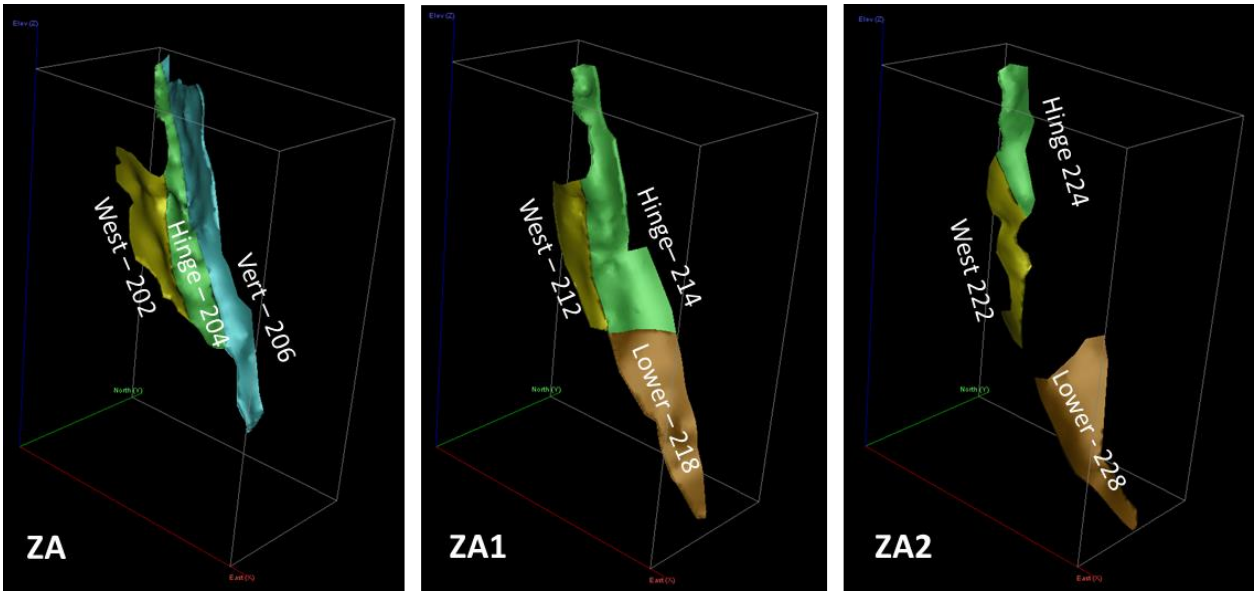


Figure 14-6: Subdomains of the Kiena Deep Zone A (ZA, ZA1 and ZA2 lenses)

14.4.2 Voids Model

The void model for the 2020 MRE was provided by Wesdome. Figure 14-4 presents a 3D view of the underground voids used for the 2020 MRE. Validations were made to make sure that any new development was considered for the 2020 MRE.

14.4.3 Overburden and Topography

The topographic surface and the overburden-rock interface were created by Wesdome in Datamine Studio RM software and are based on the collar elevation coordinates of the drillholes and the lithological description.

14.5 Data Analysis

14.5.1 Raw Assay Statistics

All raw assay data that intersected the mineralized lenses were assigned individual blockcodes. These coded intercepts were used to produce basic statistics on sample lengths and grades.

A total of 14,501 assays are included in the modelled wireframe.

Basic statistics on the raw assays are presented in Table 14-2.

Table 14-2: Basic statistics on raw assays for each zone

		Raw assays					
Zone	**Mineralized lens**	**Blockcode**	**Count sample**	**COV**	**Max (g/t Au)**	**Min (g/t Au)**	**Mean (g/t Au)**
S50	S50_100	100	477	1.43	42.17	0.000	1.99
	S50_101	101	516	1.67	52.64	0.000	2.19
	S50_102	102	829	3.07	256.67	0.000	3.62
	S50_103	103	311	4.03	134.46	0.000	1.95
	S50_104	104	237	3.23	72.37	0.005	2.25
VC1	VC1_111	111	30	2.81	178.50	0.002	11.94
	VC1_112	112	454	3.16	147.57	0.002	3.31
	VC1_113	113	89	7.24	2,578.56	0.002	37.71
	VC1_114	114	387	2.16	51.95	0.002	2.48
VC6	VC6_123	123	223	1.78	30.03	0.002	2.50
	VC6_124	124	127	2.40	35.80	0.002	1.66
South	ZS_130	130	547	2.95	96.62	0.000	1.57
	ZS_131	131	460	1.94	43.50	0.000	2.36
	ZS_132	132	554	1.66	28.80	0.000	1.60
	ZS_133	133	161	4.30	105.60	0.000	1.99
	ZS_135	135	153	2.88	36.50	0.000	1.39
B	ZB_140	140	175	3.01	58.96	0.000	2.40
Kiena Deep A	ZA	200	2,074	4.86	2,769.58	0.002	20.53
	ZA1	210	1,499	4.86	886.84	0.002	12.51
	ZA2	220	680	3.67	771.00	0.002	21.53
	H1ZA	300	194	3.78	221.34	0.005	6.40
Presqu'ile	PI_1	1110	31	2.00	21.10	0.010	3.04
	PI_2	1120	55	3.63	239.36	0.010	9.24
	PI_2A	1125	21	2.35	88.34	0.010	9.67
	PI_3	1130	23	3.60	99.35	0.005	5.67
	PI_4	1140	24	3.10	155.88	0.005	11.42
Dubuisson	DUB	1210	112	2.26	121.22	0.010	7.82
	DUB_N1	1220	77	2.44	101.41	0.010	7.92
	DUB_N2	1230	45	1.55	32.53	0.010	4.61
Martin	Mart_1310	1310	771	3.33	207.86	0.005	3.80
NorthWest	NW_1410	1410	119	1.53	24.34	0.005	2.84

		Raw assays					
Zone	**Mineralized lens**	**Blockcode**	**Count sample**	**COV**	**Max (g/t Au)**	**Min (g/t Au)**	**Mean (g/t Au)**
Wesdome	W_A	2010	1,066	2.18	118.35	0.005	3.43
	W_A1	2020	54	4.04	260.53	0.005	8.79
	W_AF	2030	163	3.11	113.25	0.005	3.07
	W_AH	2040	193	3.65	140.71	0.005	3.96
	W_AH1	2050	45	2.94	52.07	0.005	3.29
	W_AH2	2060	58	2.26	44.50	0.005	2.79
	W_AH3	2070	24	2.27	42.75	0.005	4.65
	W_B	2110	258	1.79	24.14	0.005	2.21
	W_C	2120	60	2.47	42.55	0.005	2.81
	W_D	2130	94	1.73	21.02	0.005	2.08
	W_E	2210	440	1.98	55.35	0.005	2.90
	W_E0	2220	82	2.55	80.90	0.005	4.54
	W_E1	2230	104	2.62	87.90	0.005	4.31
	W_E3	2240	83	2.37	66.73	0.005	3.55
	W_E4	2250	53	2.81	119.35	0.005	6.40
	W_F2	2260	58	1.62	39.77	0.005	4.80
	W_F4	2270	33	1.83	24.60	0.005	2.68
	W_F6	2280	19	1.69	28.42	0.005	4.63
	W_L	2310	94	3.09	85.71	0.005	3.88
	W_F	2320	50	1.66	32.75	0.050	4.31
	W_F1	2330	15	1.37	17.30	0.005	4.49

14.5.2 Compositing

Compositing of drillhole samples was performed in order to homogenize the database for the statistical analysis and remove any bias associated with the sample length that may exist in the original database. The composite length was determined using original sample length statistics and the thickness of the mineralized zones.

Figure 14-7 shows the sample length distribution within the mineralized lenses for the Kiena Deep Zone A Zone.

As a result, 14,493 composites with an average length of 1 m were generated with a minimal length of 0.25 m, when necessary, to redistribute the tails evenly between all the composites of the drillhole.

Compositing was done within each mineralized lens in order to ensure that composite samples did not cross domain boundaries.

Grades of 0.00 g/t Au were assigned to all missing intervals prior to the compositing process.

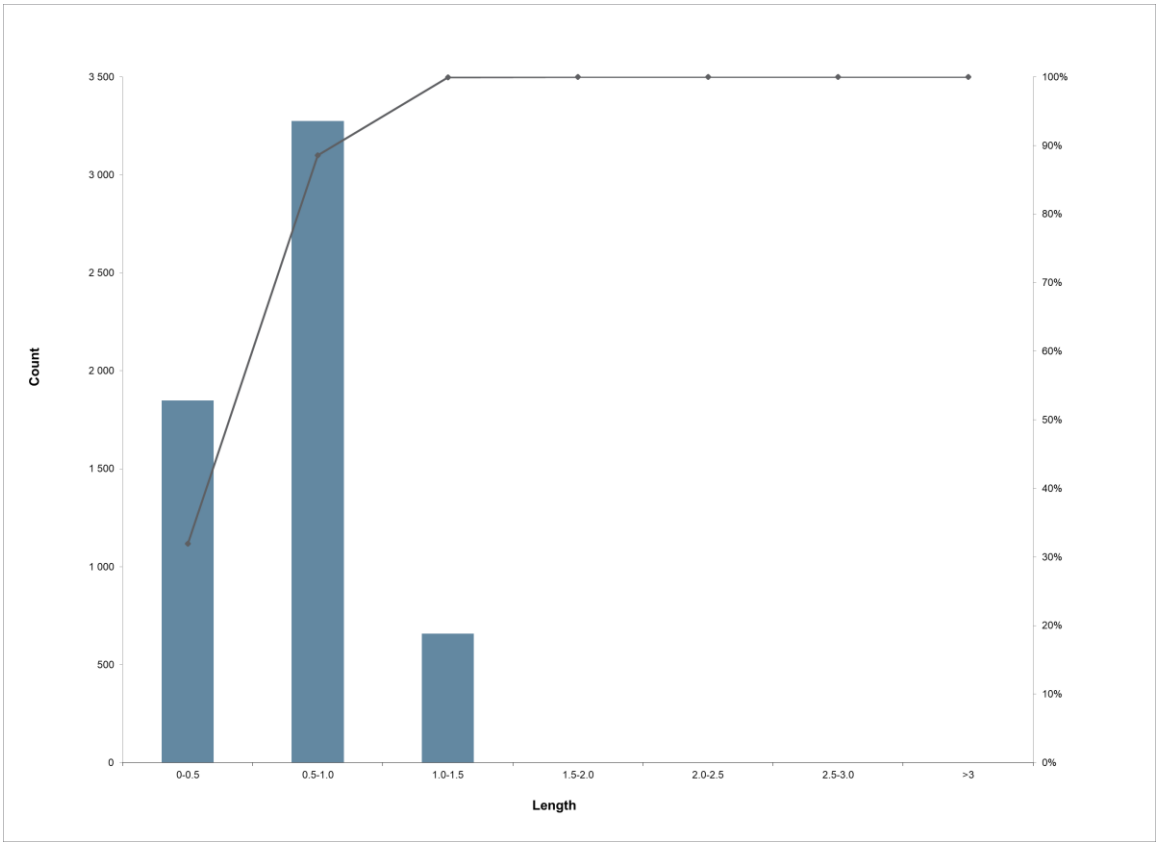


Figure 14-7: Example of sample length distribution within the Kiena Deep A Zone

14.5.3 Outlier Handling

An outlier is an observation that appears to be inconsistent with most of the data. It is common practice to statistically examine the higher grades within a population and to trim the outlier to a lower grade value based on the results of a statistical study. The capping is performed on high-grade values considered to be outliers. High-grade capping was done on the composited assay data and established on a per zone or lens type basis. To facilitate the statistical studies, the mineralized lenses were sometimes grouped by geographic location.

In addition, for some lenses of the Kiena Deep Zone A Zone, a three-step capping strategy was conducted. In this strategy, three capping values are established, and the selected capping value decreases as interpolation search distances increase. This strategy allows to restrict the impact of high grades at greater distance from the drillhole intersect by using lower capping values (Table 14-3).

The capping values were defined by searching for abnormal breaks or change of slope on the grade distribution probability plot while making sure that the coefficient of variation of the capped data was ideally lower than, or around 2.00 and no more than 10% of the total contained metal was enclosed within the first 1% of the highest-grade samples. The use of various statistical methods allows for a selection of the capping threshold in a more objective and justified manner.

Basic statistics for composited assays and capped composites are summarized in Table 14-3. For the mineralized lenses where a multi-capping strategy was applied, the statistics of the capped composites correspond to the statistics of the first capping value. Figure 14-8 to Figure 14-25 show graphs supporting the capping threshold decisions.

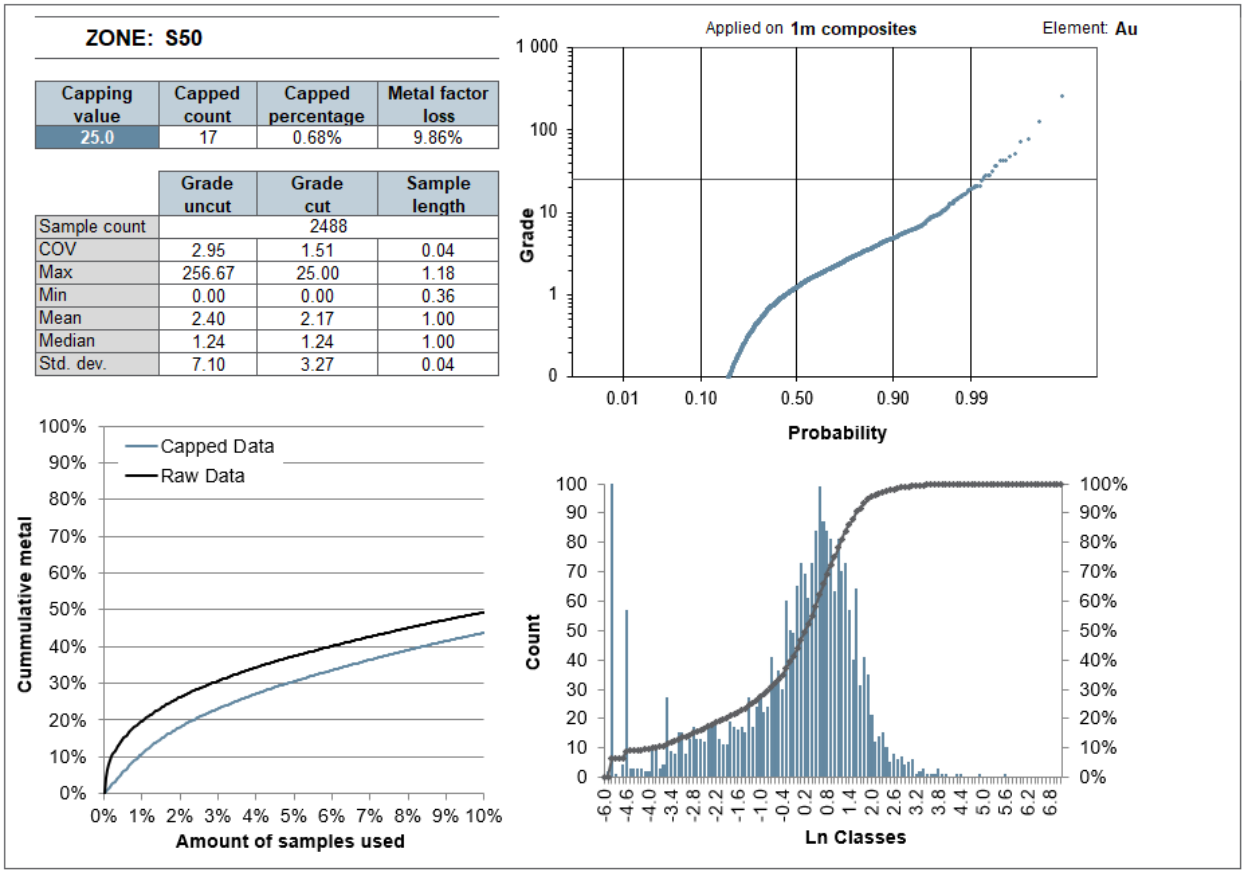


Figure 14-8: Graphs supporting capping threshold decisions on composites for the S50 Zone (blockcodes 100 to 104)

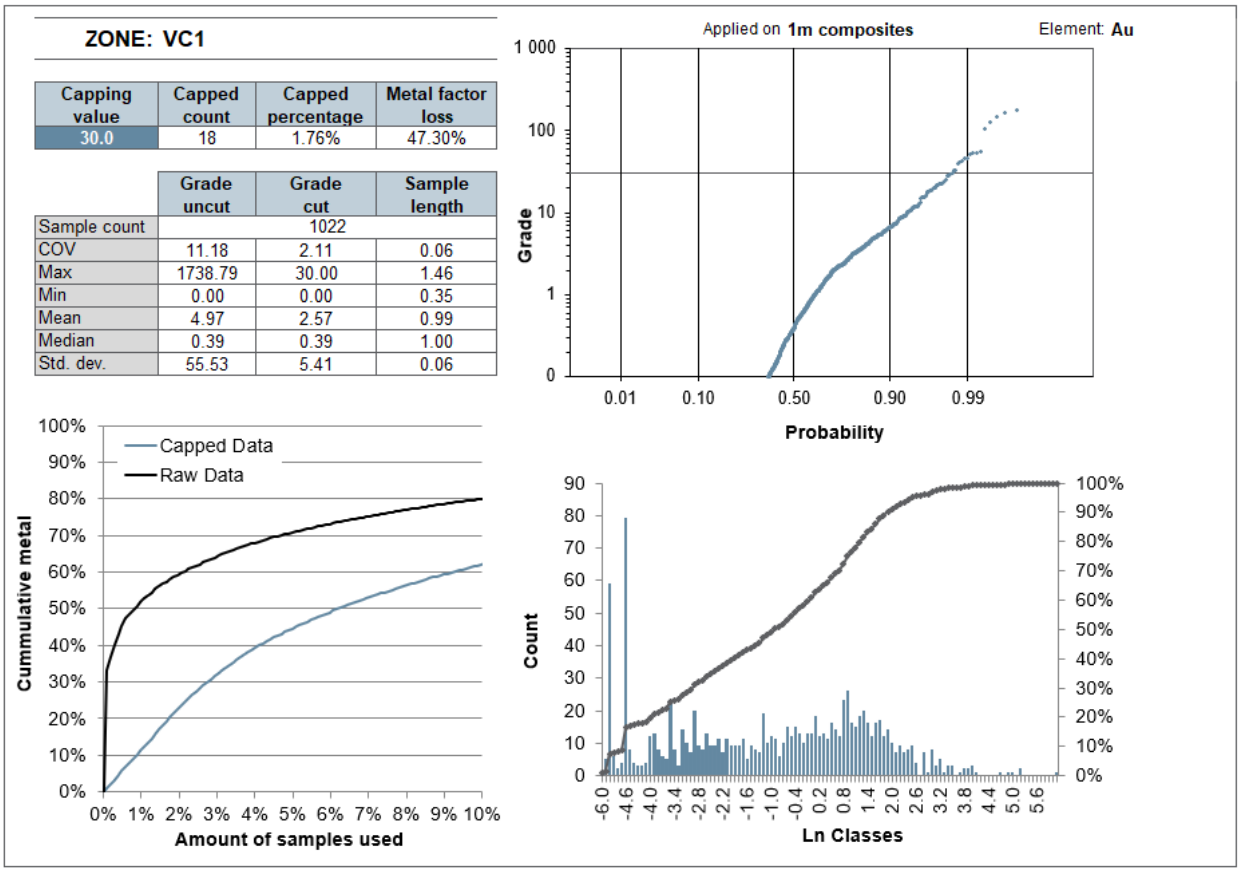


Figure 14-9: Graphs supporting capping threshold decisions on composites for the VC1 Zone (blockcodes 111 to 114)

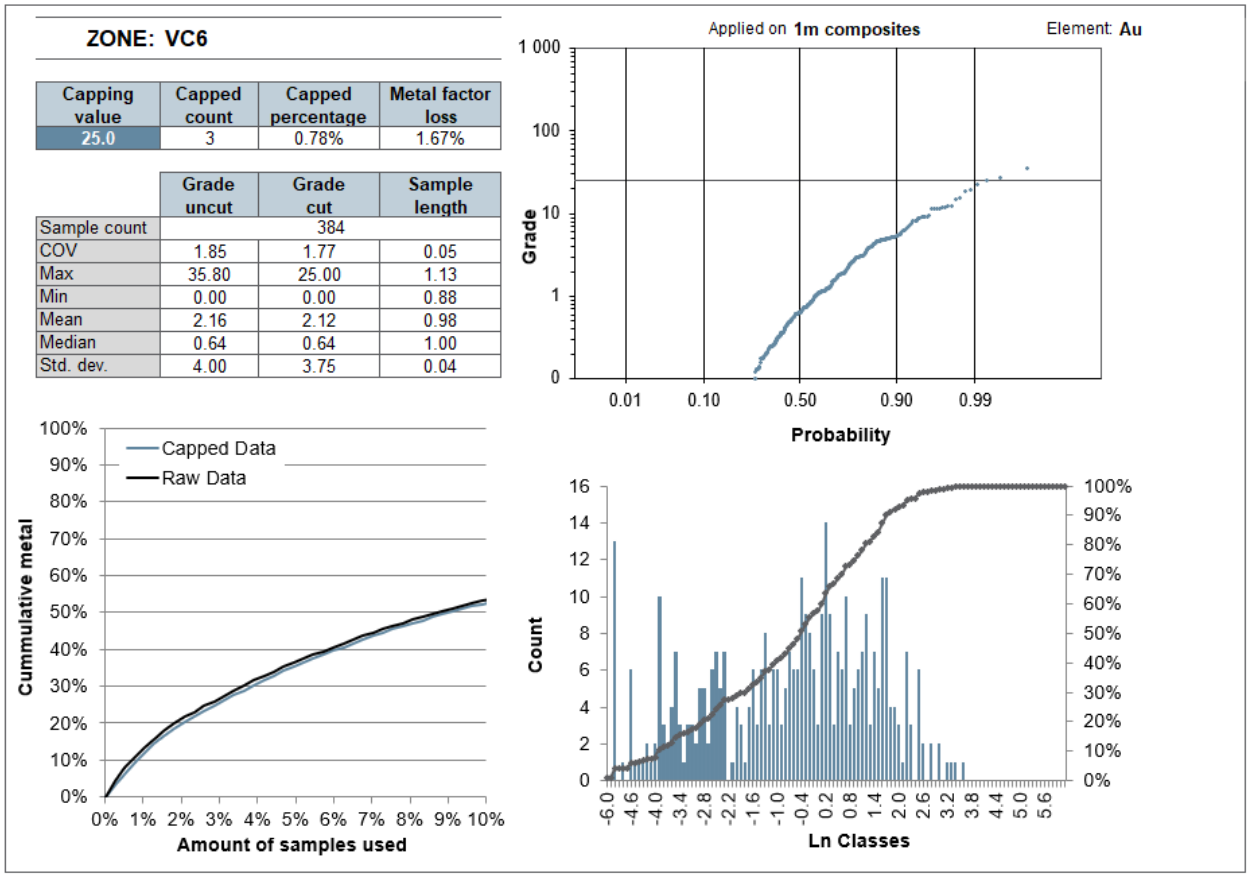


Figure 14-10: Graphs supporting capping threshold decisions on composites for the VC6 Zone (blockcodes 123-124)

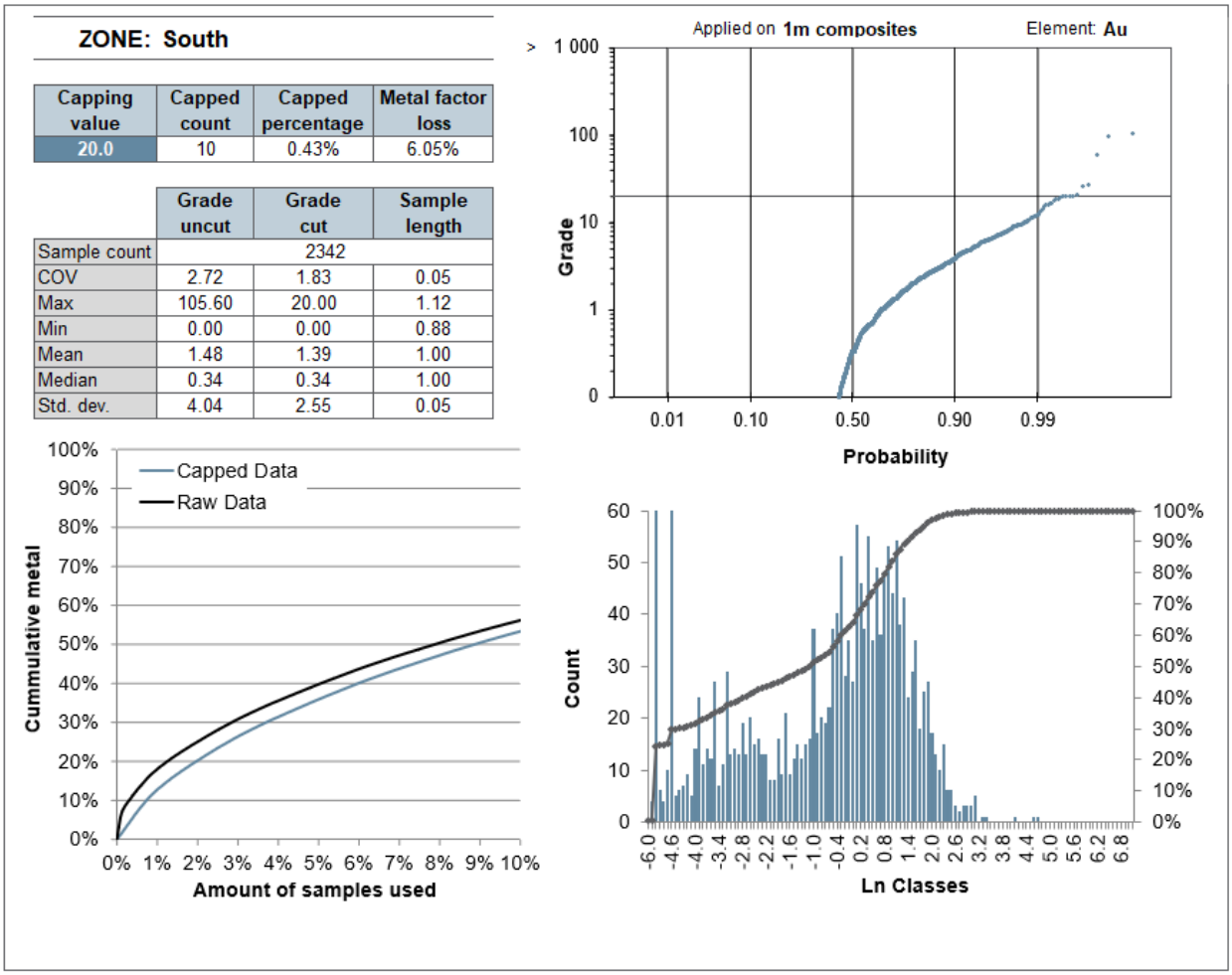


Figure 14-11: Graphs supporting capping threshold decisions on composites for the South Zone (blockcodes 130 to 133 and 135)

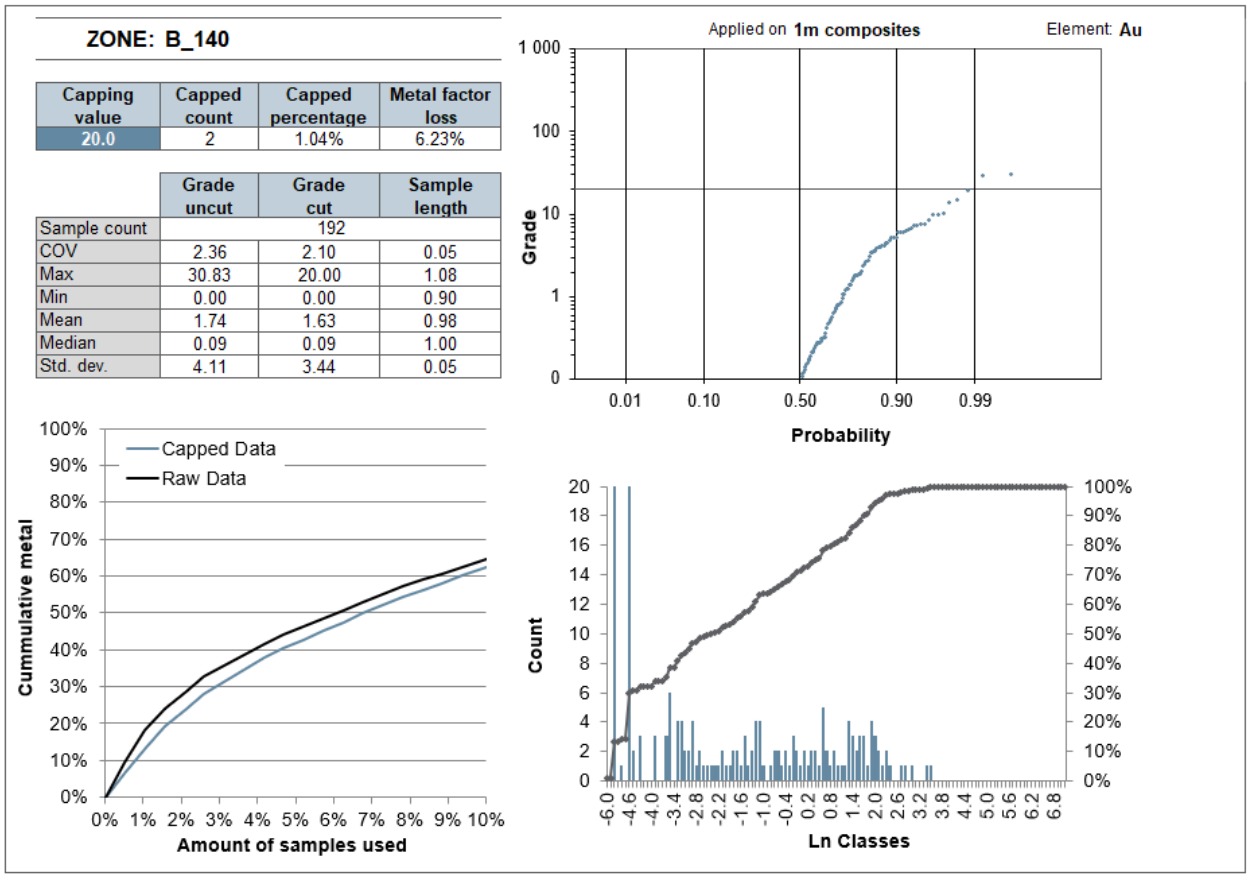


Figure 14-12: Graphs supporting capping threshold decisions on composites for the B Zone (blockcode 140)

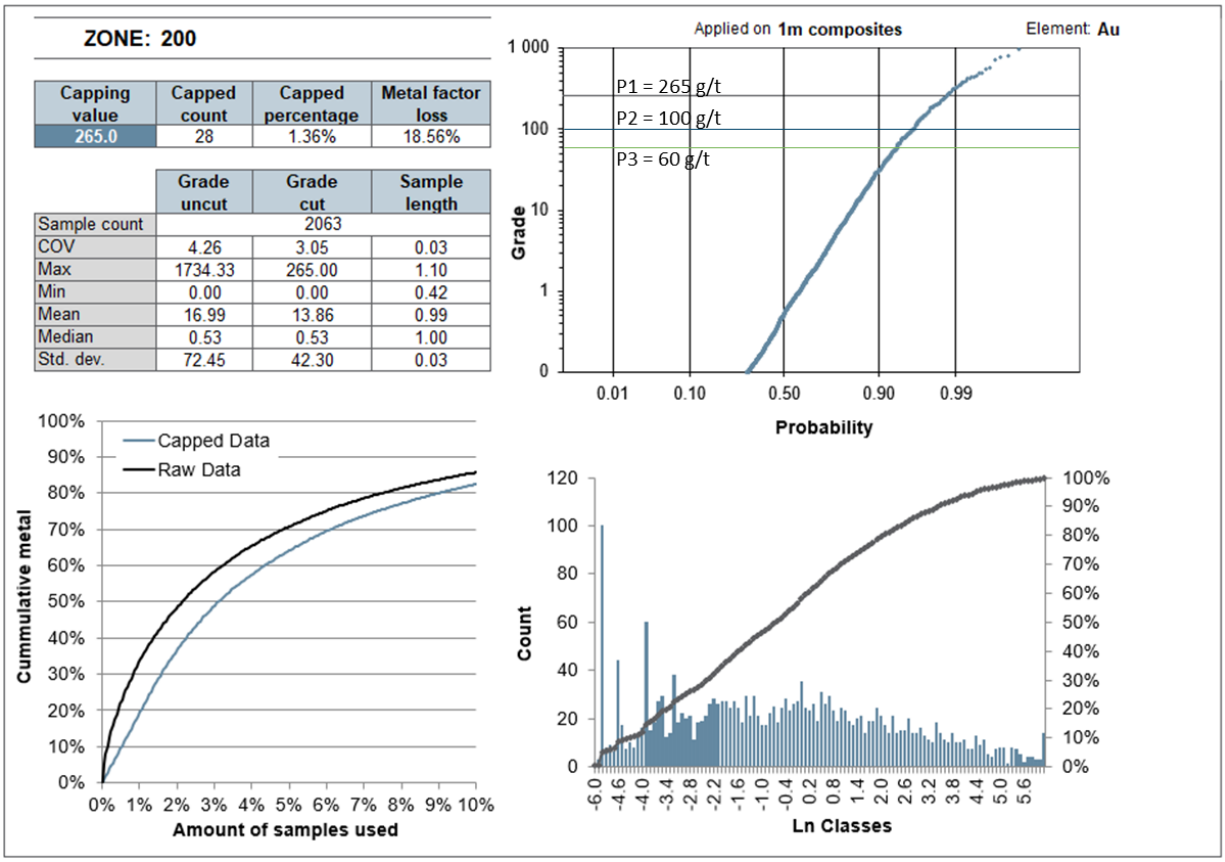


Figure 14-13: Graphs supporting capping threshold decisions on composites for the Kiena Deep A Zone A subdomains
(blockcodes 202, 204, 206)
Note that the graphs and the statistics are presented for the first capping value.

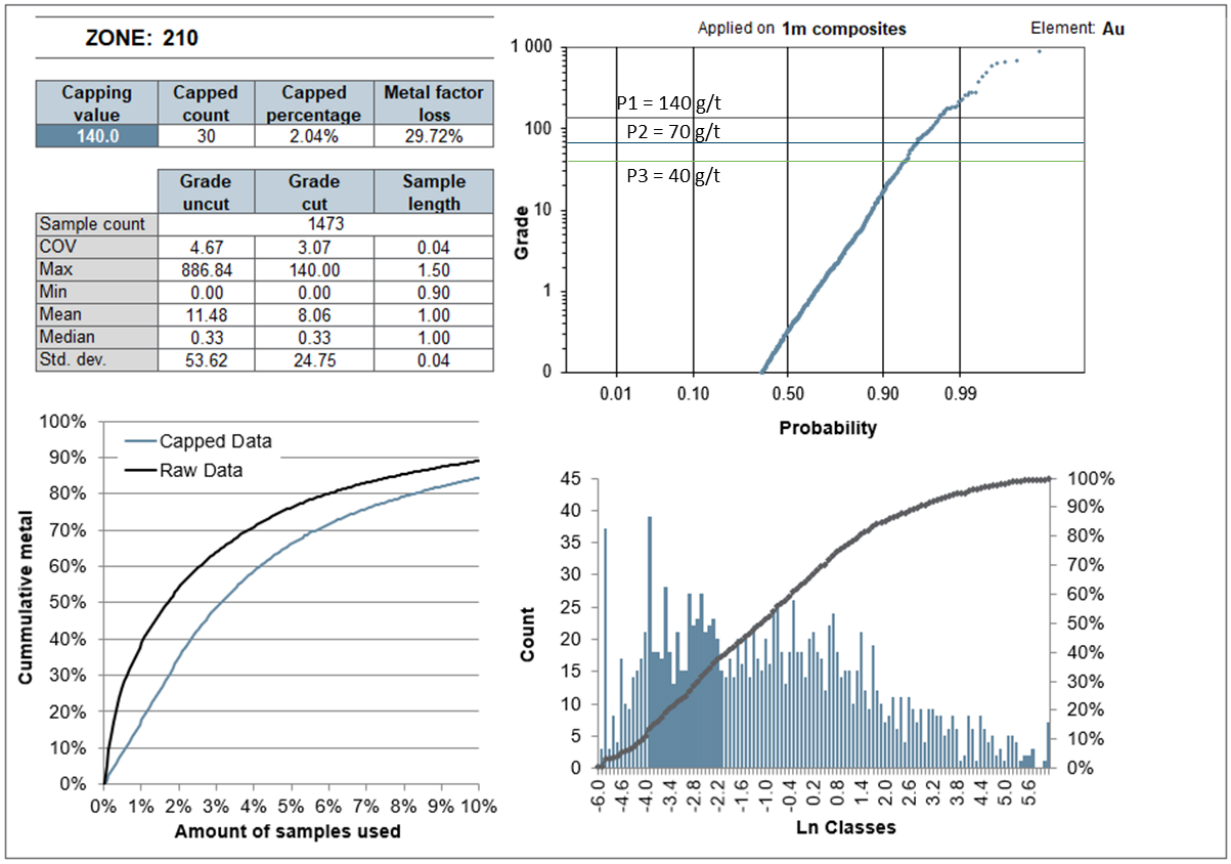


Figure 14-14: Graphs supporting capping threshold decisions on composites for the Kiena Deep A Zone A1 (blockcodes 212, 214, 218)

Note that the graphs and the statistics are presented for the first capping value.

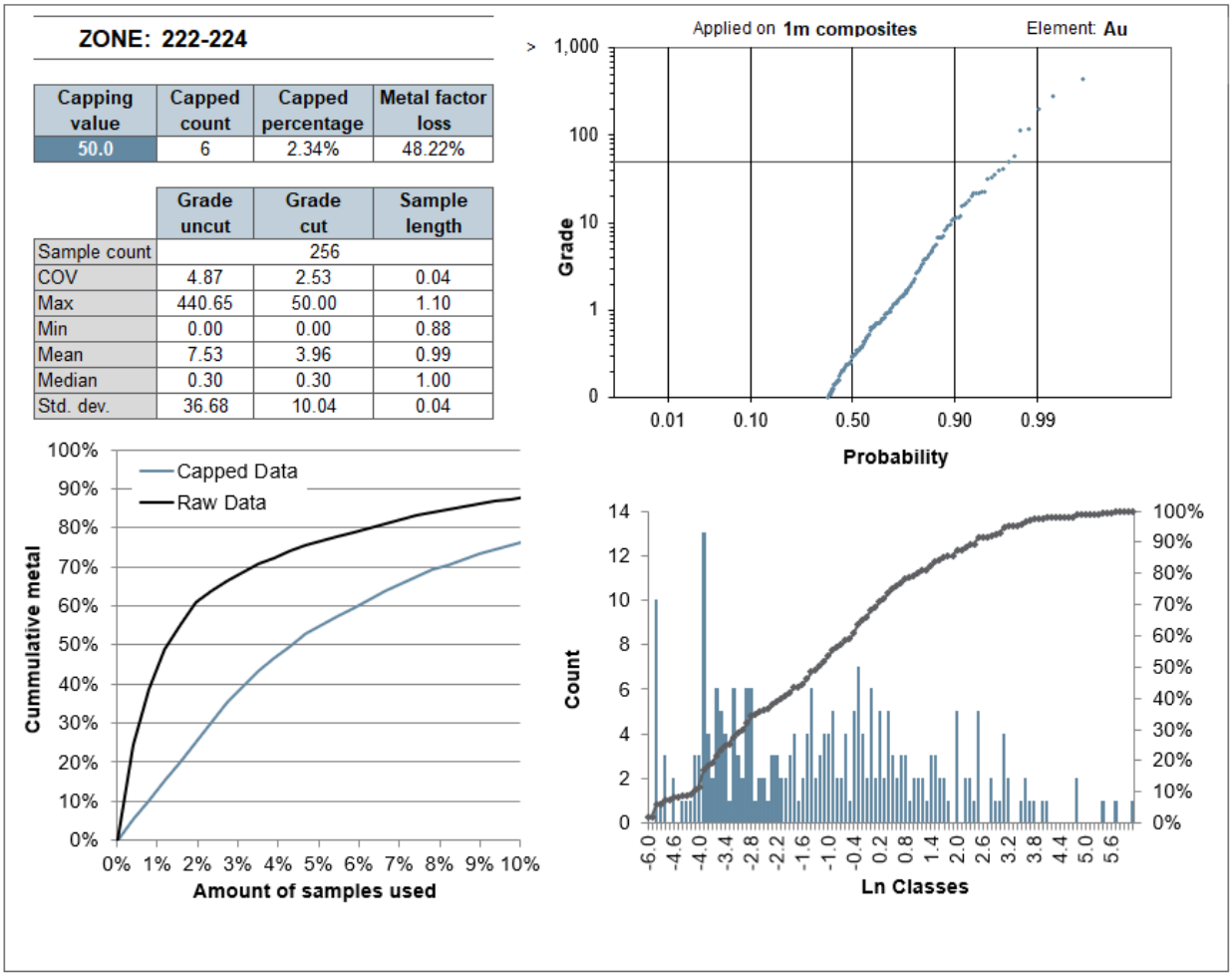


Figure 14-15: Graphs supporting capping threshold decisions on composites for the Kiena Deep A Zone A2 – upper domain (blockcodes 222-224)

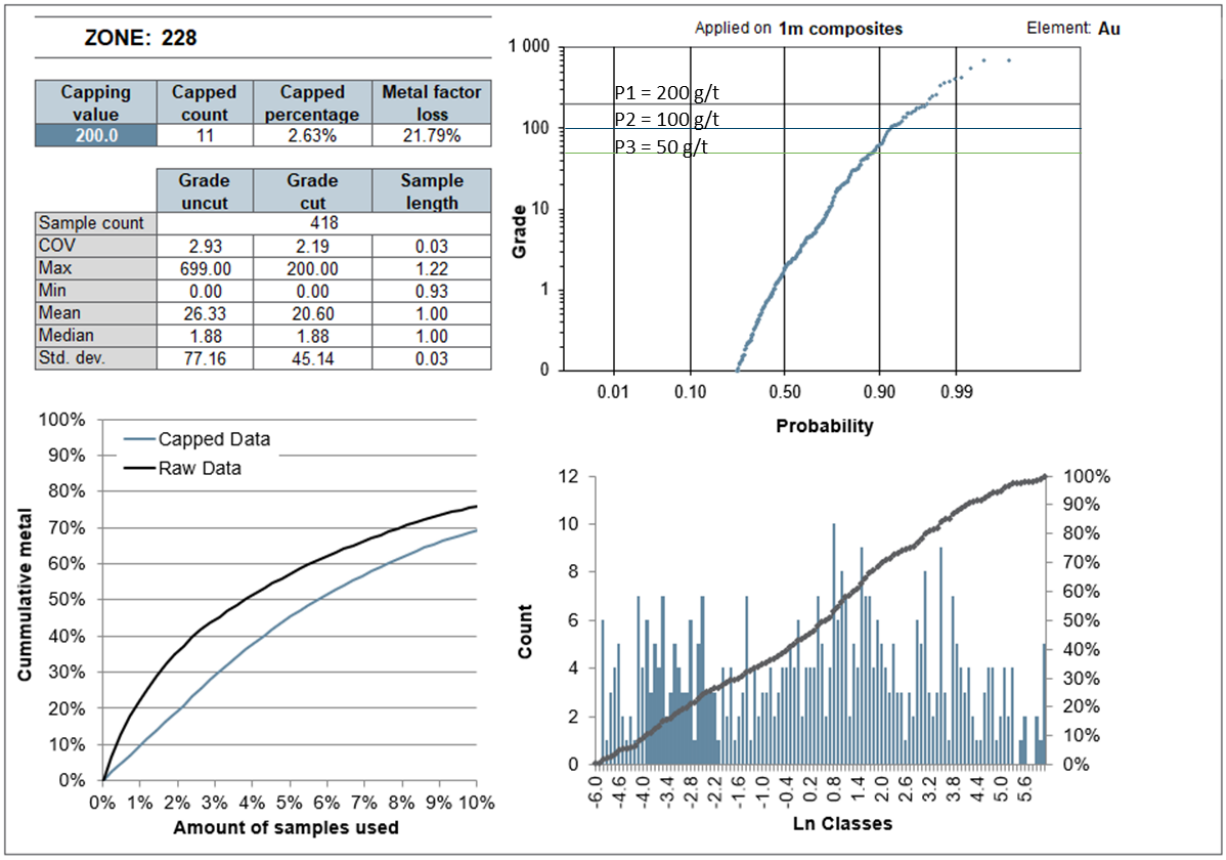


Figure 14-16: Graphs supporting capping threshold decisions on composites for the Kiena Deep A Zone A2 – lower domain (blockcode 228)

Note that the graphs and the statistics are presented for the first capping value.

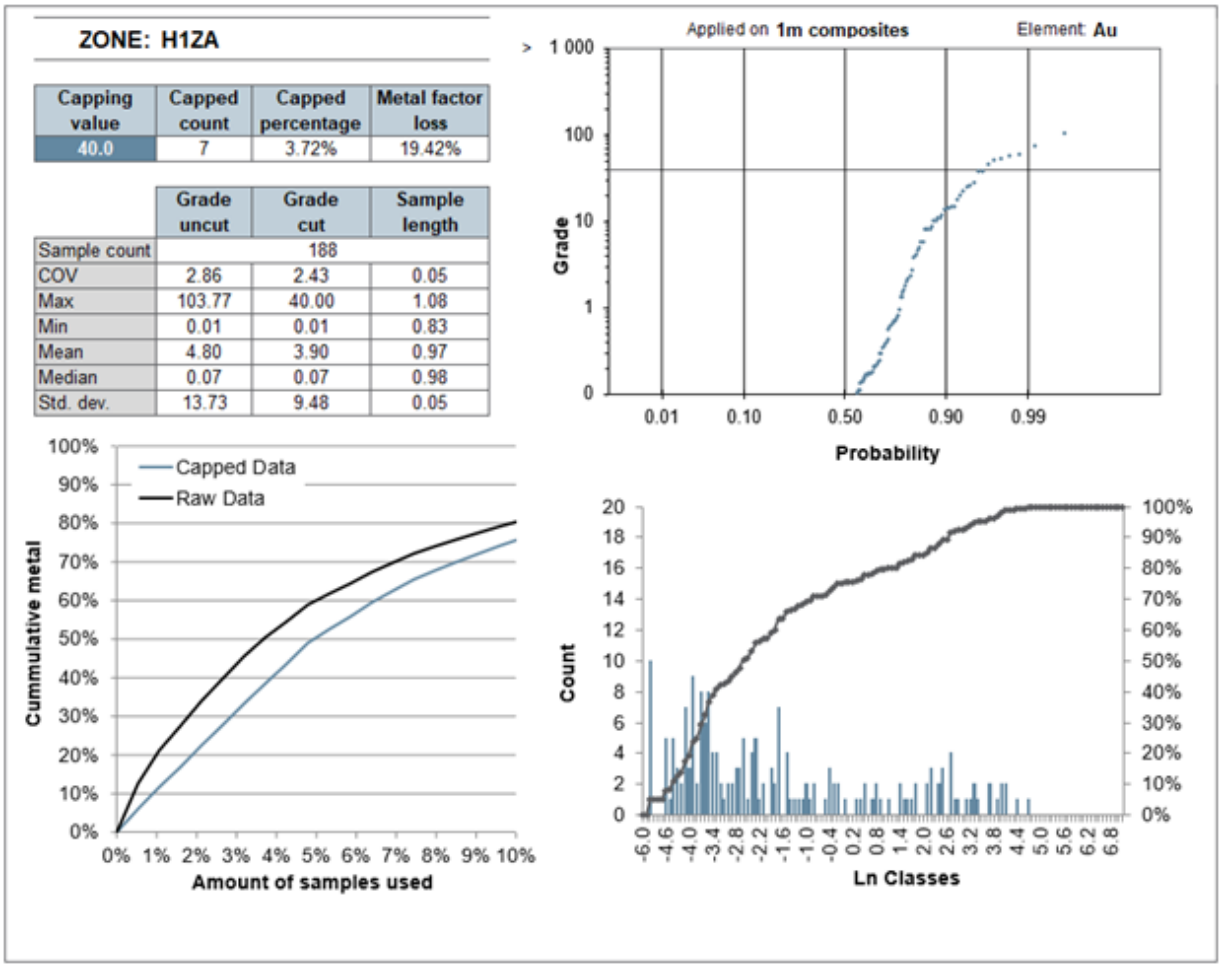


Figure 14-17: Graphs supporting capping threshold decisions on composites for the H1ZA Zone (blockcode 300)

ZONE: Presquile

Capping value	Capped count	Capped percentage	Metal factor loss
55.0	6	4.41%	26.39%

	Grade uncut	Grade cut	Sample length
Sample count	136		
COV	2.96	2.35	0.05
Max	152.27	55.00	1.08
Min	0.00	0.00	0.88
Mean	7.63	5.61	0.98
Median	0.30	0.30	1.00
Std. dev.	22.60	13.19	0.05

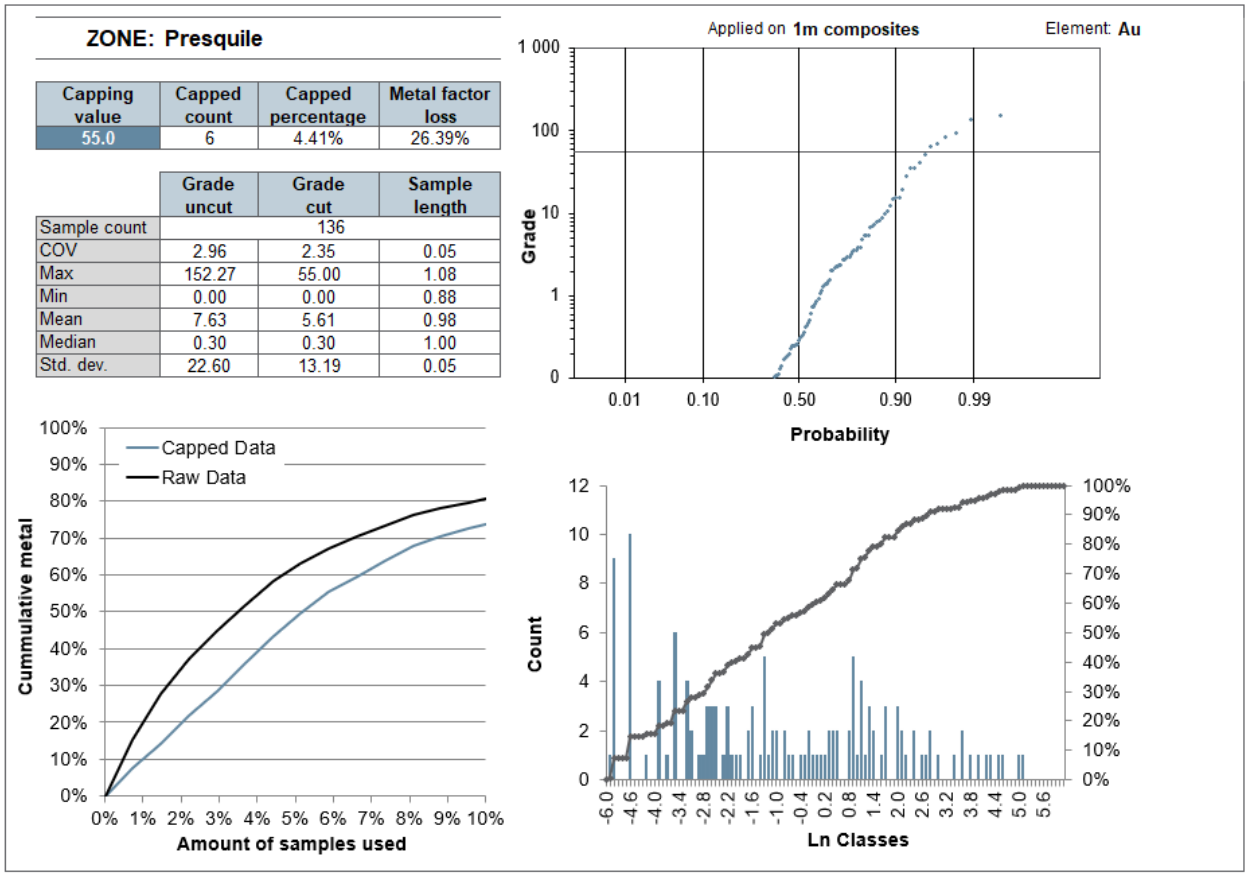


Figure 14-18: Graphs supporting capping threshold decisions on composites for the Presqu'ile Zone (blockcodes 1110 to 1140)

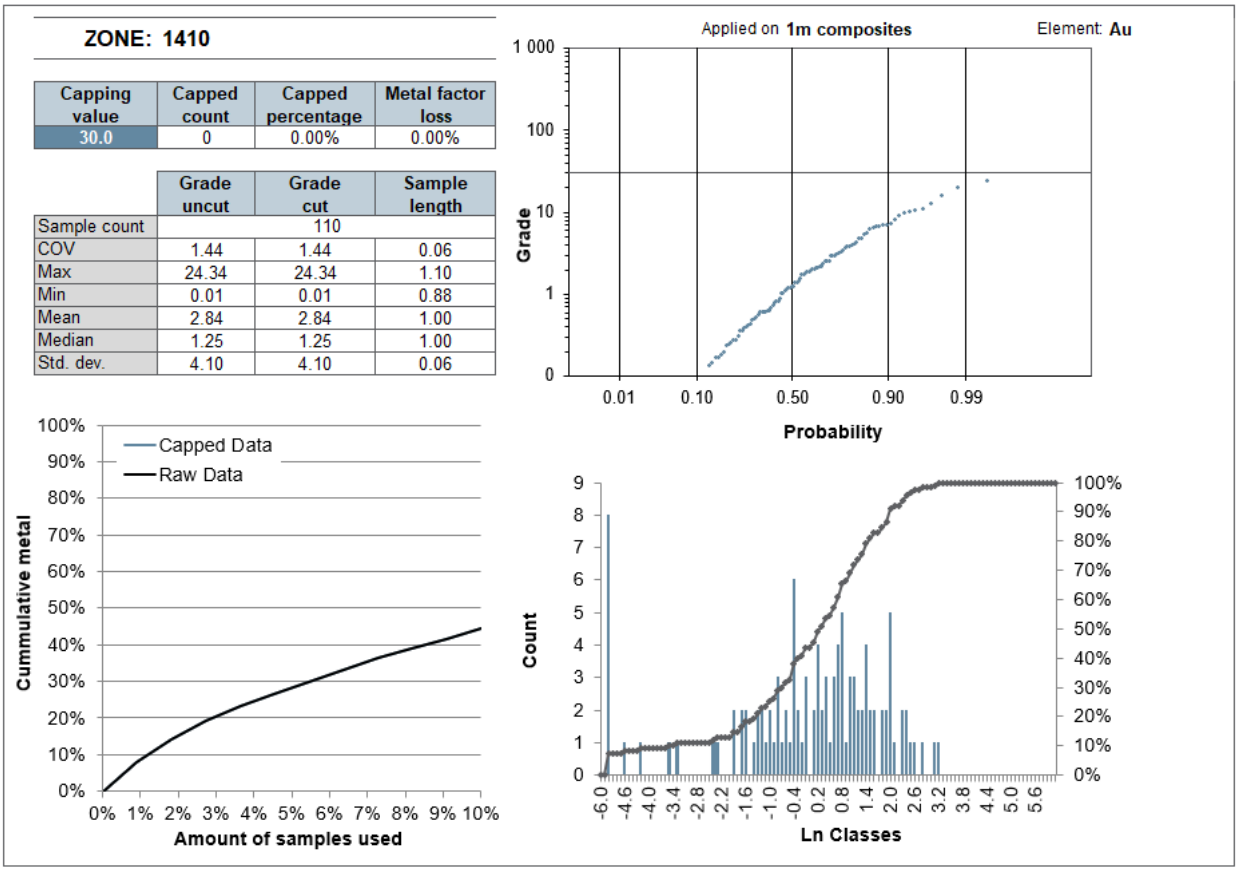


Figure 14-19: Graphs supporting the absence of capping threshold on composites for the NorthWest Zone (blockcode 1410)

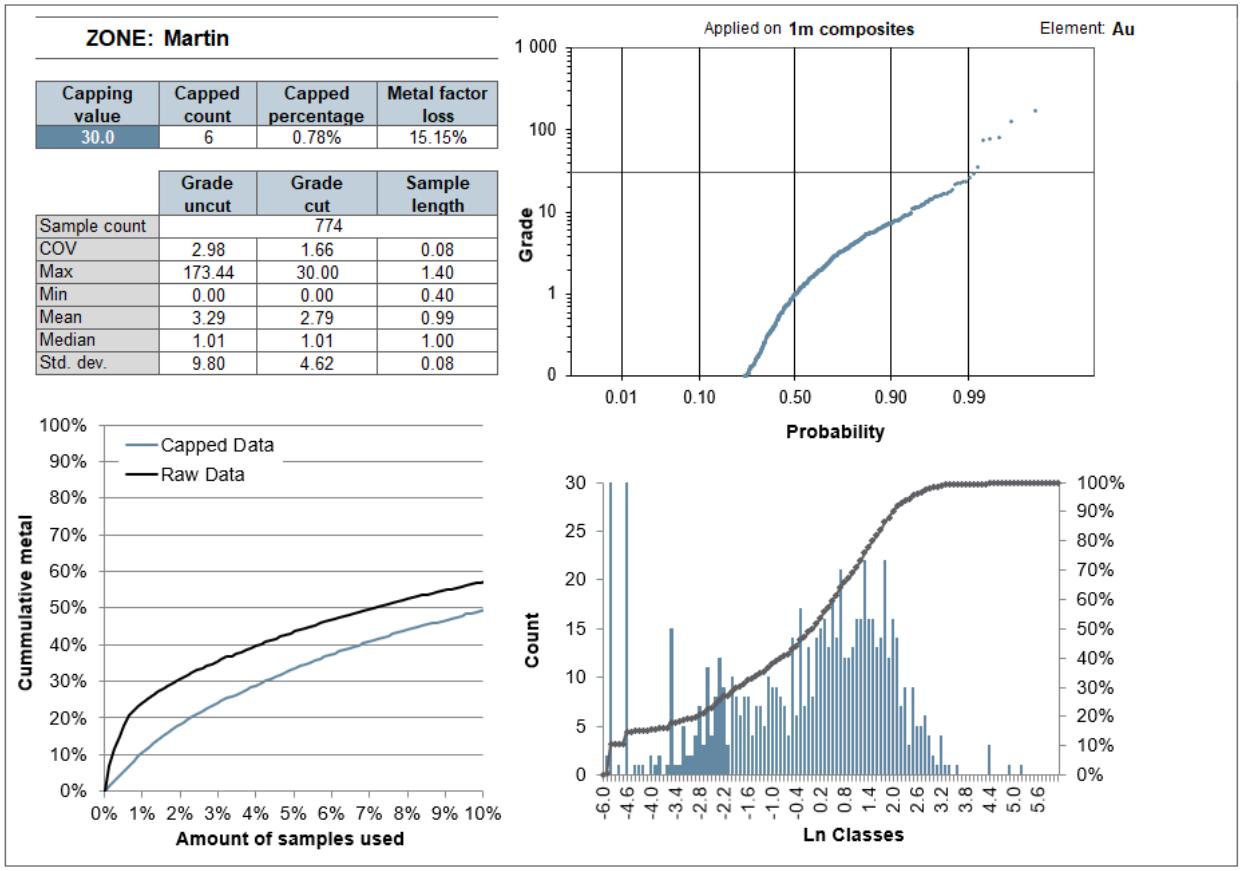


Figure 14-20: Graphs supporting capping threshold decisions on composites for the Martin Zone (blockcode 1310)

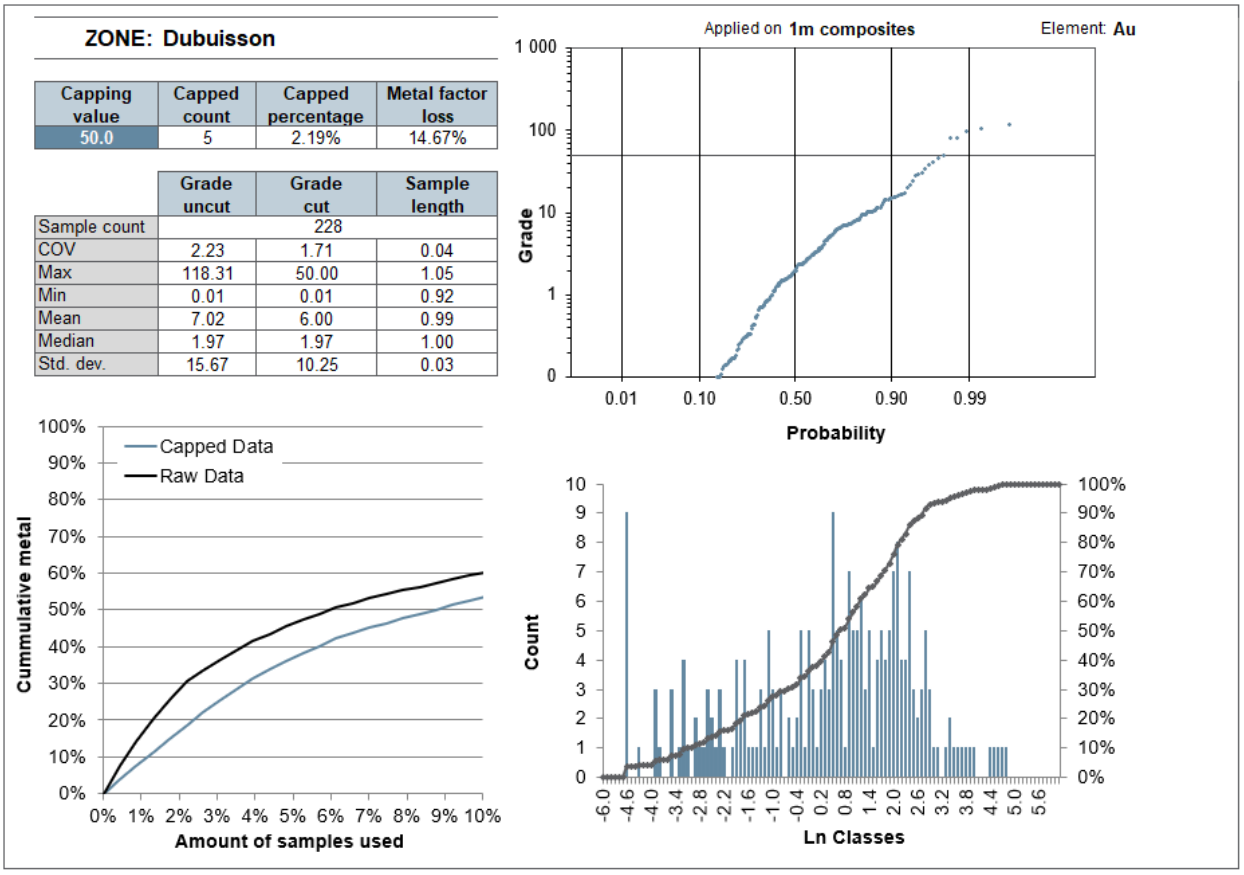


Figure 14-21: Graphs supporting capping threshold decisions on composites for the Dubuisson Zone (blockcodes 1210 to 1230)

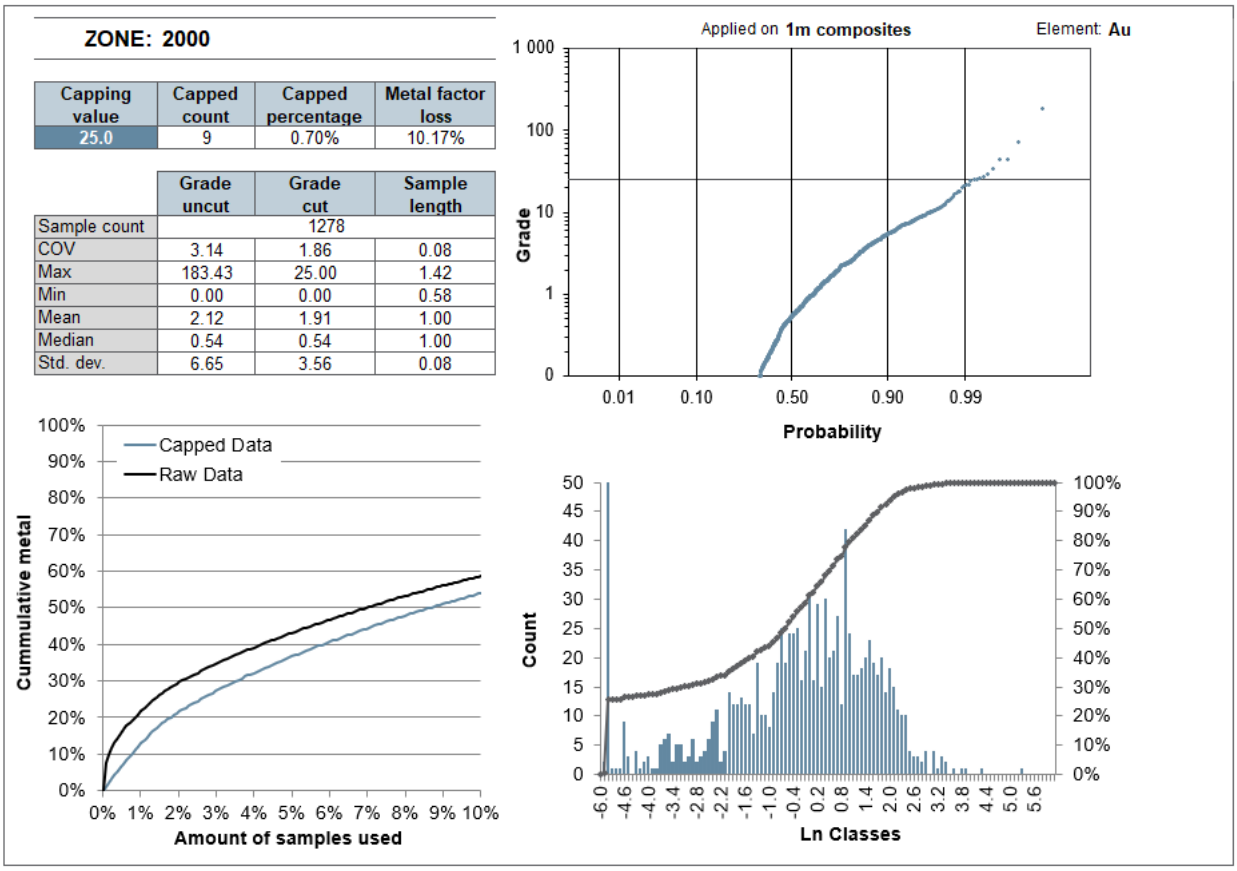


Figure 14-22: Graphs supporting capping threshold decisions on composites for the Wesdome Zone (blockcodes 2010 to 2070)

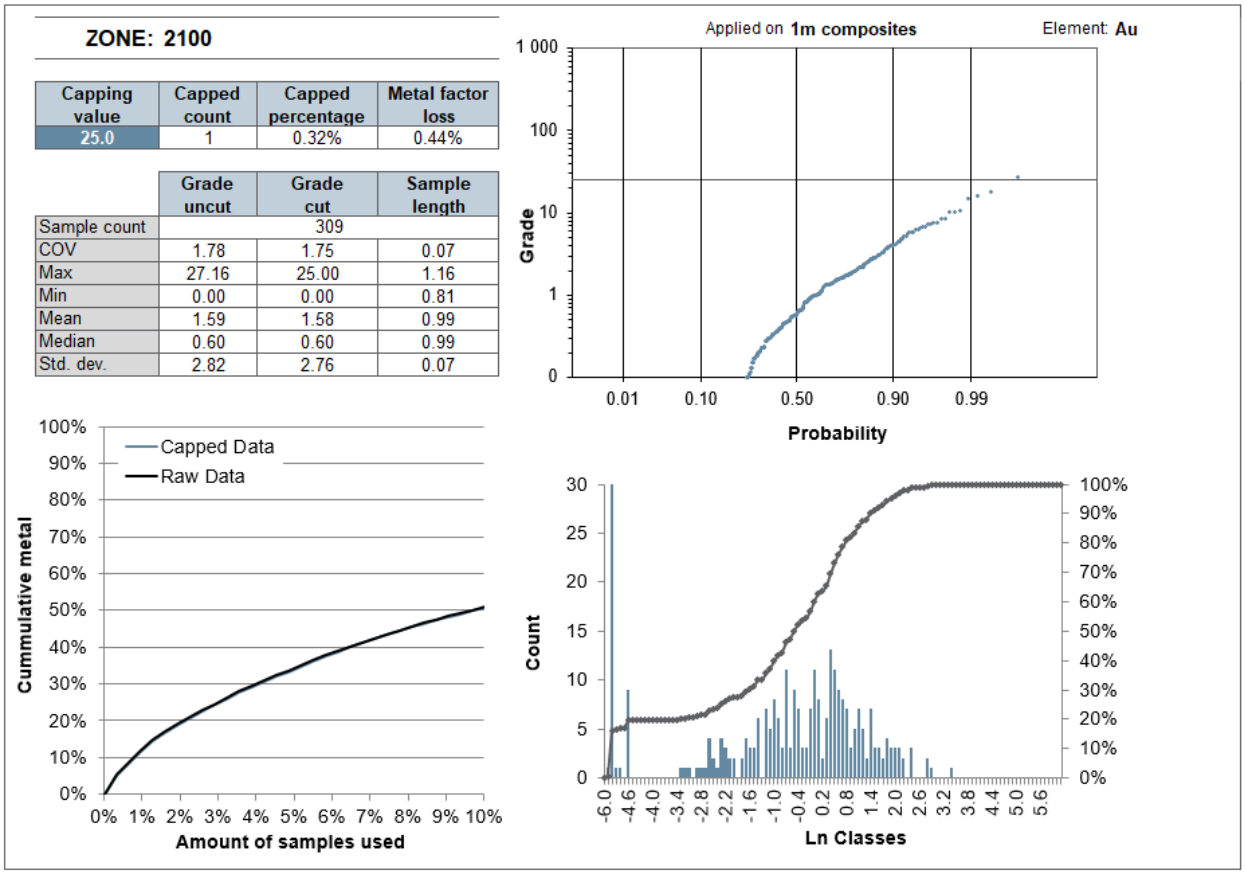


Figure 14-23: Graphs supporting capping threshold decisions on composites for the Wesdome Zone (blockcodes 2110 to 2130)

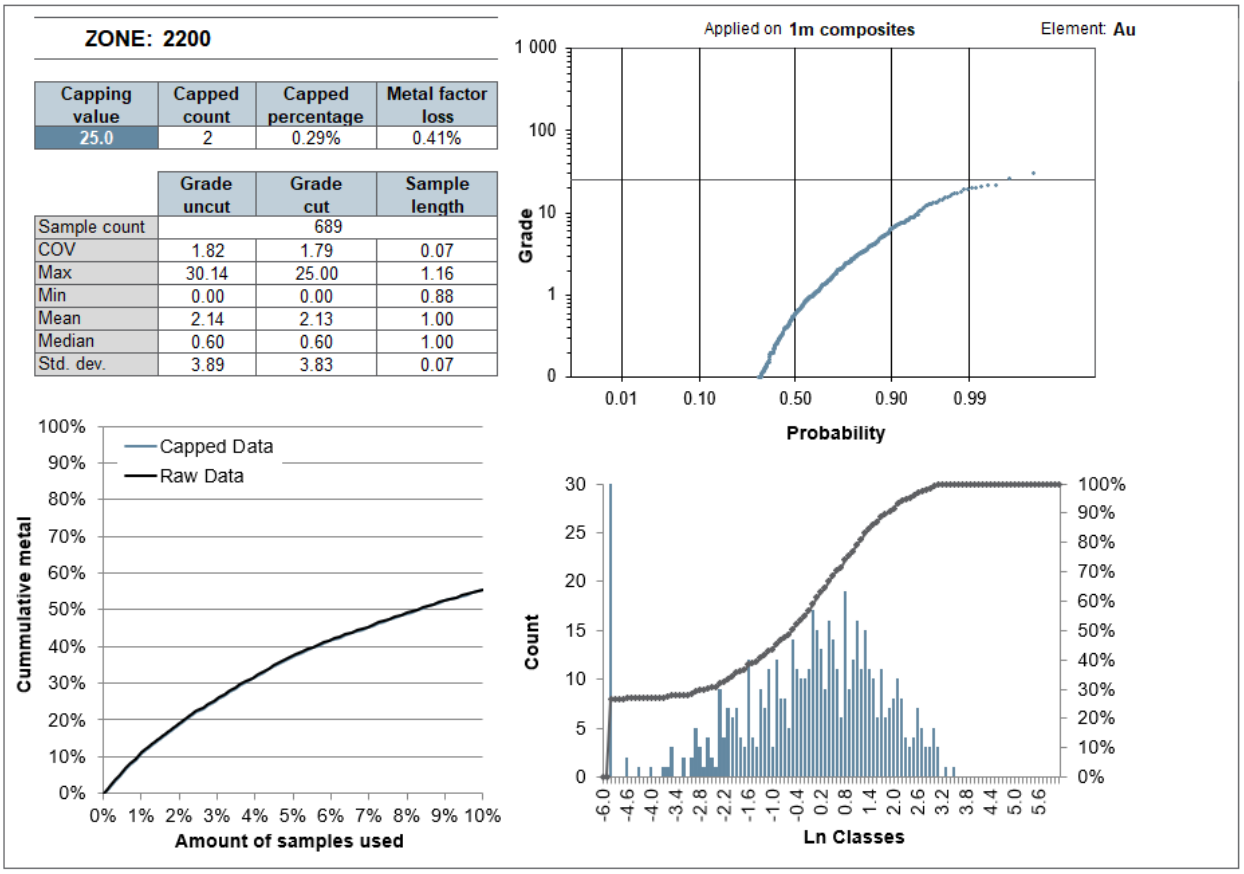


Figure 14-24: Graphs supporting capping threshold decisions on composites for the Wesdome Zone (blockcodes 2210 to 2280)

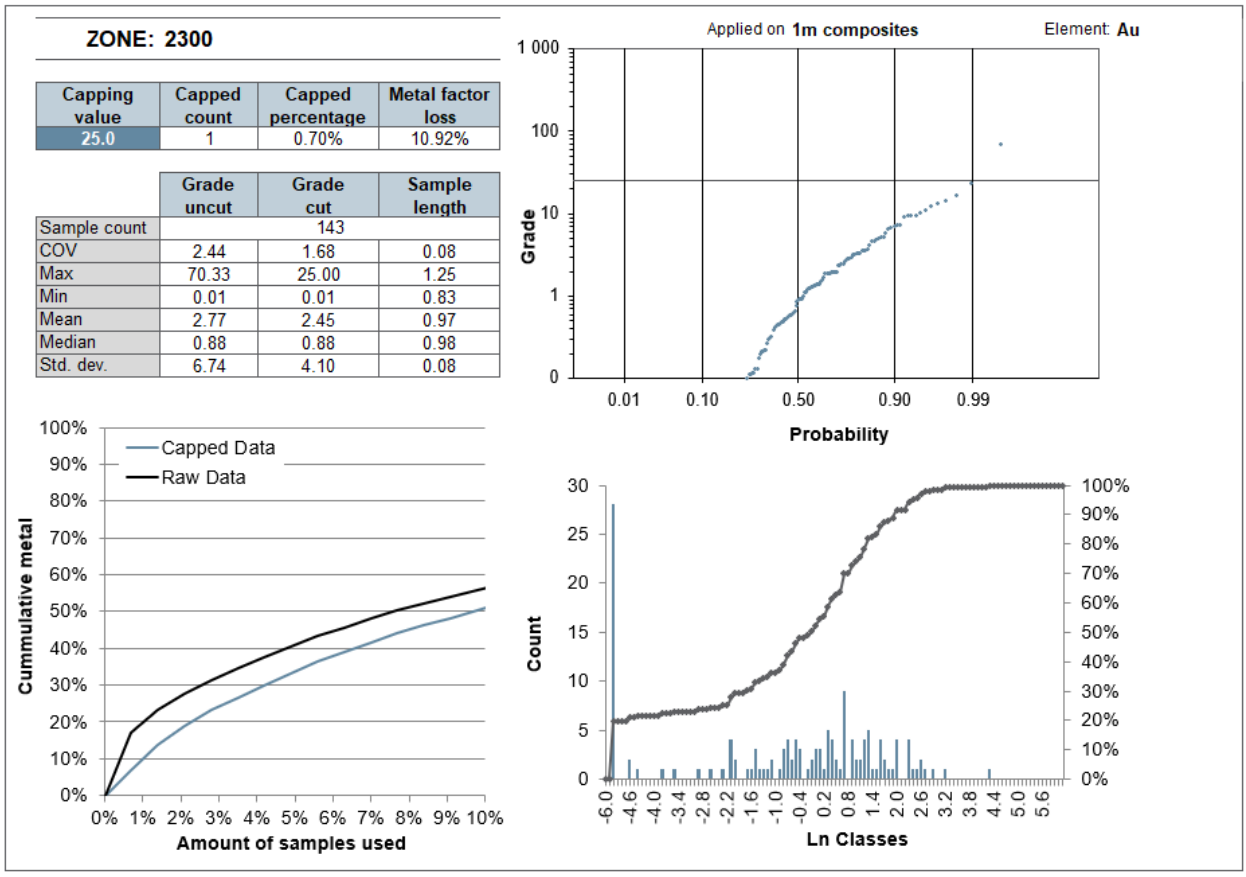


Figure 14-25: Graphs supporting capping threshold decisions on composites for the Wesdome Zone (blockcodes 2310 to 2330)

Table 14-3: Basic statistics on composites and high-grade capping value for each zone

		Composites (1 m) - Au (g/t)										
Zone	**Blockcode**	**Composite count**	**Uncut COV**	**Max (g/t Au)**	**Min (g/t Au)**	**Uncut Mean (g/t Au)**	**Capping value (Au g/t)**	**Number capped**	**Capped mean (g/t Au)**	**Capped COV**	**% Samples capped**	**Metal loss (%)**
S50	100	2488	2.95	256.67	0.000	2.40	**25**	17	2.17	1.51	0.68	9.86
	101											
	102											
	103											
	104											
VC1	111	1022	11.18	1,738.79	0.000	4.97	**30**	18	2.57	2.11	1.76	47.30
	112											
	113											
	114											
VC6	123	384	1.85	35.80	0.002	2.16	**25**	3	2.12	1.77	0.78	1.67
	124											
South	130	2342	2.72	105.60	0.000	1.48	**20**	10	1.39	1.83	0.43	6.05
	131											
	132											
	133											
	135											
B	140	192	2.36	30.83	0.000	1.74	**20**	2	1.63	2.10	1.04	6.23
Kiena Deep A	200	2063	4.26	1,734.33	0.002	16.99	**265, 100, 60**	28	13.86	3.05	1.36	18.56
	210	1,473	4.67	886.84	0.002	11.48	**140, 70, 40**	30	8.06	3.07	2.04	29.72
	222	256	4.87	440.65	0.001	7.53	**50**	6	3.96	2.53	2.34	48.22
	224											
	228	418	2.93	699.00	0.001	26.33	**200, 100, 50**	11	20.60	2.19	2.63	21.79
	300	188	2.86	103.77	0.005	4.80	**40**	7	3.90	2.43	3.72	19.42

		Composites (1 m) - Au (g/t)										
Zone	**Blockcode**	**Composite count**	**Uncut COV**	**Max (g/t Au)**	**Min (g/t Au)**	**Uncut Mean (g/t Au)**	**Capping value (Au g/t)**	**Number capped**	**Capped mean (g/t Au)**	**Capped COV**	**% Samples capped**	**Metal loss (%)**
Presqu'ile	1110	136	2.96	152.27	0.003	7.63	**55**	6	5.61	2.35	4.41	26.39
	1120											
	1125											
	1130											
	1140											
Dubuisson	1210	228	2.23	118.31	0.010	7.02	**50**	5	6.00	1.71	2.19	14.67
	1220											
	1230											
Martin	1310	774	2.98	173.44	0.001	3.29	**30**	6	2.79	1.66	0.78	15.15
NorthWest	1410	110	1.44	24.34	0.005	2.84	**-**	**-**	**-**	**-**	**-**	**-**
Wesdome	2010	1,278	3.14	183.43	0.000	2.12	**25**	9	1.91	1.86	0.70	10.17
	2020											
	2030											
	2040											
	2050											
	2060											
	2070											
	2110	309	1.78	27.16	0.003	1.59	**25**	1	1.58	1.75	0.32	0.44
	2120											
	2130											
	2210	689	1.82	30.14	0.001	2.14	**25**	2	2.13	1.79	0.29	0.41
	2220											
	2230											
	2240											

		Composites (1 m) - Au (g/t)										
Zone	**Blockcode**	**Composite count**	**Uncut COV**	**Max (g/t Au)**	**Min (g/t Au)**	**Uncut Mean (g/t Au)**	**Capping value (Au g/t)**	**Number capped**	**Capped mean (g/t Au)**	**Capped COV**	**% Samples capped**	**Metal loss (%)**
	2250											
	2260											
	2270											
	2280											
	2310	143	2.44	70.33	0.005	2.77	**25**	1	2.45	1.68	0.70	10.92
	2320											
	2330											

14.5.4 Density

Bulk density is an important parameter used to calculate tonnages for the estimated volumes derived from the resource-grade block model.

A total of 121 density measurements were conducted on the Project by Wesdome and by ALS. The samples selected were from a variety of lithologies and included a range of associated gold grades. All samples were located in the VC and the Kiena Deep Zones.

The specific gravity (SG) measurement was determined by the water displacement method. A summary of the SG data is presented in Table 14-4.

Table 14-4: Summary of the density measurements

Lithology	Count SG	Min SG	Max SG	Average SG
1P	4	2.68	2.77	2.72
3G	5	2.85	2.97	2.90
I3	3	2.71	2.74	2.73
I3A	9	2.73	2.85	2.83
M16	5	2.86	2.97	2.91
M8	39	2.70	2.95	2.83
M8Tc	5	2.80	2.87	2.82
V10	5	2.86	3.00	2.92
V3B	23	2.73	2.94	2.83
V3Py	1	2.78	2.78	2.78
V3Stwk	9	2.72	2.97	2.84
V4	3	2.87	2.98	2.94
V4Ba	3	2.82	2.91	2.86
VN	7	2.70	2.93	2.80

For the 2020 MRE, a fixed density value of 2.80 was used for all mineralized zones of the Project corresponding to the density value used when the mine was operating and close to the average of the schist unit (M8/M8Tc), which hosts the majority of the mineralized zones.

A fixed density of 2.00 g/cm^3 was assigned to the overburden and 0.00 g/cm^3 was assigned to underground workings.

14.5.5 Variogram Analysis

A semi-variogram is a common tool used to measure the spatial variability within a lens. Typically, samples taken far apart will vary more than samples taken close to each other. A variogram gives a measure of how much two samples taken from the same mineralized zone will vary in grade depending on the distance and orientation between those samples, and therefore allowing building search ellipsoids and assigning kriging weights to be used during interpolation.

Three dimensional directional variography using the Snowden Supervisor v8.11 software was carried out on the composites. Variograms were modelled in the three orthogonal directions to define a 3D ellipsoid for each unit. In some lenses where the number of composites was not sufficient to complete the variography study, the variography results of the most representative mineralized lens or of a group of mineralized lenses showing similar geological environment and azimuth/dip orientation were selected to establish the ranges of the search ellipsoids. The three directions of ellipsoid axes were set by using the variogram fans and visually confirmed with geological knowledge of the zone. Lag distances were set according to drillhole grid spacing specific to the structural domain analyzed.

Then, a mathematical model was interpreted in order to best-fit the shape of the calculated variogram for each direction. Three components were defined for the mathematical model: the nugget effect, the sill, and the range.

All variography tests were modelled with a nugget effect, as determined from the downhole semi-variograms and two spherical structures.

Table 14-5 presents the chosen variogram model parameters for each zone. Figure 14-26 to Figure 14-28 illustrate examples of the variography results.

The nugget effect values range from 5% to 30% and are typical of gold deposits.

Table 14-5: Variogram model parameters for each mineralized zone

			First structure				Second structure			
Zone	**Blockcode**	**Nugget**	**Sill**	**Range X (m)**	**Range Y (m)**	**Range Z (m)**	**Sill**	**Range X (m)**	**Range Y (m)**	**Range Z (m)**
S50	100	Same as 101								
	101	0.23	0.23	20	10	5	0.54	28	20	10
	102	Same as 101								
	103	Same as 101								
	104	Same as 101								
VC1	111	Same as 112								
	112	0.23	0.30	25	18	6	0.47	50	35	10
	113	Same as 114								
	114	0.30	0.41	25	15	5	0.29	40	30	10

			First structure				Second structure			
Zone	**Blockcode**	**Nugget**	**Sill**	**Range X (m)**	**Range Y (m)**	**Range Z (m)**	**Sill**	**Range X (m)**	**Range Y (m)**	**Range Z (m)**
VC6	123	0.14	0.30	18	10	5	0.56	35	20	10
	124	Same as 124								
South	130	Same as 132								
	131	Same as 132								
	132	0.05	0.64	15	15	5	0.31	45	32	8
	133	Same as 132								
	135	Same as 132								
B	140	0.26	0.12	16	14	5	0.62	30	29	10
Kiena Deep A	202	0.25	0.38	31	20	10	0.37	50	40	15
	204	0.15	0.60	22	16	8	0.25	40	25	15
	206	0.15	0.31	30	15	10	0.54	40	20	15
	212	Same as 202								
	214	Same as 204								
	218	0.11	0.41	35	18	10	0.48	50	29	15
	222	Same as 202								
	224	Same as 204								
	228	Same as 218								
	300	Same as 206								
Presqu'ile	1110	0.10	0.72	23	14	3	0.18	35	30	10
	1120									
	1125									
	1130									
	1140									
Dubuisson	1210	0.22	0.40	21	15	5	0.38	50	41	8
	1220									
	1230									
Martin	1310	0.15	0.37	20	20	10	0.48	46	36	15
NorthWest	1410	0.30	0.21	47	47	15	0.49	65	65	20
Wesdome	2010	0.21	0.32	44	17	7	0.47	67	41	10
	2020									
	2030									
	2040									
	2050									
	2060									
	2070									
	2110	0.30	0.31	23	23	3	0.39	60	60	10
	2120									

			First structure				Second structure			
Zone	**Blockcode**	**Nugget**	**Sill**	**Range X (m)**	**Range Y (m)**	**Range Z (m)**	**Sill**	**Range X (m)**	**Range Y (m)**	**Range Z (m)**
	2130									
	2210	0.16	0.55	25	25	4	0.29	50	50	10
	2220									
	2230									
	2240									
	2250									
	2260									
	2270									
	2280									
	2310	0.11	0.37	30	30	5	0.52	55	55	10
	2320									
	2330									

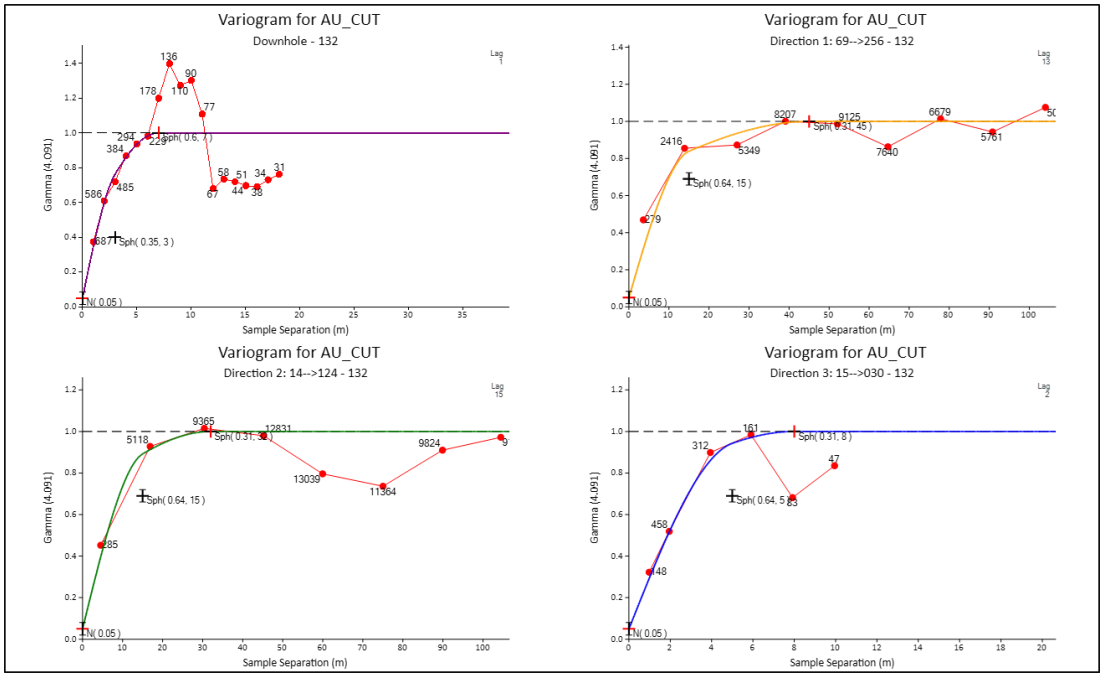


Figure 14-26: Example of the variography study for the South Zone 132

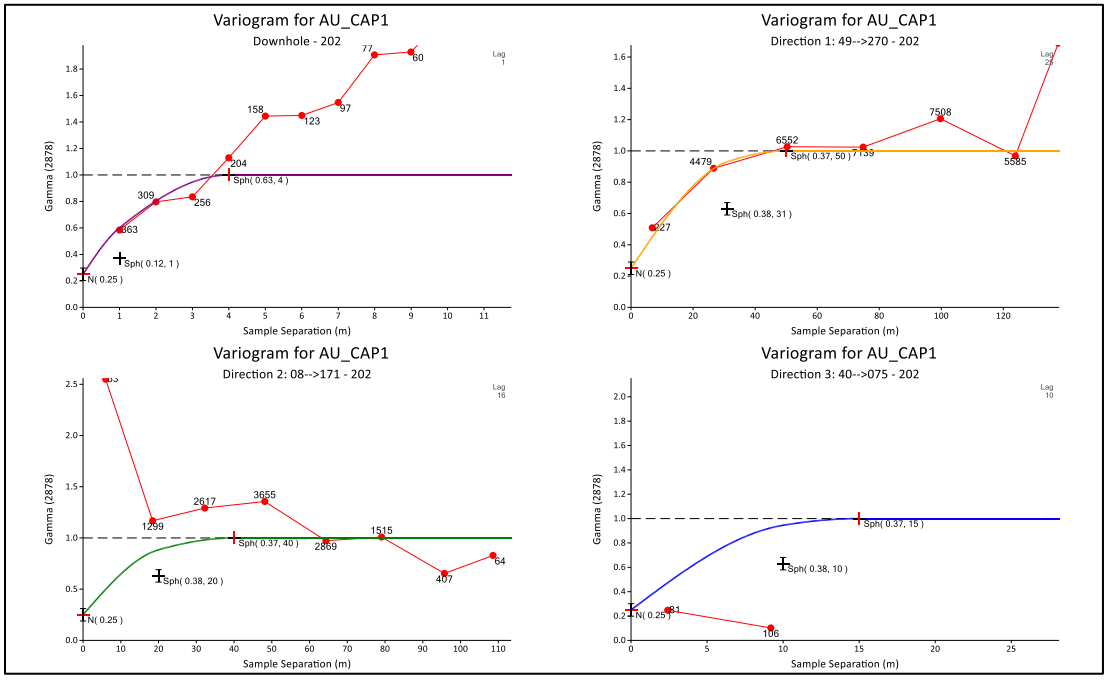


Figure 14-27: Example of the variography study for the Kiena Deep Zone A, domain 202

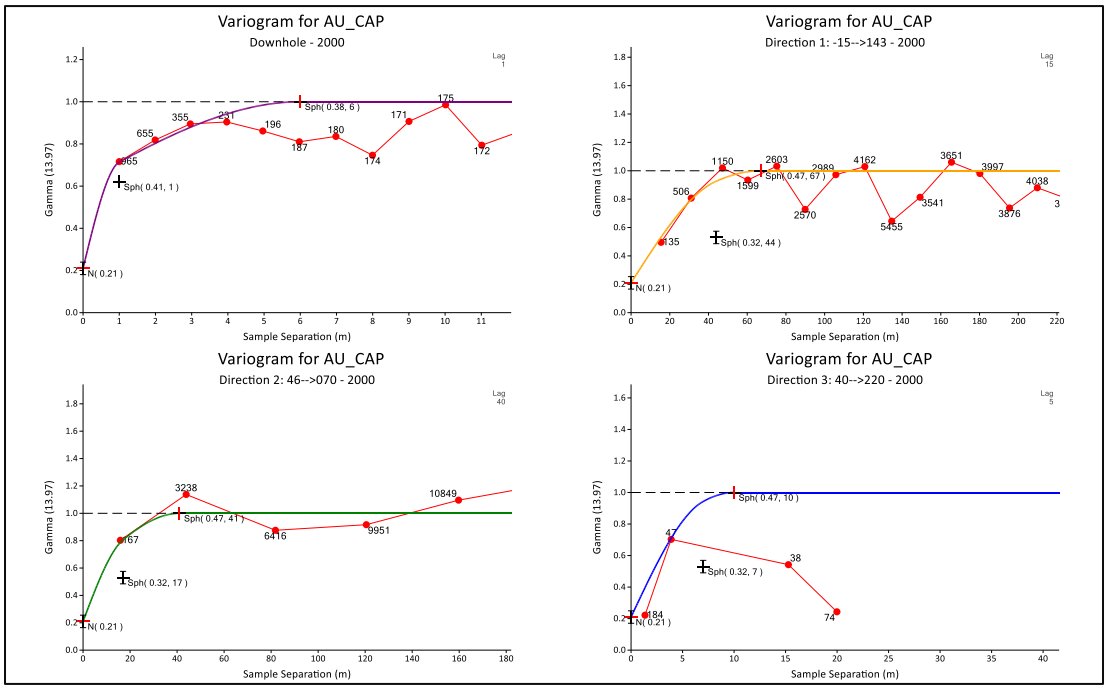


Figure 14-28: Example of the variography study for the Wesdome A Zone, lens 2000

14.6 Block Modelling

The block models for the Kiena Mine Complex were set in Datamine® Studio RM 1.6.87.

14.6.1 Block Model Parameters

For the current 2020 mineral resource estimate, six block models were created.

Table 14-6 provides the parameters used for each of the block models. Individual parent block cells have dimensions of 5 m long (X-axis) by 5 m wide (Y-axis) by 5 m vertical (Z-axis).

The size of the parent cells was chosen to best match the drilling pattern, thickness of the lenses, complexity of the geology model and underground mine planning.

The block model was coded using the sub-blocking method. Parent blocks were divided into sub-blocks when intersected wireframes. Sub-blocking allows you to subdivide the parent cells into smaller blocks to better fit the dimensions and volumes of the wireframes. A resolution of 5 m in each axis direction was applied in the division of the parent blocks for all of the block models. Consequently, the minimum sub-block size is 1.0 m long (X-axis) by 1.0 m wide (Y-axis) by 1.0 m vertical (Z-axis). All blocks falling within a wireframe were assigned the corresponding solid block code.

Table 14-6: Parameters of the Kiena Mine Complex block models

Kiena Deep S50_ZS_ZB			
Properties	**X (column)**	**Y (row)**	**Z (level)**
Origin coordinates	12,600	11,800	3,100
Parent block size (m)	5	5	5
Number of blocks	400	340	400
Block extent (m)	2,000	1,700	2,000
Min sub-cells size (m)	1	1	1

Wesdome			
Properties	**X (column)**	**Y (row)**	**Z (level)**
Origin coordinates	12,575	15,200	2,200
Parent block size (m)	5	5	5
Number of blocks	355	440	180
Block extent (m)	1,775	2,200	900
Min sub-cells size (m)	1	1	1

Presqu'ile			
Properties	**X (column)**	**Y (row)**	**Z (level)**
Origin coordinates	11,300	12,800	2,300
Parent block size (m)	5	5	5
Number of blocks	200	200	160
Block extent (m)	1,000	1,000	800
Min sub-cells size (m)	1	1	1

Martin			
Properties	**X (column)**	**Y (row)**	**Z (level)**
Origin coordinates	13,960	11,860	2,300
Parent block size (m)	5	5	5
Number of blocks	165	182	160
Block extent (m)	825	910	800
Min sub-cells size (m)	1	1	1

Dubuisson			
Properties	**X (column)**	**Y (row)**	**Z (level)**
Origin coordinates	15,850	11,700	2,100
Parent block size (m)	5	5	5
Number of blocks	340	220	200
Block extent (m)	1,700	1,100	1,000
Min sub-cells size (m)	1	1	1

NorthWest			
Properties	**X (column)**	**Y (row)**	**Z (level)**
Origin coordinates	11,890	13,120	2,300
Parent block size (m)	5	5	5
Number of blocks	228	212	160
Block extent (m)	1,140	1,060	800
Min sub-cells size (m)	1	1	1

14.6.2 Search Ellipsoid Strategy

The ranges of the ellipsoids used for the interpolation were established using the variography study and correspond to the range of the first structure for the first pass, to the second structure for the second pass and to 2 or 2.5 times the second structure for the third pass. In the Wesdome Zone, a fourth pass corresponding to four times the second structure was added as the drill spacing is larger in this zone. The third and fourth passes were designed to adequately populate all the block of the mineralized zones (Figure 14-29).

It is noteworthy to mention at this point that the classification was mostly based on drillhole spacing and, therefore, some interpolated blocks were not converted into the Inferred classification. Refer to section Mineral Resource Classification (Section 14.8) for more details.

Table 14-7 presents the orientation and ranges of the search ellipsoids for each pass.

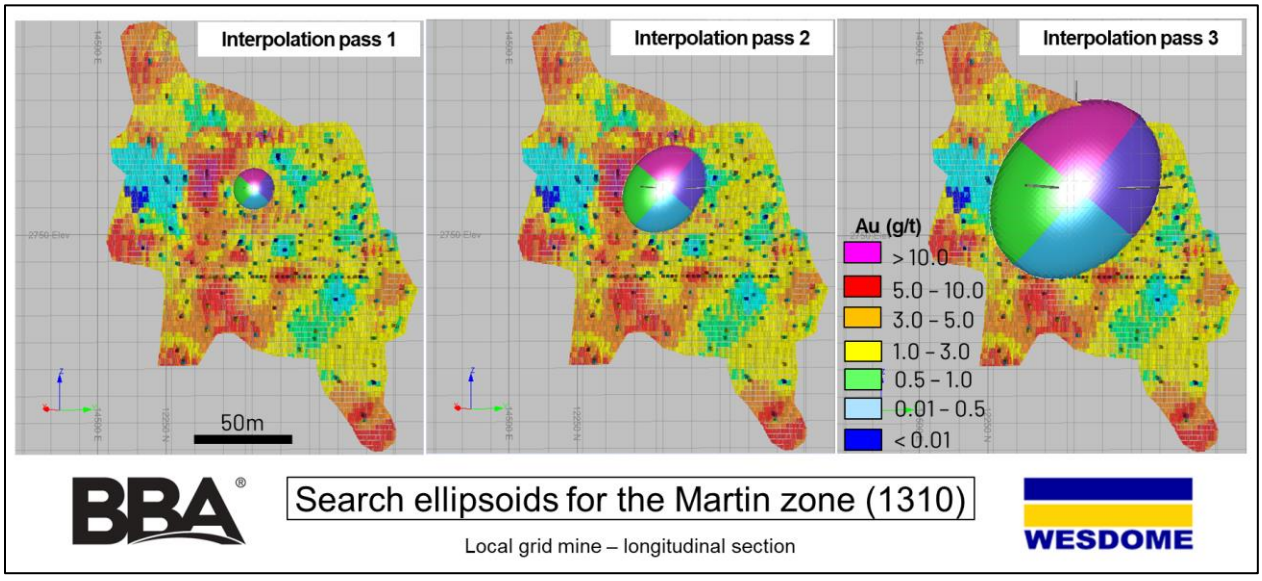


Figure 14-29: Examples of search ellipsoids for the Martin zone for the three interpolation passes

Table 14-7: Search ellipsoid ranges and orientation by interpolation passes

Zone	Blockcode	Interpolation pass	Datamine orientation			Search ellipsoid ranges			Capping
			Z	X	Z	X (m)	Y (m)	Z (m)	
S50	100 to 104	P1	55	55	110	20	10	10	25
		P2				28	20	10	
		P3				56	40	20	
VC1	111 to 112	P1	170	85	100	25	18	10	30
		P2				50	35	10	
		P3				100	70	20	
	113 to 114	P1	150	95	100	25	15	10	30
		P2				40	30	10	
		P3				80	60	20	
VC6	123 to 124	P1	160	80	130	18	10	10	25
		P2				35	20	10	
		P3				70	40	20	
South	130 to 135	P1	30	75	105	15	15	10	20
		P2				45	32	10	
		P3				90	64	20	
B	140	P1	70	60	110	16	14	10	20
		P2				30	29	10	
		P3				60	58	20	
Kiena Deep A	202	P1	75	50	100	31	20	10	265
		P2				50	40	15	100
		P3				100	80	30	60
	204	P1	115	50	90	22	16	10	265
		P2				40	25	15	100
		P3				80	50	30	60

Zone	Blockcode	Interpolation pass	Datamine orientation			Search ellipsoid ranges			Capping
			Z	X	Z	X (m)	Y (m)	Z (m)	
	206	P1	-160	90	-145	30	15	10	265
		P2				40	20	15	100
		P3				100	50	37.5	60
	212	P1	75	50	100	31	20	10	140
		P2				50	40	15	70
		P3				100	80	30	40
	214	P1	160	45	65	22	16	10	140
		P2				40	25	15	70
		P3				100	62.5	37.5	40
	218	P1	130	65	70	35	18	10	140
		P2				50	29	15	70
		P3				100	58	30	40
	222	P1	75	50	100	31	20	10	50
		P2				50	40	15	
		P3				100	80	30	
	224	P1	160	45	65	22	16	10	50
		P2				40	25	15	
		P3				100	62.5	37.5	
	228	P1	130	65	70	35	18	10	200
		P2				50	29	15	100
		P3				100	58	30	50
	300	P1	-160	90	-145	30	15	10	40
		P2				40	20	15	
		P3				100	50	37.5	

Zone	Blockcode	Interpolation pass	Datamine orientation			Search ellipsoid ranges			Capping
			Z	X	Z	X (m)	Y (m)	Z (m)	
Presquile	1110 to 1140	P1	60	55	120	23	14	10	55
		P2				35	30	10	
		P3				70	60	20	
Dubuisson	1210	P1	180	95	-100	21	15	10	50
		P2				50	41	10	
		P3				100	82	20	
	1220	P1	-150	85	100	21	15	10	50
		P2				50	41	10	
		P3				100	82	20	
	1230	P1	30	85	110	21	15	10	50
		P2				50	41	10	
		P3				100	82	20	
Martin	1310	P1	-125	85-	-130	20	20	10	30
		P2				46	36	15	
		P3				92	72	30	
NorthWest	1410	P1	55	70	115	47	47	15	NA
		P2				65	65	20	
		P3				130	130	40	
Wesdome	2010 to 2040	P1	-140	50	-160	44	17	10	30
		P2				67	41	20	
		P3				134	82	40	
		P4				268	164	80	
	2050, 2060	P1	-155	45	-150	44	17	10	30
		P2				67	41	20	
		P3				134	82	40	
		P4				268	164	80	

Zone	Blockcode	Interpolation pass	Datamine orientation			Search ellipsoid ranges			Capping
			Z	X	Z	X (m)	Y (m)	Z (m)	
	2070	P1	-155	50	-160	44	17	10	30
		P2				67	41	20	
		P3				134	82	40	
		P4				268	164	80	
	2110 to 2130	P1	-160	70	-120	23	23	10	30
		P2				60	60	20	
		P3				120	120	40	
		P4				240	240	80	
	2210 to 2280	P1	-150	70	-125	25	25	10	30
		P2				50	50	20	
		P3				100	100	40	
		P4				200	200	80	
	2310 to 2330	P1	-160	45	95	30	30	10	30
		P2				55	55	20	
		P3				110	110	40	
		P4				220	220	80	

14.6.3 Interpolation Parameters

Estimation and search parameters were evaluated through Kriging Neighbourhood Analysis (KNA).

KNA was conducted on each unit and on the most representative mineralized lenses or on a group of lenses sharing similar geological environment and azimuth/dip orientation with the Snowden Supervisor software. KNA provides a quantitative method of testing different estimation parameters (i.e. block size, discretization and min/max of composites used for the interpolation) by evaluating their impact on the quality of the results. The interpretation of these helps select the optimal value for each parameter.

Following this study, the parameters provided in Table 14-8 were chosen for the interpolation of the block models.

Table 14-8: Interpolation parameters

Zone	Interpolation pass	Minimum number of composites used	Maximum number of composites	Maximum number of composites used per drillhole	Minimum number of drillholes used
S50, VC1, VC6, South, B, Kiena Deep Zones, Presqu'ile, Dubuisson, Martin	Pass 1	12	24	4	3
	Pass 2	8	24	4	2
	Pass 3	4	24	4	1
NorthWest, Wesdome	Pass 1	9	12	3	3
	Pass 2	6	12	3	2
	Pass 3	4	12	3	2

14.6.4 Interpolation Methodology

The interpolation was performed on the capped composited data with three (or four, in the Wesdome Zone) passes characterized by increasing search ranges (Table 14-7).

For the ZA, ZA1 and 228 mineralized lenses (ZA2), a three-step capping process was applied on composites to limit extrapolation of very high-grade samples at higher distances (Table 14-7). In those cases, the first interpolation pass used composites with the highest capping value and following passes used lower capping values on composites. For example, in the Kiena Deep Zone A1 (blockcodes 212, 214 and 218), composites were capped at 140 g/t Au for the first pass, 70 g/t Au for the second pass, and 40 g/t Au for the third pass.

The block model grades were estimated using ordinary kriging (OK) methods, except for the Dubuisson N1_1220 and N2_1230 mineralized lenses that have been estimated using inverse distance squared (ID^2) methods as the ID^2 method honoured best the composites than the OK method.

Hard boundaries between the mineralized lenses were used in order to prevent grades from adjacent lenses being used during interpolation. As a block was estimated, it was tagged with the corresponding pass number.

For comparison purposes, additional grade models were generated using: 1) inverse distance squared (ID^2); 2) nearest neighbour (NN); and 3) OK on uncapped composited data.

14.7 Block Model Validation

Every step of the block modelling process was revised to ensure fair representation of the available data in the Project resource models.

More specific validations were completed on the block model including visual review of the interpolated grades in relation to the raw and composited data, checks for global and local bias, graphical validation (swath plots), statistical analysis of the model and comparison to other estimation methods.

14.7.1 Visual Validation

Block model grades were visually compared against drillhole composite grades and raw assays in cross-section, plan, longitudinal and 3D views (Figure 14-30 and Figure 14-31). This visual validation process also included confirming that the proper coding was done within the various mineralized lenses and checks for global and local bias.

The visual comparison shows that the block models are consistent and correlate well with the primary data without excessive smoothing.

Visual comparisons were also conducted between ID^2, OK and NN interpolation scenarios. The OK scenario produced a grade distribution honouring drillhole data and the style of mineralization observed on the Kiena Mine Complex for the majority of the zones and was therefore retained as the interpolation method for all the block models except for the block models of the Dubuisson N1_1220 and N2_1230 lenses.

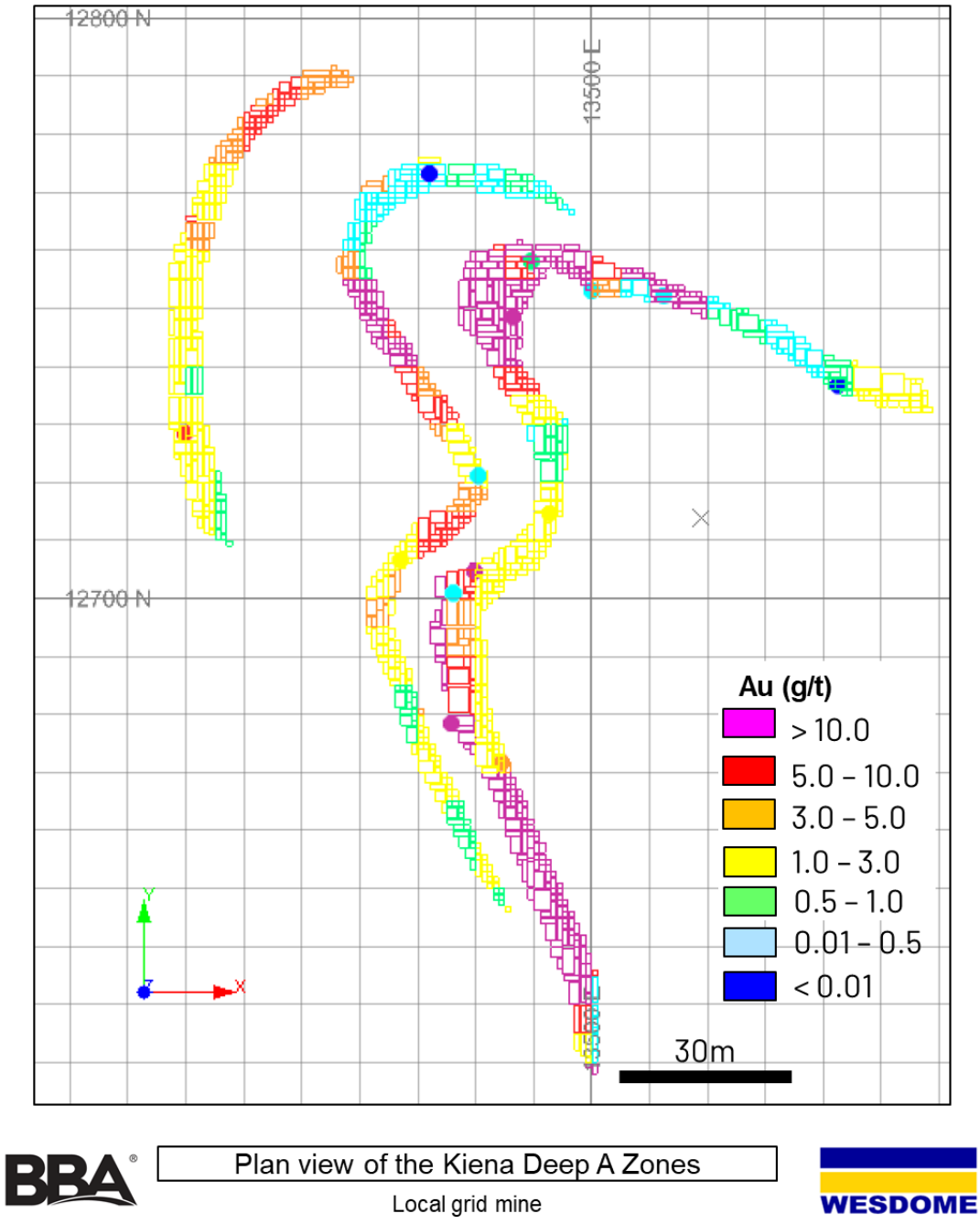


Figure 14-30: Comparative example of the grade distribution between the blocks and the composites in plan view

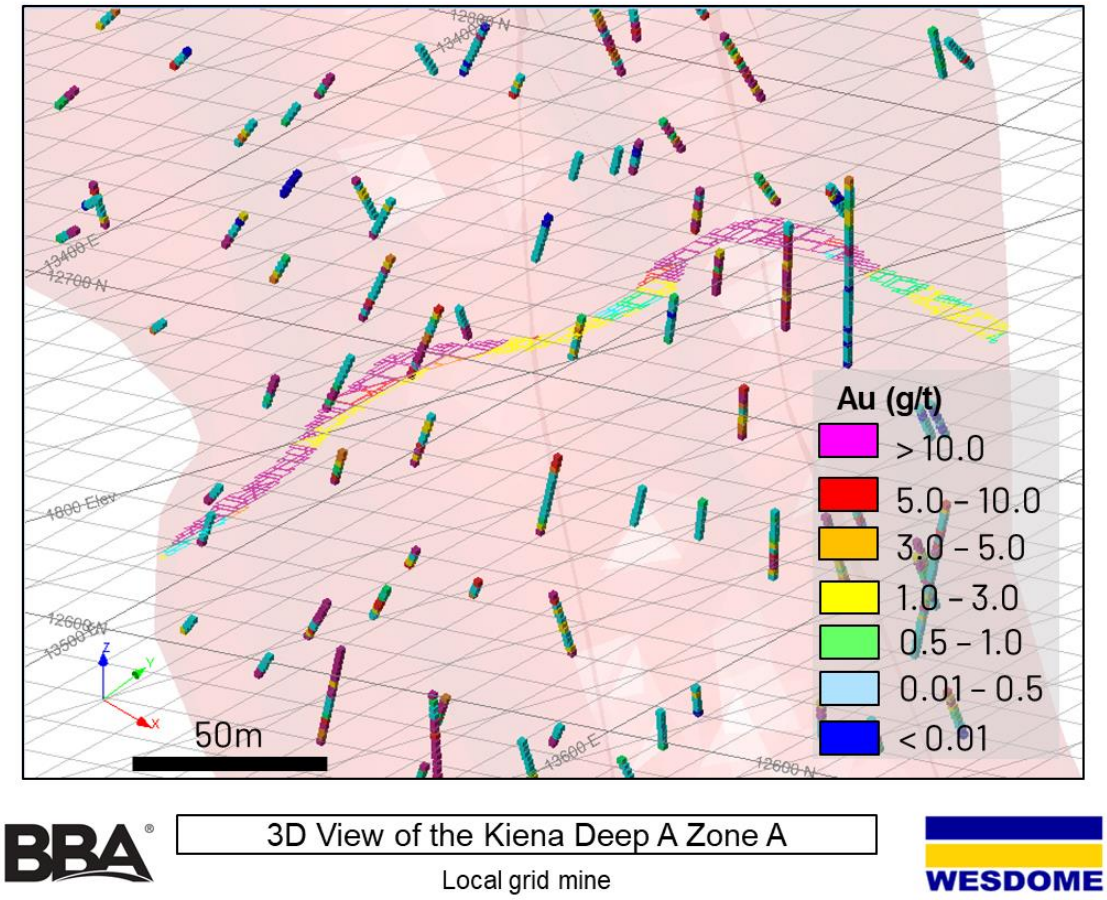


Figure 14-31: Comparative example of the grade distribution between the blocks and the composites in 3D view

14.7.2 Statistical Validation

Grade averages for the OK, NN and the ID^2 models were tabulated in Table 14-9. This comparison did not identify significant issues. As expected, block grade averages are generally lower than the composite grades and initial grades were well represented throughout the estimation process.

The average grades generated by the ID^2 interpolation method are very close to those reported from the OK interpolation method. This information provides a general indication that the resource model is reasonable.

Table 14-9: Comparison of the block and composite mean grades at a zero cut-off grade for Inferred and Indicated blocks

Lens	Blockcode	Number of composites	Capped composite mean (g/t Au) (1)	Number of blocks	OK grade model (g/t Au)	ID^2 grade model (g/t Au)	NN grade model (g/t Au)
S50_100	100	522	1.76	557	1.82	1.78	1.90
S50_101	101	559	1.93	1,759	1.81	1.75	1.83
S50_102	102	844	2.95	1,778	2.88	2.80	2.86
S50_103	103	315	1.51	1,148	1.38	1.37	1.30
S50_104	104	248	1.75	2,803	1.97	2.00	1.47
VC1_111	111	30	6.35	422	5.47	5.33	2.52
VC1_112	112	475	2.50	2,451	2.50	2.36	2.98
VC1_113	113	97	3.26	1,036	2.39	2.68	2.45
VC1_114	114	420	2.21	1,102	0.88	0.91	0.73
VC6_123	123	245	2.46	1,214	2.42	2.50	2.40
VC6_124	124	139	1.51	1,076	1.48	1.57	1.37
ZS_130	130	613	1.41	8,158	1.29	1.32	1.46
ZS_131	131	599	1.70	4,262	1.79	1.88	2.09
ZS_132	132	788	1.18	3,490	1.31	1.30	1.33
ZS_133	133	170	1.33	2,655	1.57	1.57	1.50
ZS_135	135	172	1.28	1,413	1.30	1.35	1.44
ZB_140	140	192	1.63	3,395	1.56	1.71	1.48
ZA_200	200	2063	13.86	6,713	12.62	11.70	14.19
ZA1_210	210	1473	8.06	6,608	7.91	7.39	8.79
ZA2_220	220	674	14.20	4,461	10.33	10.14	9.87
H1ZA_300	300	188	3.90	711	3.12	3.70	2.02
PI_1_1110	1110	24	3.35	728	3.00	3.02	3.07
PI_2_1120	1120	42	5.04	1,548	5.62	5.56	5.75
PI_2A_1125	1125	21	6.63	746	6.63	6.81	5.08

Lens	Blockcode	Number of composites	Capped composite mean (g/t Au) (1)	Number of blocks	OK grade model (g/t Au)	ID^2 grade model (g/t Au)	NN grade model (g/t Au)
DUB_1210	1210	115	6.53	2,189	5.72	5.74	6.08
DUB_N1_1220 (2)	1220	73	6.13	2,193	6.57	6.36	5.55
DUB_N2_1230 (2)	1230	40	4.21	1,129	5.00	4.86	5.56
MART_1310	1310	774	2.79	5,051	2.80	3.01	2.22
NW_1410	1410	110	2.84	5,239	2.71	2.77	2.70
WES_A_2010	2010	766	2.25	16,798	2.50	2.51	2.38
WES_A1_2020	2020	58	1.90	1,609	2.38	2.30	2.14
WES_AF_2030	2030	169	1.22	3,186	1.46	1.39	1.43
WES_AH_2040	2040	195	1.21	3,786	1.23	1.21	0.96
WES_B_2110	2110	181	1.73	7,973	1.52	1.49	1.34
WES_C_2120	2120	53	1.38	922	1.21	1.23	1.54
WES_D_2130	2130	75	1.40	1,537	1.54	1.43	2.63
WES_E_2210	2210	280	2.12	11,927	2.41	2.33	2.25
WES_E0_2220	2220	66	2.59	2,957	2.76	2.81	3.94
WES_E1_2230	2230	82	2.13	4,938	1.59	1.62	1.30
WES_E3_2240	2240	57	1.97	5,043	2.26	2.07	2.13
WES_E4_2250	2250	53	2.57	3,127	3.13	3.34	4.00
WES_F2_2260	2260	54	2.70	2,183	2.92	2.70	3.41
WES_F4_2270	2270	53	1.60	1,433	2.02	2.01	1.48
WES_L_2310	2310	93	2.09	2,554	1.52	1.49	1.40
WES_F_2320	2320	38	3.04	1,370	1.76	1.48	2.29

Notes to Table 14-9:

(1) When Multi-capping was used, the statistics presented in this table are the one of the AU_CAP1 field.

(2) The ID^2 interpolation method was used for these mineralized lenses for the current MRE.

14.7.3 Swath Plots

Swath plots were also generated as part of the block model validation. A swath plot is a graphical display of the grade distribution derived from a series of bands (or swaths), generated in several directions throughout the deposit. Using the swath plots, grade variations from the OK model are compared to the distribution of grade interpolated with the NN and ID^2 methods and to the composite grades. This validation method also works as a visual mean to identify possible bias in the interpolation.

Figure 14-32 to Figure 14-34 illustrate a series of swath plots in the three directions for the Wesdome 2010 lens. Generally, the grades estimated in the blocks are close to the average grades provided by the data source; no bias was found in the resource estimate in this regard.

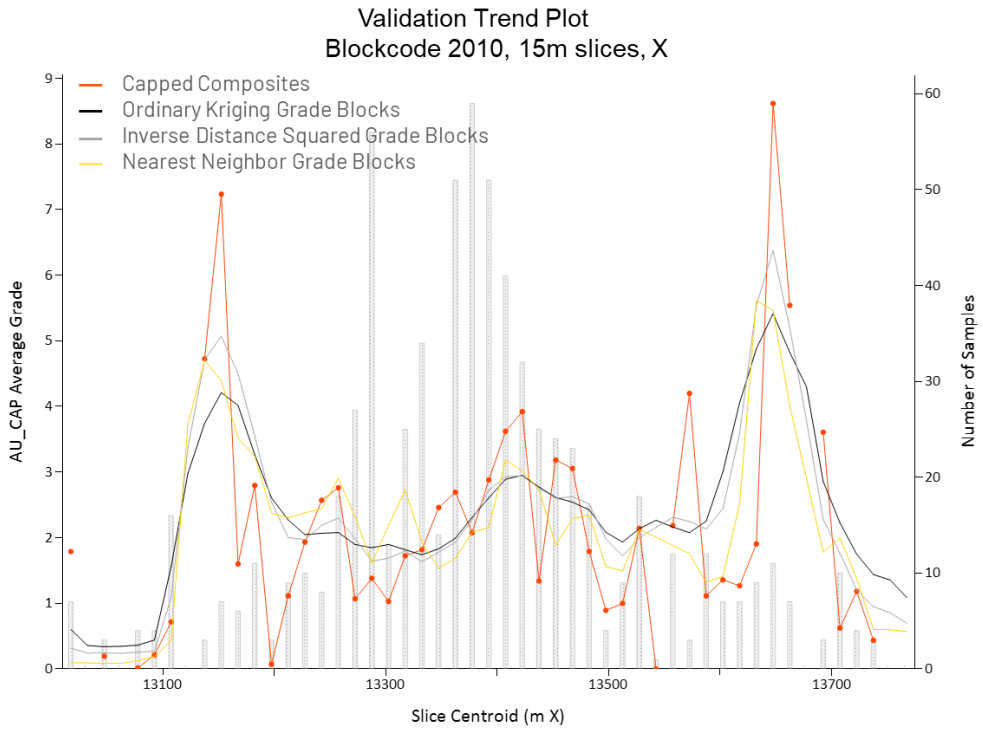


Figure 14-32: Block model validation swath plot along strike (X-direction) for the Wesdome 2010 lens

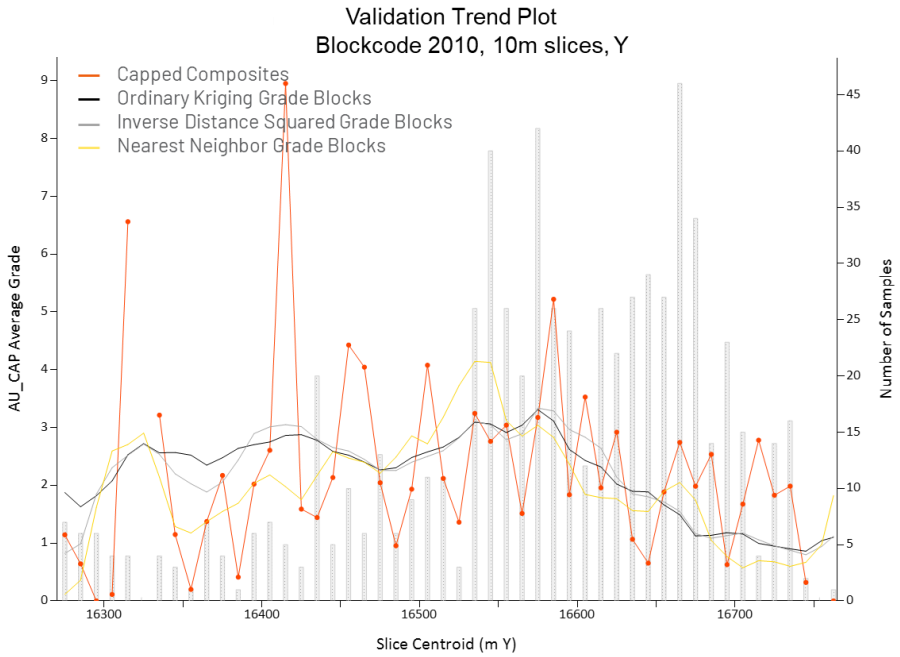


Figure 14-33: Block model validation swath plots across strike (Y-direction) for the Wesdome 2010 lens

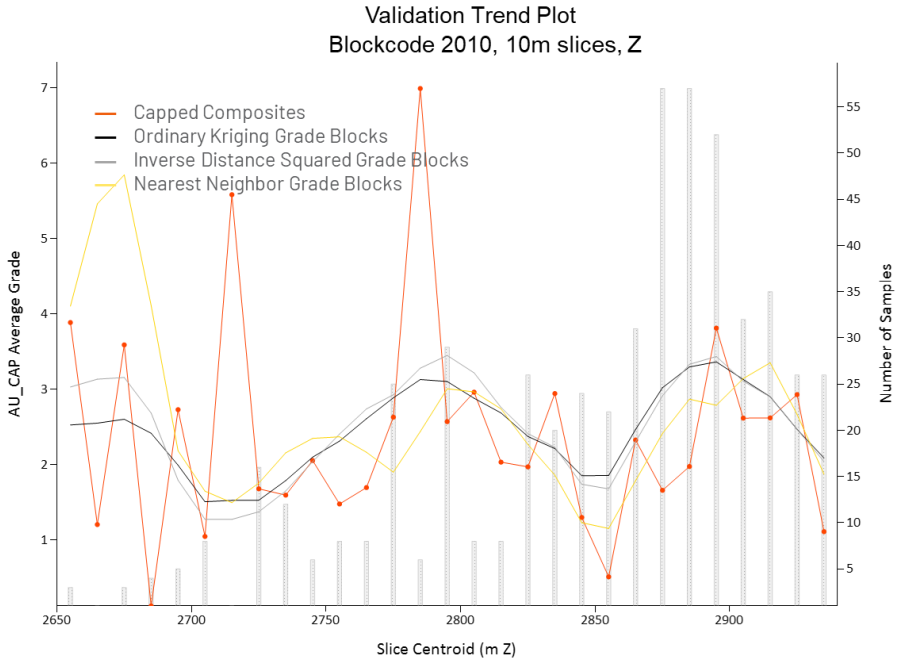


Figure 14-34: Block model validation swath plots along elevation (Z-direction) for the Wesdome 2010 lens

Based on visual and statistical reviews, it is the QPs opinion that the Kiena Mine Complex block models provide a reasonable estimate of in situ gold resources.

14.8 Mineral Resource Classification

The mineral resources for the Kiena Mine Complex were classified in accordance with CIM Standards.

14.8.1 Mineral Resource Definition

The "CIM Definition Standards for Mineral Resources and Reserves" prepared by the CIM Standing Committee on Resource Definitions and adopted by the CIM council on May 10, 2014, provides standards for the classification of Mineral Resources and Mineral Reserves estimates as follows:

Inferred Mineral Resource:

*An **Inferred Mineral Resource** is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.*

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Indicated Mineral Resource:

*An **Indicated Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.*

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Measured Mineral Resource:

*A **Measured Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.*

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

14.8.2 Mineral Resource Classification for the Kiena Mine Complex

Following the previous definitions, the estimated block grades were classified into Inferred and Indicated Mineral Resource categories using drill spacing, a minimum number of drillhole and recognition of grade and geological continuity within the lenses.

No Measured resources were defined for the Kiena Mine Complex at this stage.

The following parameters were used for the classification for all mineralized lenses in the Kiena Mine Complex except for the Kiena Deep A Zone:

- Inferred Mineral Resources were defined for blocks within the units that have been informed by a minimum of two drillholes within 40 m of a drillhole (80 m of drill spacing);
- Indicated Mineral Resources were defined for blocks within the units that have been informed by a minimum of three drillholes within 12.5 m of a drillhole (25 m of drill spacing).

For the classification of the Kiena Deep A Zone, the following parameters were used:

- Inferred Mineral Resources were defined for blocks within the units that have been informed by a minimum of two drillholes within 40 m of a drillhole (80 m of drill spacing);
- Indicated Mineral Resources were defined for blocks within the units that have been informed by a minimum of three drillholes within 15 m of a drillhole (30 m of drill spacing).

When needed, a series of clipping boundaries were created manually in longitudinal views to either upgrade or downgrade classification in order to homogenize the groups of resources by removing artificial features and isolated blocks or group of blocks due to automatically generated classification. All remaining estimated but unclassified blocks were flagged as “Exploration Potential”.

Figure 14-35 shows an example of the classification.

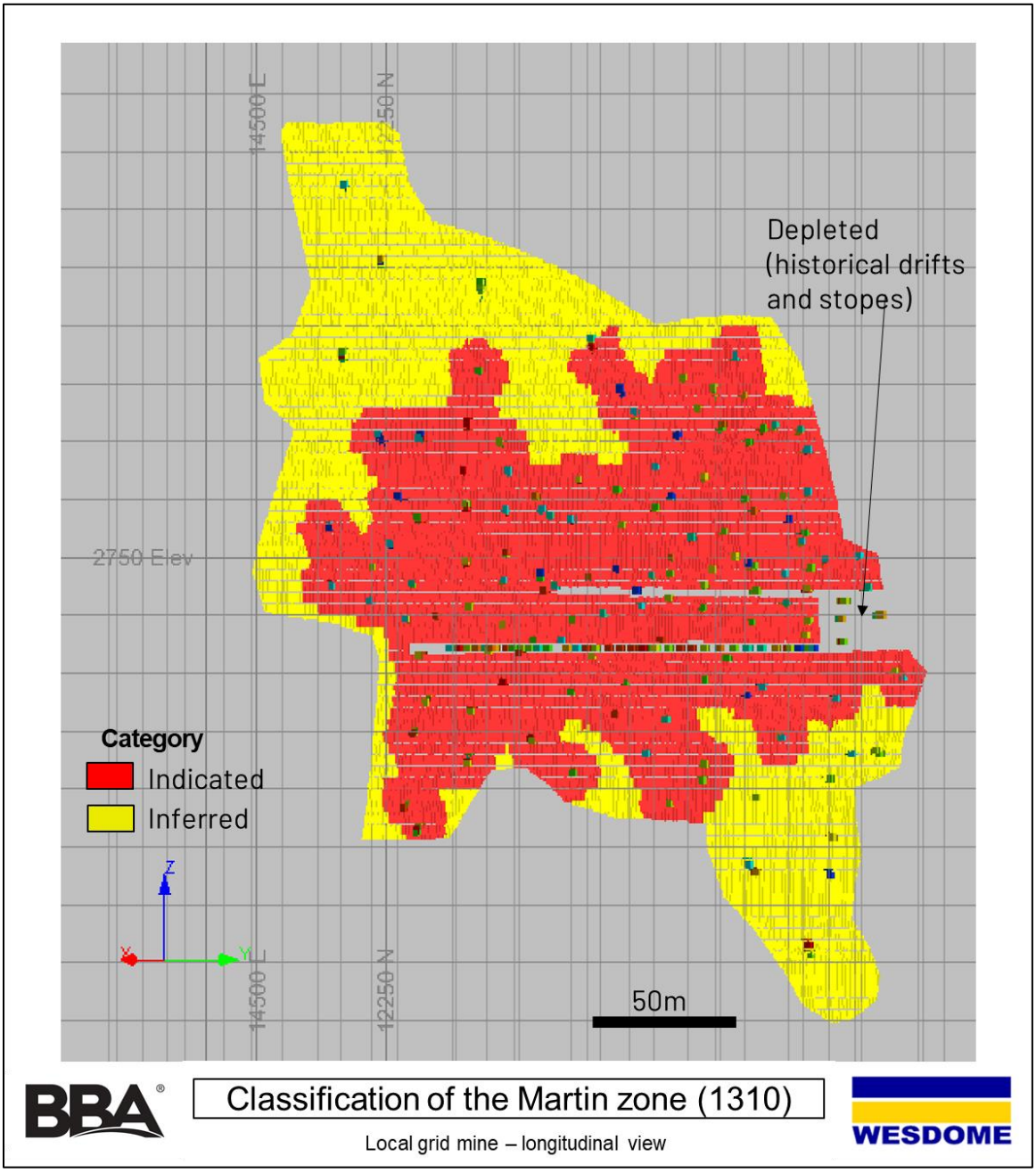


Figure 14-35: Mineral Resource classification example for Martin Zone (1310)

14.9 Reasonable Prospect for Economics

For this Project, mineral resources were estimated for an underground scenario.

According to the CIM best practice guidelines (2019), "Mineral Resource statements for **underground mining** scenarios must satisfy the "reasonable prospects for eventual economic extraction" by demonstration of the spatial continuity of the mineralization within a potentially mineable shape. In cases where this potentially mineable volume contains smaller zones of mineralization with grades or values below the stated cut-off (sometimes referred to as "must take" material), this material must be included in the Mineral Resource estimate. At a minimum, these constraints can be addressed by creation of constraining volumes."

14.9.1 Cut-off Grade

According to CIM's best practice guidelines (2019), the cut-off grade should consider the following (CIM, 2019):

- Reasonable long-term commodity price(s);
- Assumed mining methods;
- Exchange rate(s);
- Mineral process recovery; and
- Operating costs relating to mining, processing, general and administration, smelter terms, and royalties, among others.

Additional considerations include deposit location and scale, geologic and grade continuity, environmental and social considerations, and waste disposal costs.

In addition, according to the CIM best practice guideline (2019), "variations of rock characteristics, metallurgy, mining methods, processing methods, etc. within the mineral resource model may necessitate more than one cut-off grade or economic limit for different parts of the deposit".

The cut-off grade parameters presented in Table 14-10 are based on discussions with Wesdome and benchmarking against similar projects. Gold price and exchange rates were established using the three-year moving average method as of October 31, 2020.

The underground cut-off grade used for the Mineral Resource Estimate was 2.80 g/t Au for the mineralized lenses with >40° dip and 3.60 g/t Au for the shallow-dipping mineralized lenses (<40° dip).

It should be noted that all parameters are either based on similar projects or reasonable technical and economic factors. The QPs of this report section believe that the calculated cut-off grades and the parameters used are relevant for a mineral resource estimate, as they are relevant to the grade distribution of the Project and that the mineralization exhibits sufficient continuity.

Table 14-10: Cut-off grade parameters

Parameter	Unit	Dip (> 40°)	Dip (< 40°)
Operating Costs			
Mining Cost - Rock	CAD/t mined	100	150
Processing Cost	CAD/t processed	40	40
General & Administration	CAD/t processed	25	25
Transportation Costs	CAD/t processed	0	0
Revenue			
Selling Price	USD/oz	1,450.00	1,450.00
Exchange Rate	USD/CAD	1.32	1.32
Selling Price	CAD/oz.	1,914.00	1,914.00
Selling Cost	CAD/oz.	5.00	5.00
Conversion Factor	g/oz.	31.1035	31.1035
Mill parameters			
Mill Recovery	%	0.97	0.97
Cut-off grade	g/t	2.77	3.61
Cut-off grade (rounded)	**g/t**	**2.80**	**3.60**

14.9.2 Underground Material

For the current 2020 MRE, the QPs have validated that sufficient continuity is found in the reported blocks, but no mineable shape were used to declare the 2020 MRE. Clipping boundaries were however used to classify the resource, and a total of 10 mineralized lenses were also excluded from the 2020 MRE statement based on their lack of continuity above the cut-off grade or the absence of Inferred resources due to the insufficient drill spacing (Table 14-11).

Table 14-11: Mineralized lenses excluded from the MRE statement

Zone	Mineralized lens
Presqu'ile	PI_3
	PI_4
Wesdome	W_AH1
	W_AH2
	W_AH3
	W_B
	W_D
	W_E1
	W_E3
	W_F

14.10 Kiena Mine Complex Mineral Resource Estimate

The Indicated and Inferred 2020 Mineral Resource Estimate (inclusive of mineral reserves) for the Kiena Mine Complex is presented in Table 14-12:

Table 14-12: Indicated and Inferred Kiena Mine Complex 2020 Mineral Resource Estimate per zone (inclusive of mineral reserves)

Zone	Cut-off grade	Indicated Resources			Inferred Resources		
		Tonnage	Grade	Ounces	Tonnage	Grade	Ounces
	(Au g/t)	(t)	(Au g/t)	(Au oz)	(t)	(Au g/t)	(Au oz)
Kiena Deep A	2.8	1,280,000	17.50	720,000	325,700	11.49	120,400
S50	2.8	146,600	4.54	21,400	100,000	3.73	12,000
VC	2.8	137,700	4.79	21,200	169,500	5.25	28,600
ZB	2.8	-	-	-	74,000	4.12	9,800
South	2.8	63,200	4.15	8,400	226,800	3.85	28,000
Presqu'ile	2.8	-	-	-	255,600	6.70	55,100
Dubuisson	2.8	-	-	-	744,600	6.69	160,200
Martin	2.8	163,100	4.77	25,000	109,100	4.29	15,000
North West	2.8	-	-	-	285,800	4.03	37,100
Wesdome	3.6	-	-	-	1,129,400	5.27	191,400
Total		**1,790,600**	**13.83**	**796,000**	**3,420,500**	**5.98**	**657,600**

Notes to Table 14-12:

(1) The independent qualified persons for the 2020 MRE, as defined by National Instrument ("NI") 43-101 guidelines, are Pierre-Luc Richard, P. Geo., and Charlotte Athurion, P. Geo., both of BBA Inc. The effective date of this MRE is October 31, 2020.

(2) These mineral resources are not mineral reserves as they do not have demonstrated economic viability.

(3) These mineral resources are inclusive of mineral reserves.

(4) The mineral resource estimate follows CIM definitions (2014) for mineral resources.

(5) Results are presented in situ and undiluted and considered to have reasonable prospects for economic extraction, below a 100 m crown pillar.

(6) The resources include 46 mineralized lenses with a minimum true thickness of 3.0 m (2.4 m for Wesdome Zones) using the grade of the adjacent material when assayed or a value of zero when not assayed. High-grade capping varies from 20 to 265 g/t Au (when required) and was applied to composited assay grades for interpolation using an Ordinary Kriging interpolation method (ID^2 for Dubuisson lenses 1220 and 1230) based on 1.0 m composite and block size of 5 m x 5 m x 5 m, with bulk density values of 2.8 (g/cm^3). A three-step capping strategy was applied, where capping value decreased as interpolation search distance increased, in order to restrict high-grade impact at greater distance. Indicated resources are manually defined and encloses areas where drill spacing is generally less than 30 m, blocks are informed by a minimum of three drillholes, and reasonable geological and grade continuity is shown.

(7) The estimate is reported for potential underground scenario at cut-off grades of 2.8 g/t Au (>40 degree dip) and 3.6 g/t Au (<40 degree dip, Wesdome zone only). The cut-off grades were calculated using a gold price of US$1,450 per ounce, a USD:CAD exchange rate of 1.32 (resulting in C$1,914 per ounce gold price); mining cost C$100/t (>40 degree dip); C$150/t (<40 degree dip); processing cost C$40/t; G&A C$25/t.

(8) The number of metric tonnes and ounces were rounded to the nearest hundred and the metal contents are presented in troy ounces (tonne x grade/31.10348). Rounding may result in apparent summation differences between tonnes, grades and contained metals content.

(9) The QPs are not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues, or any other relevant issues not reported in this Report that could materially affect the mineral resource estimate.

Note that some minor changes were made in the final tabulation for the Kiena Deep A and South Zones since the 2020 MRE released in January 2021 corresponding to two mineralized lenses that were inadvertently not included during the final tabulation step.

Table 14-13 shows the sensitivity of the 2020 MRE of the Kiena Deep A Zone (ZA, ZA1, ZA2 and H1ZA) to grade cut-off for the in situ underground MRE.

The reader is cautioned that the numbers presented in the following table should not be misconstrued with a mineral resource statement.

Figure 14-36 shows a 3D view of the grade distribution and classification of the Kiena Deep A Zone, VC1 and VC6 Zones.

Table 14-13: Kiena Deep A Zones mineral resource estimate sensitivity table (namely lenses ZA, ZA1, ZA2 and H1ZA) - inclusive of Mineral Reserves

Cut-off grade (Au g/t)	Indicated Resources			Inferred Resources		
	Tonnage (t)	Grade (Au g/t)	Ounces (Au oz)	Tonnage (t)	Grade (Au g/t)	Ounces (Au oz)
> 4.5 g/t	1,056,500	20.43	694,000	239,500	14.37	110,600
> 4.0 g/t	1,115,300	19.58	702,000	258,700	13.61	113,200
> 3.5 g/t	1,182,100	18.68	710,100	281,700	12.81	116,000
> 3.0 g/t	1,252,200	17.82	717,400	307,400	12.01	118,700
> 2.8 g/t	**1,280,000**	**17.50**	**720,000**	**325,700**	**11.49**	**120,400**
> 2.5 g/t	1,330,700	16.93	724,300	341,900	11.08	121,800

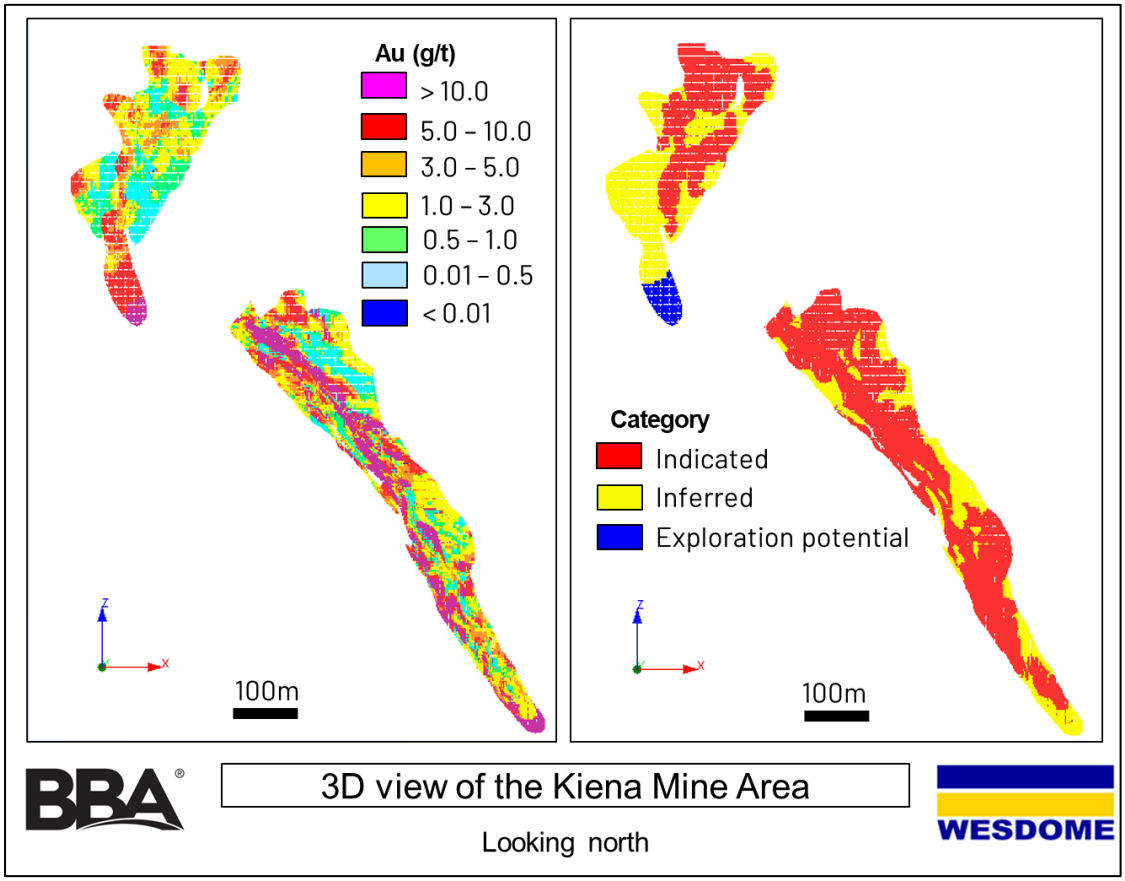


Figure 14-36: Example of 3D views showing grade distribution and classification of the mineralized lenses in the Kiena Mine area (VC1, VC6, Kiena Deep)

The Indicated and Inferred 2020 Mineral Resource Estimate (exclusive of mineral reserves) for the Kiena Mine Complex is presented in Table 14-14. In this case, mineral resources included in the mineral reserve solids designed to support the prefeasibility study (Figure 14-37 and Figure 14-38) were removed from the mineral resource estimate. Mineral reserve solids were considered for the Kiena Deep A, S50 and Martin Zones.

Table 14-14: Indicated and Inferred Kiena Mine Complex 2020 Mineral Resource Estimate per zone (exclusive of mineral reserves)

Zone	Cut-off grade (Au g/t)	Indicated Resources			Inferred Resources		
		Tonnage (t)	Grade (Au g/t)	Ounces (Au oz)	Tonnage (t)	Grade (Au g/t)	Ounces (Au oz)
Kiena Deep A	2.8	281,400	11.65	105,400	311,200	11.22	112,200
S50	2.8	69,300	3.82	8,500	99,300	3.72	11,900
VC	2.8	137,700	4.79	21,200	169,500	5.30	28,600
ZB	2.8	-	-	-	74,000	4.10	9,800
South	2.8	63,200	4.15	8,400	226,800	3.85	28,000
Presqu'ile	2.8	-	-	-	255,600	6.70	55,100
Dubuisson	2.8	-	-	-	744,600	6.70	160,200
Martin	2.8	92,100	4.38	13,000	108,200	4.28	14,900
North West	2.8	-	-	-	285,800	4.00	37,100
Wesdome	3.6	-	-	-	1,129,400	5.30	191,400
Total		**643,700**	**7.56**	**156,500**	**3,404,400**	**5.94**	**649,200**

Notes to Table 14-14:

(1) The independent qualified persons for the 2020 MRE, as defined by National Instrument ("NI") 43-101 guidelines, are Pierre-Luc Richard, P. Geo., and Charlotte Athurion, P. Geo., both of BBA Inc..

(2) These mineral resources are not mineral reserves as they do not have demonstrated economic viability.

(3) These mineral resources are exclusive of mineral reserves.

(4) The mineral resource estimate follows CIM definitions (2014) for mineral resources.

(5) Results are presented in situ and undiluted and considered to have reasonable prospects for economic extraction, below a 100 m crown pillar.

(6) The resources include 46 mineralized lenses with a minimum true thickness of 3.0 m (2.4 m for Wesdome Zones) using the grade of the adjacent material when assayed or a value of zero when not assayed. High-grade capping varies from 20 to 265 g/t Au (when required) and was applied to composited assay grades for interpolation using an Ordinary Kriging interpolation method (ID^2 for Dubuisson lenses 1220 and 1230) based on 1.0 m composite and block size of 5 m x 5 m x 5 m, with bulk density values of 2.8 (g/cm^3). A three-step capping strategy was applied, where capping value decreased as interpolation search distance increased, in order to restrict high-grade impact at greater distance. Indicated resources are manually defined and encloses areas where drill spacing is generally less than 30 m, blocks are informed by a minimum of three drillholes, and reasonable geological and grade continuity is shown.

(7) The estimate is reported for potential underground scenario at cut-off grades of 2.8 g/t Au (>40 degree dip) and 3.6 g/t Au (<40 degree dip, Wesdome Zones only). The cut-off grades were calculated using a gold price of US$1,450 per ounce, a USD:CAD exchange rate of 1.32 (resulting in C$1,914 per ounce gold price); mining cost C$100/t (>40 degree dip); C$150/t (<40 degree dip); processing cost C$40/t; G&A C$25/t.

(8) The number of metric tonnes and ounces were rounded to the nearest hundred and the metal contents are presented in troy ounces (tonne x grade/31.10348). Rounding may result in apparent summation differences between tonnes, grades and contained metals content.

(9) The QPs are not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues, or any other relevant issues not reported in this Technical Report that could materially affect the mineral resource estimate.

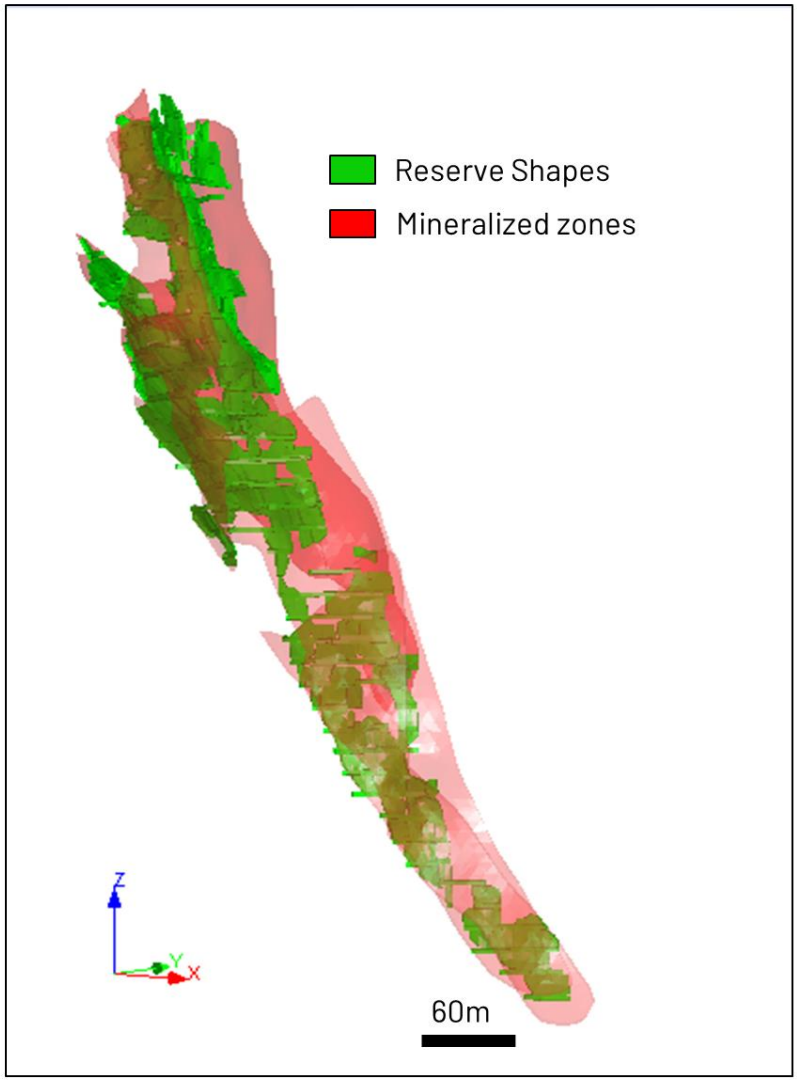


Figure 14-37: 3D view showing wireframes of reserve shapes and mineralized lenses in the Kiena Deep A Zone

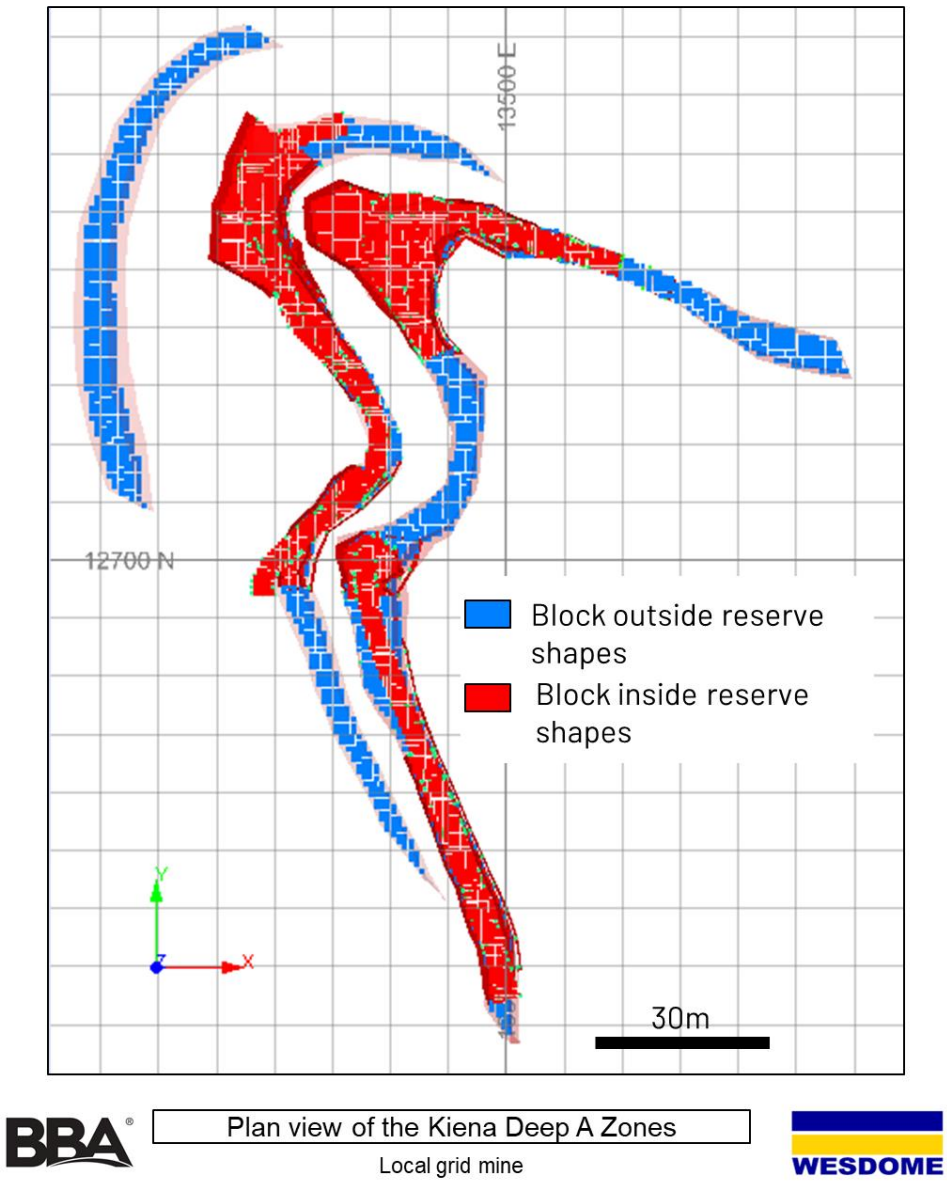


Figure 14-38: Plan view showing the block coding inside and outside the reserve shapes in the Kiena Deep Zone

15. MINERAL RESERVE ESTIMATE

The Mineral Reserve estimate for the Kiena Mine Complex was established and verified jointly by Michael Stochmal, P.Eng. (QP) and members of the Wesdome Kiena Mine Complex technical services team. The effective declaration date of the Mineral Reserve estimate is April 12, 2021.

The calculation and verification of the Kiena Mine Complex Mineral Reserve estimate began with the Kiena Mine Complex technical services team providing the following information:

- The validated mineral resource block models, representing prospective mining zones at the Kiena Mine Complex, containing material density, gold grades, block volumes, and resource confidence classifications. The validated mineral resource block models were provided in Datamine .dm format;
- The manually designed underground mineral reserve solids dependent on Indicated Mineral Resource;
- The mine as-built solids reflecting the Mineral Reserve estimate;
- The planned mine development solids required for the operation of the mine and extraction of the mineral reserve solids.

15.1 Mineral Reserve Estimate by Mineralized Zone

Table 15-1: Mineral reserve estimate by mineralized zone

Zone	Diluted Ore Tonnes (t)	Diluted Ore Grade (Au g/t)	Contained Metal (Au oz)
Kiena Deep A	1,387,100	12.95	577,300
S50	95,100	4.12	12,600
Martin	92,200	4.00	11,900
Probable Mineral Reserve	**1,574,400**	**11.89**	**601,800**

Notes:
1. CIM Definitions Standards on Mineral Resource and Reserves (2014) were followed.
2. Underground Mineral Reserves are diluted tonnes and grades; the reference point is the mill feed at the primary crusher.
3. Cut-off grade considers a gold price of C$1,914 per ounce (US$1,450 per ounce at 1.32 Exchange Rate USD to CAD) for mine design purposes, a 97% metallurgical processing recovery for both the S50 & Martin Zones and 98.5% for the Kiena Deep A Zone, life of mine operating cost of C$122.92/t mining, C$28.25/t processing, C$36.53/t General and Administration. An incremental cut-off grade excluding the mine operating cost was not considered for lateral drift development required through mineralization and would present opportunity to increase the Mineral Reserve estimate.
4. A minimum mining width of 3.0 m, and minimum footwall angle of 45° was used in the creation of all mineral reserve solids. Longitudinal long hole stoping is the predominant method considered for production mining. The life of mine mining recovery factor is 90% and combined planned and unplanned dilution factor is 27%.
5. A bulk density of 2.8 t/m^3 was used for both ore and waste rock.
6. The application to expand the mining concession to include Martin Zone is being pursued by Wesdome. Martin Zone is scheduled to begin development and production mining in 2024 to provide time for the permitting process to be approved.
7. Diluted ore tonnes and gold ounces were rounded to the nearest hundred. Numbers may not add due to rounding.

The Martin Zone mining concession is being pursued by Wesdome. The permit to mine is expected to be received however, this mining zone was scheduled to start in 2024 to mitigate any potential permitting risk. The QP is not aware of any other mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Mineral Reserve estimate.

Metal commodity prices used for the Mineral Reserve estimate are based on the three year trailing average.

15.2 Mineral Reserve Estimate Calculation Methodology

The Mineral Reserve estimate for the Kiena Mine Complex was established with the manual design of prospective mineral reserve solids with mining constraints applied. The mineral reserve solids are dependent on Indicated Mineral Resource classification represented in the block models. The Mineral Reserve estimate is based upon the long hole production mining method that was historically used at the Kiena Mine Complex. The Wesdome technical services team began the Mineral Reserve estimate by creating mineable production stope shapes on a 5-m interval within the Indicated Mineral Resource boundaries. These stope designs were prepared in Promine software and validated in Datamine. The stope sizing constraints were determined to be a minimum mining width of 3 m and a footwall angle greater than 45°, based on underground mobile equipment restrictions and long hole mining method best practice. In order to enhance stability, the maximum stope sizes were then determined based on the geomechanical analysis of the lithology associated with each mineralized zone and lens. The geomechanical analysis is further discussed in Chapter 16 – Mining Methods. The finalized stope dimensions are listed in Table 15-2 and represent the typical stope geometry for each lens respectively by mineralized zone.

Table 15-2: Production stope representative undiluted tonnes and dimensions by mineralized zone

Zone	Lens	Average Undiluted Stope Tonnes (t)	Strike Length (m)	Vertical Height (m)	Average Width (m)
Kiena Deep A	ZA	4,589	20	18	4.6
	ZA1	2,116	10	18	4.2
	ZA2	2,137	10	18	4.2
	H1ZA	1,869	10	18	3.7
S50	S50	3,810	15	18	5.0
Martin	Martin	3,360	20	20	3.0

Mineable production stope reserve solids were evaluated with due consideration for established unplanned mining dilution, constraints associated with the long hole mining method deemed as the mining recovery and the expected metallurgical processing recoveries.

The final step in the determination of the Mineral Reserve estimate was the evaluation of capital and incremental development required for the extraction of the mineable production stope reserve solids. The mine development design to access the reserve solids was prepared in Promine software and was considered in the capital and operating cost model to validate the economic viability of each prospective mining zone. Mineralized zones that were deemed unprofitable were excluded from the Mineral Reserve estimate.

15.3 Mining Dilution

Both planned and unplanned mining dilution were considered in the Mineral Reserve estimate. Mining dilution is dependent on the design of the reserve solids and the geomechanical nature of the mineralized zone. Planned dilution is unmineralized material that must be extracted alongside the mineralized material to meet geomechanical constraints, minimum mining widths or geometrical constraints. The established unplanned dilution for each mineralized zone was determined based on geomechanical analysis. A dilution grade was applied only to the S50 Zone as provided by the Wesdome Kiena Mine Complex technical services team. All lateral drift development was allocated an unplanned dilution of 5.0%. The expected planned and unplanned dilution factors considered for each mineralized zone are listed in Table 15-3.

Table 15-3: Planned and unplanned mining dilution by mineralized zone

Zone	Lens	Planned Dilution Evaluated (%)	Unplanned Dilution Factored (%)	Combined Dilution (%)	Dilution Grade (g/t)
Kiena Deep A	ZA	7.9	15.0	22.9	0.0
	ZA1	8.1	25.0	33.1	0.0
	ZA2	6.4	25.0	31.4	0.0
	H1ZA	12.0	15.0	27.0	0.0
S50	S50	6.9	15.0	21.9	0.7
Martin	Martin	7.0	15.0	22.0	0.0
Development	All	22.1	5.0	27.1	0.0
Weighted	**All**	**10.0**	**17.0**	**27.0**	**0.0**

15.4 Mining Recovery

The mining recoveries considered in establishing the Mineral Reserve estimate reflect the method of mining extraction. Table 15-4 identifies the two predominant mining methods and the recoveries accounting for production and development mining. In terms of the production mining method, the following characteristics influenced the mining recovery:

- Drill & blast constraints leaving mineral resources behind left unrecovered;
- Remote stope mucking losses due to difficult recovery at extremities, corners, and along the walls of the production stopes;
- Unplanned loss of mineral resources due to excessive waste sloughing and production stope abandonment.

Table 15-4: Mining method recoveries

Mining Method	Mining Recovery (%)
Longitudinal Long Hole Production Stoping	90.0
Lateral Drift Development	95.0
Weighted	**90.2**

Notes:
1. Select mineable production stopes had their mining recovery reduced to 80% to reflect challenging geometry for the longitudinal long hole stoping mining method.

The Mineral Reserve estimate considers the longitudinal long hole stoping method for production mining. The mining recovery factor of 90% is representative of the production stope mineral reserve solids in the respective mineralized zones.

15.5 Cut-Off Grade

The cut-off grades for the respective mineralized zones were established by the QP based on the overall mine operating costs established for the respective mineralized zones. When combined on a weighted tonnage basis, the operating costs by mineralized zones reflect the life of mine, overall mine operating cost for the Kiena Mine Complex reported in detail in Chapter 21 – Capital and Operating Costs. The overall mine operating costs by mineralized zone are presented in Table 15-5, as well as the parameters considered in calculating the cut-off grades that were considered for mine design purposes in generating the Mineral Reserve estimate. Incremental cut-off grades excluding mining costs were not considered for lateral drift development required through mineralization. Lateral development tonnes were treated as waste rock when grading was below the established cut-off grades. Waste rock development costs deemed to be capital costs were not included in the establishment of the cut-off grades. The waste rock development costs were considered separately against the prospective mineable reserves in their respective locations.

Table 15-5: Cut-off grade considerations by mineralized zone

Cost	Unit	Kiena Deep A Zone	S50 Zone	Martin Zone
Mining	C$/t ore	122.37	122.37	71.67
Processing	C$/t ore	28.25	28.25	28.25
General and Administration	C$/t ore	36.53	36.53	36.53
Water Treatment	C$/t ore	3.55	3.55	3.55
Total	**C$/t ore**	**190.70**	**190.70**	**140.00**
Revenue & Recovery				
Gold Price	C$/oz	1,914	1,914	1,914
Processing Recovery	%	98.5	97.0	97.0
Cut-off Grade				
Break-Even Mill Feed Grade	Au g/t	3.15	3.20	2.35
Cut-off Grade Considered	**Au g/t**	**3.15**	**3.20**	**2.40**

The direct underground mining costs associated with Martin Zone were estimated to be different from both the Kiena Deep A and S50 Zones based on the separation of equipment and labour requirements from the latter zones. Also, unique mining considerations apply regarding smaller lateral drift development size, two Alimak production stopes and uncemented rockfill. The Martin Zone will utilize track haulage and air equipment currently owned by the Kiena Mine Complex. Contracted development and production mining crews with their own additional proprietary equipment were considered for its extraction.

The capital waste rock development costs outlined in Table 15-6 were considered in the economic analysis by location to ensure profitability in accessing the prospective mineable reserve solids considered in the Mineral Reserve estimate. The basis for the unit costs is presented in Chapter 21 – Capital and Operating Costs. Furthermore, the capital waste rock development costs were used on an incremental basis to remove any stopes above cut-off grades that were unprofitable based on development access constraints.

Table 15-6: Capital waste rock development unit costs

Capital Waste Rock Development	Unit Cost (C$/m)
Lateral Drift	3,747
Vertical Raise (Unsupported)	3,631
Vertical Raise (Supported) Ladderway Installed	6,181

15.6 Mineral Reserve Classification

Based on technical feasibility and economic viability, the Indicated Mineral Resource forming the basis of the design of the prospective mineable reserve solids and retained in the established Mineral Reserve estimate meet expectations to be converted and treated as Probable Mineral Reserve.

The QP is of the opinion that the Mineral Reserve estimate is estimated in an appropriate manner using current mining software and procedures consistent with industry best practice.

16. MINING METHODS

16.1 Historical Mining

Historical gold production at the Kiena Mine Complex from 1981 to 2019 is approximately 1.8 million ounces.

Figure 16-1: Wesdome Kiena Mine Complex, site entrance September 2020

The first gold discoveries were made on Parker Island (site of the present Kiena Mine Complex surface infrastructure) between 1911 to 1914. Kiena Gold Mines Ltd. (Kiena Mines) was established in 1936 as part of the ventures group of Thayer Lindsley. In the years to follow, a shaft and four mine levels were developed but the Parker Vein had limited extensions. The North Zone was identified by surface drilling and a drift was extended north from the shaft on 12 Level (120 m depth). This drift encountered the VC Zone and continued north to provide a drilling platform to test the continuity of the North Zone. In 1940, the operation was abandoned due to limited reserves, economic conditions and the war effort.

Twenty years later, the property's potential was re-examined by Falconbridge Ltd. with diamond drilling in 1961 establishing the S50 Zone. By 1963, exploration drilling outlined a resource of 4.5 million tonnes averaging 6.34 g/t Au for the S50 Zone. Shaft No.1 (current hoist shaft) was sunk to a depth of 400 m and extensive underground exploration and development was completed. In 1965, the project was abandoned due to adverse mining conditions and the low gold price of US$ 35 per ounce.

In 1976, Wesdome Resources Ltd. was created, and in 1979, with an improvement in gold prices, the project was successfully developed with commercial production commencing in October 1981. In 1986, Kiena Mines was sold to Campbell Red Lake Mines, which subsequently merged with Dome Mines Ltd. and Placer Development Ltd. to form Placer Dome Canada Ltd.

In 1997, Placer Dome Canada Ltd. sold Kiena Mine to the now-defunct Montreal based junior McWatters Mining. McWatters Mining ran the mine until 2002, when it was closed due to low gold prices. Initial production was custom milled off site until Kiena Mine built their own mill, which commenced in September 1984. The mill treated 1,300 tpd from the S50 Zone. In 1999, McWatters Mining decided to develop the North Zone and expand the mill to 2,000 tpd capacity. The strategy was to increase production and lower unit costs by providing 1,000 tpd from the S50 Zone and 1,000 tpd from the North Zone. In all, between 1981 and 2002, Kiena produced 1.56 million oz of gold from 10.7 Mt grading 4.75 g/t Au, with virtually all the mineralized material coming from the S50 Zone. Between 1985 and 2002, output averaged 70,000 oz of gold annually at a cash cost of US$ 234 per ounce.

In October 1999, Dynacor Mines Inc. and Western Quebec Mines signed an agreement to consolidate the contiguous Siscoe and Siscoe-Extension (Dynacor Mines) and Wesdome, Lamothe, Lamothe-Extension, Yankee Clipper and Callahan (Wesdome Resources) properties into a new company, Wesdome Gold Mines Incorporated (Wesdome). December 2003 saw the purchase of the Kiena Mine Complex by Western Quebec Mines, which subsequently placed the property into Wesdome Gold Mines Incorporated, thereby completing and consolidating Wesdome's land package around Lac De Montigny. Then, in July 10, 2007, a merger was completed with parent company Western Quebec Mines based on 1.45 shares of Wesdome for each share of Western Quebec Mines. Wesdome was the surviving operating entity.

In 2006, Kiena Mine Complex completed a pre-production development plan and brought the mine into production in August 2006 until March 7, 2013. Wesdome suspended mining activities at the Kiena Mine Complex on June 30, 2013 due to decreasing recovered grades, persistent industry cost pressures and uncertainty in the gold price and exchange rates. At this time the Proven and Probable Mineral Reserves were reclassified as Measured and Indicated Mineral Resources. Figure 16-2 illustrates historical production at the Kiena Mine Complex from 1981 until 2019.

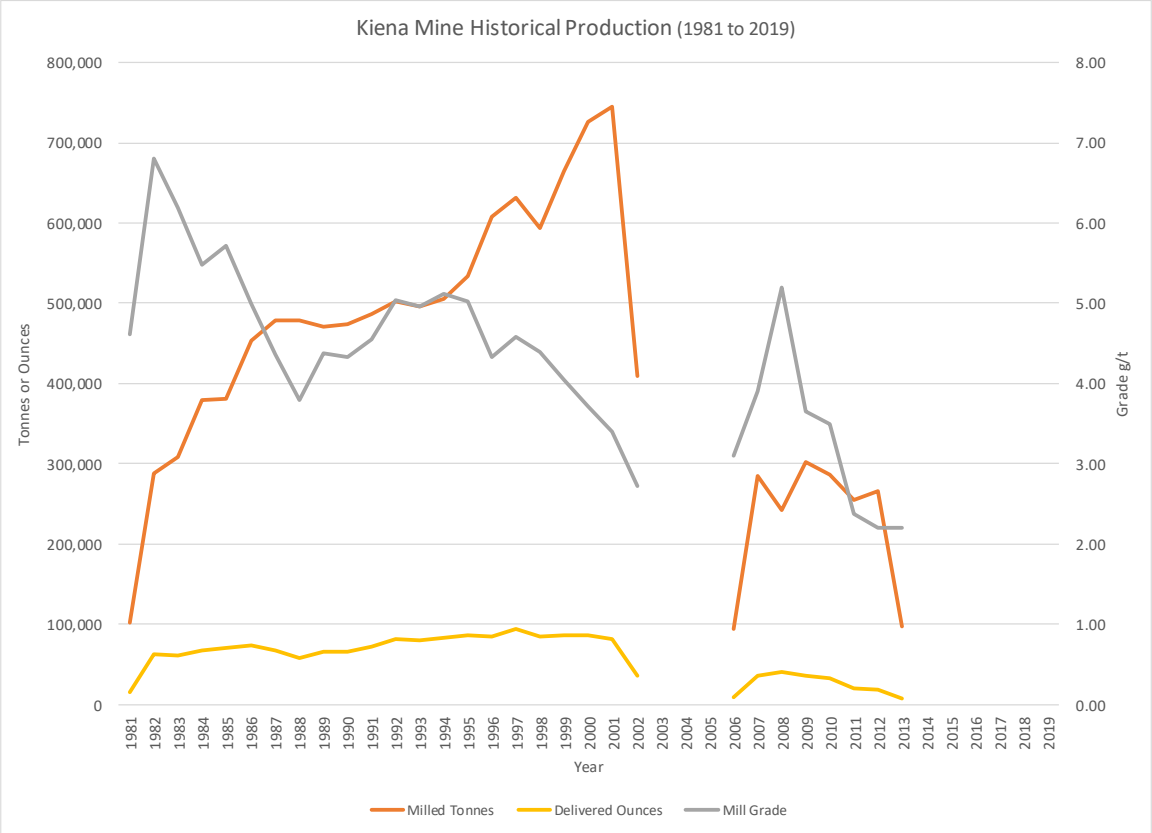


Figure 16-2: Historical production from 1981 to 2019

Since the suspension of operations, Wesdome has completed extensive underground diamond drilling to enhance the mineral resource. A preliminary economic assessment (PEA) study was finalized on June 17, 2020 (Corkal et al., 2020). Further diamond drilling influenced the Indicated Mineral Resource to be established forming the basis of the Mineral Reserve estimate in this prefeasibility study (PFS).

Figure 16-3 illustrates the three mineralized zones (Kiena Deep A, S50, Martin) forming the basis of the Mineral Reserve estimate represented in the life of mine (LOM) plan. The North Zone located under a surface crown pillar approximately 100 m thick below Lac De Montigny was mined historically. As an act of due diligence, the bulk of the voids are to be backfilled to enhance ground stability.

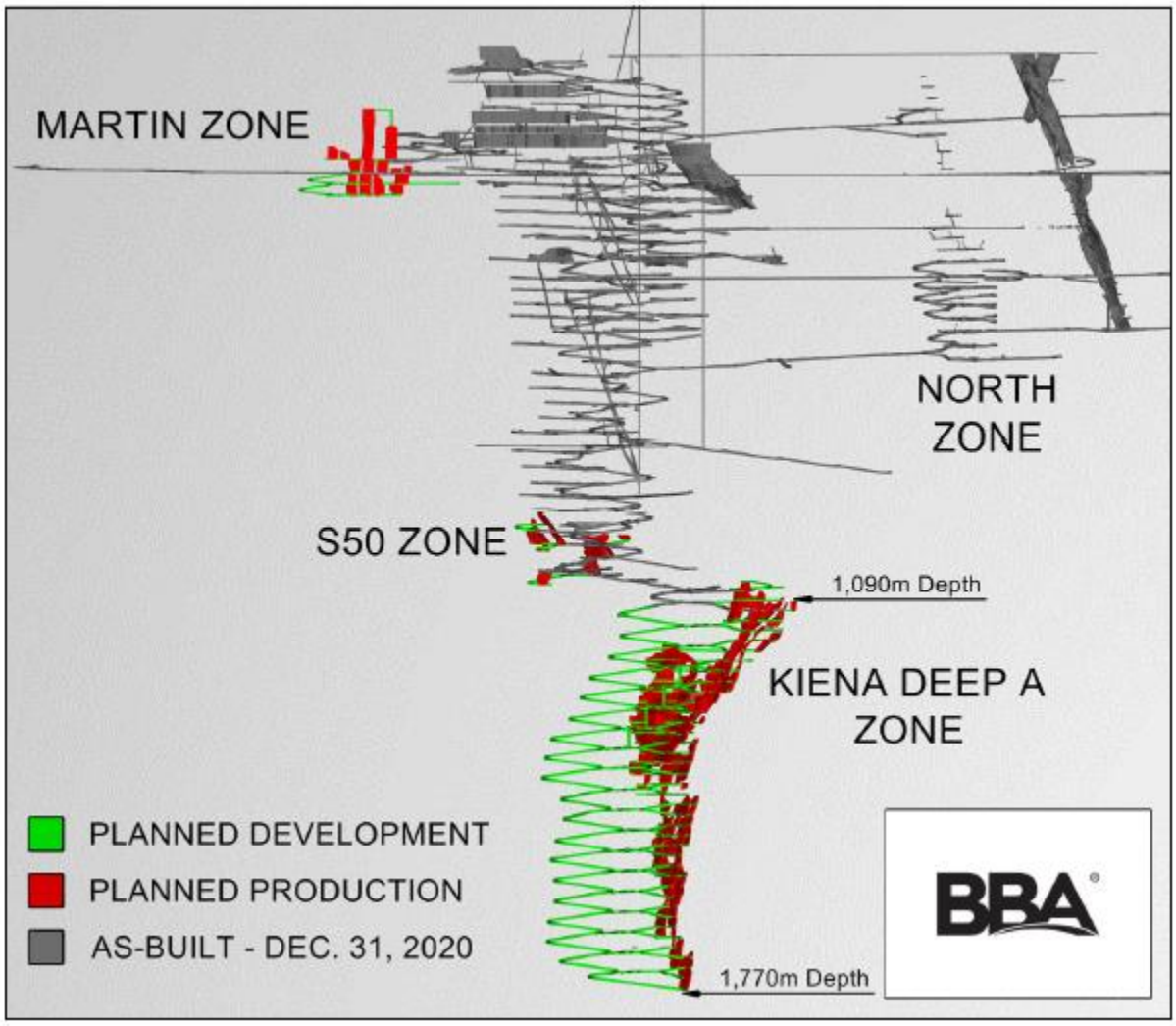


Figure 16-3: Isometric long section of the mining zones considered in the LOM plan (Looking northwest)

16.2 Existing Infrastructure

16.2.1 Hoist Shaft

The existing hoisting shaft (Shaft No. 1) is a three compartment timbered shaft developed in 1963 to a depth of 400 m. Compartments No. 1 and No. 2 house cage over skip conveyances with a 6.2-tonne capacity skip for conveying ore and waste rock and a 2.3-tonne (5,000 pound) capacity cage for conveying personnel or material respectively (Figure 16-4). Compartment No. 3 is for mine services (electrical cables, compressed air pipe, process water pipe and dewatering pipe) and ladderway. Due to the size limitation of the cage, consumable material such as pipe, screen, ground support bolts are slung below the cage. Most levels of the mine are accessible from an internal ramp that starts on 17 Level. The fresh air ventilation raise from surface to 17 Level is used to sling mobile equipment and large items underground utilizing a surface crane.

In the early 1990's, the shaft was deepened to a depth of 930 m with a fourth compartment added to enable shaft sinking without impacting production. The hoist is a Nordberg double-drum double-drive with GE motors (Figure 16-5).

Figure 16-4: Hoist shaft compartments

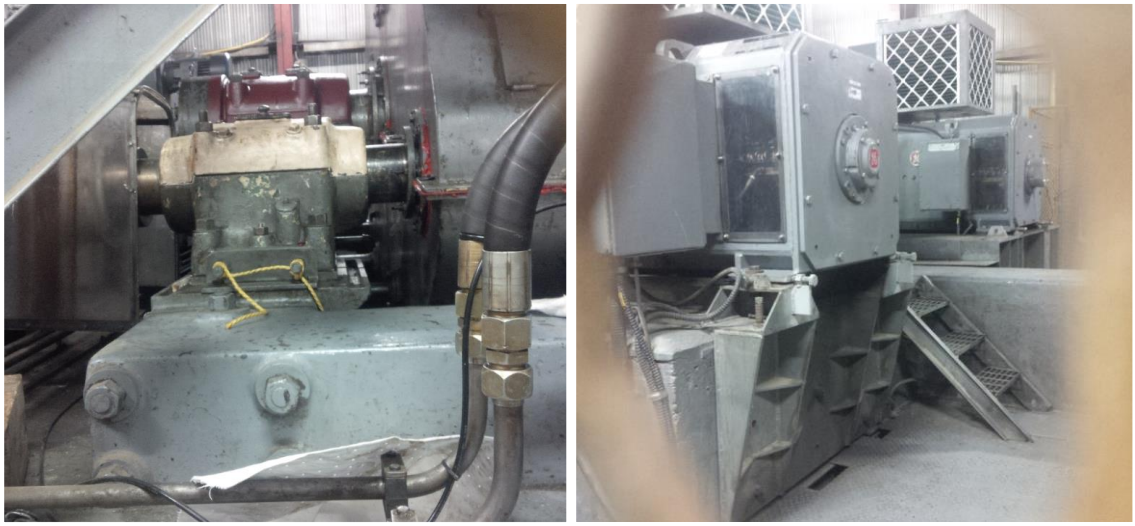


Figure 16-5: Hoist drives and motors

Hoisting of waste rock and ore is currently a manual process that requires a hoist operator at the hoist controls, as well as a skip tender at the loading pocket. The following information was considered in assessing the hoisting system availability and capacity:

- Ore and waste rock movement;
- Personnel movement;
- Daily hoist inspections and maintenance;
- Monthly, quarterly and yearly inspections;
- Conveyance changeouts;
- Rope changes;
- Unplanned outages.

Upon review, the information indicated that an average hoisting capacity of 2,000 tonnes per day can be expected.

Hoisted ore dumped into the headframe bin allocated for ore is conveyed to the surface ore storage bins located between the headframe and the crusher (Figure 16-6). Hoisted waste rock dumped into the headframe bin allocated to waste rock feeds a chute for handling by mobile equipment and storage on surface.

Figure 16-6: Surface ore storage bins

16.2.2 Mobile Equipment

Haulage trucks and load haul dumps (LHD) are currently owned and operated by a contracted mining company for ore and waste rock haulage. Most other mobile equipment is also currently owned and operated by the mining contractor. In 2021, it is intended that the Kiena Mine Complex will begin purchasing and operating their own heavy haulage mobile equipment as well as all other mobile equipment to support the LOM plan.

16.2.3 Ore and Waste Rock Movement to Loading Pocket

The path taken by ore and waste rock is dependent on the location from which it is sourced. Ore mined above 79 Level (Martin Zone) is dumped into the ore pass system that feeds the crusher on 81 Level. Ore mined below 79 Level (Kiena Deep and S50 Zones) reports to the ore truck dump on 79 Level that feeds the crusher on 81 Level. In order to prevent damage and hang-ups in the ore passes, ore size is regulated by grizzlies before entering the ore passes. Crushed ore from 81 Level then reports to the loading pocket on 86 Level for hoisting to surface. Waste rock generated above 79 Level is dumped into the rock pass system that passes through the grizzly on 81 Level. Waste rock generated below 79 Level reports to the rock truck dump on 81 Level that passes through the grizzly on 81 Level. If required, the rock breaker on 81 Level is used to address oversize. Passing through the grizzly on 81 Level, rock reports to the loading pocket on 86 Level for hoisting to surface.

16.2.4 Ore Crushing and Hoisting

A 48-inch jaw crusher is located on 81 Level. Two ore passes feed a scalper/breaker before the crusher to allow fines to bypass the crusher and to break oversize before entering the jaws (Figure 16-7). The bin below the crusher feeds directly into the loading pocket located on 86 Level for ore hoisting to surface.

Figure 16-7: Ore crusher

16.2.5 Waste Rock Disposal and Hoisting

Waste rock resulting from underground mine development that is not disposed of underground as mine backfill will be hoisted from the loading pocket on 86 Level to surface and stored at the current waste facilities.

16.2.6 Ventilation System

The ventilation system at Kiena Mine is a push-pull system.

Fresh Air

Two surface fresh air fans operated in parallel (Table 16-1) push fresh air into the mine down a fresh air raise from surface to 17 Level. From 17 Level the fresh air travels down the main ramp system down to the deepest extent of the Kiena Deep A Zone.

Table 16-1: Surface fresh air fans

Fan	Diameter	Blade setting	Horsepower (hp)	Volume KCFM	Pressure in WG
Sheldon Axico 200-8-10	79”	47º	200	170	4
Joy 2000 72-20-1200	72”	15º	250	172	4.5

Exhaust Air

A single exhaust air fan (Table 16-2) pulls air from the mine through a network of exhaust air raises.

In order to address a blockage in the exhaust air raise system between 17 Level and 33 Level, two bypass drifts were developed on 17 Level to reinstate unrestricted flow through a newly developed Alimak exhaust raise between 17 Level and 33 Level. Development to reinstate the exhaust air raise system was completed in 2020. In April 2021, a new silencer was installed on surface to reduce noise levels emitted from the exhaust air fan.

Table 16-2: Surface exhaust air fan

Fan	Diameter	Blade setting	Horsepower (hp)	Volume KCFM	Pressure in WG
Joy M- 84-58-1200	84”	36º	600	300	7.6

The current surface fresh air and exhaust air ventilation fan installations will require upgrading to support the LOM plan. In terms of capacity, it is estimated that both surface fresh and exhaust air ventilation fan installations will meet underground ventilation requirements until the end of 2022. In order to meet future ventilation demands and enhance the operational reliability of the aged surface ventilation infrastructure, new surface fresh and exhaust air fans as well as new mine air heaters must be replaced. This work is scheduled to occur in July 2022 to facilitate construction and to coincide with non-heating months.

16.2.7 Repair Facilities

Repair facilities located on 64 Level are available to complete mobile equipment repairs and services (wash bay and service bay), as well as a small shop to repair handheld drills and air pumps. A separate area exists for the welding of equipment (Figure 16-8). Electrical parts and minor electrical repairs are performed in the small facility also located on 64 Level.

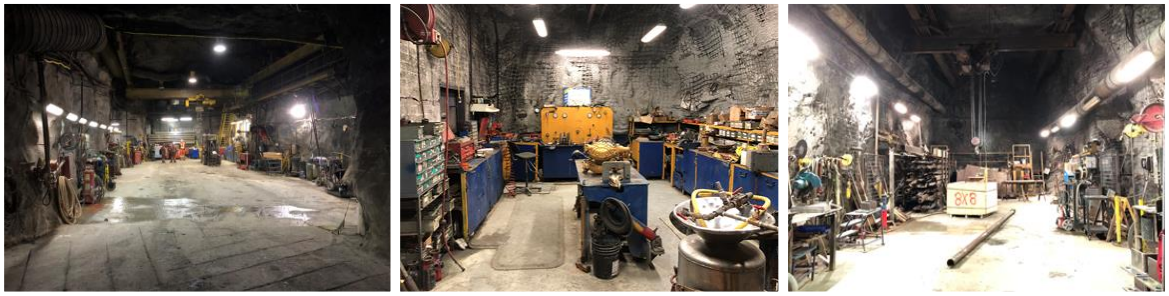


Figure 16-8: Repair facilities

16.2.8 Fuel and Oil Facilities

The main fueling station and oil storage are located on 64 Level. A fuel pipeline located in the shaft is used to fill the fuel tanks in this facility. In order to supply fuel to the satellite fueling station planned for the Kiena Deep A Zone, the fuel pipeline was extended to 82 Level and will continue to be extended to the Kiena Deep A Zone. At present, the fuel pipeline from 64 Level to 82 Level has not been commissioned.

16.2.9 Backfill Facilities

The existing hydraulic backfill system consisting of a plant located on surface and the underground pipeline distribution network will not be utilized for underground backfill purposes for future production mining. A backfill trade-off study was conducted primarily to compare two backfill types considered for the Kiena Deep A and S50 Zones and ultimately to support the LOM plan. Hydraulic and paste backfill were considered for these two mining zones. As the PFS progressed and backfill discussions occurred, it was identified that hydraulic backfill was not conducive to the expectations demanded by the quick production mining cycle time represented in the mine schedule. Compared to hydraulic backfill, paste backfill requires more control in its creation and pipeline transport underground but surpasses hydraulic backfill in its performance underground including cure time, unconfined compressive strength and placement characteristics. Expected performance underground was influential in Wesdome Kiena Mine Complex deciding to proceed with paste backfill. The backfill trade-off study conducted by QP Michael Stochmal, P.Eng., was conducted independently of the paste backfill design and cost estimation conducted by QP Frank Palkovits, P.Eng.

In addition, the backfill trade-off study evaluated secondary backfill types considered for the Kiena Deep A and S50 Zones as well as the primary backfill types considered for the Martin Zone and to backfill the voids created by historical mining in the North Zone. Hydraulic tailings disposal was selected for the North Zone while the Martin Zone will utilize uncemented rockfill. Cemented rockfill will be placed in the Kiena Deep A Zone until paste backfill becomes the primary backfill type. Cemented rockfill will be the primary backfill type in the S50 Zone making use of paste backfill as required. Refer to Table 16-3 for all backfill types considered and their estimated capital and operating costs.

Table 16-3: Backfill trade-off study

Zone	Backfill Type	Cement (dry wt%)	Status	CAPEX, OPEX C$/t Ore	CAPEX, OPEX C$/t Backfill
North	Hydraulic	1.0	Selected	0.00	4.80
Martin	Rockfill	0.0	Selected	3.86	5.74
Kiena Deep A and S50	Rockfill	5.0	Selected	13.17	19.62
Kiena Deep A and S50	Paste	2.5	Selected	22.26	34.25
Kiena Deep A and S50	Hydraulic	7.0	Not Selected	19.09	29.47

16.2.10 Dewatering Facilities

The current underground mine dewatering system at the Kiena Mine Complex is a clear water system. The current pumping stations are located on 38 Level, 52 Level, 82 Level and 94 Level. Run of mine water flows to decant sumps where slimes are separated from the dirty water. The resulting slimes are disposed of underground and the clear water is decanted into storage sumps that are serviced by multi-stage dewatering pumps (Figure 16-9). All pumping stations have backup pumps installed to maintain the continual operation of the pumping system. The main dewatering pipeline is 6-inch diameter mild steel Schedule 40 installed in the No. 3 service compartment of the hoist shaft.

Figure 16-9: Clear water mine dewatering pumps

16.2.11 Process Water Facilities

Process water for the Kiena Mine Complex is obtained from a well situated near the shore of Lac de Montigny. Water is pumped to surface facilities and to the hoist shaft for distribution underground. Located in No.3 service compartment of the hoist shaft, the main process water trunk pipeline is a 3-inch diameter mild steel Schedule 40 pipe that distributes process water to the mine levels underground (Figure 16-10). The process water trunk pipeline is equipped with pressure reducing valves (PRVs) to ensure water pressure is regulated. Beyond the hoist shaft, the main trunk pipeline extends to the deepest extent of the underground mine by following the development infrastructure.

Potable water underground is supplied in the form of bottled water for individuals and refuge stations/lunchrooms.

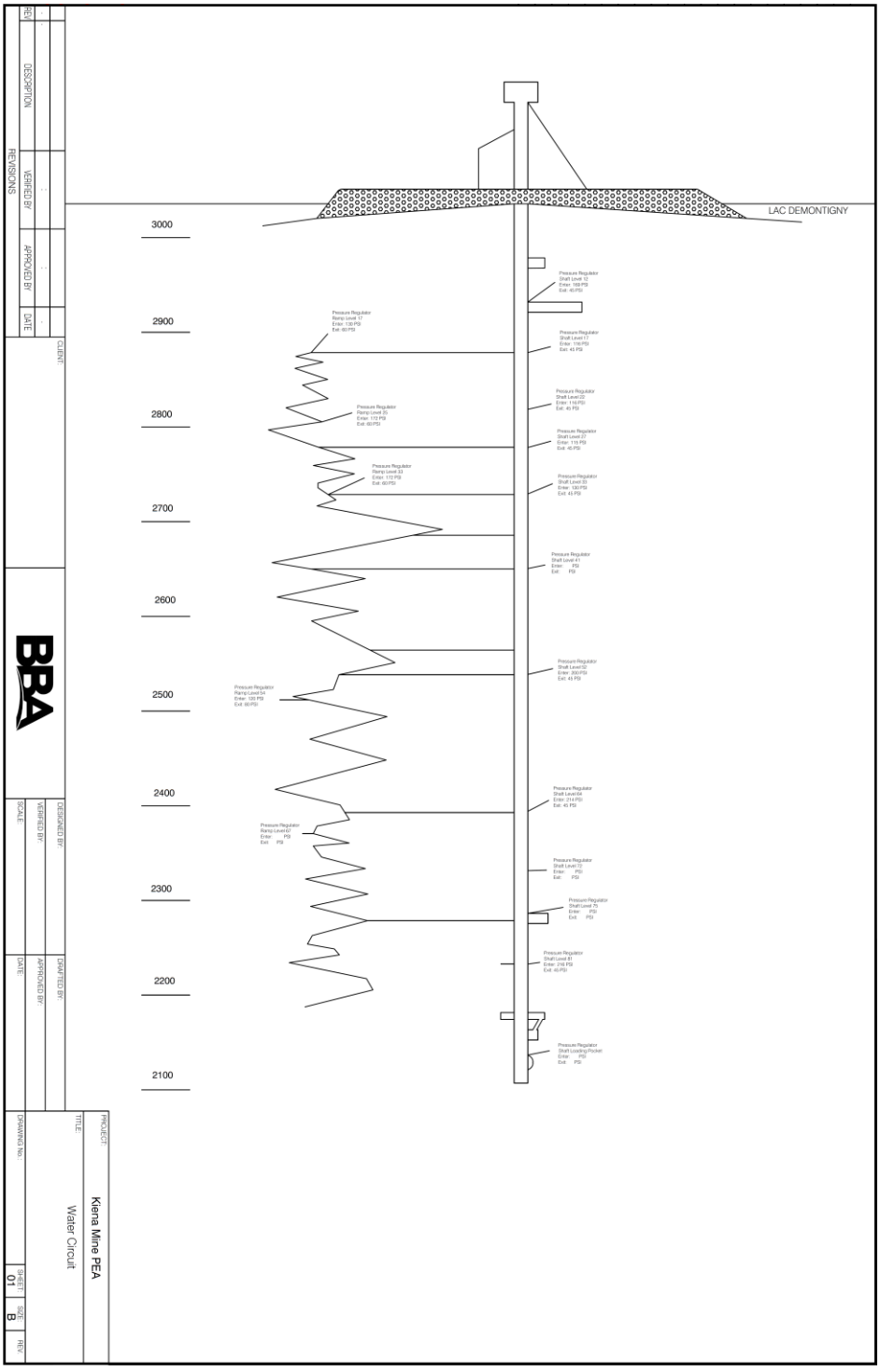


Figure 16-10: Process water distribution

16.2.12 Compressed Air Facilities

Compressed air for the underground workings at 690 kPa (100 psi) comes from the surface compressors (3 x Sullair LS-25S compressors) located in a building attached to the hoist room (Figure 16-11 and Figure 16-12). A diesel standby compressor located in a separate building is available if there are any issues with the main compressors. Underground compressed air distribution takes place through an 8-inch diameter mild steel Schedule 40 pipe located within the No. 3 service compartment in the hoist shaft. The pipeline diameter in the underground workings varies in size from 2-inch to 4-inch diameter mild steel Schedule 40 pipe.

Figure 16-11: Surface compressor building

Figure 16-12: Surface compressors

16.2.13 Electrical Facilities

Electrical power for underground operations comes from the surface electrical room located behind the hoist room. It feeds from Transformers T7 and T11 at 2.4 kV and 4.16 kV voltage respectively (Figure 16-13). There are currently four circuits dedicated to providing electricity underground.

Figure 16-13: Transformers T7 and T11

The cables for the four electrical circuits (72, 73, 1102 and 1103) feeding underground equipment are located within the No. 3 service compartment of the hoist shaft. Each feed is 1,200 A and services specific levels within the mine.

An electrical rehabilitation assessment was conducted of the main underground substation and its downstream distribution. The audit focused on identifying the requirement for maintenance testing, refurbishment, modification and replacement of the main electrical equipment supporting underground operations at 2.4 kV and 4.16 kV. In addition, the audit assessed issues with the main surface breaker tripping when overloaded. It was concluded that upgrading the mine voltage to 13.8 kV is the only sustainable option to meet the electrical needs associated with the Kiena Deep and S50 Zones. The existing electrical distribution network at 2.4 kV and 4.16 kV above 111 Level will remain in place.

An existing 1,000 kW standby diesel generator will provide emergency power to the cage hoist as required by closing the electrical circuit. Wesdome agreed not to provide emergency power to the underground mine dewatering pumps accepting that water will accumulate in the mine during a power outage. Refer to Chapter 18 – Project Infrastructure for the expectations of supplying emergency power to the Kiena Mine Complex.

16.2.14 Communications Facilities

Radio communication is available throughout the active mining areas utilizing a leaky feeder system. Hard wired analog telephones are also in use at specific locations, such as refuge stations, repair service stations and shaft stations.

Central blasting is utilized throughout the mine with the main cable located within the No. 3 service compartment 3 of the hoist shaft.

16.2.15 Refuge Facilities

Existing refuge facilities consist of stations excavated in rock with concrete walls and steel doors. Compressed air is supplied via the underground compressed air piping system.

16.2.16 Second Egress Facilities

The hoist cage is the primary mode of personnel transport underground. A ladderway also exists in the hoist shaft as an alternate means of egress. Second egress to surface is provided by ladders installed in a fresh air escapeway raise from surface to 17 Level. From 17 Level, current ramp development extending below the shaft to the deepest extent of the mine in the Kiena Deep Zone along with ladderways currently installed in a network of ventilation exhaust air and escapeway raises fulfill the requirements for second egress down to 109 Level.

16.3 Geomechanics

16.3.1 Lithology

The primary lithologies associated with the Kiena Deep A Zone and other zones are: Basalt, Komatiite, and Schist. Komatiite and Schist are associated with a “Schist Zone” which envelopes the Basalt. Secondary lithologies occur, such as komatiitic basalt, gabbro and granodiorite that are associated with the Basalt host rock. Regional faulting (Marbenite Fault) lies within the Schist Zone.

The Kiena Deep A Zone consists of three or more coplanar lenses. The Kiena Deep A Zone dip steepens with depth, from approximately 50°(±) to 70°(±). In plan, the Kiena Deep A veins follow an arched shape, due to an isoclinal fold associated with the Marbenite Fault.

The S50 and Martin Zones have a tabular shape, located in Basalt rock at the Schist contact. Fault zones can be present in the hanging walls.

16.3.2 Rock Strength

Laboratory testing of representative rock core samples from the Kiena Deep A Zone was conducted in 2020, refer to Table 16-4. The values for Schist (M8) reflect historical testing conducted in 2012.

Table 16-4: Rock core test data

Property	Basalt †		Basaltic Komatiite †		Komatiite †		Schist (M8) ‡	
	Mean	SD	Mean	SD	Mean	SD	Mean	SD
Uniaxial compressive strength (UCS) MPa	73.7	6.6	59.4	23.7	22.97	4.8	25.3	9.9
Youngs Modulus (E) GPa	106.3	14.4	67.5	49.9	39.2	13.5	31.8	5.0
Poisson Ratio (γ)	0.22	0.11	0.23	0.09	0.31	0.25	0.36	0.05
Tensile Strength MPa	12.5	4.2	10.0	5.8	3.6	1.2	3.8	0.3

† =data from MIRARCO 2020
‡ = historic data (Kiena 2012)
SD = Standard Deviation

16.3.3 Rock Mass Characterization

Rock mass characterization was interpreted from geological log reports and drill core photographs of spatially representative diamond drillholes (DDH). Representative rock mass characterization for primary lithologies associated with each prospective mining zone are summarized in Table 16-5.

Table 16-5: Typical rock mass characterization from DDH logs

Zone	Lithology Q' (typical Q')		
	Basalt	Komatiite	Schist
Kiena Deep A	5 – 10 (~10)	1.5 – 4 (~3)	1 – 2.5 (~2)
S50	4.4 – 9 (~9)		1 – 2.5 (~2)
Martin	3.1 – 9.4 (~8.3)		

Representative development headings on 98 and 111 Levels in the vicinity of the S50 and the Kiena Deep A Zones respectively were inspected in 2020. Rock mass characteristics obtained from exposures of representative Basalt and Komatiite lithologies were characterized visually according to the Geological Strength Index, GSI (Hoek et al., 1995). Representative GSI ranges for the S50 and Kiena Deep A Zones are shown in Figure 16-14. Photographs of diamond drill core from the Martin Zone Basalt indicate comparable GSI ranges.

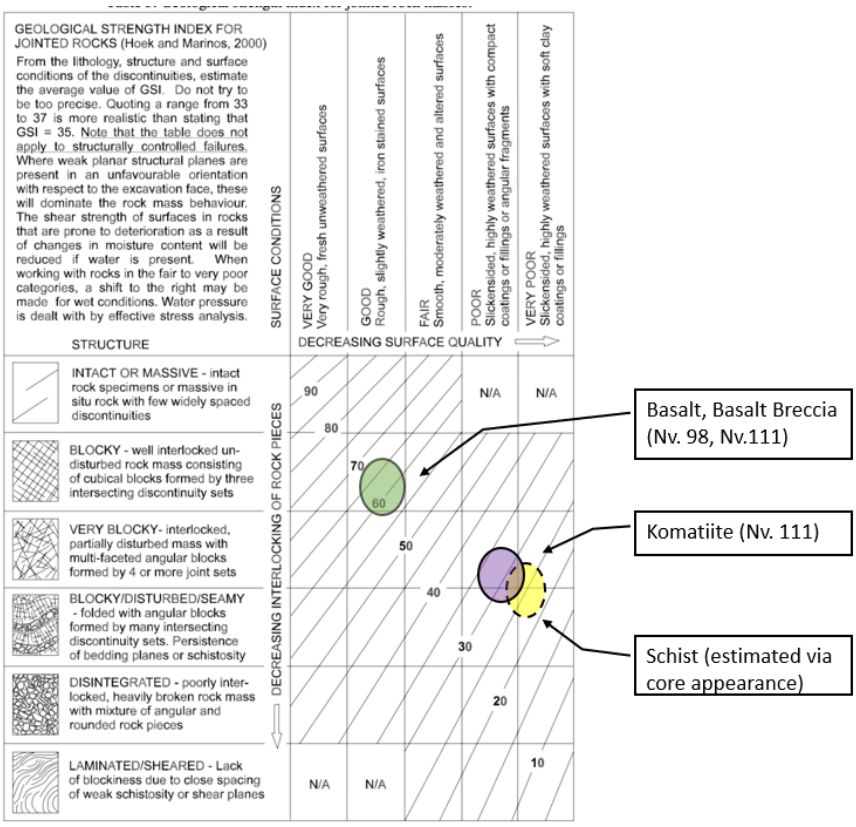


Figure 16-14: Geological strength index (GSI) characterization, S50 and Kiena Deep A Zones

16.3.4 Underground In situ Stress

Historically, underground in situ stress measurements were taken at the Kiena Mine Complex. The most recent in situ stress measurements were taken in 1995 at a depth of 810 m. Measurements summarized in Table 16-6 indicate a maximum horizontal to vertical stress ratio (K-ratio) of K = 2.06.

Table 16-6: Stress tensor, Corthésy et al (1996)

Tensor	Measured at depth of 810 m (MPa)	Stress tensor at depth H = vertical depth below surface	K-ratio
σ1	51.8 (sub-horizontal)	σ1 = 0.064H	2.06
σ2	40.5 (sub-horizontal)	σ2 = 0.050H	1.61
σ3	25.1 (sub-vertical)	σ3 = 0.031H	

The anticipated principal stress tensor trend (Azimuth 035°; sub-horizontal), is based on mean principal stress assumptions from mines within the Cadillac Fault region (McKinnon and Labrie, 2006; Corthésy et al. 1997). The orientation of the anticipated stress tensor relative to the Kiena Deep A and S50 Zones is illustrated in Figure 16-15.

The Kiena Deep A Zone has a folded geometry; the southern and northern lobes trend sub-parallel and sub-perpendicular respectively, to the anticipated principal stress tensor.

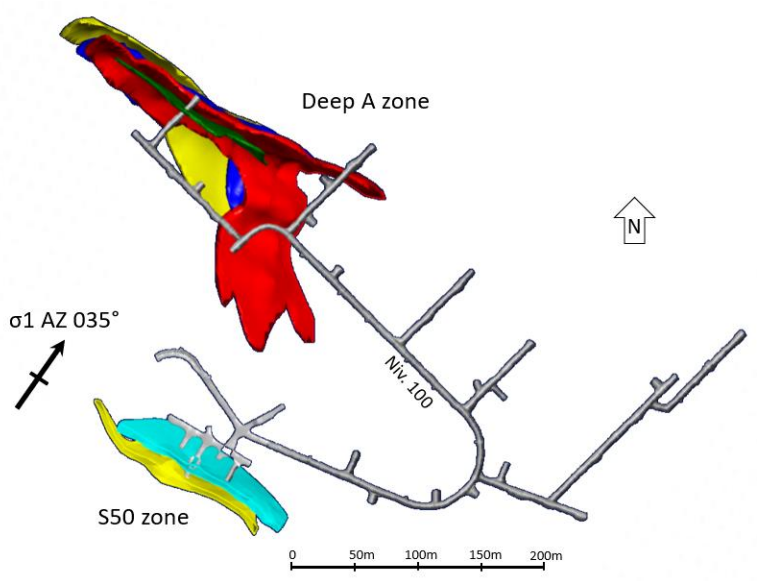


Figure 16-15: Orientation of principal stress tensor relative to Kiena Deep and S50 Zones
Section shown: Depth = 1,924m, ±250m

16.3.5 Anticipated Rock Mass Behaviour

The Kiena Mine Complex zones are characterized by two rock types with differing response to stress. The Basalt host rock is relatively stiff and is expected to exhibit elastic deformation characteristics in zones of good quality rock masses. The weaker "Schist Zone" of altered ultramafic rock (ranging from Komatiite to Talc-Schist) will exhibit yielding / ductile type deformation behaviour in response to elevated stress.

Basalt

Based on the available information, it is anticipated that the rock mass behaviour of Basalt will be influenced primarily by stress. However, spacing of discontinuities (rock mass quality) and locally by fault and shear features are influential factors. The frequency, persistence and the characteristics of the major geological structures and rock mass fabric will control the rock mass behaviour around mine openings.

At depth and increased extraction in the lower Kiena Deep A Zone, mining induced stresses are expected to concentrate around the excavation abutments. Evidence of the mining induced stresses is anticipated to be evident as localized fracturing and light to moderate spalling in the high range rock masses.

Schist and Komatiite

Low quality rock such as Schist and to a lesser extent Komatiite are encountered in many manifestations in underground hard rock mines. Ranging from weak and poorly consolidated rock to highly altered and or foliated rock. The combination of elevated stress (far field and induced) with a weak rock mass results in non-linear, squeezing or creeping behaviour.

Lateral drift development driven in low quality rock pose ongoing challenges to a mine operator. Ground stability issues such as caving/unravelling of under supported ground, drift closure due to surface deformation or buckling processes and persistent degradation of exposed rock surfaces are commonly encountered. As some deformation is inevitable, managing rather than preventing deformation is the goal of the rock mass support.

Production stopes mined in Schist are sensitive to factors including stope dimensions, length of time between start of stope mucking and completion of backfilling (rate of mining cycle), excess blasting concussion and stope overbreak of adjacent and underlying stopes.

16.4 Stope Empirical Design

The Stability Graph method (Potvin and Hadjigeorgiou, 2001) was used to assess preliminary stope dimensions and the stability of stopes within the Kiena Deep A Zone for the three A-Lenses as well as the S50 and Martin Zones. The Stability Graph takes into consideration the key factors that influence stope stability, including rock mass quality, the orientation of the principal structures, in situ stress and the geometry of the stope. The empirical stope designs for hanging wall (HW) and footwall (FW) stope dimensions are summarized below.

16.4.1 Kiena Deep A Zone Stope Design

- A Lens:
  - Design considerations: HW = Basalt, FW = Komatiite;
  - Stope dimensions: 18 m vertical height, 20 m stope width (on strike);
  - Design constraint: Stability of the FW Komatiite.
- A1 Lens:
  - Design considerations: HW = Komatiite, FW = Komatiite;
  - Stope dimensions: 18 m vertical height, 13 m stope width (on strike), 10m stope width considered in stope design;
  - Design constraints: Stability of the HW Komatiite. HW cable-bolt reinforcement may not be effective. Stability will be time dependent.
- A2 Lens:
  - Design considerations: HW = Schist, FW = Schist;
  - Stope dimensions: 18 m vertical height, 10 m stope width (on strike);
  - Design constraints: Stability of the HW Schist. HW cable-bolt reinforcement will not be effective. Stability will be very time dependent.

16.4.2 S50 Zone Stope Design

- Design considerations: HW = Basalt, FW = Schist, ore zone dip = - 60°;
- Stope dimensions: 20 m vertical height, 15 m stope width (on strike);
- Design constraints: Strike length is controlled by stability of the Schist exposed in the FW. Anticipated FW instability may be managed with stope cycle time.

16.4.3 Martin Zone Stope Design

- Design considerations: HW = Basalt, FW = Basalt, steeply dipping;
- Stope dimensions: 20 m vertical height, 30 m stope width (on strike), 20 m stope width considered in stope design.

16.5 Ground Support Strategy

In ground support applications, a goal of rock reinforcement is to limit the amount of closure around the mine opening while assisting the surrounding rock to reach a new state of equilibrium. Ideally, ground support characteristics should match the rock mass demand. In Schist, progressive deformation of the weak rock mass due to elevated ground stresses is to be expected.

Preliminary ground support design was based on established empirical approaches using anticipated rock mass quality, as indicated in Figure 16-16. Numerical modelling was used to further refine the support design, taking into consideration the mining induced stress variation, anticipated deformation intensity and the potential for structural wedge instability.

The anticipated ground support requirements for the Kiena Mine Complex are summarized in Table 16-7. All lateral development with exception of the main ramp is assumed to be 4 m W x 4 m H. The main ramp in the Kiena Deep A Zone considered to be 5 m W x 4.8 m H.

The main ramp and most of the lateral access development are in Basalt. It is located on the hanging wall side of zones. Anticipated rock behaviour is relatively stiff rock mass (elastic rock behaviour).

Komatiite and Schist (Schist Zone) are located on the foot wall of the zones. Anticipated rock behaviour is a yielding / deforming rock mass (non-linear rock behaviour).

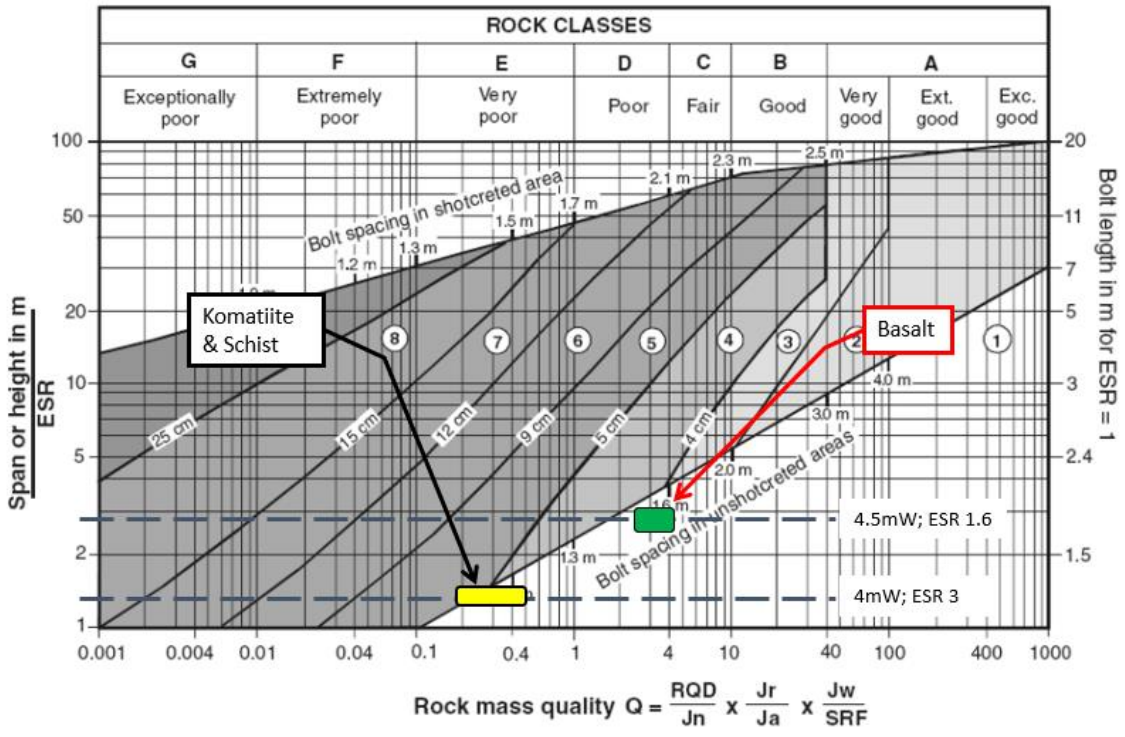


Figure 16-16: Empirical ground support design for drift development based on Q rock mass classification (Barton, 2004)

Table 16-7: Ground support requirements as a function of lithology

Lithology	Primary Support		Secondary support
	Roof	Wall	Roof (wide spans)
Basalt	1.8 m long resin grouted Rebar (20 mm Ø), installed across the roof and upper corners.	1.8 m Split-Set (35 mm Ø) or resin grouted Rebar in lower wall.	Spans exceeding 6 m 6m to 9m spans: install 3 m to 4 m long Super Swellex type bolt (MN24) or cable-bolts: 2 m x 2.4 m, dice 5 pattern. Secondary support to be installed in the excavation core, represented as a diameter of 60% of excavation span. **Brow Support** Cable-bolts, 5 m long, plated.
	Typical bolt pattern: 1.2m x 1.2m Surface cover: Install weldmesh screen (#6 gauge, 100 mm x 100 mm) across the roof and down both walls to 1.5 m above floor.		
Schist Zone	2.1 m long Swellex (Mn12) type bolt.	2.1 m long Swellex (Mn12) type bolt.	**Pre-support (spiling)** install long Split-Set spiles (3.6 m long, 46 mm Ø) in advance of each development round. **Secondary support** 5m to 8m spans: Install 3 m to 4 m long Super Swellex-type bolt (Mn24), plated. Installed on a 1.8 m x 1.8 m pattern Secondary support is to be installed in the excavation core, represented as a diameter of 80% of the excavation span. **Brow Support** Super Swellex-type bolt, 3 m+ long. Installed with zero-gauge screen strap across roof.
	Typical stagger bolt pattern: 1.0 m x 1.0 m Install weldmesh screen (#6 gauge, 100 mm x 100 mm) across the roof and down both walls to approx. 0.8 m above floor. In development parallel to schistosity: Install zero-gauge screen strap vertically over the weldmesh on both drift walls. Install with every third row of wall tendons (3 m spacing maximum).		

16.6 Mining Sequence

16.6.1 Kiena Deep A Zone Mining Sequence

The two limbs of the folded Kiena Deep A Zone appear to have differing stabilities and dilution. When looking at the fold axis, the left hand limb (South limb) is oriented sub-parallel / oblique to the anticipated trend of σ1 (Azimuth ~ 035°). The right hand limb (North limb) is oriented sub-perpendicular to σ1 trend. As a result, the South limb is unfavourably oriented in terms of stress accumulation and dilution potential. The North limb is more favourably oriented.

Mining of Lens A1 stopes prior to Lens A stopes minimizes stress generated challenges to the mining within the Komatiite/Schist rock mass. Sequencing of the Schist Zone stopes (Lens A1 and A2) in the South limb ahead of Basalt hosted stopes (Lens A) is recommended.

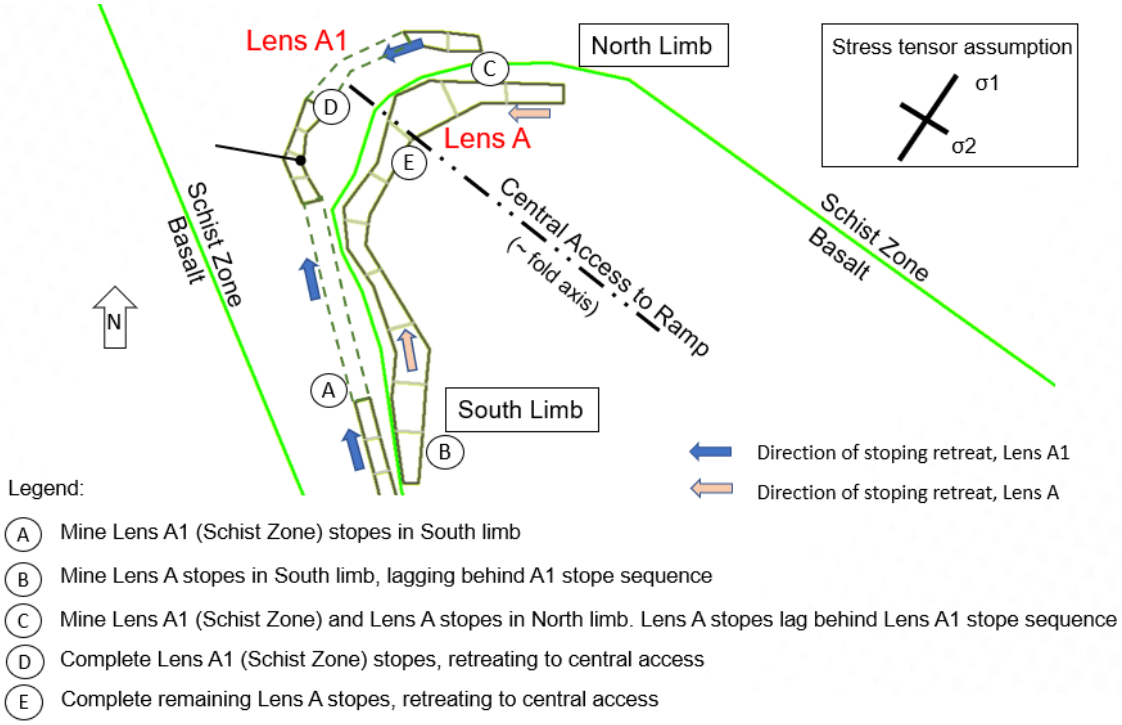


Figure 16-17: Kiena Deep A Zone mining sequence

The mining sequence for the Kiena Deep A Zone illustrated in Figure 16-17 represents the primary consideration for the LOM plan. Mining of the individual lenses by longitudinal retreat long hole stoping starts from the southern and northern extremities advancing towards a central access (central core). On a given mining level, Lens A1 stope is mined first and mining progresses upward. Lens A stope mining lags Lens A1 stope on the given mining level and lags by at least one mining level vertically. The opportunity to mine Lens A stope ahead of Lens A1 stope in the North limb is the secondary consideration for the LOM plan. Where Lens A2 exists in the footwall behind Lens A1 (or Lens A), mine Lens A2 ahead of Lens A1.

Mining fronts consisting of five mining levels each 18 m in height are proposed for the Kiena Deep A Zone. The uppermost mining level in the mining front will be used to mine the stopes in the sill pillar located under previously mined and backfilled stopes in the mining front above. Elevated stresses are expected to accumulate in the diminishing pillar, ahead of the advancing Lens A stope mining direction.

Ground conditions in the central core will deteriorate as pillar dimensions within the lenses are reduced to less than 40 m (approximately two A lens stopes). Planned second pass ground support will be required for late stage mining in the central core pillars.

The concave or folded profile of Lens A results in decreased stress within the hanging wall behind the mined stopes. Stress-relieved conditions, potentially affecting the adjacent access infrastructure, are anticipated as the remaining central pillars of Lens A are mined.

16.6.2 S50 Zone Mining Sequence

Historical mining of S50 Zone in wider areas utilized a transverse long hole stope mining method. A thin isolated lens, approximately 5 m in width, located in the footwall of the main S50 Zone was mined longitudinally. Cut and fill mining was utilized to recover the narrow, lower portion of the historical S50 Zone.

Future S50 Zone mining are discrete lenses located below the historical workings. Existing mine infrastructure (ramp and level development) provides access to these lenses.

The relatively narrow S50 Zone ore lenses will be mined longitudinal retreat utilizing a long hole stope mining method. The premise for sequencing is to retreat on an individual ore lens and complete mining on a given level before advancing to the next level above.

In instances where the ore width requires two production stopes, the hanging wall stopes will be mined first, then the footwall stopes.

16.6.3 Martin Zone Mining Sequence

Two Alimak production stopes will be mined at the upper extent of the Martin Zone while lateral drift development is advanced to reach the production stopes at depth. All other production mining will utilize longitudinal retreat long hole stoping on a bottom up mining sequence.

16.7 Mining Method

16.7.1 Overhand Long Hole Production Mining

Most of the future production mining supporting the LOM plan, approximately 80%, will utilize the historical overhand long hole production stope mining method as illustrated in Figure 16-18.

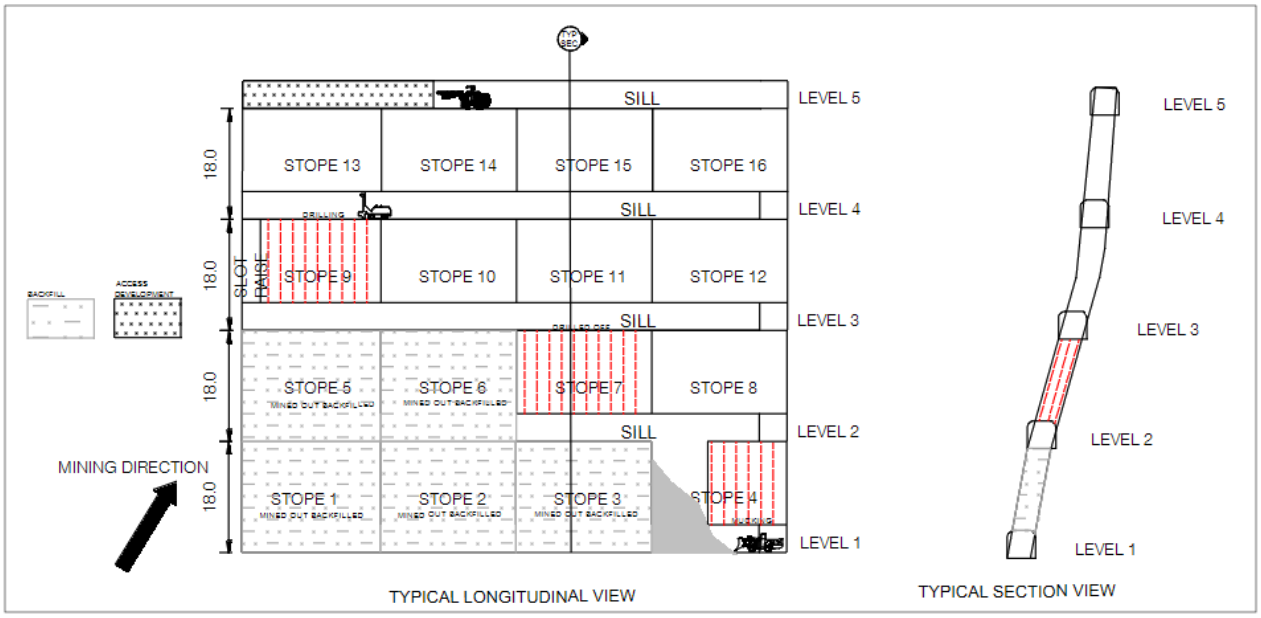


Figure 16-18: Overhand long hole production mining

Overhand long hole production stope mining will be utilized in a bottom up sequence advancing towards the mining level above. Top hammer production drills will be used for down hole production drilling.

Production mining activities associated with a production stope include the following:

- Slot raise drilling;
- Production hole drilling;
- Slot raise blasting and mucking;
- Production hole blasting and mucking;
- Barricade construction;
- Backfilling.

16.7.2 Underhand Long Hole Production Mining

Underhand long hole production stope mining will be utilized when mining under previously mined and backfilled stopes. This will occur in the Kiena Deep A Zone when mining fronts advance towards a completed mining front (refer to Figure 16-19). This mining method represents approximately 18% of the production mining supporting the LOM plan. The same top hammer production drills used in the overhand mining method will be utilized for up hole production drilling expected in the underhand mining method. Production mining activities identified in the overhand mining method also apply with the underhand mining method.

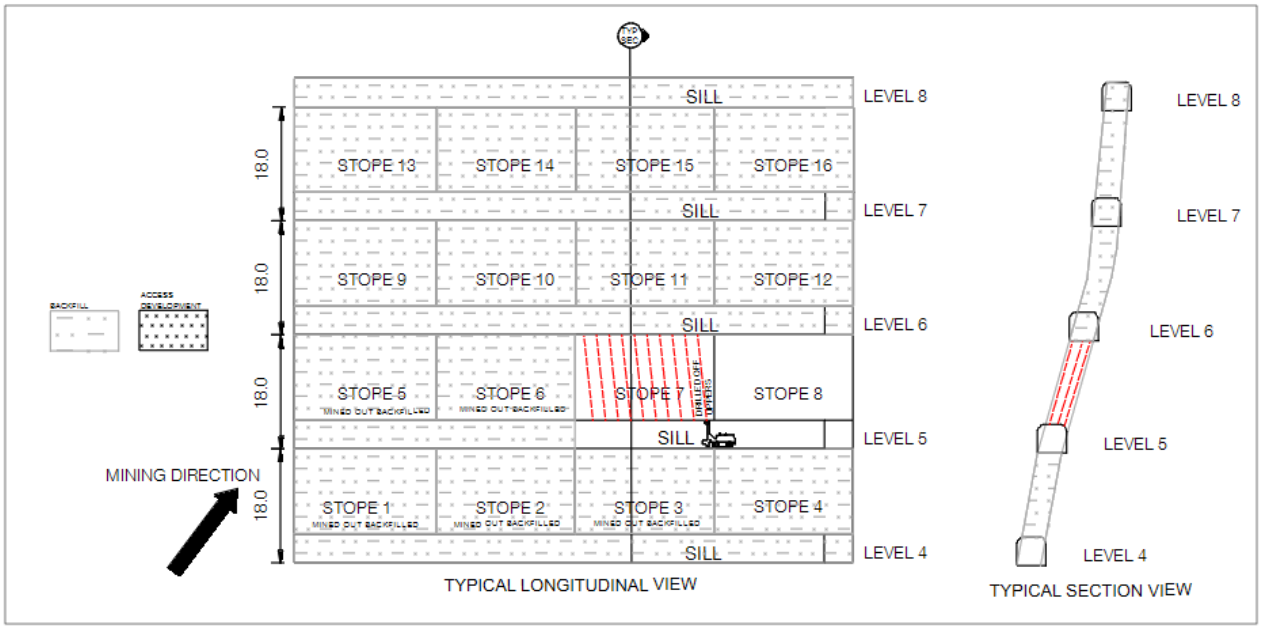


Figure 16-19: Underhand long hole production mining

16.7.3 Alimak Production Mining

Alimak production stope mining, two stopes in total representing approximately 2% of the production mining supporting the LOM plan, will be utilized in the Martin Zone. A mining contractor will be engaged for both development and production mining in the Martin Zone and are expected to supply all Alimak equipment.

16.8 New and Upgraded Infrastructure

16.8.1 Hoist Shaft

The existing hoisting shaft will be utilized for mining of the Kiena Deep A, S50 and Martin Zones. The hoist drives are scheduled for replacement in 2021 to improve productivity and facilitate future maintenance. Mechanical and infrastructure maintenance including crash gates and doors at the loading pocket will be conducted at this time. In parallel, instrumentation will be added to support the control and automation of the ore and waste rock hoisting system.

16.8.2 Mobile Equipment

The new fleet of underground and surface mobile equipment to be purchased to support the LOM plan is identified in Table 16-8.

Table 16-8: New fleet of mobile equipment

Mobile Equipment	Class	Total	2021	2022	2023	2024	2025	2026	2027	2028
LHD	4yd^3 Bucket	**2**		2						
	6yd^3 Bucket	**2**		2						
Haulage Truck	30T Diesel	**4**		4						
	42T Battery	**3**			2	1				
Jumbo Drill	One Boom	**1**		1						
	Two Booms	**1**		1						
Production Drill Top Hammer	LH II	**3**		2		1				
Scissor Lift	SLX5100M4D	**3**		3						
ANFO Loader	MC100G	**2**		2						
Bolter	Boltec S	**2**		2						
Personnel Carrier	HZJ79	**3**	3							
Boom Truck	BT2	**1**		1						
Grader	CAT UG20M	**1**		1						
Cement Transmixer	Cassette	**1**	1							
Tractor with Fork Attachment	6110M	**2**	2							
Tractor with Basket	6110M	**2**	2							
Surface Loader	CAT 926M	**1**	1							

Most of the mobile equipment are diesel powered. Battery powered haulage trucks will be integrated into the fleet as mining extends deeper in the Kiena Deep A Zone. In total, three battery electric vehicles (BEV) will be part of the prime haulers.

A trade-off study was conducted to assess the economic viability of BEV and although not considered, future studies could include the impact on the carbon emission inventory and to assess the expected noise reduction underground. The trade-off study evaluated 42-tonne capacity haulage trucks both diesel and battery powered. The width of the haulage truck requires the main ramp in the Kiena Deep Zone to be driven 5.0 m in width to comply with clearance requirements imposed by Quebec mining regulations. Integrating battery powered haulage trucks into the mobile equipment fleet was deemed economically viable as it eliminated the requirement for a mine air (fresh air) ventilation cooling plant with an in-house estimated capital cost of C$11 M. The mine air ventilation cooling plant would be required with a fully diesel powered haulage truck fleet operating at depth.

In terms of haulage trucks, the intent is to utilize a fleet comprised of 30-tonne diesel powered only from 2021 to 2023 conducive to existing development infrastructure and then incorporate the 42-tonne battery powered haulage trucks starting in 2023. As of 2023, both diesel and battery powered haulage trucks will be operational to meet haulage requirements.

16.8.3 Ore Crushing and Waste Rock Breaking

The ore crusher located on 81 Level and the associated ore pass infrastructure will be utilized in mining the Kiena Deep A, S50 and Martin Zones. Structural repairs to infrastructure including ladders, stairs and the dust collector at the ore crusher will be completed in 2021.

The rock breaker located on 81 Level was recently rebuilt. Waste rock generated from underground mine development that is not disposed of underground as mine backfill will be hoisted to surface and stored at the current waste rock facility.

16.8.4 Construction

In order to support the LOM plan, surface and underground construction items listed in Table 16-9 are required.

Table 16-9: Surface and underground construction

Construction Item	Unit	Total	2021	2022	2023	2024	2025	2026	2027	2028
Surface Fresh Air Ventilation Fans and Heaters	each	**2**		2						
Surface Exhaust Air Ventilation Fans	each	**2**		2						
Surface Ventilation Electrical House	each	**1**		1						
Surface and Underground Paste Backfill	lot	**1**		1						
Kiena Deep Second Egress - Escapeway	m	**697**	115	131	133	152	130	36		
Dewatering Solids Management Station	each	**1**	1							
Ventilation Regulating Control Wall	each	**20**	3	3	4	5	3	2		
Refuge Station	each	**5**	1	1	1	1		1		
Latrine Station	each	**5**	1	1	1	1		1		
Haulage Truck Loading Chute/Pass	each	**1**		1						
Service Repair Station	each	**1**		1						
Explosive Magazine	each	**1**			1					
Fuse Magazine	each	**1**			1					
Fuel and Oil Station	each	**1**			1					

The 42-tonne battery powered haulage trucks considered are too wide to travel to the existing truck dumps on 79 and 81 Levels and the service facilities on 64 Level while complying with clearance requirements imposed by Quebec mining regulations. The loading chute/pass located between 107 and 109 Levels will be used to transfer ore and waste rock hauled by the 42-tonne haulage trucks to the 30-tonne diesel powered haulage trucks. Also, the service repair station to be established in the Kiena Deep Zone will be used to service the wider battery powered haulage trucks.

16.8.5 Ventilation System

The Kiena Mine Complex requires an upgrade to the mine air ventilation infrastructure located on surface. Infrastructure includes the purchase and installation of two fresh air fans, two fresh air heaters, two exhaust air fans and an electrical house. Commissioning is scheduled in July 2022 and is represented in the LOM plan. General arrangement drawings were generated at a conceptual level.

Mine Air Cooling Evaluation

A heat load evaluation was conducted to assess the requirement for a mine air (fresh air) cooling system as part of the ventilation system for the Kiena Deep A Zone. As the deepest mining zone containing most of the Mineral Reserve estimate in the LOM plan, it requires most of the mobile equipment for ore and waste rock haulage.

Two fleets of haulage trucks were considered, a fleet of haulage trucks that is diesel powered only requiring a 4.5 MW mine air cooling system estimated to have a capital cost of C$11 M and a combined fleet of diesel and battery powered haulage trucks that eliminated the requirement for mine air cooling. In the combined fleet, diesel powered haulage trucks can operate down to 148 Level as required and only battery powered haulage trucks operate below 148 Level.

The cut-off temperature considered as a requirement for mine air cooling was 28.5°C wet bulb. This temperature was selected as it is used by a neighbouring mine as the stop work temperature for medium work. The Kiena Deep A Zone is a relatively compact working environment with a single fresh air source and a large mobile equipment fleet expected to be operated within an elevation range of approximately 700 m. A fully diesel powered mobile equipment fleet would contribute over 60% of all heat gains underground.

As battery compared to diesel powered mobile equipment emit less heat during operation, it is possible that replacing diesel powered mobile equipment below a specific mining level could eliminate the requirement for mine air cooling. The mining level where the wet bulb limit of 28.5°C was first realized was established by considering diesel powered mobile equipment only. The ventilation modelling established 148 Level to be the mining level limit for diesel powered haulage trucks to operate in the Kiena Deep A Zone.

Below 148 Level, only battery powered haulage trucks were considered to assess their influence on the wet bulb temperature. Refer to the yellow star in Figure 16-20 that illustrates the location of 148 Level established as the threshold mining level for diesel powered haulage trucks to operate in the Kiena Deep A Zone.

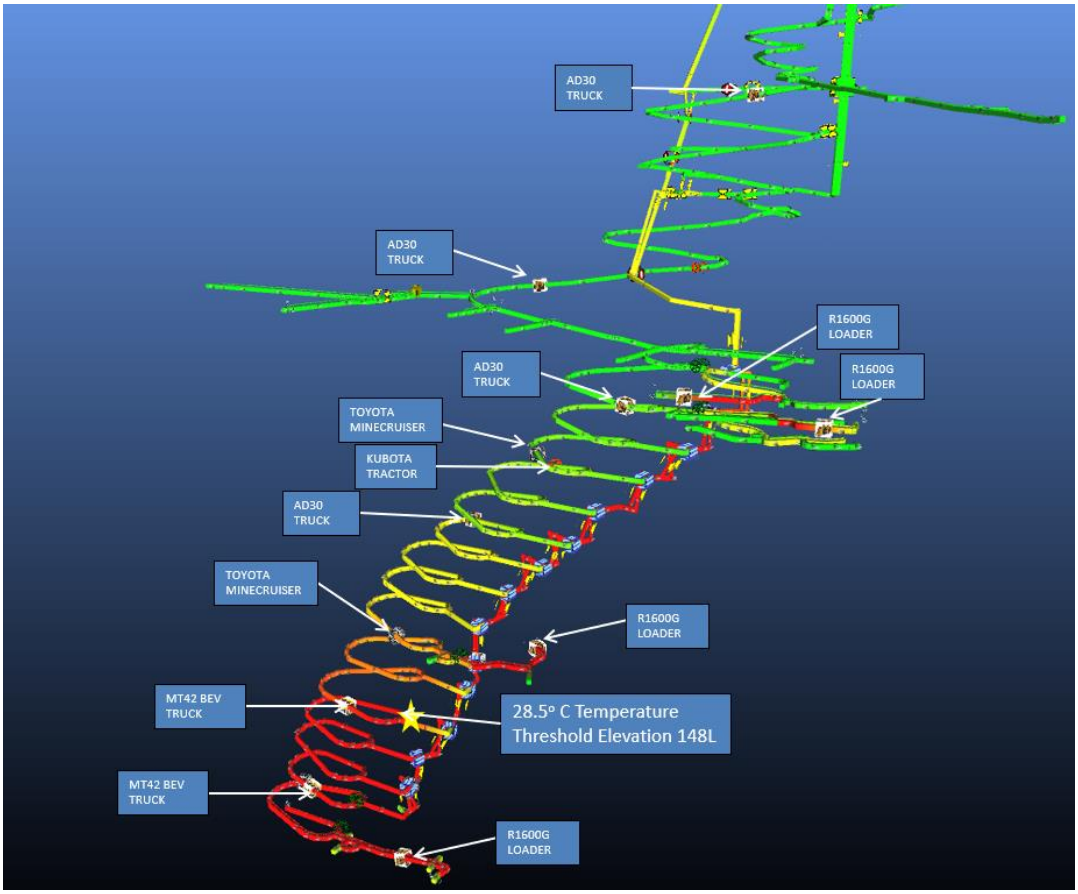


Figure 16-20: Threshold mining level to operate diesel powered haulage trucks

16.8.6 Repair Facilities

Existing underground service repair facilities will continue to be used as the primary areas to conduct maintenance services associated with fixed infrastructure as well as repairs to underground mobile equipment. A new underground service repair facility is required and will be constructed in the Kiena Deep A Zone. It will serve as an additional repair area for underground mobile equipment, more specifically to accommodate the largest haulage truck (42-tonne) to be operating in the zone. The width of the 42-tonne haulage truck limits it from travelling to the primary service repair facilities located on 64 Level.

16.8.7 Fuel and Oil Facilities

New fuel and oil facilities are required for the Kiena Deep A Zone. A satellite fueling station and oil storage will be established. Fuel will be delivered to the station through an extension of the fuel pipeline from its current location on 82 Level.

16.8.8 Rockfill Facilities

An LHD was allocated for the haulage and placement of rockfill into mined out stopes. A cement transmixer was purchased in 2021 and is currently on site for use in the preparation of cemented rockfill. Mine development rock will be coated with cement slurry transported in the transmixer and mixed in a designated pit excavation. Once mixed, the cemented rockfill be hauled and placed into mined out stopes.

16.8.9 Paste Backfill Facilities

Reconciliation of In situ Material to Mineable Reserve Estimate

The LOM plan assumed no rib or sill pillars will remain in the mining blocks where paste backfill is considered. Using paste backfill will enable ore to be recovered adjacent to the paste fill masses with minimum dilution. When extracting sill pillars, underhand mining below paste backfilled areas will be utilized. It is assumed that reinforcement in the form of stronger fill and or mats will be used.

Design Basis

The project philosophy is to maximize underground productivity within a safe and sustainable manner, that includes tailings management. The design included herein is based on a blended sand/tailings paste to accelerate mining cycle time. Use of sand at 65% of the backfill requires ~87% of the tailings to be stored on surface; this compares to 50% on surface and 50% placed underground when sand is not used to augment the backfill. For any recipe less than 65% sand will require a vacuum filter for the tailings, while a 65/35 blended sand/tailings paste requires the use of a paste thickener operating at ~65 wt% solids underflow in order to make the paste recipe.

Testing of the tailings for backfill quality indicates that the sand/tailings blends provide the safe and productive requirements necessary for the economically successful exploitation at the Kiena Mine Complex. Use of tailings alone in paste fill is possible in old voids and limited to those stopes not critical in the mining cycle, where 7-day cycle times are not required. Both blended paste and 100% tailings paste are amenable to backfill at the Kiena Mine Complex; optimization makes it possible to apply the various recipes as needed on a stope-by-stope basis. This is also important because cyclone classification (as previously in the sand fill system at Kiena) is not necessary, further mitigating the impact on the TSF placement (varying flow rates when pumping either slimes or whole tailings when not backfilling).

The paste plant will consist of a new thickener, agitated tailings storage tank, sand system, binder system and mixing along with the boreholes and underground piping (UDS). Ancillary systems include compressed air line, fresh water, power and communications. No allowance has been made for washrooms, sewage, etc. A plant control room and electrical MCC room has been outlined. A laboratory for paste testing is also allocated within the building.

A sand blend with tailings has been incorporated due to the high variability of the ore types and the predominance of talc-chlorite which has been stated as being problematic for tailings dewatering and strength development, within the targets of 7-day strength and an economic trade-off of sand, binder and TSF costs.

The planned lateral retreat extraction sequence along strike should be scheduled so that the planned paste filled sublevel long hole stopes are mined after the placed paste fill has been allowed to cure and to achieve a required strength that will achieve safe and efficient productivity. That goal is currently targeted at 7-day turnaround time at which point blasting of the next stope begins. The required strength is largely dependent on dimensions of the filled stope, however for this project, the average width considered is 5m x 10m to 20m lengths over 18m height is 140 kPa while 200 kPa is required for 10 m widths.

Binder usage with a blended paste having 65% sand and 35% tailings by mass will require approximately 2% binder to achieve the free-standing strength of 140 kPa for 5 m stope width x 20 m high, it is estimated that 2.5% will achieve 200 kPa for 10 m mining; 100 kPa is considered safe for non-liquefaction. An increased binder content of 3% was considered for undercut stopes.

Waste rock can be placed with the paste pour into stopes so long as a 1:1 ratio of paste and rock is not exceeded (waste rock tonnes should not exceed the paste tonnes). This ratio allows a large volume of paste to inundate the waste rock and encapsulate the rock with cementitious material.

Barricades can be made of waste rock rather than shotcrete, as many mines in Canada and abroad have been doing so safely for up to 25 years without incident. Additional practices such as sealing of the brow with shotcrete or spray foam, as well as use of rammer jammers to push waste rock tightly to the back have been incorporated in many operations. Site by site specifications have been made to reflect stope dimensions and orientation of dip, rock mass, and other aspects. Shotcrete barricades may be used, including recently developed Sturda walls.

Paste Plant Location

The paste plant will be located on the footprint of the former hydraulic fill plant that will be demolished. This site has been preferentially selected due to limited footprint, ease of drilling boreholes through former mining zones and reaching 107 Level and to the 112 Level, which will be ready by February 2022. The overall layout is shown in Appendix A, while the boreholes are shown in Figure 16-21. A sand stockpile will be to the south of the paste plant, where the existing stockpile is located. The sand stockpile footprint will be increased by relocating the hazardous materials storage building further to the west.

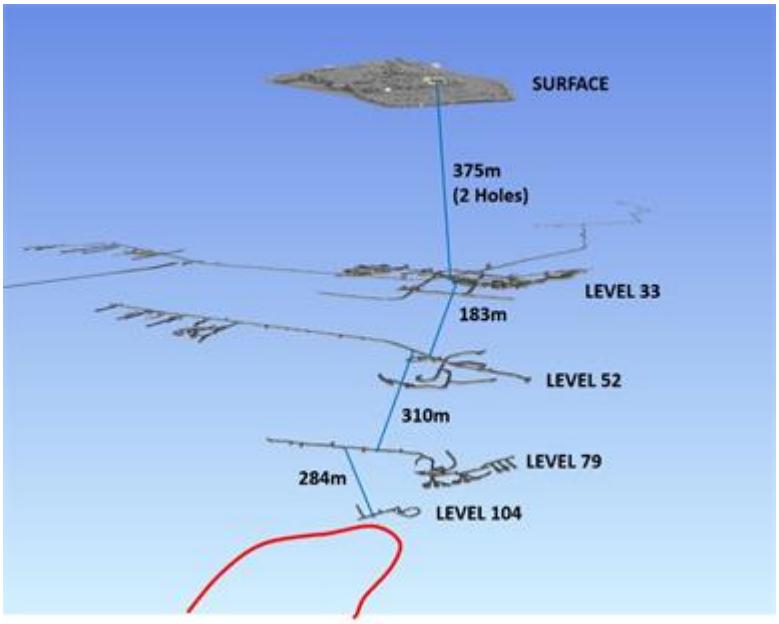


Figure 16-21: Boreholes underground

The underground distribution system (UDS) consisting of the 6-in Schedule 80 wear-resistant cased boreholes and level piping is a critical design aspect herein, given the >1,700 m depth to planned mining though less than 200 to 300 metres laterally from the shaft bottom. The necessity of a well-controlled low slump paste product to mitigate wear of the boreholes is the primary design consideration that enables low-slump paste to flow at a reduced velocity down inclined boreholes, which have been laid out in a cascading pattern from surface to 33 Level, and then to subsequent levels to the current mining zones. The boreholes will be lined with abrasion resistant piping, and a redundant system will be in place prior to start-up. Access to the North Zone will be via 12 Level from which the surface to 33 Level lined borehole will be intersected.

The former roadway entering along the main causeway through to the Mill will be re-established to enable safe and efficient sand haul truck and binder truck access and return, while reducing traffic through the main control (employee and warehousing materials delivery) gates.

Tie-ins to the existing water, compressed air and power systems will be needed. The natural gas line will need to be relocated. Noise generated using a front-end-loader to re-handle sand will be mitigated through the employment of noise barriers (by others).

Sand stockpile will be covered to mitigate precipitation from significantly altering moisture content and reducing the potential for frozen chunks. A sand screen deck will be used to eliminate both frozen chunks and oversize particles that may have entered via worn screens at the sand supply location, or from the Kiena operator digging below the stockpile into subgrade material. Furthermore, any mechanical pieces (bucket tooth, bolts, etc.) can be removed.

Productivity

The paste backfill system is designed for 100 tph solids at a nominal flow rate of 60 m^3/hr within a 150 mm (6-in diameter pipe). The filling rate exceeds the mining rate through the application of sand blend, such that backfill will require 48 to 72 hours of pouring per week to fill the stopes. This does not include time for curing, flushing, or preparation. A typical paste fill system by comparison in most mining applications considers 60% utilization. The variation herein is due to the milling rate of 1,500 tpd (~63 tph) versus the filling rate (100 tph) which yields ~37% utilization.

Underground Tailings Disposal - North Zone

Located under a surface crown pillar approximately 100 metres below Lac De Montigny are historical mined out workings in the North Zone. The voids total approximately 278,000 m^3. Engineered hydrostatic bulkheads were recently constructed to allow the voids to be backfilled with low-strength tailings backfill. Underground disposal of the tailings in the voids will be conducted through a network of boreholes and pipelines to enable up to 270,000 tonnes of tailings to be deposited, assuming 80% filling efficiency and 1.21 dry t/m^3 placement density. The placement density is a function of both time and initial slurry density at time of placement.

In order to mitigate the risk of liquefaction, Wesdome has elected to add cement to the tailings to provide additional strength. A binder addition rate of 1 dry wt% was estimated for this task. Increasing the density of the placed fill will enhance the effectiveness of this strategy.

Entry boreholes collared on surface at the location of the paste plant will be drilled to the lower mining horizons. Lateral pipelines will be installed to reach the pour locations.

16.8.10 Dewatering Facilities

Underground mine dewatering below 94 Level will utilize a dirty water pumping system. As the mine extends deeper, the amount of ground water inflowing into the mine is not expected to increase and the new pumping system is designed to handle the water generated by the proposed mining equipment, dust suppression and backfill.

The dirty water system intended for Kiena Deep A and S50 Zones will incorporate sump pumps and modular skid-mounted dirty water pumping systems that will be relocated as mine development progresses until installation at their final locations. All dirty water below 94 Level will be pumped to the solids management system on 94 Level. Solids will be removed from the dirty water for reclaim or underground disposal. The clear water generated will either be pumped to surface using the existing clear water pumping station on 94 Level or recycled back to the mine as process water.

16.8.11 Process Water Facilities

The process water system will be expanded on 94 Level to include a pumping arrangement for recycling clear water resulting from the solids management system considered with mine dewatering. As the mines extends deeper, the required pipeline will be installed and is costed in the cost of lateral drift development.

16.8.12 Compressed Air Facilities

The infrastructure supporting the compressed air system was evaluated and deemed adequate to provide the compressed air requirements to operate the underground mine. Mine development requires the compressed air distribution system to be extended. The required pipeline and installation costs are included in the cost of lateral drift development.

16.8.13 Electrical Facilities

Electrical deficiencies identified in the underground electrical rehabilitation assessment and refurbishment plan need to be addressed for existing installations above 111 Level.

A new 25 kV-13.8 kV transformer will be installed on surface by the end of 2021 to provide 13.8 kV power to the S50 and Kiena Deep A Zones. From surface, a 15 kV 350 MCM cable will be installed in the service compartment of the hoist shaft. At shaft bottom, the 15 kV 350 MCM cable will continue to the S50 Zone and extend deeper to the Kiena Deep A Zone.

In total, eighteen and three 13.8 kV-0.6 kV skid mounted substations are estimated for Kiena Deep A and S50 Zones respectively to supply power to support the LOM plan. Each substation will be installed in their own respective electrical station. The first substation to be established and connected to the 15 kV 350 MCM cable will be on 111 Level in the Kiena Deep A Zone.

16.8.14 Battery Charging Facilities

Battery charging stations will be required to ensure batteries are available to power the three battery electric vehicle (BEV) 42-tonne haulage trucks forming part of the new underground mobile equipment fleet. As of 2023, two chargers and charger posts are required with an additional charger and charger post required in 2024.

16.8.15 Communications Facilities

The existing communications network backbone and leaky feeder infrastructure will be advanced to support mining operations. Emergency Voice over Internet Protocol (VoIP) phones will be installed in new refuge stations, service stations and electrical facilities. Likewise, provisions were made for closed circuit television (CCTV) to be installed at backfill pour sites, new repair service stations and refuge stations.

16.8.16 Refuge Facilities

Kiena Deep A Zone will require five refuge stations. The refuge stations will be developed and constructed at intervals to satisfy a 1,000-m distance and 15-minute walking criteria imposed by Quebec mining regulations. The first new refuge station is being developed and constructed on 112 Level.

16.8.17 Second Egress Facilities

An escapeway will be provided in one of the two ventilation exhaust air raises being developed from 109 Level down to 177 Level in the Kiena Deep A Zone. Ladderways will be installed in the ventilation exhaust air raise to be ground supported. The escapeway along with the ramp development advancing to 177 Level extends the second egress system for the Kiena Mine Complex.

16.9 Mine Development

Lateral drift development in waste rock and ore advancing in the S50 and Kiena Deep A Zones is scheduled to be sustained at an overall average rate of 12 metres per day (mpd) until 2025. At present, the Kiena Mine Complex utilizes mining contractors for lateral drift development. Procurement of equipment and labour recruitment has begun to establish Kiena Mine Complex proprietary development crews by the end of 2021. The Kiena Mine Complex technical services team has designed ramps, level access drifts, level infrastructure and ore sills to achieve the Mineral Reserve estimate. Most of the level access development for S50 and Martin Zones was completed prior to Kiena Mine Complex coming out of care and maintenance. As such, the Mineral Reserve estimate in these mining zones will be readily accessible with limited lateral development expected in the future. The lateral development scheduled in the Kiena Deep A Zone represents all lateral development required to access the Mineral Reserve estimate in the Kiena Deep A Zone.

Ramp, level access and ore sill designs were completed for all mining zones. In order to minimize lateral development requirements, a sublevel system was considered for the Kiena Deep A Zone. This approach allows each main ramp level access to service two sublevels with a ramp providing access to each sublevel. A typical development layout for the Kiena Deep A Zone is provided in Figure 16-22. The upper sublevel ore sills are shown while the lower sublevel ore sills are hidden for clarity purposes.

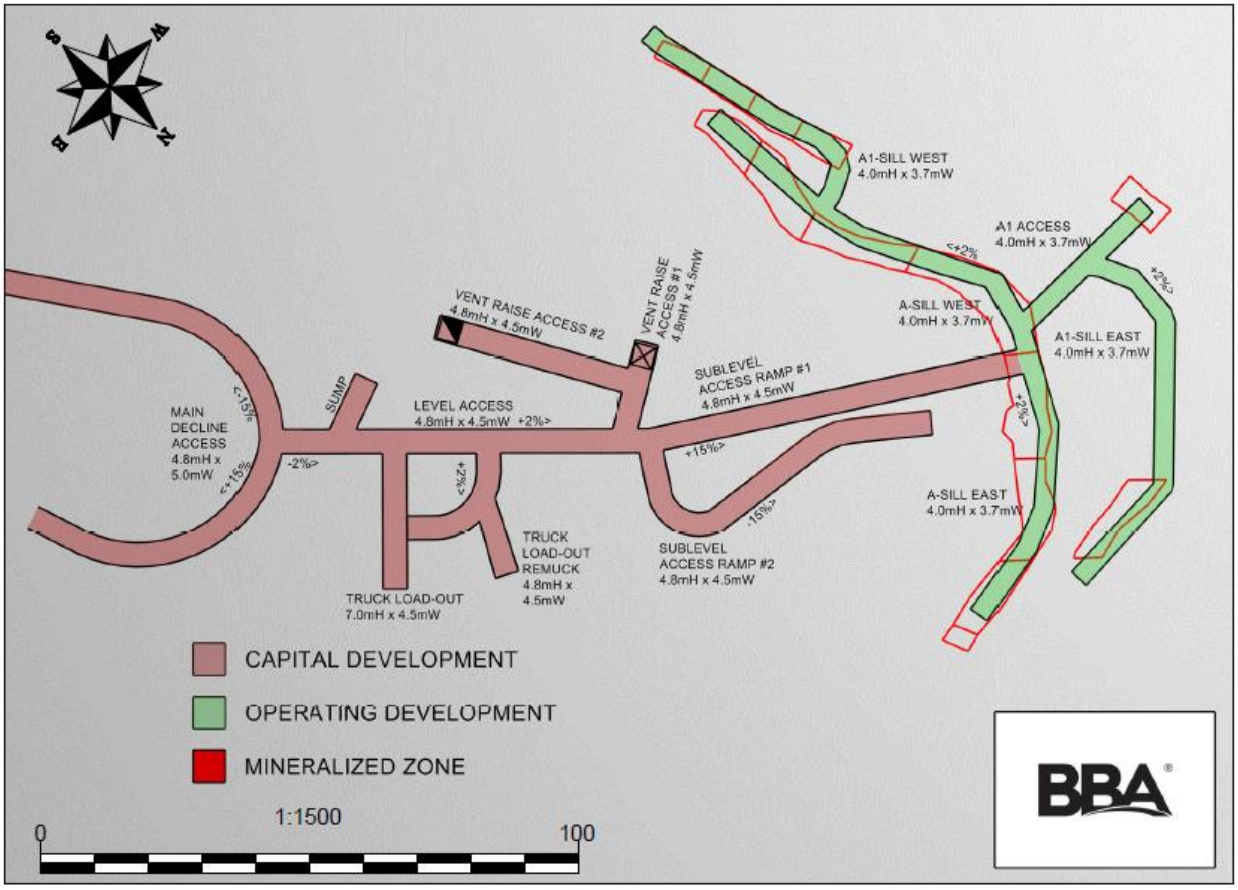


Figure 16-22: Typical development layout for the Kiena Deep A Zone

Vertical raise development in waste rock is scheduled in the Kiena Deep A Zone to establish the ventilation exhaust air raises. Two ventilation exhaust air raises will be developed from 109 Level down to 177 Level. Vertical raise development is dependent on lateral drift development and constrains production mining both for ventilation and second egress considerations.

The mine development requirements in both waste rock and ore for the LOM plan are identified in Table 16-12.

Lateral drift dimensions considered in the LOM plan are influenced by existing lateral drift sizing, compliance to Quebec mining regulations associated with trackless mobile equipment and scheduling for both development and production mining. Vertical raise dimensions considered in the LOM plan are influenced by both ventilation and second egress requirements. Table 16-10 identifies the mine development types and their design dimensions.

Table 16-10: Mine development types and design dimensions

Development Type	Width (m)	Height (m)
Ramp – Existing	4.0 to 4.5	4.5
Ramp – New, Kiena Deep Zone	5.0	4.8
Level Access	4.5	4.8
Ramp – Sublevel Access	4.5	4.8
Truck Loadout	4.5	7.0
Ventilation Access Drift	4.5	4.5
Electrical Substation	4.5	4.5
Sump	4.5	4.5
Ore Sill (Adjusted for Execution)	3.7 (4.0)	4.0
Ventilation Exhaust Raise, Diameter	3.3	3.3

16.10 Life of Mine Plan

The LOM plan was developed jointly between the QP and the Kiena Mine Complex technical services team. The LOM plan was finalized on April 12, 2021. Productivity rates were influenced by labour noted in Chapter 21 (Capital and Operating Costs), the new fleet of mobile equipment, fixed infrastructure requirements and mining cycle constraints. Due consideration was given to hoist shaft conveyance and milling throughput capabilities. Table 16-11 identifies the rates, durations and factors considered in the LOM plan.

Table 16-11: Rates, durations and factors considered in the LOM plan

Rate and Duration	Unit	Value
Lateral Development Ore	mpd per heading	2.4
Lateral Development Waste Rock	mpd per heading	3.0
Vertical Development Waste Rock, 2 Ventilation Exhaust Raises – 70 m total	d	56
Long Hole Production Drilling	mpd per drill	150
Long Hole Production Blasting	d	2 to 6
Long Hole Production Mucking	tpd per scoop	300
Cemented Rockfill	tpd	200
Cemented Rockfill Curing	d	14
Paste Backfill	tpd	1,200
Paste Backfill Curing	d	7
Factor	**Unit**	**Value**
Long Hole Drilling	t/m	3.5
Paste Backfill Factor	Volumetric	0.65

In order to enhance ground stability, production stope sequencing established by geomechanical analysis and presented earlier in this chapter for each mining zone was applied in the mine schedule (LOM plan). Production stope sequencing in the Kiena Deep A Zone considered eight mining fronts comprised of five mining levels each. Figure 16-23 illustrates the mining fronts established in the Kiena Deep A Zone.

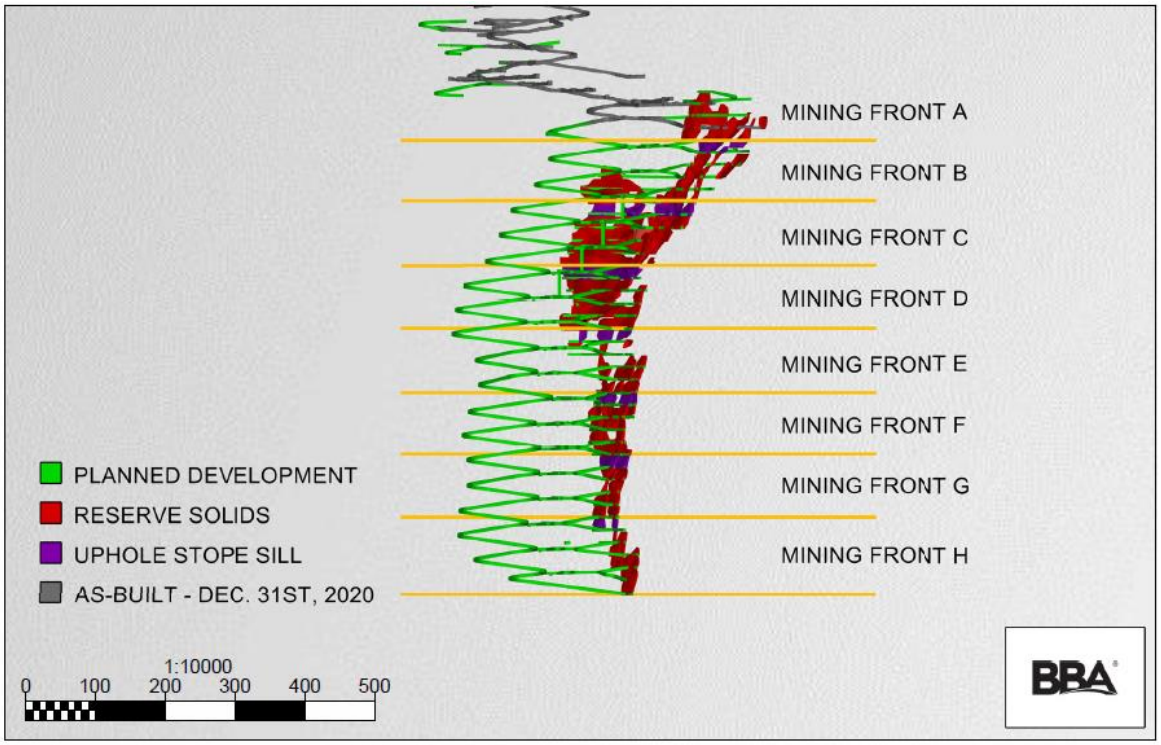


Figure 16-23: Isometric long-section of the mining fronts in the Kiena Deep Zone (facing northwest)

Mine development in the Kiena Deep A Zone will be advanced to establish the bottom mining level of the shallowest mining front and progress deeper to establish the bottom level of each successive mining front thereafter. Production mining will progress bottom up in each mining front using the overhand long hole stoping method. Seven undercut stoping levels are expected using the underhand long hole stoping method.

The initial stopes mined in the shallowest mining front only (Mining Front A), will be backfilled with cemented rockfill (CRF) until the paste backfill system is commissioned in July 2022. Due consideration will need to be given with the CRF backfilled stopes expected to be undercut by Mining Front B. Quality records of the CRF placed in the first stopes of Mining Front A must be maintained for future stope designs in Mining Front B. Undercut stopes expected in the deeper mining fronts (Mining Fronts B to H) will be paste backfilled. An increased binder content was considered for these undercut stopes to enhance stability. The cure time of the undercut backfilled stopes is prolonged as paste backfill exposure will occur only when the mining sequence of the five mining levels below promotes exposure. Refer to Section 16.8.9 (Paste Backfill Facilities) for paste backfill recipe considerations.

Refer to Table 16-12 representing the LOM plan for the Wesdome Kiena Mine Complex. Subsequently, Table 16-13 represents the LOM plan for the Kiena Deep A Zone, Table 16-14 represents the LOM plan for the S50 Zone and Table 16-15 represents the LOM plan for the Martin Zone.

Refer to Figure 16-24 for the total ore tonnes and total recovered gold ounces representing the Mineral Reserve estimate and captured in the LOM plan for the Wesdome Kiena Mine Complex.

Table 16-12: LOM plan for the Kiena Mine Complex

	Unit	2021	2022	2023	2024	2025	2026	2027	2028	Total
Total Lateral Development	m	4,186	4,071	4,311	4,088	3,123	1,005	169	-	**20,953 m**
Lateral Development Waste	m	3,313	2,683	3,052	2,998	2,297	745	116	-	**15,204 m**
Lateral Development Waste	t	188,583	156,406	178,027	168,858	130,704	33,323	6,480	-	**862,381 t**
Lateral Development Ore	m	872	1,388	1,259	1,090	826	261	53	-	**5,749 m**
Lateral Development Ore	t	35,450	56,403	51,162	42,262	33,571	9,870	2,154	-	**230,872 t**
Lateral Development Ore Grade	g/t	11.12	10.71	9.47	9.90	12.13	3.52	3.20	-	**10.18 g/t**
Vertical Development Ore	m	-	-	-	141	-	-	-	-	**141 m**
Vertical Development Ore	t	-	-	-	3,678	-	-	-	-	**3,678 t**
Vertical Development Ore Grade	g/t	-	-	-	5.3	-	-	-	-	**5.34 g/t**
Vertical Development Waste	m	230	262	267	304	259	72	-	-	**1,394 m**
Vertical Development Waste	t	5,794	6,599	6,727	7,649	6,533	806	-	-	**34,108 t**
Total Development Waste	t	194,376	163,005	184,754	176,507	137,237	34,129	6,480	-	**896,488 t**
Long Hole Drilling Metres	m	11,595	32,604	41,964	67,635	63,486	73,927	89,455	2,158	**382,824 m**
Long Hole Production Ore	t	40,581	104,763	146,947	231,567	224,300	245,741	335,030	10,955	**1,339,884 t**
Long Hole Production Ore Grade	g/t	9.91	14.92	11.30	12.18	14.40	13.70	9.69	5.52	**12.20 g/t**
Total Rockfill	t	-	-	-	-	-	5,189	13,384	-	**18,573 t**
Total Cemented Rock Backfill	t	28,451	38,154	-	-	-	5,477	31,883	-	**103,965 t**
Total Wet Paste Backfill (1.82 t/m^3)	t	-	37,716	107,591	142,988	166,210	167,268	176,417	15,055	**813,245 t**
Total Backfill	t	28,451	75,870	107,591	142,988	166,210	177,934	221,685	15,055	**935,783 t**
Total Ore	t	76,031	161,166	198,109	277,507	257,871	255,612	337,183	10,955	**1,574,434 t**
Average Ore Grade	g/t	10.48	13.44	10.82	11.74	14.10	13.30	9.65	5.52	**11.89 g/t**
Mineable Gold Ounces	oz	25,612	69,659	68,935	104,756	116,933	109,323	104,616	1,944	**601,778 oz**
Recovered Gold Ounces	**oz**	**25,204**	**68,614**	**67,901**	**103,102**	**115,179**	**107,641**	**102,829**	**1,915**	**592,385 oz**

Table 16-13: LOM plan for the Kiena Deep A Zone

Kiena Deep Zone										
	Unit	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**Total**
Total Lateral Development	m	3,804	4,071	4,311	3,822	3,123	-	-	-	**19,131 m**
Lateral Development Waste	m	3,011	2,683	3,052	2,858	2,297	-	-	-	**13,901 m**
Lateral Development Waste	t	175,201	156,406	178,027	165,437	130,704	-	-	-	**805,775 t**
Lateral Development Ore	m	794	1,388	1,259	964	826	-	-	-	**5,231 m**
Lateral Development Ore	t	32,260	56,403	51,162	39,182	33,571	-	-	-	**212,578 t**
Lateral Development Ore Grade	g/t	11.8	10.7	9.5	10.3	12.1	-	-	-	**10.7 g/t**
Vertical Development Ore	m	-	-	-	-	-	-	-	-	**-**
Vertical Development Ore	t	-	-	-	-	-	-	-	-	**-**
Vertical Development Ore Grade	g/t	-	-	-	-	-	-	-	-	**-**
Vertical Development Waste	m	230	262	267	274	259	-	-	-	**1,292 m**
Vertical Development Waste	t	5,794	6,599	6,727	6,905	6,533	-	-	-	**32,558 t**
Total Development Waste	t	180,994	163,005	184,754	172,342	137,237	-	-	-	**838,332 t**
Long Hole Drilling Metres	m	9,426	32,604	41,964	60,064	63,486	68,478	57,395	2,158	**335,575 m**
Long Hole Production Ore	t	32,991	104,763	146,947	205,069	224,300	230,134	219,354	10,955	**1,174,513 t**
Long Hole Production Ore Grade	g/t	11.1	14.9	11.3	13.1	14.4	14.4	12.8	5.5	**13.3 g/t**
Total Rockfill	t	-	-	-	-	-	-	-	-	**-**
Total Cemented Rock Backfill	t	22,925	38,154	-	-	-	-	-	-	**61,079 t**
Total Wet Paste Backfill (1.82 t/m^3)	t	-	37,716	107,591	142,988	166,210	167,268	154,796	15,055	**791,624 t**
Total Backfill	t	22,925	75,870	107,591	142,988	166,210	167,268	154,796	15,055	**852,703 t**
Total Ore	t	65,251	161,166	198,109	244,251	257,871	230,134	219,354	10,955	**1,387,091 t**
Average Ore Grade	g/t	11.4	13.4	10.8	12.6	14.1	14.4	12.8	5.5	**12.95 g/t**
Mineable Gold Ounces	oz	23,988	69,659	68,935	99,263	116,933	106,491	90,083	1,944	**577,296 oz**
Recovered Gold Ounces	**oz**	**23,628**	**68,614**	**67,901**	**97,774**	**115,179**	**104,893**	**88,732**	**1,915**	**568,636 oz**

Table 16-14: LOM plan for the S50 Zone

S50 Zone										
		2021	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**Total**
Total Lateral Development	m	381	-	-	-	-	279	169	-	**829 m**
Lateral Development Waste	m	303	-	-	-	-	251	116	-	**670 m**
Lateral Development Waste	t	13,382	-	-	-	-	14,988	6,480	-	**34,850 t**
Lateral Development Ore	m	79	-	-	-	-	28	53	-	**160 m**
Lateral Development Ore	t	3,190	-	-	-	-	1,154	2,154	-	**6,498 t**
Lateral Development Ore Grade	g/t	4.0	-	-	-	-	2.6	3.2	-	**3.5 g/t**
Vertical Development Ore	m	-	-	-	-	-	-	-	-	**-**
Vertical Development Or	t	-	-	-	-	-	-	-	-	**-**
Vertical Development Ore Grade	g/t	-	-	-	-	-	-	-	-	**-**
Vertical Development Waste	m	-	-	-	-	-	-	-	-	**-**
Vertical Development Waste	t	-	-	-	-	-	-	-	-	**-**
Total Development Waste	t	13,382	-	-	-	-	14,988	6,480	-	**34,850 t**
Long Hole Drilling Metres	m	2,169	-	-	-	-	2,299	20,849	-	**25,317 m**
Long Hole Production Ore	t	7,590	-	-	-	-	7,523	73,495	-	**88,608 t**
Long Hole Production Ore Grade	g/t	5.0	-	-	-	-	3.3	4.2	-	**4.2 g/t**
Total Rockfill	t	-	-	-	-	-	-	-	-	
Total Cemented Rock Backfill	t	5,526	-	-	-	-	5,477	31,883	-	**42,886 t**
Total Wet Paste Backfill (1.82 t/m^3)	t	-	-	-	-	-	-	21,621	-	**21,621 t**
Total Backfill	t	5,526	-	-	-	-	5,477	53,504	-	**64,507 t**
Total Ore	t	10,780	-	-	-	-	8,677	75,649	-	**95,106 t**
Average Ore Grade	g/t	4.7	-	-	-	-	3.2	4.1	-	**4.12 g/t**
Mineable Gold Ounces	oz	1,625	-	-	-	-	899	10,085	-	**12,609 oz**
Recovered Gold Ounces	**oz**	**1,576**					**872**	**9,782**		**12,230 oz**

Table 16-15: LOM plan for the Martin Zone

Martin Zone										
	Units	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**Total**
Total Lateral Development	m	-	-	-	266	-	726	-	-	**992 m**
Lateral Development Waste	m	-	-	-	140	-	494	-	-	**634 m**
Lateral Development Waste	t	-	-	-	3,421	-	18,335	-	-	**21,756 t**
Lateral Development Ore	m	-	-	-	126	-	232	-	-	**358 m**
Lateral Development Ore	t	-	-	-	3,079	-	8,716	-	-	**11,795 t**
Lateral Development Ore Grade	g/t	-	-	-	4.5	-	3.6	-	-	**3.9 g/t**
Vertical Development Ore	m	-	-	-	141	-	-	-	-	**141 m**
Vertical Development Ore	t	-	-	-	3,678	-	-	-	-	**3,678 t**
Vertical Development Ore Grade	g/t	-	-	-	5.3	-	-	-	-	**5.3 g/t**
Vertical Development Waste	m	-	-	-	30	-	72	-	-	**102 m**
Vertical Development Waste	t	-	-	-	743	-	806	-	-	**1,550 t**
Total Development Waste	t	-	-	-	4,165	-	19,142	-	-	**23,307 t**
Long Hole Drilling Metres	m	-	-	-	7,571	-	3,150	11,211	-	**21,932 m**
Long Hole Production Ore	t	-	-	-	26,498	-	8,084	42,181	-	**76,763 t**
Long Hole Production Ore Grade	g/t	-	-	-	5.2	-	3.5	3.3	-	**4.0 g/t**
Total Rockfill	t	-	-	-	-	-	5,189	13,384	-	**18,573 t**
Total Cemented Rock Backfill	t	-	-	-	-	-	-	-	-	**-**
Total Wet Paste Backfill (1.82 t/m^3)	t	-	-	-	-	-	-	-	-	**-**
Total Backfill	t	-	-	-	-	-	5,189	13,384	-	**18,573 t**
Total Ore	t	-	-	-	33,256	-	16,800	42,181	-	**92,237 t**
Average Ore Grade	g/t	-	-	-	5.1	-	3.6	3.3	-	**4.0 g/t**
Mineable Gold Ounces	oz	-	-	-	5,492	-	1,934	4,449	-	**11,875 oz**
Recovered Gold Ounces	**oz**				**5,327**		**1,876**	**4,315**		**11,518 oz**

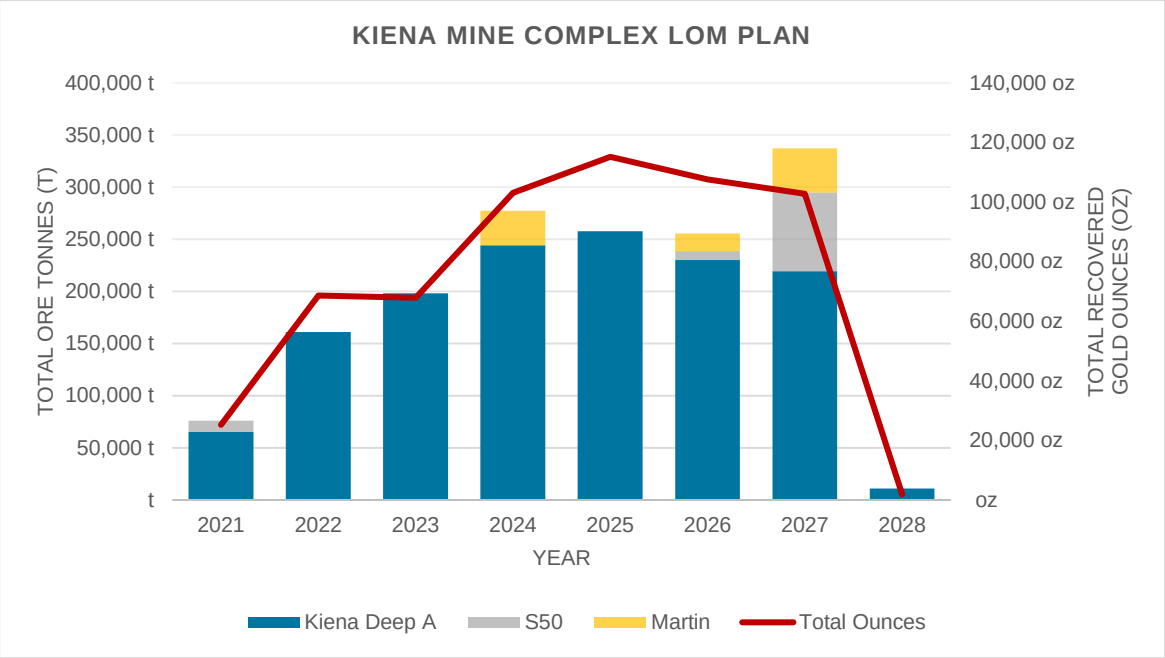


Figure 16-24: Ore tonnes by mining zone and recovered gold ounces

17. RECOVERY METHODS

The operation at the Kiena Mine processing plant began in September 1984. A conventional gold recovery process involving cyanidation and conventional carbon-in-pulp (CIP) was used. The plant was designed for a throughput of 1,250 tpd, and up to 2,000 tpd once crushing capacity was added. The restart, aiming at processing 1,000 tpd of mineralized material from the Kiena Deep deposit along with material from S50 and Martin zones will be done using the existing Kiena concentrator with minor modifications. The following sections provide a detailed description of the Kiena processing plant.

17.1 Process Description and Simplified Flowsheets

The principal process steps will include grinding, cyanide leaching, gold adsorption and desorption on activated carbon, electrolysis and cyanide destruction.

The crushing circuit includes a cone crusher and a screen. The aim of the crushing circuit was to allow a throughput increase from the original nameplate capacity. The envisioned throughput being under the original design, the crushing is not anticipated to be required. The grinding circuit consists of an open-circuit SAG mill followed by a ball mill in closed circuit with two stages of cyclones. The ground ore is then thickened and sent to the leaching circuit which then feeds the CIP circuit. A simplified process flowsheet of the crushing, grinding, leaching and CIP circuits are shown on Figure 17-1.

The loaded carbon is sent to the carbon stripping circuit, which allows desorption and electrowinning leading to melting and casting of doré bars as well as carbon acid washing and regeneration. A simplified process flowsheet of the carbon stripping, washing and regeneration as well as the refinery are shown on Figure 17-2.

The CIP tails will go through a cyanide destruction step before being thickened and sent to the tailings management facility (TMF) or paste backfill plant. The water needs for the process will be supplied by fresh water from the lake, process water and mine water. Internal recirculation of water will come from the thickeners overflow and additional water will be reclaimed from the TMF. The simplified process flowsheets of the tailings and water management circuits is shown on Figure 17-3. The secondary crushing circuit and hydraulic backfill plant are not used in the operations scenarios presented in the PFS.

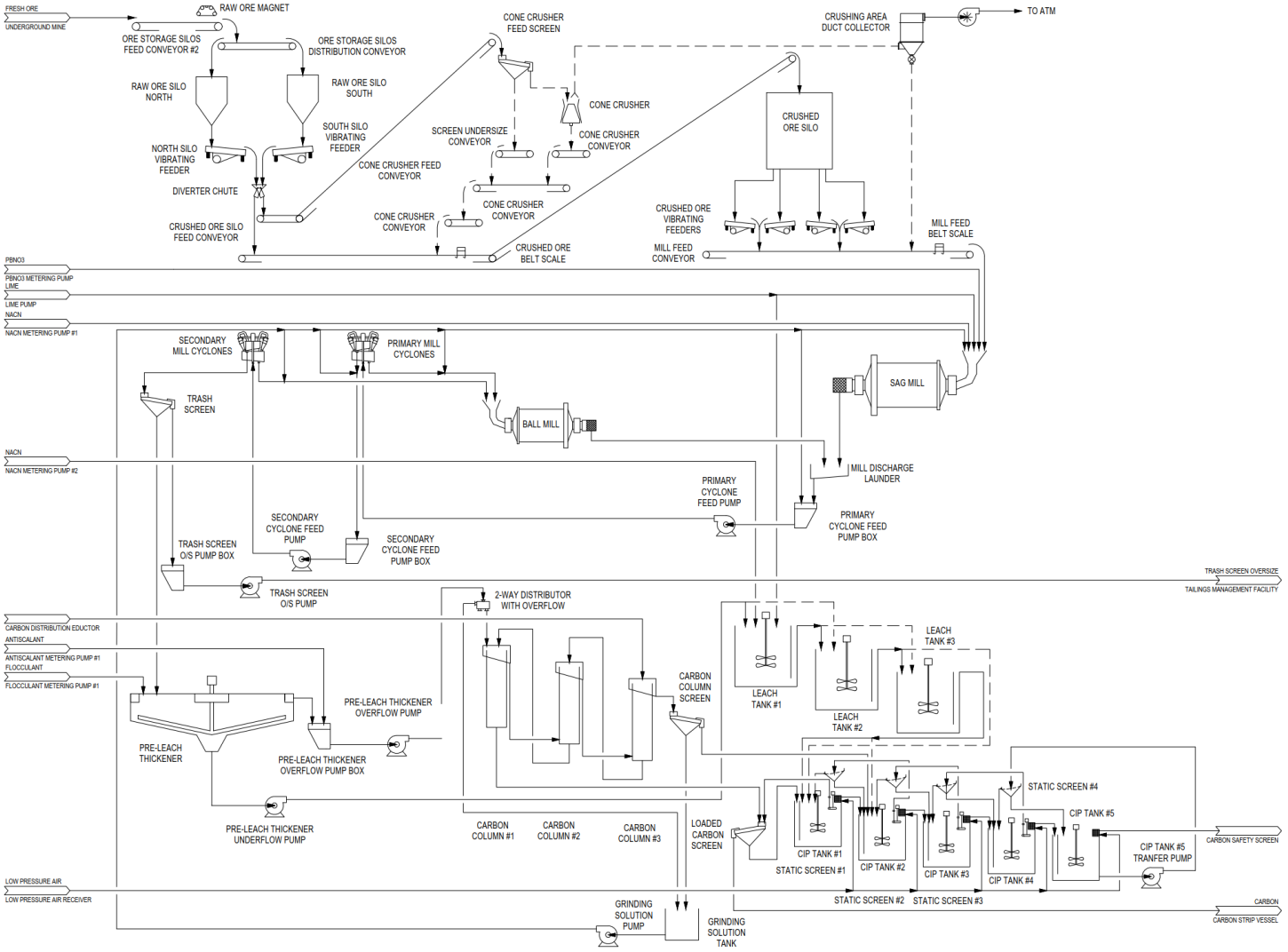


Figure 17-1: Crushing, grinding and leaching simplified process flow diagram

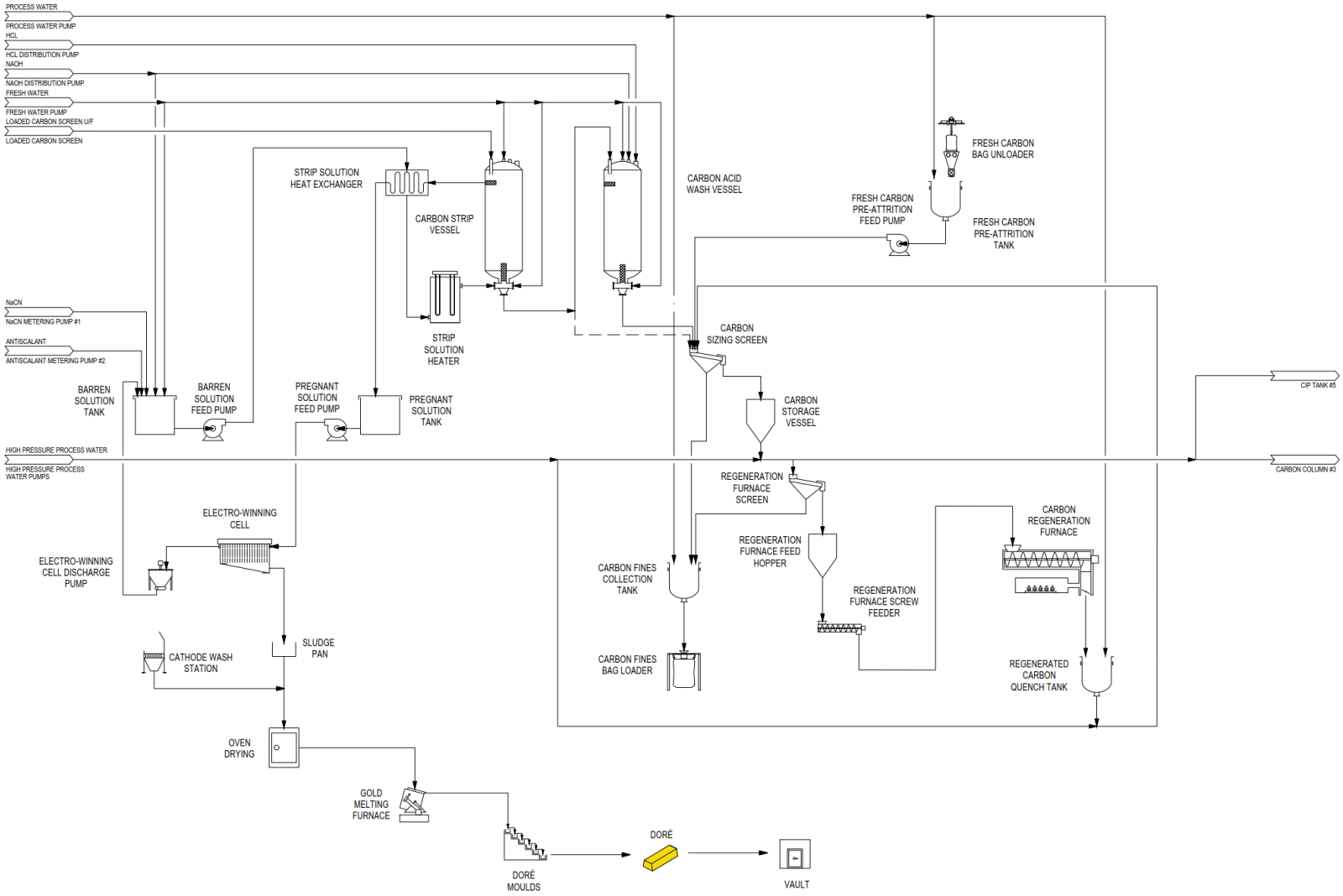


Figure 17-2: Carbon stripping and washing and refining simplified process flow diagram

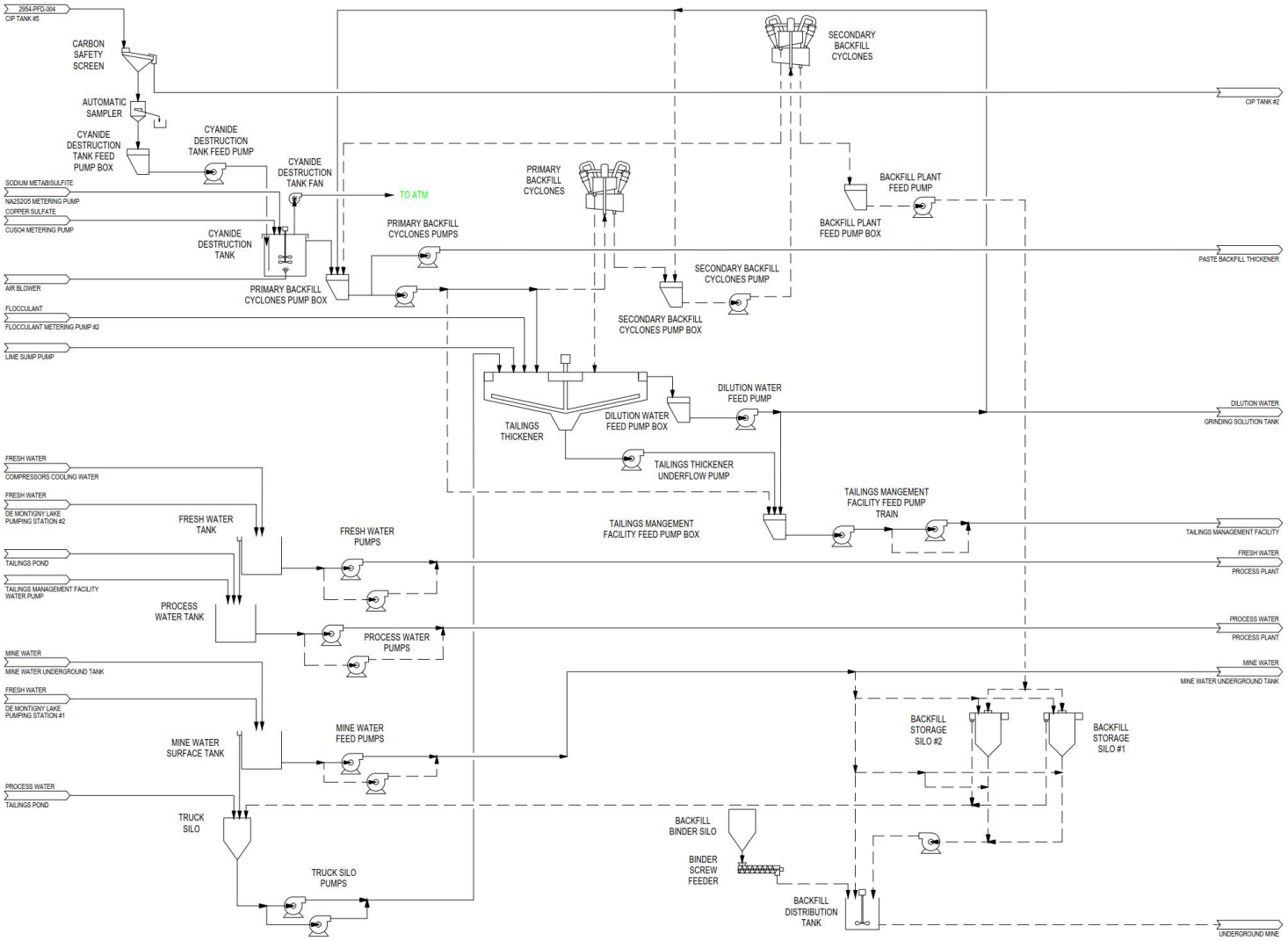


Figure 17-3: Tailings, water management simplified process flow diagram

17.2 Design Criteria

The design criteria were developed according to the metallurgical data obtained from the metallurgical testwork program on the Kiena Deep deposit and data from the bulk sample processing. The process design criteria and mass and water balance were used to evaluate the process performance. The Kiena Mine Complex was originally designed for a plant throughput of 1,250 tonnes per day and up to 2,000 tonnes per day with the secondary crushing circuit. The production target is set at 1,000 tonnes of dry ore per day to ensure adequate retention time in the leaching and cyanide destruction circuits. The yearly mine production and corresponding processing plant operating schedule is presented on Table 17-1. Mineralised material will come from Kiena deep (A, A1 and A2), S50 and Martin zones.

Table 17-1: Mine production and processing plant operating schedule

Year	Mine Production (tpa)	Processing Plant Operating Schedule (d/w)	Processing Plant Utilization (%)
2021	79,716	4	21.8
2022	161,938	4	44.4
2023	197,386	4	54.1
2024	276,505	5	75.5
2025	258,406	5	70.8
2026	250,420	5	68.6
2027	329,715	7	90.3
2028	10,965	4	3.0

During the first three years of operation, the process plant will operate on a schedule of four days on and three days off (4/3). That schedule considers 72 hours per week of effective operation. From 2024 to 2026, the schedule will change to five days on and two days off. The operating schedule will then change to seven-day per week operation for a one-year period and finally, in 2028, it will go back to the 4/3 schedule. During the down times, the water and tailings pumping systems will remain running to avoid freezing or sanding out of pipelines. The processing tailings will be pumped into the North Tailings Pond and the processing plant flowsheet will be modified to reinclude cyanide destruction and the capacity to send approximately 15% of the tailings to be used for paste backfill on average over the life of the mine (see Chapter 16). The main process design criteria are presented in Table 17-2.

Table 17-2: Main process design criteria

Parameter	Unit	Value
Plant - General		
Ore Moisture	% w/w	4.0
Ore Specific Gravity	t/m^3	2.80
Average Feed Au Grade	g/t	11.73
Targeted Au Recovery - Kiena Deep (A, A1, A2)	%	98.5
Targeted Au Recovery - S50 and Martin	%	97.0
Processing Plant Nominal Feed Rate (Dry)	tph	41.7
Grinding		
Ball Mill Work Index - Kiena Deep (A, A1, A2)	kWh/t	10.7
A x b - Minimum - Kiena Deep (A, A1, A2)	-	51.9
A x b - Maximum - Kiena Deep (A, A1, A2)	-	64.0
Abrasion Index (Ai) - Average - Kiena Deep (A, A1, A2)	g	0.196
Specific Power - Average - Kiena Deep (A, A1, A2)	kWh/t	22.2
SAG Mill Consumed Power	kW	550
Recirculating Load - Nominal	%	250
Ball Mill Consumed Power	kW	375
Grinding Circuit Product Size (P_{80}) - Maximum	µm	80
Leaching and Gold Recovery		
Leaching Circuit Retention Time	h	42.8
Carbon Concentration in CIP Tank	g/L	20
CIP Circuit Retention Time	h	5.5
Carbon Strip Vessel Capacity	t	3.0
Carbon Strip Vessel Bed Volume	m^3	6.0
Carbon Stripping Frequency	#/d	0.5
Carbon Stripping Solution Flowrate (bed volume)	BV/h	1.0
Cyanide Destruction		
Cyanide Destruction CN_{WAD} Target	mg/L	1
SO_2 Dosage	g SO_2/g CN_{WAD}	8.0
Copper Sulfate Dosage	g Cu/t Ore	50.0
Cyanide Destruction Tank Retention Time	h	0.86

17.3 Process Equipment Description

17.3.1 Ore Storage and Crushing Circuit

The crushing circuit is fed by 150 mm rock coming from the Birdsboro Buchanan jaw-crusher located underground. The ore is hoisted to surface and stored in two raw ore silos, each capable of holding 600 t. Vibrating feeders under the silos reclaim the ore and feed it to a diverter chute that either feeds the secondary crushing circuit or the crushed ore silo conveyor that leads to the crushed ore silo that can hold 1,800 t. The secondary crushing circuit is composed of a feed conveyor, a screen, a standard cone crusher and product conveyors. With the current production plan, the secondary crusher is not required and will be bypassed.

17.3.2 Grinding Circuit

The mineralized material is ground in a semi-autogenous grinding (SAG) mill operating in an open circuit, followed by a ball mill operating in closed circuit with two stages of classification cyclones. Cyanide and lead nitrate are added at the SAG mill along with milk of lime. The underflow of the primary and secondary cyclones is redirected to the ball mill for further grinding. The cyclone overflow is then pumped to a vibrating trash screen. The screen undersize is directed to the pre-leach thickener and the oversize is pumped with water to the tailings pump box. The pulp from the thickener underflow is transferred to the leaching circuit while the thickener overflow is partially pumped to a series of three carbon columns to adsorb the gold in solution. The last carbon column in the series overflows onto a safety carbon screen where the oversize carbon that would mistakenly escape from the column is sent to the second CIP tank. Once every two days, the loaded carbon of the first column is sent to the loaded carbon screen, the carbon from the downstream columns is transferred to their respective upstream column and unloaded carbon is added in the last column. The third column discharge screen undersize reports to the grinding solution tank along with the remainder of the thickener overflow. The grinding solution is recycled back to the grinding circuit as grinding solution.

The main characteristics of the major pieces of equipment of the grinding circuit are listed in Table 17-3.

Table 17-3: Major equipment characteristics of the grinding circuit

Equipment	Qty	Characteristics
SAG Mill	1	3.35 m diameter x 5.48 m long, 1,000 hp motor
Ball Mill	1	3.20 m diameter x 3.66 m long, 900 hp motor
Primary Cyclones	5	380 mm diameter
Secondary Cyclones	2	508 mm diameter
Trash Screen	1	Vibrating, 2.4 m x 1.2 m 35 mesh opening
Pre-leach Thickener	1	19.8 m diameter
Carbon Columns	3	1.2 m diameter x 3.7 m high, 0.8 t carbon capacity
Mill Solution Tank	1	4.5 m diameter x 5 m high, 79.5 m^3 volume

Given that the envisioned throughput will be lower than the original design, the possibility of bypassing the ball mill, thus going to a single-stage SAG mill circuit, has been evaluated. Comminution testwork results (see Section 13.2.3) indicate that the ore is soft and that the power requirements, to achieve the targeted P_{80} of 80 µm falls within the capacity of the SAG mill. It was also determined that single mill operation would allow the throughput to be increased up to 44 tph, which leads to believe that the power utilization of both mills would be quite low if operating the SAG mill and ball mill circuit at the nominal throughput of 41.7 tph. The processing of the bulk sample in November 2020 allowed to obtain data of the full-scale operation. From this data, it was observed that at a feed rate of around 950 tpd and a P_{80} of around 67 µm, the combined SAG and ball mill power draw was in the 925 to 950 kW range, or 20 to 24 kWh/t, which indicates a higher power consumption than what could be expected from the comminution testwork results. As a result, it is considered that for the PFS both mills will be operated but that the opportunity of using the SAG mill only should be considered in the future.

The grinding media consumption for the SAG and ball mill was estimated using historical consumption and data from the bulk sample processing. The average media consumption for both grinding applications is presented in Table 17-4.

Table 17-4: Grinding media consumption

Area	Size (mm)	Consumption (kg/t)
SAG Mill	127	0.45
Ball Mill	51	0.95

17.3.3 Leaching, CIP and Gold Recovery Circuits

Thickener underflow is pumped to a series of three leach tanks. Cyanide, milk of lime and lead nitrate are added to the leach feed. Air is also added through the tanks' agitator shafts. The leaching tanks are operated in series for further gold dissolution. Leach tank #1 overflow feeds the next tank (leach tank #2). Leach tank #3 overflow feeds the first CIP tank.

The CIP circuit consists of five CIP tanks in series. The tanks are equipped with static screens that prevent carbon from exiting the tank. The slurry is fed into CIP tank #1 whereas the unloaded carbon is fed into CIP tank #5. The discharge of the CIP tanks is directed to a safety screen to recover any carbon particles escaping from the tanks.

The CIP tank #1 pulp is sent to the loaded carbon screen on a daily basis in order to recover loaded carbon contained in the pulp. The loaded carbon screen oversize is transferred to the carbon strip vessel while the passing pulp is returned to CIP tank #1. Carbon from CIP tank #2 is then transferred to CIP tank #1 by pumping slurry onto a static screen that directs the carbon to CIP tank #1 while the slurry is returning to CIP tank #2. The same principle is then used for the other CIP tanks. When the carbon from the first carbon column is sent to the screen, the carbon transfer water reports to the screen undersize and then to CIP tank #1.

The loaded carbon falls from the screen oversize to the carbon strip vessel, where the gold desorption takes place using the Zadra process. The gold adsorbed on the carbon contained in the carbon strip vessel will be desorbed by the strip solution that comes from the barren solution tank and then goes to the pregnant solution tank.

Once desorption is complete, the carbon is transferred to the carbon acid wash vessel where it is washed with hydrochloric acid. After acid neutralization with caustic solution, it is forwarded to the carbon sizing screen and partly fed to the regeneration furnace. Finally, the regenerated carbon falls into the quench tank and is screened again on the carbon sizing screen and reused in the process. The carbon going to the undersize of the screen is too fine to be reused in the process and is collected into a 1 m^3 bag to be sent to a gold-recycling plant.

The electrowinning cell is fed with the pregnant solution and gold is recovered by plating on stainless steel wool cathodes. The resulting gold-rich sludge is dried in the drying oven and, after mixing with the appropriate flux, it is sent to the gold melting furnace. The smelted product is poured in the doré moulds where metallic gold sinks in the moulds and slag overflows. The doré bars obtained from casting are stored in the vault before being sent to gold refiners. The slag is recovered and is sent to a gold-recycling plant.

The main characteristics of the major pieces of equipment of the leaching, CIP and gold recovery circuits are listed in Table 17-5.

Table 17-5: Major equipment characteristics of the leaching, CIP and gold recovery circuits

Equipment	Qty	Characteristics
Leaching Tanks	3	10 m diameter x 10.6 high, 809 m^3 effective vol. per tank. 60 hp agitators
Low Pressure Compressor	2	150 hp motor, 2300 Nm^3/h at 275 kPa capacity each
CIP Tanks	5	4.3 m diameter x 4.6 m high, 62.4 m^3 volume, 7.5 hp agitators
Safety Screen	1	Vibrating, 2.4 m x 1.2 m, 28 mesh opening
Elution Vessel	1	3 t capacity, 6 m^3 volume
Barren Solution Tank	1	4.0 m diameter x 4.5 m high, 56.5 m^3 volume
Pregnant Solution Tank	1	4.0 m diameter x 4.5 m high, 56.5 m^3 volume
Strip Solution Heater	1	360 kW electric elements in tube
Acid Wash Vessel	1	1.4 m diameter x 1.4 m high, 2.2 m^3 volume
Carbon Regeneration Furnace	1	Vertical natural gas furnace
Loaded Carbon Screen	1	Vibrating, 2.4 m x 1.2 m, 28 mesh opening
Carbon Sizing Screen	2	Vibrating, 2.4 m x 1.2 m, 20 mesh opening
Electrowinning Cell	1	2.6 m x 0.95 m x 1.25 m, 600 mm x 600mm anodes and cathodes
Gold Melting Furnace	1	Induction type furnace 30 kg doré production per batch

17.3.4 Cyanide Destruction Circuit and Tailings Treatment

The existing cyanide destruction circuit is to be refurbished and modified. The cyanide destruction circuit is fed by the CIL tails that are adjusted to 45% (w/w) solids by process water addition. The cyanide destruction reaction occurs in one tank and uses a combination of sodium metabisulfite (SMBS), copper sulfate and air to achieve a target of less than 1 ppm CN_{WAD}. Lime will be used to control the pH.

The tailings thickener increases the tails pulp density with the addition of flocculant before it is transferred to the TMF feed pump box by the tailings thickener underflow pump. The TMF feed pump train is used to transfer the tails to the TMF. The tailings thickener overflow is sent to the dilution water feed pump box. The water is pumped through a distribution system that takes a portion to the grinding solution tank, a portion to the thickener feed pump box to decrease density, and the rest goes to the TMF feed pump box. The backfill cyclones are not required and will not be used.

When the paste backfill plant is running (see Chapter 16) or when it's stock tank needs to be filled, the cyanide destruction tails is pumped to the paste backfill plant from the primary backfill cyclones feed pump box. The tailings thickener is then put in recirculation and reclaim water is added to the TMF feed pump box to keep the tailings line running. When the paste backfill plant is down or does not need feed, all cyanide destruction tails are pumped directly to the tailings thickener.

The main characteristics of the major pieces of equipment of the cyanide destruction and tailings circuits are listed in Table 17-6.

Table 17-6: Major equipment characteristics of the cyanide destruction and tailings circuits

Equipment	Qty	Characteristics
Cyanide Destruction Tank	1	4.0 m diameter x 5.0 m high, 56.5 m^3 volume
Blower	1	200 hp motor, 3370 Nm^3/h capacity at 62 kPa
Tailings Thickener	1	19.8 m diameter

17.3.5 Water Management

The water management circuit is composed of four tanks: the fresh water tank, the process water tank, the mine water surface tank and the truck silo (used for water storage). The fresh water tank is fed with fresh water from the De Montigny Lake pumping station #2. Fresh water is pumped to the process and the excess in the tank overflows is pumped into the process water tank, which is also fed with reclaim water from the north tailings cell. The mine water surface tank is fed by the mine water underground tank and the De Montigny Lake pumping station #1. The mine water feed pumps feed the mine water underground tank. The mine water surface tank overflows into the truck silo. The truck silo water is pumped to the tailings thickener by the truck silo pumps.

In order to minimize the amount of fresh water used, process water will be used for most applications throughout the plant. The process water tank will be primarily supplied by reclaim water from the TMF and fresh water overflow kept to a minimum. Fresh water will be used in the carbon stripping circuit, for lead nitrate, flocculant, copper sulfate and SMBS preparation and for compressors cooling. It is expected that approximately 85 m^3 of fresh water per day will be used in the process.

17.3.6 Reagents Systems

The expected consumptions of all reagents unless specified otherwise is based on the metallurgical testwork results as described in Chapter 13. Sodium cyanide solution is delivered by tanker truck and transferred into the NaCN storage tank. From there, it is pumped by the cyanide transfer pump that allows sending the solution into the mixing cyanide tank. The cyanide mixing tank is used to reduce the solution strength that is transferred by opening a valve to the cyanide distribution tank. Dosing pumps are used to feed cyanide into the SAG mill, leach tank #1 and the barren solution tank.

Flocculant is wetted in the eductor and fed into the mixing tank. When the flocculant preparation is complete, the content of the mixing tank is transferred into the distribution tank. Metering pumps are used to feed flocculant to the thickeners with further water addition in the in-line mixers.

Lead nitrate is mixed with water in the mixing tank. From there, it is pumped into the distribution tank. Metering pumps are used to feed lead nitrate to the SAG mill and leach tank #1.

Quicklime is delivered by truck and transferred into the lime silo. From there, it is mixed with water in the lime slaker. A lime pump distributes the milk of lime through a loop recirculating back into the distribution tank.

Copper sulfate is mixed with water in the mixing tank. From there, it is pumped into the distribution tank. A metering pump is used to feed copper sulfate to the cyanide destruction tank. New equipment will be required for this system.

SMBS is mixed with water in the mixing tank. From there, it is pumped into the distribution tank. A metering pump is used to feed SMBS to the cyanide destruction tank. New equipment will be required for this system.

Hydrochloric acid is delivered by tanker truck and transferred into the HCl Storage Tank. From there, it is pumped into the dilution tank where it is mixed with water. A pump transfers the diluted acid into the carbon acid wash vessel. The expected consumption is based on historical consumption.

Caustic Soda is delivered by a tanker as a 50% NaOH solution. The solution is stored in a distribution tank. A pump transfers the caustic solution into the cyanide mixing tank, the barren solution tank and the carbon acid wash vessel. The expected consumption is based on historical consumption.

Activated carbon will be fed into the fresh carbon pre-attrition tank. The existing system will be used, and the expected consumption is based on historical consumption.

The delivery form and expected reagents consumption is presented in Table 17-7.

Table 17-7: Reagents delivery form and consumption

Reagent	Delivery Form	Consumption kg/t
Sodium Cyanide	Bulk – Liquid	0.86
Flocculent – Pre-Leach Thickener	800 kg bags	0.025
Flocculent – Tailings Thickener	800 kg bags	0.025
Lead Nitrate	1,000 kg bags	0.075
Quick Lime	Bulk – Solid	2.2
Copper Sulfate ($CuSO_4*5H_2O$)	1,000 kg bags	0.196
Sodium Metabisulfite	1,000 kg bags	2.90
Hydrochloric Acid	Bulk – Liquid 32%	0.15
Caustic	Bulk – Liquid 50%	0.10
Activated Carbon	500 kg bags	0.01

17.4 Energy Requirements

The average electrical energy requirements for the process plant were determined from Kiena's equipment list and bulk sample processing. Each motorized item of equipment was assigned utilization and load factors to derive the data presented in Table 17-8. Given that the mine throughput will not be sufficient to maintain continuous operation at 1,000 tpd, processing plant shutdowns will be planned regularly. Critical equipment will remain in operation during the shutdowns such as the water and tailings systems.

Table 17-8: Process power consumption

Area	Installed Power (kW)	Daily Power Consumption (kWh/d)	
		In Operation	In Shutdown
Ore Storage and Grinding	1,860	27,600	0
Leaching, CIP and Recovery	817	7,400	3,500
Cyanide Destruction and Tailings	356	5,700	2,700
Reagents, Water and Compressed Air	757	13,100	7,500
Total	3,790	53,800	13,700

17.5 Risk and Opportunity

The following opportunities for the processing plant have been identified:

- Comminution testwork results suggest that it could be possible to operate the SAG mill as a single stage. This would improve the grinding efficiency resulting lower energy costs and would also reduce the grinding media cost;
- As there is no need for tailings filtration with the current life of mine, the tailings circuit could be rearranged in order to put the tailings thickener before the cyanide destruction circuit. This would allow to recover cyanide bearing water to be reused in the process resulting in lower cyanide consumptions and lower cyanide destruction reagents consumption (SMBS, copper sulfate, lime).

The project presents the following risks:

- The presence of nickel in the ore has led to incomplete cyanide destruction during the testwork. It is possible that reagents addition needs to be increased in the cyanide destruction circuit;
- Martin Zone has not been subjected to testwork nor been processed in the past. It is possible that the recovery associated with this zone is lower than expected or that it contains certain minerals or has certain characteristics that are detrimental to the process.

17.6 Conclusion

A complete review of the equipment has been made to verify the capacity and condition with respect to the expected mill throughput, reliability, operation schedule and metallurgical performances. From this review, it has been found that the existing equipment allows reaching the throughput and performance objectives. This was confirmed from the processing of the bulk sample in November and December 2020. The following aspects must also be considered and were taken into account in the cost and performance evaluation:

- Most water needs will be met by recycling process water. Process plant fresh water requirements will be around 85 m^3/day;
- The process plant operating schedule will need to vary over time to operate in the most efficient manner to match with the mine production schedule;
- The existing surface crushing circuit operation is not required;
- The grinding circuit's available power is higher than required and its efficiency is lower than what could be expected from testwork, but it allows reaching the targeted size reduction. The power consumed during the bulk sample processing was used for estimating the grinding power cost;
- The current cyanide destruction reactor is adequate but reagent addition systems (SMBS and copper sulfate) will need to be installed;
- The cyanide destruction reagent addition will have to be higher than what was used during the testwork to reach the objective of reducing the CN_{WAD} concentration below 1 ppm.

18. PROJECT INFRASTRUCTURE

18.1 Introduction

The Kiena Mine Complex produced gold up until June 2013 and has been under care & maintenance since then. The complex was suspended with the belief that operations would resume eventually. Wesdome has maintained a site presence since 2013 and development was and is still being done providing access to underground geological drilling platforms while using the existing infrastructure. It is therefore expected that most of the surface infrastructure can be reused where required to meet any needs similar to those during development. The following sections outline the changes required for the Project.

18.2 Site Description

The Kiena Mine Complex is an inactive underground gold mine located about 9 km west of Val-d'Or, Quebec. The site was operating intermittently between 1984 and 2013, and the mining operations ceased on June 30, 2013. The site is located next to Highway 117 and is accessible via the Kienawisik road. The nearest airport with scheduled flights from Montreal is Val-d'Or Airport.

The site layout including existing tailings storage facility (TSF), polishing pond, waste rock stockpile and mill site is shown on Figure 18-1. The TSF is flanked by Lac de Montigny on the north and east sides, Petit Lac on the west side, and CN railway tracks and Hydro-Quebec lines on the south side. The TSF is contained by perimeter dykes and comprises two cells: North Cell and South Cell. The Mill site is located at Parker Island with access via a 900-m long causeway. The island contains the access to the underground mine, the mill, and associated infrastructure and buildings.

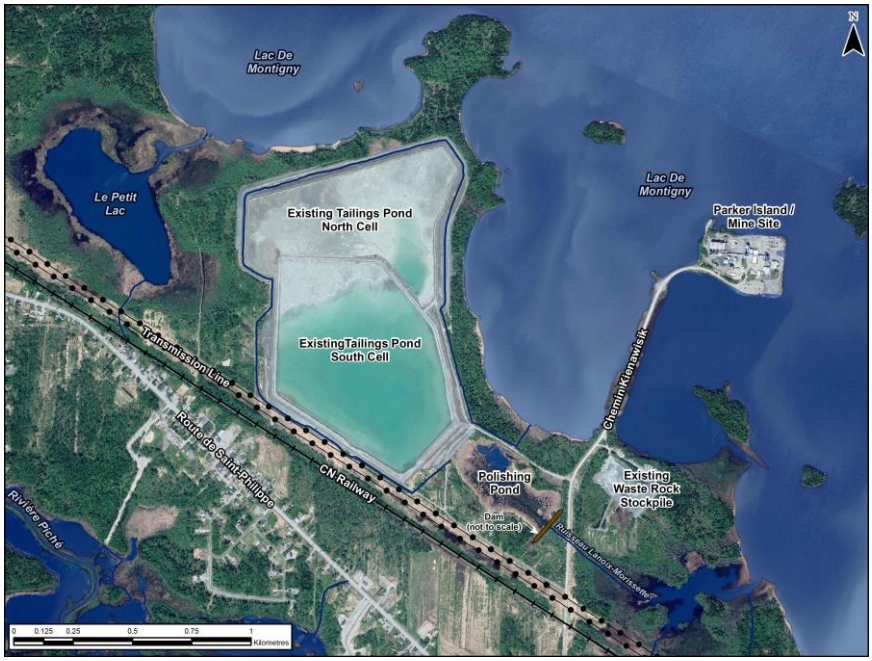


Figure 18-1: Kiena Mine Mite - Existing site layout

Mining and milling operations are proposed to start in July 2021 with life of mine (LOM) 2021-2028 and total tailings tonnage of 1.57 Mt. Approximately 0.47 Mt tailings will be used for backfill operation and the remaining 1.10 Mt tailings will be required to be stored at the existing TSF or at a new tailings dry stack facility. It is expected that the existing TSF can be upgraded and raised to store the LOM tailings (1.10 Mt) to TSF. As such, tailings dry stack is not required to manage the LOM tailings.

18.3 Plot Plans

Refer to Appendix A for the overall plot plan, the mine site plot plan, and the demolition plot plan.

18.4 General Access and On-Site Roads

18.4.1 Site Access

The Kiena Mine Complex is located next to Highway 117, 10 km west of Val-d'Or, on De Montigny Lake. It can be accessed by trucks via the Kienawisik Road by turning north from Hwy 117.

The nearest airport with scheduled flights from Montreal is Val-d'Or Airport.

18.4.2 On-Site Roads

On site, an existing road is present to provide access throughout the site, as well as a parking lot on the island.

The Project will require the following modifications (refer to the mining site plot plan in Appendix A):

1. The stretch of road that links the end of the main road (at the end of the causeway) to the secondary road will be rehabilitated to restore direct access to the mill and to the sand stockpile. The existing gate will be automated and equipped with a CCTV camera to allow for remote access control and operation by the security guard from the guard house.
2. The access to the portion of the secondary road located between the new paste plant and the sand stockpile will be restricted with new manually operated gates. This will prevent traffic in the area when the wheel loader is operating.

18.5 Civil Works

The main changes to the surface infrastructure include the addition of a paste plant, effluent treatment plant, underground mine transformer, and the replacement of the mine ventilation system. The following civil works will be required as part of these modifications:

- Demolition/removal of the existing mine ventilation system (including the heat exchanger system), site preparation and foundation for the new ventilation system;
- Demolition of the existing hydraulic fill plant, site preparation and foundation for the new paste plant;
- Site preparation and concrete slab for the effluent treatment units;
- Site preparation and concrete slab for the new mine and effluent treatment plant transformers.

18.6 Power Supply and Distribution

18.6.1 Power Demand Estimates

The mine consumed about 7.8 MVA at peak demand when it was last operating in 2013. The peak loads for the Project were estimated by taking the peak load from 2013 as a baseline, and by subtracting the loads to be removed and adding the new loads. The loads that will be eliminated include the hydraulic backfill plant, the old mobile equipment loads, and the old underground ventilation loads. The new loads on the surface are the mine intake and exhaust ventilation fans (accounted for as the difference between the new ventilation plant requirements and the old), the effluent treatment plant, and the paste plant. The new underground loads are the mobile equipment (including three chargers for battery electric trucks), the new dewatering system, new underground ventilation, and underground services.

The load baseline, additions and removals for the Kiena site are presented in Table 18-1. The resulting peak power yearly demand for the Kiena site is shown in Table 18-2.

Table 18-1: Load baseline, additions and removals

Description	Starting Year	Load (kVA)
Baseline (from 2013)	2013	7,778
Existing hydraulic fill plant	2021	-105
Existing U/G mobile loads	2021	-1,053
Existing U/G ventilation	2021	-684
New ventilation system on surface – additional load vs. existing	2022	1217
New effluent treatment plant	2022	947
New paste plant	2022	320
U/G development (jumbo and bolter)	2021	496
U/G production (top hammer drill)	2021	331
U/G BEV chargers, three (3) units	2023	525
U/G diamond drilling	2021	620
U/G dewatering – Kiena Deep / S50 – Phase 1	2021	490
U/G dewatering – Kiena Deep / S50 – Phase 2	2025	490
U/G dewatering – Martin	2024	74
U/G ventilation – Kiena Deep / S50	2021	903
U/G ventilation – Martin	2024	149
U/G Lighting / Refuge / SatStat	2021	190

Table 18-2: Peak power demand by year

Year	2021	2022	2023	2024	2025	2026	2027	2028
Peak Power (kVA)	8,964	11,448	11,973	12,196	12,686	12,686	12,686	12,686

18.6.2 Electrical Power Source

The electrical power to the Kiena Mine Complex is supplied by Hydro-Quebec via a 25 kV line which connects outside of the site gate, on Kienawisik Road, at the mine's main substation. This substation is composed of a main 25 kV switch connected upstream to the Hydro-Quebec line and downstream to the mine's main 25 kV breaker. On the Kiena property, power is then conducted through an overhead line downstream of the main breaker to several substations where it is stepped down to supply power to the various consumers on the surface and underground.

Hydro-Quebec has informed the Project that the current distribution network can only supply up to 10 MVA to the Kiena mine site, without major modification on Hydro-Quebec's network. According to the load estimates (see Table 18-2), the peak load will exceed this limit in 2022 with the addition of the new mine ventilation system and paste plant. To meet the power requirements above 10 MVA, Hydro-Quebec indicated that a new dedicated 25 kV line with an estimated length of 13 km needs to be added to their distribution network.

18.6.3 Site Power Distribution

The existing power distribution system feeding the existing loads remains unchanged. However, modifications and additions are required to feed the new loads. The following list outlines the tie-in strategy for the new loads:

1. The new underground loads (mobile equipment, ventilation, dewatering, and services) will be fed by a new transformer (7.5 MVA) located on the surface and connected to the main 25 kV site distribution power line – refer to Chapter 16 for details.
2. The mine ventilation system upgrade on the surface (new intake and exhaust fans) will be fed by the existing transformer T11 (T11 will have the required spare capacity once the new mine transformer is commissioned).
3. The effluent treatment plant will be fed by a new transformer (1.5 MVA) connected to the main 25 kV site distribution power line.
4. The paste plant will be fed by the existing transformer T7.

A single line diagram showing the modifications to the power distribution system on surface can be found in Appendix D.

18.6.4 Emergency Power Supply

The Kiena mine site comprises an existing 1,000 kW standby diesel generator. In the event of a power outage, it will be configured to supply critical loads in the concentrator building, the paste plant, and the administrative building. In case of an emergency, standby power can also be switched over to the hoist. Since the effluent treatment plant is located off the island in an area with no emergency power, an emergency generator (250 kW, 600V) will be added at the effluent treatment plant site.

18.7 Communications

Site communications infrastructure includes:

- Internet access (both satellite and hardwired from a local provider);
- Cell phone service;
- A fibre-optic control network using the Modbus Plus technology;
- An emergency communication line for the mine;
- Three separate CCTV networks, one for the site security, one for the gold room, and one for process monitoring.

18.8 Buildings

The existing buildings include the process plant, administrative building, warehouses and maintenance shops, the dry house / engineering office building, the assay laboratory, security building, hoist room, headframe core facility, and the reagents and hazardous material storage facility.

The Project requires the addition of the paste plant building as described in Chapter 16. No new administration or service buildings are envisioned for the future operation of the site; however, the Project will require the following modifications:

- The reagents and hazardous material storage facility will be relocated approximately 50 m further to the west from its current location, between the secondary road and the mine site fence. This will increase the available footprint for the sand stockpile;
- A trailer fitted out as a mine rescue station will be permanently located behind the administrative building.

18.9 Accommodation

As the Kiena Mine Complex is located within the limits of the town of Val-d'Or, there are no requirements for camps or lodging.

18.10 Temporary Construction Infrastructure

The following laydown areas will be built to provide storage space for construction materials and trailers:

1. The primary laydown area, with an estimated surface area of 2,500 m^2, located off the island, west of the existing clean rock stockpile. This area will also act as the contractor management area.
2. The secondary laydown area, with an estimated surface area of 1,000 m^2, located on the island north of the warehouses.

18.11 Tailings Storage Facility

18.11.1 Existing Tailings Storage Facility

As noted in Section 18.2, the existing TSF is contained by perimeter dykes and comprises two cells: North Cell and South Cell (see Figure 18-2). The starter perimeter dykes comprised zoned embankments with wide central low permeability silty clayey core and upstream and downstream rockfill shells. Geotextile was placed on the downstream side of the clayey core to act as a separator/filter between the core and the downstream rockfill shell. The first two phases of the dyke raises were performed by downstream construction method. Some of the later dyke raise(s) were constructed by upstream method on the tailings beach and comprised of silty clay / tailings core and waste rock shells with geotextile fabric on the downstream side of the sore and located.

Subsurface soil conditions at the TSF site are characterized by the presence of varved grey silty clay stratum underlain by a clayey silt stratum and then till. A brownish crust is typical in the upper portion of the varved grey silty clay. The combined thickness of varved silty clay and clayey silt strata generally ranges between 5 m and 15 m at the TSF. The tailings had been deposited into the TSF by slurry spigoting. Therefore, the tailings within the TSF (which act as the foundation of some of the dyke raises) is expected to be in loose to very loose condition and potentially susceptible to both static and seismic liquefaction.

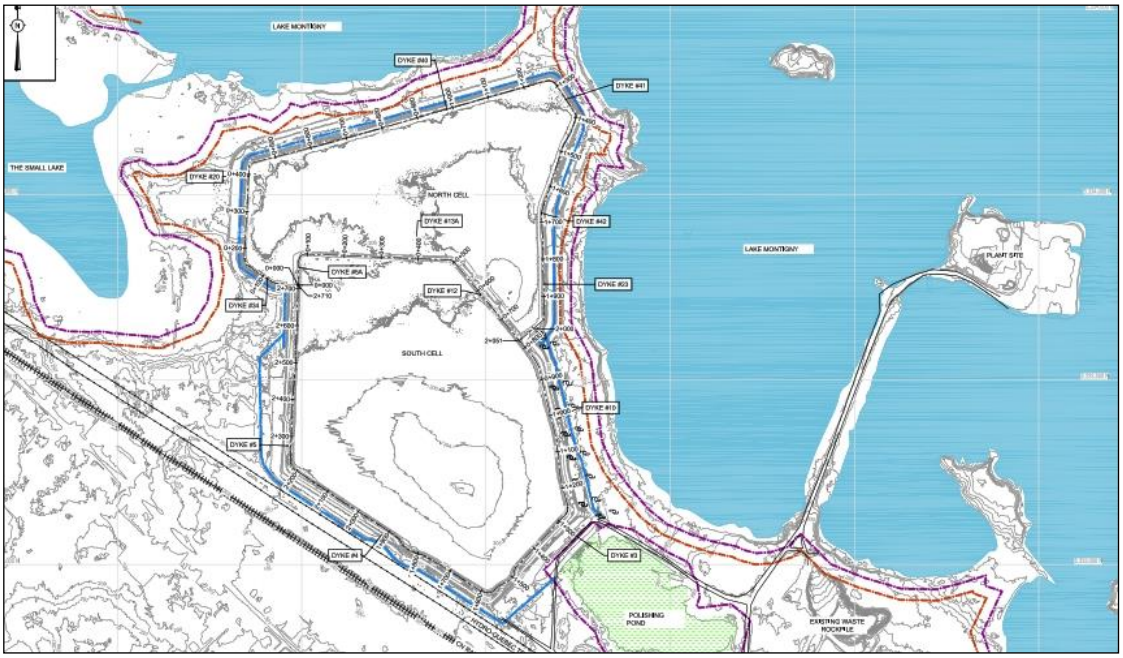


Figure 18-2: Existing tailings storage facility

18.11.2 Climate and Hydrology

The closest weather station to the project site is Environment and Climate Change Canada's station at Val-d'Or Airport, located 13.3 km southeast from the project site. The most recent available Canadian Climate Normals data for this station (1971–2000) indicate that the average annual precipitation is 914 mm, with approximately 635.2 mm (69%) falling as rain and the remainder falling as snow. The majority of rainfall occurs in the summer and fall months, with peak rainfall occurring during September (99.8 mm). The majority of snow falls from October to April, with peak snowfall in December (61 cm). The daily average temperature for January is -17.2°C and the daily average temperature in July is 17.2°C. Temperatures fall below freezing from November through March.

Val-d'Or annual 24-hour precipitation maximums from 1961 to 2017, and snowfall data from 1951 to 2018, were used to determine storm depths for TSF design. Quebec's Ouranos Consortium on Regional Climatology and Adaptation to Climate Change (Ouranos) provides climate change projections for Quebec. Precipitation projections are available for total annual and seasonal (spring, summer, fall, winter) precipitation, as well as 5-day precipitation maximums, on a regional basis for the 2041 – 2070 time period (beyond the project LOM). Ouranos does not provide climate change projections for 30-day snowmelt, or for 24-hour (1-day) rainfalls; however, the latter value is available through Climate Atlas of Canada (Climate Atlas). For a high carbon (RCP 8.5) climate change scenario, Climate Atlas' projected mean 1-day precipitation maximum for Val-d'Or for 2021-2050 is 39 mm. This is lower than the mean 24-hour precipitation maximum used for TSF design (43.3 mm). As noted, data on the Environmental Design Flood (30-day, one in 100-year snowmelt plus 24-hour, one in 2,000-year rainfall) are not available from the climate change sources. However, in terms of the data available (mean annual maximum 1-day rainfall), the project climate data used for TSF design are relatively recent and yield a higher value than Climate Atlas projection. For this reason, the water management plan and infrastructure described in this study are expected to be adequate in terms of dealing with the effects of climate change within the duration of the project design life (2021-2028). However, it should be noted that climate change impacts should be considered as part of closure design in future project work.

18.11.3 Seismic Hazard

The Kiena Mine Complex is located at about latitude 48°08' N and longitude 77°54' W within the seismic source zone IRB (Iapetan Rift Background source zone) incorporated in the National Building Code of Canada (NBCC) (Adams and Halchuck 2003). The IRB is a source zone of relatively low to moderate seismicity with maximum credible earthquake of magnitude 7.0. The peak horizontal ground accelerations (PGA) for the site obtained from the NBCC2015 are summarized in Table 18-3. The PGA corresponding to 5,000-year design seismic event was estimated to be 0.12 g by extrapolation of the values provided in Table 18-3.

Table 18-3: Peak ground acceleration for Kiena Mine site

Probability	40% / 50 year	10% / 50 year	5% / 50 year	2% / 50 year
Return period (years)	1:100	1:475	1:1,000	1:2,475
PGA (g), mean values	0.013	0.032	0.049	0.081

18.11.4 Safety Status of Existing TSF and Upgrade Option

Dam safety assessment of the existing TSF facility indicated that most of the perimeter dykes required upgrade to meet current dam safety guidelines and standards. Wood developed TSF dykes' upgrade design option to increase the TSF's tailings storage capacity as required to accommodate the LOM tailings quantity and improve the stability of the existing perimeter dykes to meet current dam safety requirements. Analyses were performed on a representative section of Dyke 40 to assess the relative merits of a number of suitable stabilization options with due consideration for technical, economic and proximity to the lake (30 and 60-m setbacks from lake high water mark), etc. Stabilization options respected the 30-m setback requirement for all dykes. However, the 60-m setback requirement could not be respected at segments of Dykes 40, 41 and 23 as the existing toe is already within the 60-m setback. The findings of these analyses formed the basis of improvement of the perimeter dykes. Considering very loose to loose condition of the slurry deposited tailings, the dam safety assessment and remediation design were performed assuming liquefaction of tailings.

The proposed upgrade involves construction of downstream toe buttress where possible and ground improvement combined with toe buttress where toe area is limited due proximity of the lake shore. Based on review of potential ground improvement options, Wood's experience at sites with similar conditions and discussions with a ground improvement contractor, Deep Soil Mixing (DSM) was chosen as the preferred ground improvement method. The layout of the proposed upgrade option with toe berm extension and DSM is shown in Figure 18-3. Sections 18.11.5 and 18.11.6 provide additional information for the proposed upgrade design.

It is also noted that the existing TSF does not have the Emergency Spillway required to satisfy current dam safety guidelines and standards.

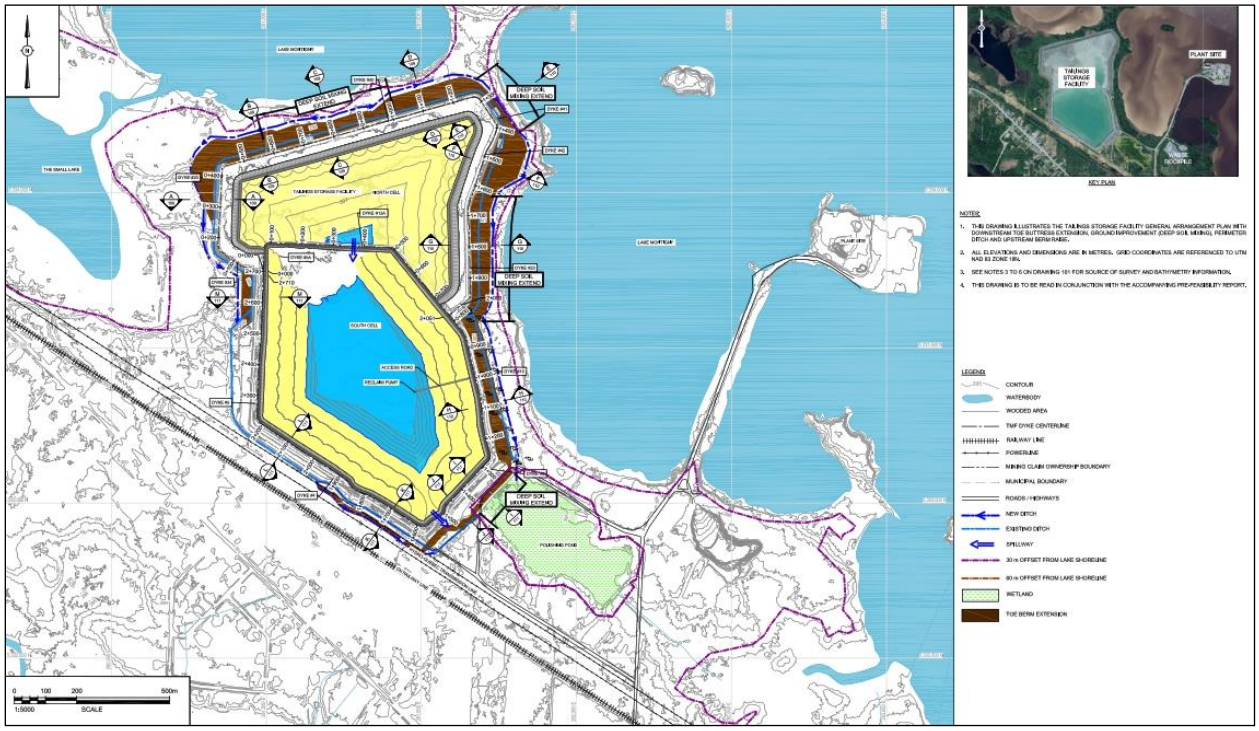


Figure 18-3: TSF upgrade and LOM tailings deposition plan

18.11.5 Tailings and Water Management Plan

The tailings and water management plan at the upgraded TSF was developed considering the following:

- The existing North Cell dyke crest is at elev. 306 with tailings beach almost at crest level;
- The existing South Cell dyke crest is at elev. 308 with tailings beach elevation ranging between 305.5 and 306.5;
- An Emergency Spillway is required for the TSF to contain Environmental Design Flood (EDF) and safely pass Inflow Design Flood (IDF). Directive 019 requires minimum freeboard of 1.5 m above the emergency spillway invert;
- Considering the requirements for spillway and 1.5 m freeboard, operating the North and South Cells separately will not be efficient. That is, both cells will require allowance for respective EDF storage as well as 1.5 m freeboard above spillway invert elevation. Furthermore, it will require active water management between the cells using pumps.

Based on the above discussions and TSF Closure consideration, it is proposed to raise and maximize tailings deposition in the North Cell and allow water management within South Cell. An Emergency Spillway is proposed at Dyke 3 of the South Cell. An Internal spillway is proposed at Dyke 13 to allow for passive flow from North to South Cell.

At closure, it is proposed to breach Dyke 3 Emergency spillway and grade and cover tailings in both cells for erosion protection and allow continuous passive flow into the polishing pond located south of the TSF.

The deposition plan was developed assuming an average deposited tailings void ratio of 1.17 (dry density 1.30 t/m^3, Specific Gravity 2.82, Solid Content 0.46) and deposited tailings slopes of 1% and 4% above and below the pond level, respectively. Operating pond volume of approximately 180,000 m^3 was considered within the South Cell. The tailings deposition plan shown in Figure 18-3 was developed to store 1.3 Mt of tailings. This provides flexibility with the backfill operation by providing additional tailings storage of 0.20 Mt beyond the PFS LOM tailings to TSF of 1.10 Mt. LOM tailings production and tailings to TSF are summarized in Table 18-4. Based on the tailings deposition plan, years 2021-2025 tailings can be stored in North Cell with the construction of a 3-m high upstream tailings berm over compacted tailings beach. Remaining tailings for years 2026-2027 can be stored in the South Cell by raising the South Cell perimeter dykes by 1.0 m. Operating pond and EDF can be managed within the South Cell.

Water will be pumped from South Cell using a pump system to a water treatment plant (WTP) located at the northeast end of polishing pond. The pump system will be located within the South Cell pond with access road connecting to Dyke 10. Treated water will be discharged into the polishing pond, which will then be released into the environment via the existing discharge outlet. The reclaim water for mill operation, if required, will be from TSF South Cell via the pump system.

Table 18-4: Tailings production – TSF required capacity

Year	Ore milled (Mt)	Tailings (Mt)	Tailings to TSF (Mt)	Cumulative Tailings to TSF (Mt)
2021	0.076	0.076	0.076	0.076
2022	0.161	0.161	0.107	0.183
2023	0.198	0.198	0.126	0.309
2024	0.278	0.278	0.197	0.505
2025	0.258	0.258	0.171	0.676
2026	0.256	0.256	0.169	0.845
2027	0.337	0.337	0.248	1.093
2028	0.011	0.011	0.007	1.100

18.11.6 Tailings Storage Facility Design

18.11.6.1 Design Considerations and Criteria

Slurry tailings are proposed to be conveyed to the TSF via pipeline and spigoted from the perimeter dykes. The spigots will be raised progressively as the perimeter dykes are raised to provide containment. The key design considerations and criteria for the TSF are summarized below:

- The TSF dykes are classified as “Very High” hazard classification given the environmental and economic consequences of a potential failure under both Canadian Dam Association Guidelines (CDA, 2013 and 2019) and Global Industry Standard on Tailings Management (GISTM, 2020);
- The EDF is defined as the volume of runoff generated from the 1:2000-year 24-hour storm and 1:100-year 30-day snowmelt based on Directive 019 (2012);
- The TSF will contain the Environmental Design Flood (EDF) runoff above the maximum operating water level (MOWL) with no discharge to the environment;
- The Probable Maximum Flood (PMF; crue maximale probable) is adopted as the Inflow Design Flood (IDF). The Emergency Spillway is sized to safely pass IDF;
- The runoff collection ditches are designed for 1 in 100-year 24-hour design storm event based on Directive 019 requirement;
- Directive 019 requires TSF dykes to be designed for a minimum 1 in 2,475-year seismic event. Considering CDA (2013, 2019) and GISTM (2020) requirements for Very High consequence classification, 1 in 5,000-year seismic event is adopted as the design earthquake;
- Based on previous production at the Kiena Mine Complex in 2013, tailings and waste rock are anticipated to be non-potentially acid generating (NPAG) with a generally low risk of neutral metal leaching.

18.11.6.2 TSF Plan and Typical Dyke Sections

The plan layout of TSF downstream toe buttress including Deep Soil Mixing (DSM) zone, where required, and upstream berm raise is shown on Figure 18-3. Typical TSF dyke cross sections for North and South Cells are shown on Figure 18-4 and Figure 18-5, respectively. The Pre-Feasibility Study (PFS) Design Report (Wood, 2021) provides typical dykes sections and upstream berm raise details for all TSF dykes.

The primary construction material is rockfill (waste rock) and tailings. Ground improvement by DSM is required at dyke segments where space is limited between the dyke toe and the lake shore. In order to support the upstream berm construction at North Cell over tailings beach, the tailings beach is to be compacted by means of track-packing compaction using a bulldozer to achieve necessary density and strength improvement.

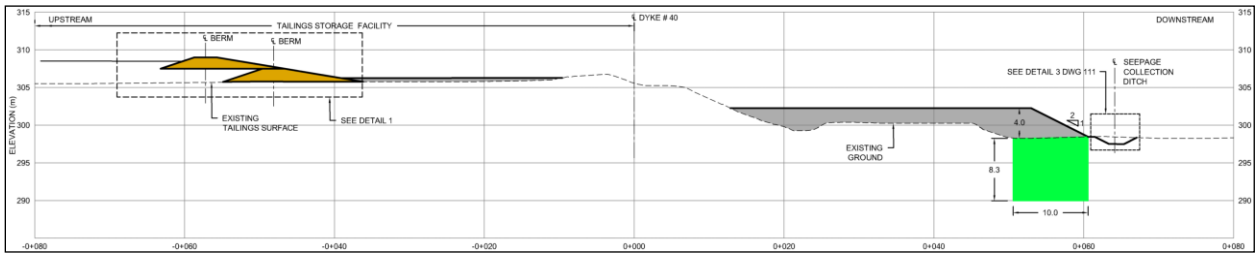


Figure 18-4: Typical TSF Dyke section at North Cell

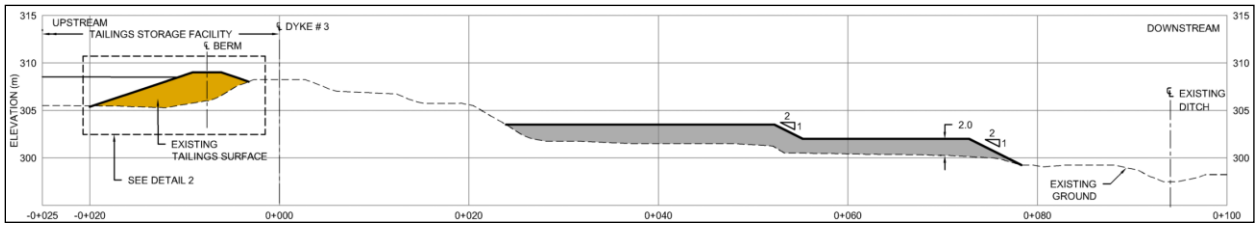


Figure 18-5: Typical TSF Dyke section at South Cell

18.11.6.3 Stability Analyses

Limit equilibrium stability analyses were performed to size the downstream toe buttress extension, determine the size and location of DSM zone and upstream berm configuration using the software Slope/W Version 2016 using the Morgenstern-Price method of analysis. The material parameters, analysis methodology and results for all applicable loading conditions can be found in the PFS Design Report provided by Wood (2021). Selected long-term stability analysis results for the South Cell (Dyke 10) and North Cell (Dyke 40) are illustrated in Figure 18-6 and Figure 18-7, respectively.

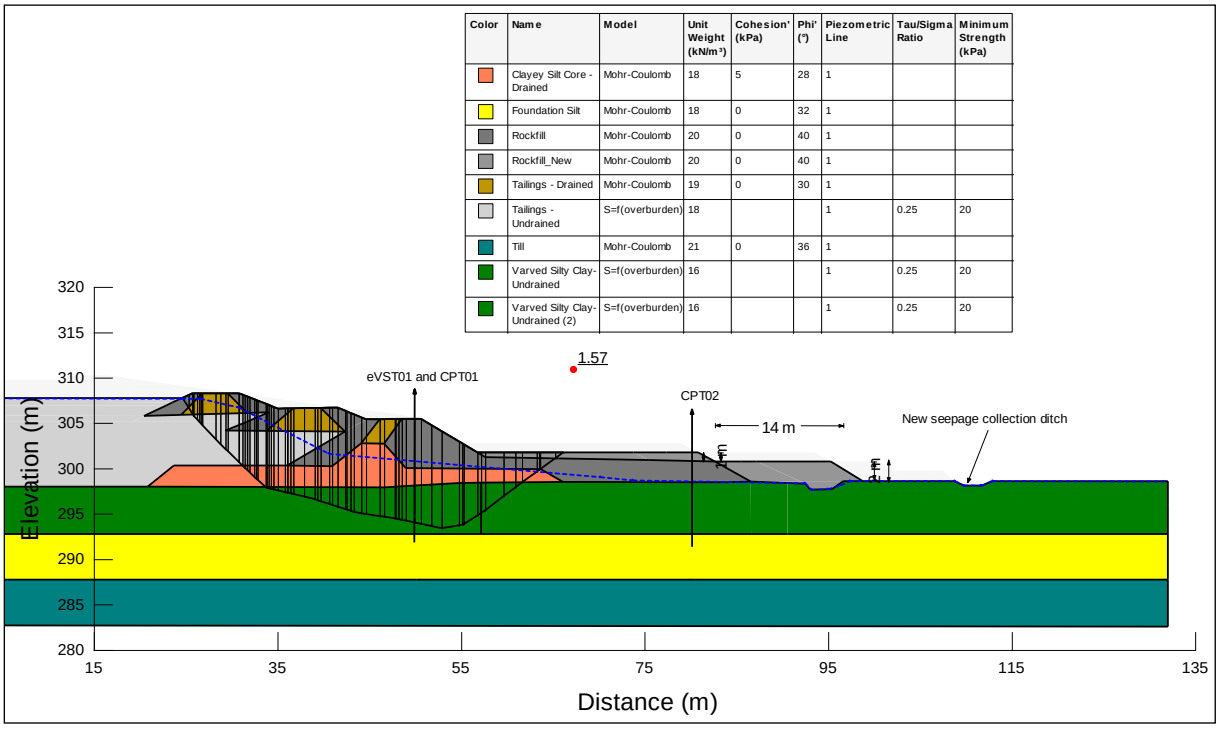


Figure 18-6: South Cell Dyke 10 - Long-term stability analysis

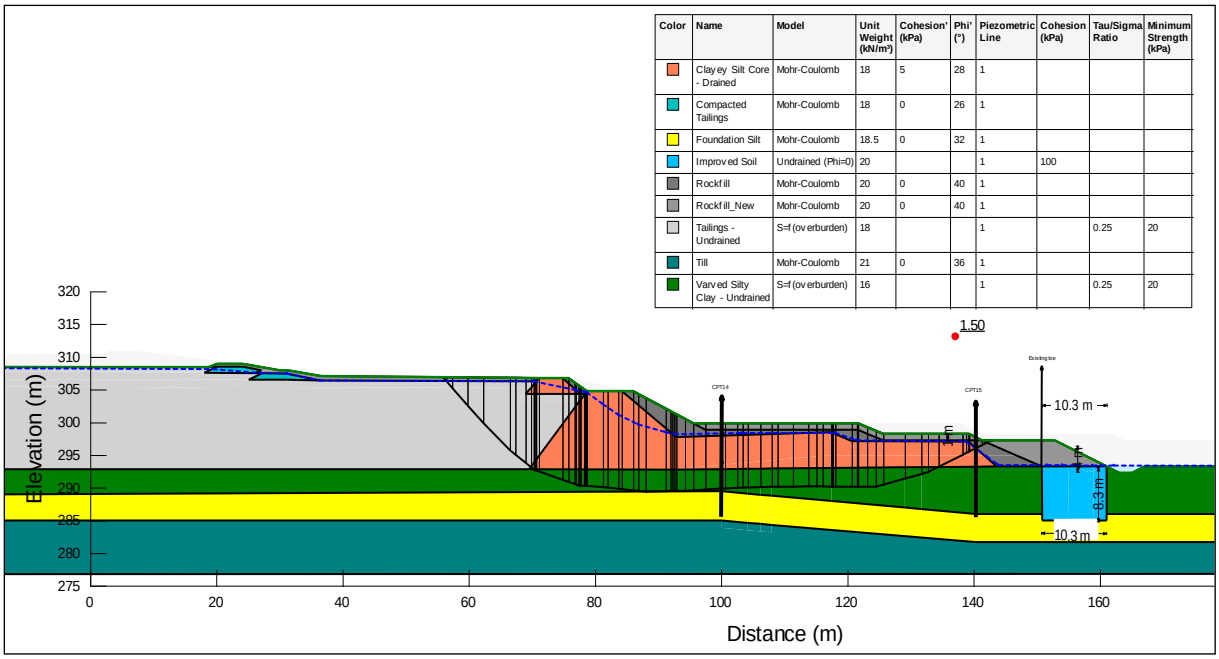


Figure 18-7: North Cell Dyke 40 - Long-term stability analysis

18.11.6.4 Seepage Analyses

Steady state seepage analyses were performed for two sections representing North and South Cells (Dyke 40 and Dyke 3, respectively) to assess phreatic surface across the dykes and seepage losses from TSF. The analyses were performed using the finite element software Seep/W (2016 version). The material parameters, analyses methodology and results for NOWL and EDF conditions can be found in the PFS Design Report by Wood (2021).

The result of the seepage analysis for Dyke 40 is presented in Figure 18-8. The seepage losses from the North and South cells have been estimated to be approximately 13 and 11 m^3/day, respectively, corresponding to the normal operating pond.

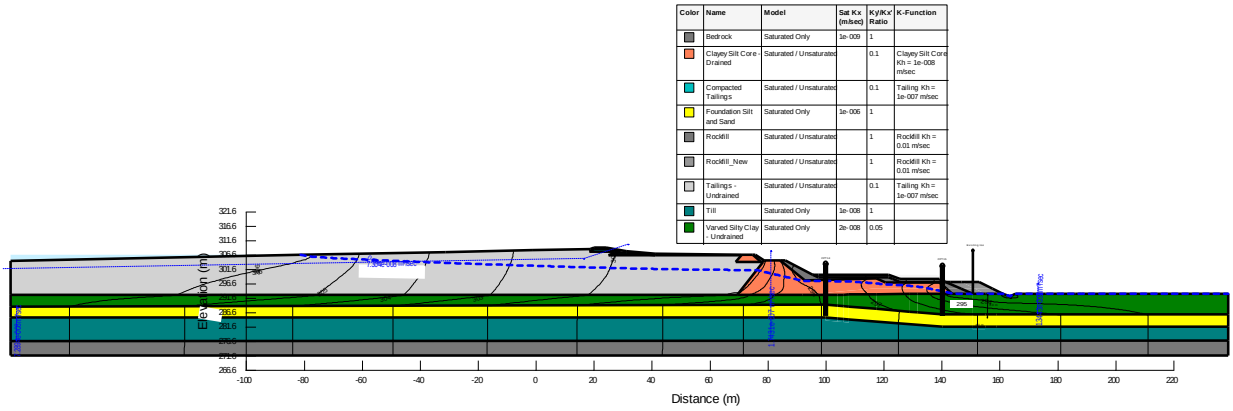


Figure 18-8: North Cell Dyke 40 - Seepage analysis

18.11.6.5 Seepage Runoff Collection Ditches

Runoff from the TSF dykes and seepage will be collected in perimeter ditches sized to handle one in 100-year 24-hour rainfall event. Water captured in the perimeter ditches will be directed to the polishing pond. Layout of perimeter collection ditches is shown in Figure 18-3.

18.11.7 TSF Water Management and Maximum Operating Levels

The proposed TSF water management plan involves the following:

- Years 2021–2025:
  - Tailings deposition in North Cell;
  - Operating pond and EDF managed at South Cell;
  - Passive flow from North Cell to South Cell via Internal Spillway;
  - Emergency Spillway at South Cell Dyke 3 with 1.5 m freeboard.

- Years 2026–2027:
  - Tailings deposition in South Cell;
  - Operating pond and EDF managed at South Cell;
  - Passive flow from North Cell to South Cell via Internal Spillway;
  - Emergency Spillway at South Cell Dyke 3 with 1.5 m freeboard.

South Cell water level will be managed by treating and discharging to the polishing pond. A pump system (located within the South Cell) will be used to convey water from South Cell to the WTP. Detailed hydrotechnical analyses for TSF water management can be found in the PFS Design Report by Wood (2021). The maximum operating pond and EDF pond levels for years 2021–2025 and years 2026–2027 periods are summarized in Figure 18-9.

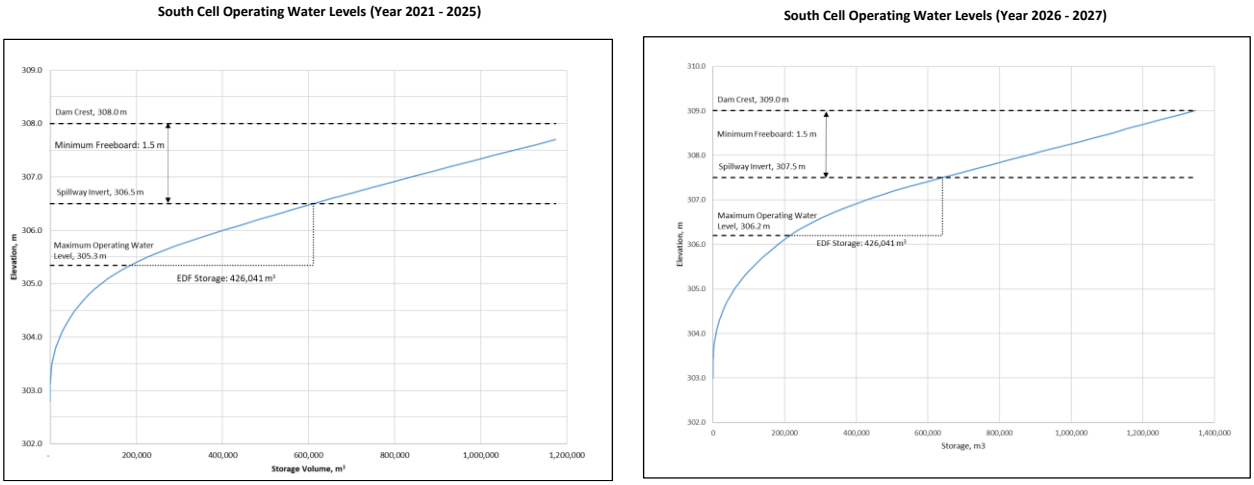


Figure 18-9: TSF operating pond and EDF levels

18.11.7.1 TSF Dyke Remediation, Raise Stages and Construction Materials

The proposed TSF remediation and upstream berm raise construction schedule is summarized below in Table 18-5. Key construction activities include:

- Clearing, grubbing and stripping of dykes' downstream toe area;
- Construction of new seepage and runoff collection ditches. Ditches to be lined with rockfill for erosion protection;
- Construction of toe berm extension using waste rockfill and deep soil mixing;
- Compaction and construction of rockfill cover over tailings beach between proposed new upstream berm raise and existing dykes' crest;

- Staged construction of new upstream berm raises (two 1.5-m raises) on compacted tailings beach at North Cell using tailings as fill material. Downstream slope to be covered with non-woven geotextile and rockfill;
- Construction of Internal and Emergency Spillways;
- Construction of upstream 1-m berm raise at South Cell. Downstream slope to be covered with non-woven geotextile and rockfill.

It is estimated that approximately 400,000 m^3 of waste rockfill will be required for the LOM TSF constructions. Available LOM waste rockfill, including the existing waste rock stockpile, is estimated to be 650,000 m^3. The excess waste rockfill will be used for underground mine backfill operation, stockpiled at existing waste stockpile area or used to cover the TSF.

Existing double wall pipelines are in place to pump tailings from Mill to TSF and to convey water between mill and TSF. New pipelines will be required to convey water between TSF and WTP. The new pipelines will be installed along the existing road corridor. The water treatment plant will be located at the northeast end of the polishing pond.

Table 18-5: Summary of tailings storage facility remediation and dyke raises

Tailings Storage Facility	Internal Spillway Invert Elev. (m)	Emergency Spillway Invert Elev. (m)	Perimeter Dyke Crest Elevation (m)	Downstream Slope Remediation		Upstream Dyke Raise	
				Toe Buttress Extension	Deep Soil Mixing	Raise (m)	Length (m)
North Cell Remediation	-	-	306.0	Dykes 20, 40, 41, 42, 23	Dykes 40, 41, 42, 23	-	
South Cell Remediation	-	-	308.0	Dykes 10, 3, 4, 5	Dyke 3	-	
North Cell (2021-2025)	306.0	306.5	309.0			3.0	2,000
South Cell (2021-2025)			308.0			-	-
North Cell (2026-2027)	306.0	307.5	309.0			-	-
South Cell (2026-2027)			309.0			1.0	1,900

18.11.8 Recommendations, Risks and Opportunities

Recommendations for future design works include the following:

- Advanced geotechnical laboratory testing on intact Shelby tube tailings sample and reconstituted tailings samples to determine liquefaction resistance and liquefied shear strength;
- Advanced geotechnical laboratory testing on undisturbed varved silty clay samples (Shelby tube samples) to determine static shear strength, liquefaction resistance and cyclic soften shear strength;
- Perform site-specific seismic hazard assessment and estimate design earthquake parameters (1 in 10,000-year earthquake);
- Develop design earthquake time histories and perform advanced seismic deformation analyses of critical dyke section for the design earthquake;
- Comprehensive geochemistry assessment of tailings and waste rock.

It is noted that these recommendations are currently being addressed as part of ongoing work.

The following risks have been identified:

- Ice build-up at tailings beach due to ON/OFF milling plan and tailings deposition at the TSF. This could lead to reduced tailings storage capacity. It may also impact stability and safety of upstream berm raise (second 1.5m raise) at the North Cell. There is also potential for freezing of tailings lines, however this will be mitigated by recirculating the water between Mill and TSF;
- PFS design assumed the foundation soil units would not liquefy during design earthquake event based on preliminary / screening level analyses. If found to liquefy during detailed design, larger toe buttress and ground improvement would be required;
- Seismic hazard for Canada is updated by Geological Survey of Canada (GSC) every 5 years. Future updates by GSC may see higher seismic hazard for the site which may lead to the requirements to upgrade TSF design;
- Tailings and waste rock are assumed to be non-potentially acid generating (NAPG) with a generally low risk of neutral metal leaching;
- TSF upgrade (toe berm and ground improvement) and upstream raise construction need to be completed as soon as possible to allow for restart in July 2021;
- Wet weather conditions are expected to impact the upstream berm raise construction in the North Cell. As such, upstream berm raise needs to be constructed during the dry summer months;
- The southeast corner of the existing North Cell pond needs to be filled in with dredged or new tailings as soon as possible to allow time for consolidation of new tailings.

The following opportunities have been identified:

- As part of the TSF tailings management study, PFS level design was developed for a Filter Plant and a Tailings Filter Stack (TFS) located at the existing waste rock stockpile area. These designs have been included in the PFS Design Report (Wood, 2021). These designs can be utilized, as required, if additional tailings storage is required beyond the current LOM;
- Proposed TSF South Cell raise design was developed to accommodate current LOM tailings to TSF. There is potential for additional tailings storage by raising the South Cell perimeter dykes. It is noted that additional raise to the South Cell dykes will require further reinforcement of the dykes than the currently proposed reinforcement design.

18.12 Water Treatment

The current authorization certificate allows sending the tailings to the TSF without cyanide destruction, thus relying on natural degradation to remove cyanide. Since the SO_2-Air cyanide destruction process was stopped, Kiena has been able to meet the effluent environmental requirements without any water treatment. However, it was decided that the cyanide destruction process be restarted for the following reasons:

- The tailings pond dykes will need to be raised to manage the extra tailings that will be produced, Wesdome will need to update its authorization certificate and it appears unlikely that an authorization certificate, allowing natural cyanide degradation, would be issued;
- Because the tailings lines run along the causeway near De Montigny Lake, the risk of a line rupture resulting in a spill into the lake would be far less damaging to the environment if cyanide is destroyed in the plant, upstream of the tailings lines;
- A portion of the tailings produced will be used as paste backfill which requires cyanide destruction.

The use of a cyanide destruction process produces cyanate ions that evolve to ammonia which then partly vaporizes. Residual concentrations are usually high enough to exceed lethality limits of rainbow trout and daphnia magna. This condition is not acceptable for the effluent of the tailings pond thus the water needs to be processed to remove the ammonia. As a result, a water treatment plant will be required.

An effluent water quality assessment was performed by ASDR. One toxicity element was identified, probably linked to ammonia, from the 46 historical tests performed from 2003 to 2009. No other toxicity element was identified from the water 348 samples analyzed from 1998 to 2009.

The effluent quality criteria were determined from the available assays as well as experience. The main quality criteria are presented in Table 18-6.

Table 18-6: Effluent quality criteria

Contaminant	Toxicity Limit (mg/L)
Total Ammoniacal Nitrogen (as N)	24.0
Cyanate	15.0
Total Cyanide	0.16
Cyanide WAD	0.09
Free Cyanide	0.044
Thiocyanate	2.1
Total Copper	0.10
Dissolved Copper	0.08

The initial WTP design was based on the removal of dissolved metal with co-precipitation with ferric sulphate solution and the removal of nitrogen compounds (ammonia and cyanate) with the MBBR (Moving Bed Biofilm Reactor) process. Both processes produce sludge that was to be returned to the tailings pond. The design was based on preliminary estimations of the contaminants in the feed and limited contaminant evolution in the tailings pond:

- Copper and nickel in the mill effluent are mainly complexed with cyanide and will dissociate as pH decreases thus precipitate to an unknown extent;
- The cyanate ion will evolve to carbonate and ammonia ions and it is assumed that about 30% of the cyanate will have been transformed by the time it reaches the WTP;
- The ammonia ion is present in the mill effluent and its concentration increases with pH evolution. It is considered that no vaporization occurs in the tailings ponds.

Substantial risks remained in the resulting design considering that:

- There is no available data on the tailings pond water quality fed with a tailings stream resulting from cyanide destruction. As a result, it is not possible to accurately know the amount of ammonia in the effluent that will require treatment;
- To consider that there is no evolution of the contaminants' concentration would result in oversizing the equipment. On the other hand, the WTP would have the capacity to remove those contaminants whatever is happening in the tailings pond;
- The dissolved copper, and any other heavy metals in the water, will be in the form of cyanide complexes that cannot be removed by the iron coprecipitation process. Na_2S addition would be required for heavy metals to precipitate.

The initial design had a high probability of having over capacity resulting in a higher than needed investment. Furthermore, it is not possible to accurately predict the behaviour of the contaminants thus know what the right design should be. As a result, a mobile treatment plant, based on filtration and reverse osmosis, is to be used. It will allow the removal of heavy metals cyanide complexes and reject a stream containing contaminants back to the South Cell. This solution is to be implemented for at least the first year and this PFS assumes that the mobile plant is used for the life of mine. It is estimated that due to the lower mine throughput in the early years thus lower reagent consumption, no excessive accumulation of contaminants in the South Cell will occur considering its size.

The mobile water treatment plant will consist of:

- A filtration unit meant to remove suspended solids.
- A two-pass reverse osmosis unit that is meant to remove dissolved contaminants to levels below the desired targets.

Furthermore, species such as cyanide will degrade naturally to some extent in the North and South Cells before being processed by the water treatment plant. An extensive sampling plan will be implemented during the mobile water treatment plant operation to determine the design criteria of the permanent water treatment plant if it is needed, or if the mobile plant can be kept in operation.

19. MARKET STUDIES AND CONTRACTS

19.1 Market Studies

No market studies have been conducted by Wesdome Gold Mines Ltd., or its consultants, in relation to the gold doré that will be produced by the Kiena Mine Complex project. Gold is a freely traded commodity on the world market for which there is a steady demand from numerous buyers.

19.2 Assumptions

Table 19-1 outlines the terms used in the economic analysis.

Table 19-1: NSR assumptions used in the economic analysis

Assumptions	Unit	Value
Au payable	%	99.975
Au refining charge (including transportation cost)	CAD/oz	3.00
Gold price	USD/oz	1,600.00

19.3 Contracts

Wesdome Gold Mines Ltd. currently has a gold refining agreement in place with the Royal Canadian Mint in Ottawa, Ontario, Canada.

20. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The purpose of this section is to provide summary information on the Project and environmental setting, the legislative framework applicable to the Project, and key environmental and socio-economic risks.

20.1 Project and Environmental Setting

20.1.1 Project Setting

The Kiena Mine is an inactive underground gold mine located about 9 km west of Val-D'Or, Québec. The Kiena Mine TSF is flanked by Lac de Montigny to the north and east, Petit Lac to the west, and CN railway tracks to the south. The current project setting, along with infrastructure and development areas are shown on Figure 20-1.

Figure 20-1: Project setting

Access to the underground mine is currently, and will continue, to be from Parker Island located in Lac de Montigny, which is connected to the mainland via a 900-m long causeway. Since the start of mining operations, Parker Island has been expanded by approximately one third of its area, mostly to the west, presumably using waste rock. The island contains the access to the underground mine, the processing plant, and associated infrastructure and buildings (Figure 20-2).

Figure 20-2: Parker Island

20.1.2 Natural Environment

Wood completed a preliminary focused natural environment baseline study in November 2020 (Dibley and Ruthven, 2020) to assist with determination of the need for future environmental studies and to support a determination of permitting / regulatory requirements. The study area focused on the new area of potential development: adjacent to the existing TSF, as well as east of the access road (Kienawisik Road) where a future dry tailings stack may be developed. Further detail is provided in Dibley and Ruthven (2020). Wesdome is also completing additional, more detailed environmental studies to support the mine development and meet regulatory requirements for the approvals processes.

20.1.2.1 Terrestrial Habitat and Species

Five distinct habitat classifications were observed during field investigations: Coniferous Swamp, Deciduous and Mixed Forest (dominated by Trembling Aspen), Deciduous Swamp, Marsh, Thicket Swamp, in addition to habitat reflecting heavily modified areas from previous industrial activities.

Thirty-three species of plants were identified during field investigations, as well as an additional six plants identified only to genus. All species identified are common and expected species of boreal forest habitats.

Eight species of bird and five species of mammal were observed during field investigations. All observed are common boreal forest species.

No Species at Risk animals or Species at Risk plants were identified during the preliminary field investigation completed in late fall 2020. There is the potential for a number of Species at Risk animals and plants to be present based on data available for the region. The built-up nature of much of the Kiena Mine lands may result in wildlife, including Species at Risk, avoiding much of the site. Studies to further investigate the potential presence of Species at Risk could be considered going forward to support the environmental approvals process.

20.1.2.2 Aquatic Habitat and Species

The November 2020 field program also investigated a number of minor, natural and man-made waterbodies / watercourses to identify whether fish were present. Brook Stickleback, a resilient, commonly found, small-bodied fish, were present in a number of the man-made waterbodies (ditches and small ponds) near the TSF and near the potential area where a filter stack could be developed. No other fish including Species at Risk fish were present at the time of the investigation (Dibley and Ruthven, 2020).

20.1.3 Human Environment Setting

A brief summary of the human environment aspects of the area was prepared to support the PFS (Griffiths, 2021). The Kiena Mine is located within the limits of the City of Val-d'Or in the Abitibi-Témiscamingue region of western Québec. The Abitibi-Témiscamingue region is located along the Québec - Ontario border. The economy of the area is based on mining, logging and agriculture. The population of Val-d'Or increased by 4% between the 2006 and 2016 census while the Québec population grew by 8%. The median age of residents of Val-d'Or was 42.5 in the 2016 Census, which is higher than in 2011, meaning that the population appears to be ageing. The region has a strong history in mining and the local population is anticipated to have appropriate training and education.

Val-d'Or has a small Indigenous population. In the 2016 census, less than 5% of the population self-identified as Aboriginal and most of these identify as First Nations. The nearest Indigenous communities are Conseil de la Première Nation Abitibiwinni, Pikogan, which is approximately 75 km north of Val-D'Or and Nation Anishnabe du Lac Simon, 32 km southeast of Val-d'Or.

20.2 Preliminary Assessment of Effects of Reopening

Proposed reopening of the mine will provide local economic benefits and jobs, and will require some physical changes to the site, including as follows:

- Reactivation of the underground mine which has been maintained in a dewatered condition;
- Modification to the existing processing facilities located on Parker Island which is connected to the mainland by means of an access road (Kienawisik Road) on a causeway;
- Establishment of effluent treatment as needed;
- Stabilization of the existing two-cell TSF and rearrangement to provide for additional tailings storage.

Potential development of a dry storage facility for tailings (filter stack), at approximately the location of the existing waste rock stockpile on the mainland east of Kienawisik Road.

Professionals of Wood Environment & Infrastructure Americas reviewed the potential effects of reopening the mine to the natural, biological and human environment in support of this PFS. This preliminary assessment is described in Ruthven (2021) and Griffiths (2021) and is summarized in the sections that follow.

20.2.1 Impacts to the Atmosphere

Emissions to the air occur from most industrial operations including underground mines. The key emission sources relevant to the Kiena Mine are:

- Point source emissions from industrial stacks, heating and mine ventilation (shafts and vent raises);
- Fugitive emissions from equipment operation, vehicle movements, material handling and mineral waste (tailings and waste rock) placement;
- Greenhouse gas emissions from fossil-fuel combustion including from underground mining equipment and surface vehicles.

Key air quality parameters of interest include total particulate matter, fine particulates, sulphur oxides, nitrogen oxides, metals and silica (human health). Air dispersion modelling will be required to support environmental permits to predict air quality concentrations from sources associated with the mine.

For an underground mining operation, fugitive dust from vehicle movement, and tailings and waste rock placement is likely to pose the greatest concern for effects to the natural environment including off-site, based on the assumption that point source emissions meet all regulatory requirements. Fugitive dust releases to the environment can be controlled through mitigation measures, such as regular use of water sprays and potentially other suppressants. Significant effects to the natural environment for changes to air quality from the mine reopening are not expected.

Project-related greenhouse gas emissions include both Scope 1 Direct GHG Emissions and Scope 2 Purchased Electricity. Scope 1 emissions are predominantly from onsite fuel combustion (i.e., heating of return air and engine tailpipe emissions), as well as emissions from other mining and ore processing activities. The effects will be continuous throughout the construction, operation and eventual closure phases of the mine. The annual greenhouse gas emissions from the mine are a very minor contribution to the Canadian greenhouse gas inventory and are considered not to have a significant effect on the local natural environment, but may need to be assessed to support permitting.

Reopening of the Kiena Mine may cause increased noise levels beyond the existing background noise levels and is expected to need to be assessed to support permitting. Noise can be expected to be created from the processing operations and exhaust fan on the Parker Island (including new equipment), as well as the tailings and waste rock storage on the mainland and general traffic. Noise sources on the island (existing and future) have the potential to increase noise levels in the surrounding receptor areas, above regulatory levels if left unmitigated.

On the mainland, the existing TSF and potential tailings filter stack may be significant sources of noise. As the Kiena Mine is in a relatively developed area, significant effects to the natural environment from the increased noise levels are not anticipated, although wildlife may avoid the immediate area due to noise disturbance.

20.2.2 Potential Impacts to Land and Wildlife

Parker Island was originally a natural island that has been expanded over time by adding fill. It is a brownfield industrial setting as visible from the photo below. The redevelopment is expected to occur within the existing industrial footprint. No direct effect to the limited amount of vegetated land remaining on the island is expected from the reopening of the mine.

Stabilization of the existing TSF will result in limited additional disturbance of land. A majority of this land has been heavily modified by mine activity in the past. Where natural habitat remains, it is primarily meadows, thickets and small wetlands, or to a lesser extent, deciduous or mixed forest, deciduous swamp or marsh (Dibley and Ruthven, 2020). Apart from the limited direct loss of habitat, no material effects on birds or wildlife are expected from the mine reopening.

20.2.3 Potential Impacts to Water and Aquatic Resources

20.2.3.1 Covering of Existing Waterbodies with Tailings

A field investigation completed in Fall 2020 in the area of proposed development on the mainland, determined that a number of the man-made ditches and ponds that are proposed to be either removed during dam construction, or could potentially be covered by a filtered tailing stack if developed, contained Brook Stickleback a small-bodied fish (Dibley and Ruthven, 2020). No other fish species were collected during the field investigation. It is our understanding that all of the waterbodies that are anticipated to be removed or covered by mineral waste as a result of the mine reopening are man-made drainage features (ditches) or man-made ponds; constructed solely as or for mine infrastructure and not intended to support fish. Trapping and relocation of the minnows prior to construction could be considered to mitigate impact to fish in these areas. There may, however, remain regulatory requirements.

One small, natural pond containing Brook Stickleback was identified south of the potential tailings filter stack (Pond 10; Dibley and Ruthven, 2020). Based on discussions with Wesdome, it is understood that the pond is likely to be avoided to reduce regulatory requirements. All of the remaining man-made ponds are understood to be isolated from adjacent fish bearing waters.

Apart from the limited direct loss of man-made ponds and ditches related to the dam expansion and disposition of mineral wastes from reopening, no material effects to fish and aquatic resources are expected from the mine reopening. Previous geochemistry results and data from ongoing investigations indicate that acid rock drainage and metal leaching is not expected to be of concern from the Kiena Mine tailings (Walker, 2021).

20.2.3.2 Potential for Surface Water Flow / Lake Effects

The Project site is bounded by Lac de Montigny (virtually surrounding Parker Island, and immediately north of the mainland (Figure 20-1). The proposed underground development consists of mining of four new zones either beside the existing workings or beneath the existing workings. All four proposed zones are at more than 100 m depth below the lake bed, and with the exception of the South Zone, considerably deeper beneath Lac de Montigny.

Hydrogeology investigations indicated that the re-establishment of mining operations is not expected to have an effect on the lake level, particularly as most of the proposed new underground development is at a greater depth and below the existing dewatered workings. The presence of mine-induced drawdown at depth in bedrock is not anticipated to result in a significant environmental impact to local surface water features including the lake (Gautrey, 2021). This is confirmed in part by the low overall pumping rate from the mine, which has extensive dewatered underground workings already. In addition, any limited water taken from the lake, will be returned to the lake as treated effluent.

The stabilization of the existing TSF will require the relocation of existing man-made water management infrastructure. Future establishment of the filter tailings stack if developed, would also result in some localized runoff capture. Since the runoff from both facilities after treatment will be discharged to Lac De Montigny, there is no net hydrologic change beyond the immediate area.

20.2.3.3 Potential for Degraded Water Quality

Tailings from the mill for the LOM plan are proposed to be stored in the TSF. Cyanide destruction as well as other forms of water treatment are expected to be required and will be put in place, in order that effluent from the mine meets regulatory requirements prior to discharge to Lac de Montigny and beyond the dilution zone. As with all industrial operations, there is the potential for some localized water quality effects at the discharge location until the effluent is mixed with the surrounding lake waters; however, these effects are not expected to be significant.

Geochemical investigations of tailings from the previous mining operation showed that the tailings were not potentially acid generating (Corkal, 2020). Wood is conducting an acid rock drainage/metal leaching (ARD/ML) investigation on behalf of Wesdome. Results to date are similar to previous work (Walker, 2021). Should the new tailings be identified as potential ARD/ML, additional mitigation will be required during operation and/or at closure depending on the nature of the risk, to ensure that local waters are protected in the long term.

Assuming that proposed water treatment measures are effective, including for treatment of residual ammonia related to use of blasting agents underground, no risk areas have been identified with respect to potential impacts to water from the mine reopening.

20.2.4 Potential Social and Community Impacts

Based in part on the engagement information provided by Wesdome, a list of potential key social / community impact issues has been identified (Table 20-1). Further information is provided in Griffiths (2021).

Table 20-1: Potential social / community issues and potential mitigation

Category	Key identified issues	Preliminary mitigation / responses
Public health and safety	Potential traffic issues at the mine road entrance.	Traffic safety measures to be implemented to improve safety.
Air Quality	Dust levels may be higher than from the previous operation if not mitigated, if a filter stack is developed in the future for additional tailings storage on the mainland.	Dust suppression measures, especially on service roads. Effective ongoing maintenance at the mine site to reduce dust.

Category	Key identified issues	Preliminary mitigation / responses
Artificial Light	Flashing lights of equipment during the night.	Minimize work at night as practical.
Noise	Noise from equipment and operations above ground including tailings management and construction on the mainland.	Placement of a noise reduction berm if practical.
Aquatic Habitat and Surface Water Quality	Impact of electrical cables on aquatic life; mining in the lake; drilling holes in the lake.	Environmental management to minimize impacts on the lake. Note that there is no mining “in the lake”, although there was past exploration drilling from the ice.
	Potential impact on Lac de Montigny.	Collector ditches around the mine site and tailings to retain water from operations and prevent inadvertent discharges into the lake. Treatment of effluent to meet regulatory requirements.
Groundwater Quality	Potential for groundwater quality issues associated with historic TSF and future tailings storage.	Investigate the potential for groundwater impairment from the facility. Install and monitor additional groundwater wells if appropriate.
Communications	The tight timeframe of the Project may cause challenges in communicating with stakeholders.	An annual public information meeting is held and an information flyer distributed two times per year. Project website will be available in 2021. Employment opportunities will be advertised on social media. Information will be provided on how / where / whom to obtain answers to questions at any time. The confidentiality of communications will be assured .
	Stakeholders may feel that they have not been informed prior to the start of Project activities.	Advance notices will be provided to the public, employees and property owners when work is proposed.
	Inadequate information sharing with stakeholders could lead to a perception that Wesdome is not taking responsibility for perceived impacts.	Project website will be available in 2021.
Process	Request information on consultation requirements.	All infrastructure components have the necessary permits and authorizations. The current evaluation work is not an environmental assessment and does not meet the request of stakeholders.

20.2.5 Effects Summary

Parker Island, where the mine and mill are located, is a heavily built-up area with limited natural environmental aspects and no natural habitat in the area of redevelopment. No material change to the natural environment is expected at this location from reopening of the mine, apart from an increase in atmospheric emissions which are anticipated to be similar to the previous operation (a reduction in local air quality and an increase in noise). These aspects are not expected to have a material effect on the natural environment, including terrestrial and aquatic species.

Most of the area proposed for development on the mainland has been heavily modified by previous mine-related activities / facilities. The natural terrestrial habitat conditions on the mainland in the area of proposed development are typical of boreal forest locations. It is expected that any terrestrial species that are displaced due to the development will relocate. The only fish collected within the man-made drainage features bordering the TSF and in the area of the potential future tailings filter stack were Brook Stickleback, a resilient and tolerant small-bodied fish. An attempt could be made to relocate these fish prior to construction if required. Further, project design, notably collection of contact / seepage water in a centralized location and treatment of effluent, limits any potential impacts to water quality and quantity (flow) as a result of mine reopening.

In conclusion, while there will be a limited direct loss of habitat from the TSF expansion, no significant effect to the natural environment on the mainland, including to terrestrial and aquatic species is expected.

As the Kiena Mine Complex is an existing mine operation that has been on care and maintenance since 2013, the reopening of the mine will be a continuation of the previous mining activities. With the various existing and proposed mitigation measures, there should be no major project risks from a public perspective. It is recommended that Wesdome enhance communications with the public and stakeholders to build positive relationships and maintain community support for the mine.

20.3 Regulatory Framework

20.3.1 Environmental / Impact Assessment Requirements

20.3.1.1 Federal

The Kiena Mine is an existing, suspended, mining operation. A Federal Impact Assessment for the reopening could potentially be required under two scenarios:

- If the reopening project meets the requirements for an expansion under the *Impact Assessment Act*, 2019; or
- If the Project is designated by the Federal Minister of Environment and Climate Change Canada as requiring an Impact Assessment.

If the following condition of the Physical Activities Regulations (SOR/2019-285) pursuant to the *Impact Assessment Act* is met, documentation must be provided to the Impact Assessment Agency so that they can consider whether or not an Impact Assessment is required:

> *19 The expansion of an existing mine, mill, quarry or sand or gravel pit in one of the following circumstances:*
>
> *(c) in the case of an existing metal mine, other than a rare earth element mine, placer mine or uranium mine, if the expansion would result in an increase in the area of mining operations of 50% or more and the total ore production capacity would be 5,000 t/day or more after the expansion;*
>
> *(d) in the case of an existing metal mill, other than a uranium mill, if the expansion would result in an increase in the area of mining operations of 50% or more and the total ore input capacity would be 5,000 t/day or more after the expansion;*

As the planned production rate of the Kiena Project will remain under 5,000 tpd and the area of expansion does not meet the 50% criterion, the Physical Activities Regulation does not apply.

While the Minister could designate the reopening of the Kiena Project to complete an Impact Assessment. Based on the QP's expertise and knowledge in this area, this is not expected to occur. Designation by the Minister is highly uncommon.

20.3.1.2 Provincial

In Québec, the principal environmental statute governing the environmental assessment process is the *Environmental Quality Act* (CQLR c Q-2), administered by the *Ministère de l'Environnement et de la Lutte contre les changements climatiques* (MELCC). An associated regulation (Q-2, r.23.1) describes the activities that are subject to the Provincial environmental assessment process. According to Section 22, Part II in the Schedule I, mining projects existing before March 23, 2018 that are subject to the environmental impact assessment and review process include:

(6) Any expansion of 50% or more of the mine operation area in the following cases:

> *(b) the maximum daily extraction capacity of a mine referred to in subparagraph 2 (2,000 tonnes),… is reached or exceeded;*
>
> *(c) the mine is situated in whole or in part within an urbanization perimeter indicated in the land use planning and development plan applicable to the territory concerned or to an Indian Reserve and at less than 1,000 m from such a perimeter or such a Reserve.*

The author understands as the mine operation area will not be expanded by more than 50%, the Project is not subject to the Provincial EA process.

20.3.2 Environmental Permit Requirements

Three primary provincial agencies will be involved with mining project approvals and permits in Québec are:

- *Ministère de l'Environnement et de la Lutte contre les changements climatiques* (MELCC);
- *Ministère de l'Énergie et des Ressources naturelles* (MERN);
- *Ministère des Forêts, de la Faune et des Parcs* (MFFP).

Additional agencies that may be involved in permitting include:

- Fisheries and Oceans Canada (DFO);
- Environment and Climate Change Canada (ECCC).

20.3.2.1 Government of Québec

20.3.2.1.1 Environmental Quality Act

The Kiena Mine is a previously operating mine and complies with regulation requirements under the *Environmental Quality Act*. The latest authorizations to operate the Kiena Mine were issued in 2001, authorizing the extraction and the milling of 2,040 tpd ore. The authorization was transferred in 2004 to Wesdome Inc. A modification for the expansion of the TSF was obtained in 2010.

The activities associated with the re-opening of the mine will require modifications to the existing authorization and must also conform with the latest version of Directive 019. Also, an authorization under the *Environmental Quality Act* will be required to carry out any works in a constant or intermittent watercourse, lake, pond, marsh, swamp or bog. Pursuant to the Act respecting the conservation of wetlands and bodies of water, the issuance of an authorization is subject to the payment of a financial contribution to compensate for adverse effects on the wetlands and bodies of water concerned.

The mill will require a de-pollution attestation from the MELCC, which identifies the environmental conditions that must be met by the industrial facilities when carrying out its activities. The certificate compiles all the environmental requirements related to the operation of an industrial facility and is renewable every five years. The operator must apply for a de-pollution attestation within 30 days following the issuance of the authorization under EQA for the operation of its project.

20.3.2.1.2 Mining Act

The Kiena Mine site is the object of active mining rights (mining concession CM 494 and multiple mining claims). The *Mining Act* contains provisions requiring companies to reclaim the land that is affected by project development, and applies to both mineral exploration activities and mining operations. The Act requires submission of a rehabilitation and restoration plan, and provision

of a financial guarantee for the cost of reclamation work. A rehabilitation and restoration plan for the site was submitted in 2015. It was revised in 2020 as part of the administrative requirement to update a plan every five years. A new version of the rehabilitation and restoration plan is expected to be submitted in 2021 as part of the reopening activities. Note that the MERN must also approve the location of the new filter stack if a decision is made to progress this facility. The request for approval must demonstrate the absence of an economically exploitable deposit underlying the sector intended to receive dry tailings. As a result, condemnation drilling will need to be completed prior to application.

20.3.2.1.3 Act Respecting the Conservation and Development of Wildlife

When work is carried out in fish habitats, an authorization is required under the Act Respecting the Conservation and Development of Wildlife. A monetary compensation is also requested in parallel with this authorization.

20.3.2.2 Government of Canada

20.3.2.2.1 Fisheries Act

Under the *Fisheries Act*, work that may result in the death of fish or the harmful alteration, disruption or destruction (HADD) of fish habitat requires an Authorization pursuant to paragraphs 34.4(1)(b) and 35(2) of the Act respectively. Based on the planned infrastructure and activities at the Kiena mine site a federal Authorization may not be required. However, it is recognized that Brook Stickleback were found in the man-made ponds / ditches that will require disturbance to stabilize the existing TSF and as such, confirmation by DFO may be needed to confirm the Projects understanding that these waterbodies are not considered natural waterbodies frequented by fish.

DFO issues advice on specific measures to implement for projects near water, in order to protect fish and fish habitat. A Request for Review under the Fish Habitat Protection Provisions of the *Fisheries Act* can be submitted to DFO for it to determine whether the foreseen mitigation measures are sufficient to avoid or mitigate the death of fish or the HADD of fish habitat.

The Kiena Mine will be subject to the Metal and Diamond Mining Effluent Regulations (MDMER) (SOR/2002-222) pursuant to article 36 of the *Fisheries Act* and administered by ECCC for the control and monitoring of the site's effluents.

As Wesdome's current plan is to avoid deposition of a mine waste (such as tailings, overburden, waste rock or untreated effluent) in a natural water body that is frequented by fish, an amendment to Schedule 2 of the MDMER is not expected to be required. Listing of waterbodies on Schedule 2 of the MDMER is a lengthy regulatory process and would require development of a fish habitat compensation plan and an assessment of alternatives for mine waste disposal. Confirmation is required from DFO or ECCC that the man-made ditches and ponds are not considered natural waterbodies frequented by fish and that a Schedule 2 listing will not be required.

20.3.2.3 Key Existing Permits

Table 20-2 lists the principal valid environmental approvals.

Table 20-2: Key existing environmental permits

Authorization / Permit	Year	Agency	Comments
Authorization for the re-opening of the mine (#7610-08-01-70065-23)	1981	MELCC	Extraction of 1,000 tpd ore
Authorization for septic installation (#7610-08-01-70065-00)	1981	MELCC	Service buildings for up to 135 people
Authorization for the ore concentration plant (#7610-08-01-70065-24)	1984	MELCC	Milling rate of 1,250 tpd ore Construction of the tailings storage facility and the polishing pond
Authorization for the expansion of the tailings storage facility (#7610-08-01-70065-28)	1992	MELCC	Including: natural degradation of cyanide instead of the SO_2-Air process; pumping of 125,000 m^3/year of fresh water from Lac de Montigny
Authorization for the exploitation of borrow pit (#7610-08-01-70065-29)	1994	MELCC	Clay extraction
Authorization for the exploitation of borrow pit (#7610-08-01-70065-33)	1999	MELCC	Clay extraction
Modification of the authorization #7610-08-01-70065-23	2001	MELCC	Extraction rate at 2,040 tpd
Modification of the authorization #7610-08-01-70065-24	2001	MELCC	Milling rate at 2,040 tpd
Modification of the authorization #7610-08-01-70065-23	2004	MELCC	Expansion of the waste rock stockpile
Modification of the authorization #7610-08-01-70065-28	2005	MELCC	Raising north cell from elevation 3,048 m to 3,050 m
Modification of the authorization #7610-08-01-70065-28	2010	MELCC	Raising south cell from elevation 3,050 m to 3,051.5 m
Rehabilitation and restoration plan, and addendum	2015, 2017	MERN	Wesdome is currently updating plan

20.3.2.4 Permits / Amendments Needed

A number of new environmental permits or amendments to existing environmental permits are expected to be required for the reopening of the Kiena Mine and additional tailings storage, as provided in Table 20-3. This listing is based in part on direct communications with regulators by Wesdome.

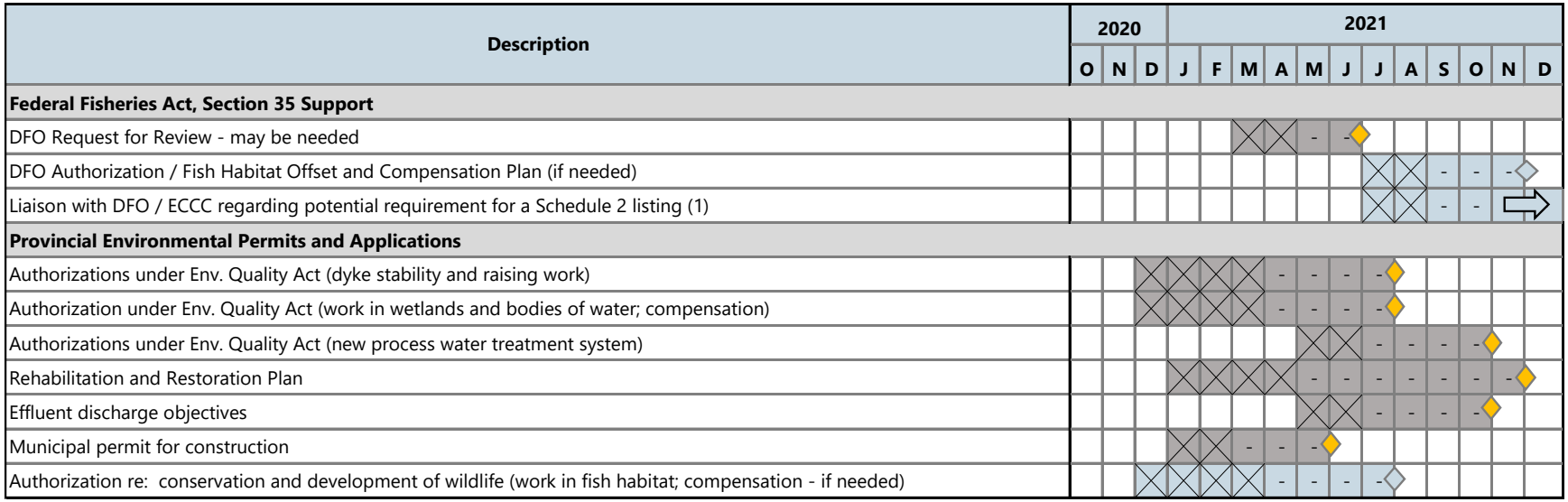
Table 20-3: Anticipated environmental permits or amendments to existing permits

Agency	Permit / Approval	Act	Relevant Components
MELCC	Authorizations / Modification to Authorizations	*Environmental Quality Act*	Construction works at the tailings storage facility for dyke stability and raising work in various phases.
MELCC	Authorization	*Environmental Quality Act*	Work in wetlands and bodies of water; a monetary compensation will also be required.
MFFP	Authorization	*Act respecting the conservation and development of wildlife*	Work in fish habitat; compensation will also be required.
MFFP	Authorization	*Sustainable Forest Development Act*	Tree cutting
MELCC	Authorization	*Environmental Quality Act*	Establishment and operation of new water treatment. A request for effluent discharges objectives should be part of this authorization.
MERN	Rehabilitation and restoration plan	*Mining Act*, Regulation Respecting Mineral Substances other than Petroleum, Natural Gas and Brine	Mine closure and restoration, including financial assurance.
DFO	Request for review	Fish Habitat Protection Provisions of the *Fisheries Act*	Construction works that may affect the drainage ditch / waterbodies where the presence of fish (minnows) has been noted.
DFO and ECCC	Confirmation of Schedule 2 requirements, if any	*Fisheries Act*	Liaison with DFO to identify if consultation with ECCC is required on this aspect.

Municipal permits for construction and deforestation may also be required. These applications will be submitted as part of the ongoing process of the reopening of the site and should not impact the Project critical path schedule.

20.3.3 Regulatory Schedule

Figure 20-3 provides an overview of the preliminary Federal and Provincial permitting schedule for the Kiena Mine, updated on April 30, 2021 by Guillaume Angers, Wesdome.



Note: A De-pollution Attestation must be filed 30 days following the issuance of the authorization associated with the reopening of the mine.

Expected to be required
Not expected to be required
Timeline uncertain
Preparation of the request
Analysis by the authority
Delivery of the authorization

Figure 20-3: Preliminary permitting schedule

20.4 Preliminary Closure Plan for Reopening

20.4.1 Concepts

Closure of the reopened Kiena Mine will be governed by the requirements of the Provincial *Mining Act* and the Regulation Respecting Mineral Substances other than Petroleum, Natural Gas and Brine. The rehabilitation and restoration plan for the existing mine, which is on care and maintenance, was approved by the MERN on June 19, 2017 (an amendment to the plan approved September 30, 2015).

Table 20-4 provides a preliminary approach to reclamation, consistent with the existing plan for the Kiena Mine (Wesdome 2020a).

Table 20-4: Preliminary rehabilitation and restoration approach, key components

Project element	Rehabilitation and restoration approach
Underground Mine	**Current approach:** Openings to underground will be sealed in accordance with Regulations.
	Additional Effort required from Reopening: No material additional effort anticipated or change to proposed approach.
TSF	**Current approach:** Prior to TSF, closure of internal pond(s) will be minimized, and excess water will be discharged to the environment after treatment if needed. Once accessible and if needed, the tailings beach will be contoured to minimize ponding and promote natural drainage through the spillways. Organic amendment will be added to the surface of the tailings management area, and revegetated to provide erosion and dust control.
	Additional Effort required from Reopening: Significant additional effort is not expected to rehabilitate the TSF with the updated design, as the majority of the effort is related to stabilization of the dykes with no significant increase to the areal extent of the facility. The new design is more amenable to closure with improved dyke stability and long-term gravity drainage through an overflow spillway. The dams forming the existing TSF do not generally meet current dam safety guidelines with respect to factors of safety. The new tailings deposition plan improves the factors of safety by incorporating a toe berm and/or soil improvements to improve the dams to meet current dam safety guidelines. It will also allow for eventual gravity drainage from the North Cell to the South Cell, and eventually, to overflow through a spillway at closure to the polishing pond. If excess rock is available, the North Cell may be covered by rock for long-term erosion management benefit.
Polishing Pond	**Current approach:** The polishing pond will be enhanced as a wetland by reducing water level of the existing dyke.
	Additional Effort required from Reopening: No material additional effort anticipated or change to proposed approach.

Project element	Rehabilitation and restoration approach
Stockpiles	**Current approach:** The flat surfaces of the waste rock stockpile will be covered with a 30 cm layer of overburden (to be captured in the gaps between rock blocks), and revegetated by hydroseeding and natural colonization. No other stockpiles are expected to be present.
	Additional Effort required from Reopening: No additional rock stockpiles are expected to be present at closure, and the existing waste rock stockpile is anticipated to be either utilized to backfill the underground, or in TSF dam construction.
Buildings, Machinery, Equipment and Infrastructure including Pipelines	**Current approach:** Salvageable machinery, equipment and other materials will be dismantled and taken off site (sale, reuse or disposal). Buildings and surface facilities will be dismantled and if not amenable to sale or reused, with be sent to an authorized waste disposal site.
	Additional Effort required from Reopening: Although additional facilities will be created, they are of limited scale, and will be reclaimed in the same manner as existing structures.
Petroleum Products, Chemicals and Explosives	**Current approach:** All petroleum products, chemicals and explosives remaining will be removed and handled according to regulations.
	Additional Effort required from Reopening: No material additional effort anticipated or change to proposed approach.
General Site (Parker Island)	**Current approach:** Compacted ground surfaces will be scarified and revegetated. Overburden will be placed as needed.
	Additional Effort required from Reopening: No material additional effort anticipated or change to proposed approach (no additional footprint).

Notes:

(1) The rehabilitation and restoration approach described herein is consistent with the existing Amended Rehabilitation and Restoration Plan for the site which is currently being revised by Wesdome for the reopened operation.

Progressive remediation continues to be proposed in the latest plan, and will be implemented towards the end of the life of mining operations. This will reduce the effort required after mining operations cease.

Wesdome propose to submit an amendment to the existing rehabilitation and restoration plan during 2021. The preliminary approaches described in Table 20-4 may be revised during final revision of the plan already prepared by Wesdome in draft, or if required by regulators.

20.4.2 Final Closure Cost Estimate

In support of the proposed reopening, Wesdome is preparing a revision to the Rehabilitation and Restoration Plan for the mine. Wood completed a review of the updated draft proposed cost for final reclamation (Wesdome 2020b), recognizing that Wesdome and its local consultant are most familiar with local unit costs. The cost estimate:

- Considers inflation from the previous 2015 estimate;
- Includes administration (monitoring and post-restoration maintenance);
- Includes indirect costs (engineering and supervision) and a contingency;
- Excludes resale value for equipment and buildings.

One of the primary reclamation cost areas for the mine redevelopment will be the TSF. Accordingly, Sheila Daniel, P. Geo. and QP from Wood, reviewed the Wesdome projected cost for the required revegetated cover for the TSF to be established at closure, and have revised utilizing local unit costs provided by Wesdome based on the updated TSF design. The QP recommends utilizing the existing Wesdome reclamation cost estimate (Daniel, 2021), adjusted for the new TSF cover costing resulting in a total final reclamation cost of C$9,001,974.

Note that these costs assume that the proposed remediation of the TSF dams to improve long-term stability is completed as part of the reopening plan / operating costs. If for whatever reason that work is not completed, the restoration costs would increase materially.

20.4.3 Restoration Bond Requirement

Wesdome has previously provided a bond to the *Ministère de l'Énergie et des Ressources Naturelles* totalling C$7,027,000. To support reopening, and with the assumption of the total cost estimate above (C$9,001,974), the bond will need to be increased by C$1,974,974 on the required regulatory schedule. The carrying cost for this bond should be considered in the over project estimate as an upfront, pre-construction cost.

21. CAPITAL AND OPERATING COSTS

The capital and operating cost estimates presented in this PFS for the Kiena Mine Complex project are based on the restart of production at the mine after a period of inactivity and comprise various refurbishments, upgrades, and the construction and operation of new facilities.

All capital and operating cost estimates cited in this Report are referenced in Canadian dollars.

21.1 Capital Costs

21.1.1 Introduction

The estimate has been developed in native (quoted) currencies for labour, equipment and bulk materials and converted to CAD as per the following exchange rates, fixed by Wesdome as per their corporate guidance for 2021 and beyond.

Table 21-1: Exchange rates

Currency Code	Currency Name	Rate[(1)]	% Content
USD	US Dollar	1.32	0.2%
CAD	Canadian Dollar	1.00	99.8%

(1) Multiply native currency by rate to obtain values in CAD

21.1.2 Overview

Major permanent equipment pricing is based on technically and commercially evaluated budgetary quotes from vendors; pricing for minor equipment is based on a combination of budgetary pricing, informal quotes and in-house pricing. Measured against other classifications this type of estimate is considered a Class 4 estimate as defined by the Association for the Advancement of Cost Engineering (AACE) in Recommended Practice No. 47R-11ER. As such AACE provides a broad range for accuracy within each estimate class.

Some elements, packages, or areas may not achieve the target level of accuracy individually; the ultimate accuracy achieved has been evaluated based on careful consideration of the level of definition achieved in major engineering deliverables, execution strategy and pricing. The target accuracy of +/-25% has been validated for the entire project scope and estimate in its entirety.

The estimate has been arranged according to the agreed upon work breakdown structure (WBS), by discipline and work element coding (WEC).

Engineering deliverables have been developed to a level of detail consistent with the requirements of the estimate. Additionally, deliverables have been structured to reflect the adopted estimate WBS.

Table 21-2 presents an overview of capital costs by WBS.

Table 21-2: Capital costs summary by WBS (in thousands C$)

WBS Description	Grand Total
Infrastructure	3,668
Administration and Services	104
Mine	130,260
Processing Plant	2,419
Tailings Storage Facility & Water Management	53,418
Total Direct Costs	**189,869**
Owner's Costs	4,657
Project Indirect Costs	14,736
Project Contingency	22,544
Total Indirect Costs	**41,937**
Grand total	**231,806**

21.1.3 Estimate Responsibilities

The overall capital cost estimate combined inputs from several consultants including BBA, Soutex, Mine Paste, and Wood. Table 21-3 lists the general scope for each consultant. BBA (WP4) was responsible for integrating the inputs from each consultant into the overall estimate, and for calculating the contingency. The owner's costs and indirect costs for a given scope are the responsibility of each consultant, however their calculation was coordinated by BBA (WP4).

Table 21-3: CAPEX estimate responsibilities

Scope	Responsible Entity
Mining	BBA (WP1)
Processing & Lab	Soutex (WP2)
Water Treatment	Soutex (WP2)
Tailings Management	Wood (WP3)
General Infrastructure	BBA (WP4)
Paste Plant	Mine Paste (WP5)
Contingency	BBA (WP4)

A summary of capital costs by responsible entity and WBS can be found in Table 21-4.

Table 21-4: Capital costs summary by responsible entity and WBS (in thousands C$)

WBS Description	BBA (WP1)	Soutex (WP2)	Wood (WP3)	BBA (WP4)	Mine Paste (WP5)	Grand Total
Infrastructure				3,668		3,668
Administration and Services				104		104
Mine	123,872				6,388	130,260
Processing Plant		1,678		742		2,419
Tailings Storage Facility & Water Management		23,095	20,429	329	9,565	53,418
Total Direct Costs	**123,872**	**24,773**	**20,429**	**4,842**	**15,953**	**189,869**
Owner's Costs	1,047	141	2,690	300	479	4,657
Project Indirect Costs	6,894	690	2,687	1,538	2,927	14,736
Project Contingency				22,544		22,544
Total Indirect Costs	**7,941**	**831**	**5,377**	**24,382**	**3,406**	**41,937**
Grand total	**131,813**	**25,604**	**25,806**	**29,224**	**19,359**	**231,806**

21.1.4 Exclusions

The following items have been excluded from the estimate:

- All forward escalation;
- All facilities not identified in the summary description of the Project;
- Schedule acceleration costs;
- Foreign exchange fluctuations;
- Risk events (Project Risk Register);
- Unidentified underground conditions;
- Force majeure;
- Operating spares;
- Financing charges and interest;
- Recoverable value added taxes (VAT);
- Sunk costs prior to January 1, 2021.

21.1.5 Capital Cost Estimate Structure

The estimate has been structured by discipline code and WBS.

The estimate direct costs are captured in four principal cost categories: labour, permanent equipment, permanent material, and subcontract, which have been identified with a pricing basis according to the following parameters:

- Firm – pricing based on firm and/or awarded PO or contract;
- Budgetary – budgetary pricing with corresponding technical and commercial review;
- Scaled – vendor pricing adjusted for capacity;
- In-house – pricing from previous studies, escalated PO, etc.;
- Estimated – unit price applied to developed quantities.

21.1.6 Labour Costs

Installation labour costs are based on a forty-hour work week (5 days x 8 hrs/day) deploying a single day shift. Wage rates for craft have been established based on the 2020 Québec construction industry labour agreement of hourly labour costs for industrial projects in accordance with the collective institutional/commercial and industrial sectors.

Composite crew wage rates were established for each commodity based on a craft mix comprised of foreman, journeymen, apprentices and general labour across all construction trades. The composite crew rates include the following costs:

- Craft base rates, fringe benefits and overtime;
- Mobilization & demobilization of contractor items;
- Non-manual labour (general foreman, superintendent, project manager, etc.);
- Small tools and consumables;
- Ownership and operational costs of construction equipment (inclusive of fuel);
- Construction cranes up to 90T;
- Health, safety and environmental requirements;
- Site supervision and administration;
- Contractor temporary site facilities;
- Overhead and profit.

Construction equipment is developed and assigned by specific crew. Hourly equipment costs include the material portion (depreciation, interest, cost of repair and maintenance, insurance, permits and taxes) and operating portion (fuel, lubricants and filters). Sources for rates include the yearly Québec Government publication entitled “Taux de Location de Machinerie Lourde” used by the Ministry of Transport for civil contracts related to public works, roads and highways, contractor pricing and/or pricing received by crane suppliers. The cost of the operator is excluded from the hourly operating cost and included in the crew mix.

It is assumed that 100% of craft reside within 120 km from the site for craft related to civil, concrete, structural, architectural, mechanical, piping and electrical works, as such, travel or accommodations are excluded from the crew rates.

Table 21-5: Composite crew rates per commodity

Typical Crew	Labour Rate		Construction Equipment	Crew Rate
	Direct	Indirect		
Heavy Civil	$66.45	$31.35	$98.22	$196.00
Standard Civil	$67.37	$31.78	$72.36	$171.50
Light Civil	$66.97	$30.62	$62.26	$159.85
Concrete	$67.50	$31.06	$14.03	$112.60
Structural	$67.99	$34.48	$32.49	$134.95
Heavy Architectural	$67.24	$32.50	$10.60	$110.35
Heavy Mechanical	$70.04	$35.32	$21.67	$127.05
Piping	$67.78	$34.17	$18.88	$120.85
Electrical	$70.00	$34.28	$8.72	$113.00
Electrical Power Transmission Lines	$69.94	$35.88	$75.60	$181.40
Automation and Telecom	$68.99	$33.76	$5.74	$108.50

21.1.7 Mining

Mining development capital costs were established considering the capital waste rock development unit costs identified in Chapter 15 – Mineral Reserve Estimate applied to the expected lateral drift and vertical raise lengths to be developed in waste rock identified in the life of mine plan presented in Chapter 16 – Mining Methods. The lateral waste rock development unit cost was established from first principles using Client approved productivities for both underground mobile equipment and labour. Salaries established by Wesdome for the Kiena Mine Complex were considered in the unit cost representing labour. The material consumable costs provided by the Kiena Mine Complex as firm pricing negotiated for purchasing, and from the respective suppliers as budgetary pricing, were also considered in establishing the capital waste rock lateral drift development unit cost. The capital waste rock vertical raise development unit costs considered mining contractor firm pricing provided by Wesdome as well as in-house pricing for both the supported ground, mine air exhaust ventilation raise with ladderway installed and the unsupported ground, mine air exhaust ventilation raise only. Both mine air exhaust ventilation raises are located in the Kiena Deep Zone.

The established fleet of underground and surface mobile equipment presented in Chapter 16 – Mining Methods was based on the requirements to achieve the life of mine plan also presented in Chapter 16 – Mining Methods and Client preferences. The capital costs associated with the fleet of mobile equipment are based primarily on budgetary quotes provided by the respective suppliers and on firm purchase pricing provided by the Kiena Mine Complex for mobile equipment already purchased and on-site or in the process of being purchased and delivery to site is expected soon.

Fixed mining infrastructure accounts for electrical, ventilation, underground dewatering, underground process water, underground communications and underground stations such as refuge, latrine, maintenance service, fuel and material handling chute/pass identified in Chapter 16 – Mining Methods. Both underground and surface construction are required in establishing the fixed mining infrastructure. Capital costs are based primarily on budgetary pricing by the respective equipment suppliers, contractors providing the labour as well as in-house pricing.

Table 21-6 presents a breakdown of capital costs by year and WBS.

Table 21-6: Mining capital costs summary (in thousands C$)

Description	2021	2022	2023	2024	2025	2026	2027	Grand Total
Mine - Infrastructure	6,207	9,413	1,948	1,017	1,711	279	12	20,588
Mine - Mining Fleet	1,124	15,893	4,227	2,738	0	0	0	23,983
Mine - Mine Development	14,196	15,254	13,461	13,161	9,946	3,494	435	69,945
Mine - Telecom	485	407	453	446	297	103	14	2,206
Mine - Electrical	2,712	927	1,206	1,206	754	344	0	7,150
Direct Costs Total	**24,725**	**41,895**	**21,294**	**18,569**	**12,708**	**4,221**	**461**	**123,872**
Owner's Costs - General	329	376	126	93	97	25	1	1,047
Project Indirect Costs - EPCM Services	2,468	1,250	538	461	246			4,963
Project Indirect Costs - Commissioning Services	345	376	116	68	93	10	0	1,007
Project Indirect Costs - Project Indirect Costs - Common Distributables	316	345	106	62	85	9	0	923
Indirect Costs Total	**3,458**	**2,347**	**886**	**684**	**521**	**44**	**1**	**7,941**
Grand Total	**28,182**	**44,241**	**22,180**	**19,252**	**13,229**	**4,265**	**462**	**131,813**

21.1.8 Processing & Lab

The processing plant will remain, for the most part, unchanged. Some refurbishing will be done along with the addition of some new equipment. The new equipment includes:

- The cyanide destruction feed pump box;
- The cyanide destruction feed pump;
- The sodium metabisulfite (SMBS) preparation and distribution system;
- The copper sulfate preparation and distribution system;
- The paste backfill plant feed pump.

Budgetary quotes were received for the cyanide destruction feed pump box and the SMBS preparation and distribution system. A scale factor was applied to the SMBS system quote for the estimation of the copper sulfate system cost. The cyanide destruction feed pump and paste backfill feed pump cost were estimated from quotes for similar projects. Manpower cost for installation as well as an allowance for piping were included in the estimates.

The refurbishing costs include:

- New SAG mill and ball mill motor starters;
- Grinding circuit sampler replacement;
- Leach tanks cover replacement;
- Truck silo cover replacement;
- Tailings thickener containment repairs;
- Instrumentation air compressor;
- Density and pressure gauges replacement;
- Acid scrubber replacement.

Firm quotes were received by Wesdome for the emergency showers and the density gauges. The leach tanks and truck silo cover replacement as well as the tailings thickener containment repairs were estimated by Wesdome for similar work done in the past. These are turn-key estimates. The grinding circuit sampler and instrumentation air compressor, the acid scrubber and tailings line pressure gauge are estimates based on similar projects. Installation costs were included in the estimates.

Allowances for operators training and commissioning support were included in the processing plant CAPEX estimate.

Table 21-7 presents a breakdown of capital costs by year and WBS.

Table 21-7: Processing & lab capital costs summary (in thousands C$)

Description	2021	Grand Total
Processing Plant - General	500	500
Processing Plant - Grinding	423	423
Processing Plant - Leaching	50	50
Processing Plant - Tailing Detox	204	204
Processing Plant - Process Water and Reagents	500	500
Total Direct Costs	**1,678**	**1,678**
Owner's Costs - General	59	59
Owner's Costs - Project Development	50	50
Project Indirect Costs - EPCM Services	285	285
Project Indirect Costs - Commissioning Services	171	171
Project Indirect Costs - Project Indirect Costs - Common Distributables	65	65
Total Indirect Costs	**631**	**631**
Grand total	**2,308**	**2,308**

21.1.9 Water Treatment

The pumping system to feed the mobile water treatment plant includes:

- Two raw water pumps with drives;
- A raw water pump barge with foundations;
- 200 m of heat-traced insulated HDPE piping.

Budgetary quotes were received for the pumps and drives and for the barge as well as for the mobile effluent treatment equipment. The foundation and piping costs were estimated. The study assumes that the effluent treatment plant will be leased over the life of mine, while the feed pumps, barge, and associated piping will be purchased. The effluent treatment plant lease has been capitalized for the duration of the LOM.

Table 21-8 presents a breakdown of capital costs by year and WBS.

Table 21-8: Water treatment capital costs summary (in thousands C$)

Description	2021	2022	2023	2024	2025	2026	2027	2028	Grand Total
Tailings Storage Facility & Water Management - Effluent Treatment	2,035	3,120	3,120	3,120	3,120	3,120	3,120	2,340	23,095
Direct Costs Total	**2,035**	**3,120**	**3,120**	**3,120**	**3,120**	**3,120**	**3,120**	**2,340**	**23,095**
Owner's Costs - General	32								32
Project Indirect Costs - EPCM Services	158								158
Project Indirect Costs - Commissioning Services	6								6
Project Indirect Costs - Project Indirect Costs - Common Distributables	5								5
Indirect Costs Total	**201**								**201**
Grand Total	**2,236**	**3,120**	**3,120**	**3,120**	**3,120**	**3,120**	**3,120**	**2,340**	**23,296**

21.1.10 Tailings Management

The estimated construction quantities and capital costs have been developed based on the PFS-level Life of Mine (LOM) Tailings Storage Facility (TSF) design, the current understanding of the site conditions, and the permitting requirements. The quantity and cost estimates are based on neat line quantities and material take-offs from the typical sections and details, neat line AutoCAD modelling, and unit rates from similar projects. Lump sum estimates have been applied where necessary.

The estimated quantity and capital cost include the following main work items:

- Upgrade of existing TSF perimeter dykes which involves site clearing and grubbing, realignment of existing perimeter ditches, foundation ground improvement by Deep Soil Mixing at select segments of the TSF dykes and toe berm extension at most of the perimeter dykes;
- Construction of Internal Spillway, Emergency Spillway and TSF North Cell upstream berm raise. North Cell upstream berm raise involves two 1.5-m raises;
- Construction of TSF South Cell dyke raise and Emergency Spillway raise.

The TSF construction quantities and cost estimates were developed based on the following:

- Construction quantities are based on LOM TSF dykes remediation and raise plan summarized in Table 18-5;
- Clearing of TSF downstream area will extend to 5 m beyond the toe berm extension and new ditch footprint and grubbing and stripping will extend 2 m beyond the toe berm and new ditch footprints;
- TSF dyke toe berm extension fill volumes assume grubbing and stripping of the footprint area of 0.5 m thickness;
- The volumes calculated are neat line take-offs. No wastage has been included in the quantities;
- Toe berm extension fill volumes include 10% allowance for settlement;
- Upstream berm raise fill volumes include 10% allowance for settlement;
- Geosynthetics quantities include anchorage and 10% increase for overlaps, seaming and wastage;
- Earth works unit rates are assumed based on experience with similar projects in the region;
- Unit rate for the deep soil mixing ground improvement is based on a preliminary quote received from Advanced Geosolutions Inc. (AGI);
- Existing polishing pond and related infrastructures are assumed to be adequate as-is.

Table 21-9 presents a summary of the quantities feeding the capital cost estimate.

Table 21-9: Quantity summary

Cost Element	Unit	Quantity
TSF Remediation / Upgrade Construction		
Clearing (Toe berm extension and relocated ditch footprints)	m^2	193,000
Grubbing and stripping of topsoil (Toe berm extension footprints)	m^3	72,000
Excavation, haulage and shaping of realigned new ditch	m^3	30,000
Zone 2a - Rockfill (minus 100 mm), Ditch erosion protection layer	m^3	16,000
Zone 2 - Rockfill (minus 200 mm), Toe berm extension	m^3	295,000
Deep soil mixing (foundation ground improvement at select dyke locations)	m^3	100,000
North Cell Upstream Berm Raise and Spillways		
Grade and compact (proof roll) tailings beach area between upstream berm and existing crest	m^2	62,000
Zone 2 - Rockfill (minus 200 mm), cover over tailings beach	m^3	38,000
Grade and compact (proof roll) upstream berm footprint	m^2	60,000
Zone 1a - Tailings, Upstream berm fill	m^3	66,000
Zone 5 - Non-woven geotextile, Terrafix 600, on downstream slope of upstream berm	m^2	50,000
Zone 2 - Rockfill (minus 200 mm), upstream berm downstream slope cover	m^3	32,000
Zone 4 - Sand and gravel, Road surface	m^3	1,600
Internal Spillway	LS	1
Emergency Spillway	LS	1
South Cell Upstream Berm Raise		
Grade and compact (proof roll) upstream berm footprint	m^2	30,000
Zone 1a - Tailings, Upstream berm fill (borrowed from tailings beach)	m^3	60,000
Zone 5 - Non-woven geotextile, Terrafix 600, downstream slope of upstream berm	m^2	16,000
Zone 2 - Rockfill (minus 200 mm), upstream berm downstream slope cover	m^3	9,000
Zone 4 - Sand and gravel, Road surface	m^3	900
Emergency Spillway, raise by 1.0 m	LS	1

Table 21-10 presents a breakdown of the tailings management capital costs by year and WBS.

Table 21-10: Tailings Management capital costs summary (in thousands C$)

Description	2021	2022	2023	2025	2026	Grand total
Tailings Storage Facility & Water Management - Tailings Pond and Dykes	18,107		1,002		1,320	20,429
Total Direct Costs	**18,107**		**1,002**		**1,320**	**20,429**
Owner's Costs - General	634		35		46	715
Owner's Costs - Reclamation and Closure	1,975					1,975
Project Indirect Costs - EPCM Services	1,687	300	200	300	200	2,687
Total Indirect Costs	**4,296**	**300**	**235**	**300**	**246**	**5,377**
Grand total	**22,403**	**300**	**1,237**	**300**	**1,566**	**25,806**

21.1.11 Paste Plant

Costs for all major equipment within the paste plant were obtained from Mine Paste Ltd.'s current database and, where necessary, updated with email budgetary values. Factoring was applied to develop the overall project costs based on similar plant layouts for all civil, structural, mechanical piping, electrical, instrumentation and architectural components as well as labour.

For the underground distribution, the cost for drilling and casing was based on email values provided by contractor and pipe supplier for a project implemented in 2020 in Canada.

Table 21-11 presents a breakdown of the paste plant capital costs by year and WBS.

Table 21-11: Paste plant capital costs summary (in thousands C$)

Description	2021	2022	Grand Total
Mine - Infrastructure		6,388	6,388
Tailings Storage Facility & Water Management - Backfill Plant		9,565	9,565
Total Direct Costs		**15,953**	**15,953**
Owner's Costs - General		479	479
Project Indirect Costs - EPCM Services	806	1,595	2,401
Project Indirect Costs - Commissioning Services		244	244
Project Indirect Costs - Project Indirect Costs - Common Distributables		282	282
Total Indirect Costs	**806**	**2,600**	**3,406**
Grand Total	**806**	**18,553**	**19,359**

21.1.12 General Infrastructure

The General Infrastructure scope mainly includes modifications and additions to the surface electrical distribution, demolition, civil works, and various surface infrastructure upgrades.

All quantities generated for the estimate exclude contingency. Growth allowances have been applied to the MTO's and are managed with a unique column within the details of the estimate.

Allowances for fabrication losses, cut and waste losses and wastage have been applied to the material unit price.

Quantities were provided by engineering and were based on key engineering deliverables that were reviewed jointly between the estimators and discipline engineers.

Budgetary pricing was received from vendors for major mechanical and electrical equipment. Technical reviews were performed by package. Equipment pricing is exclusive of freight (except in instances where equipment pricing has been quoted DAP), spare parts or vendor assistance for installation and commissioning. These costs are captured separately in the indirect costs.

Table 21-12 presents a breakdown of G&A capital costs by year and WBS.

Table 21-12: G&A capital costs summary (in thousands C$)

Description	2021	2022	Grand Total
Infrastructure - Site Preparation	47		47
Infrastructure - Electrical	1,029	2,593	3,621
Administration and Services - Permanent Buildings	104		104
Processing Plant - General	742		742
Tailings Storage Facility & Water Management - Effluent Treatment	213		213
Tailings Storage Facility & Water Management - Backfill Plant	115		115
Total Direct Costs	**2,249**	**2,593**	**4,842**
Owner's Costs - General	79	91	170
Owner's Costs - Field Programs	130		130
Project Indirect Costs - EPCM Services	592	231	823
Project Indirect Costs - Construction - Facilities and Utilities	223		223
Project Indirect Costs - Construction - Operation and Maintenance	84		84
Project Indirect Costs - Commissioning Services	29	17	46
Project Indirect Costs - Overhead Expenditures/ Monetary Fees Input	321		321
Project Indirect Costs - Project Indirect Costs - Common Distributables	26	16	42
Total Indirect Costs	**1,483**	**355**	**1,838**
Grand Total	**3,733**	**2,948**	**6,680**

21.1.13 Indirect Costs

Indirect costs were primarily factored from direct costs based on historical experience. Table 21-13 below summarizes the basis of indirect costs.

Table 21-13: Indirect costs basis

Area	Sub-Area	Primary Source
80 - Owner's Costs - General		The Owner's costs include, but are not limited to, the expenses such as administration and management personnel, project management team personnel, construction management, purchasing costs, and environmental expenses.
82 - Owner's Costs - Project Development		Plant operator training
84 - Owner's Costs - Field Programs		Mine and surface geological studies (soil testing)

Area	Sub-Area	Primary Source
89 - Owner's Costs - Reclamation and Closure		Reclamation and closure costs calculated and associated with changes to the tailings storage facility.
91 - Project Indirect Costs - EPCM Services		EPCM costs were factored based on direct costs, excluding mine development, mine mobile equipment's and ETP rental costs
92 - Project Indirect Costs - Construction - Facilities and Utilities		Cost for construction offices for the first two years of construction.
93 - Project Indirect Costs - Construction - Operation and Maintenance		Cost for construction sanitary trailer maintenance for the first two years of construction.
95 - Project Indirect Costs - Commissioning Services	POV - & Mechanical Acceptance	POV and mechanical acceptance is based on 3% of the equipment supply value.
	Commissioning Spares	Commissioning spares is based on 2% of the equipment supply value.
	First Fills	Initial fills are based on 1% of the equipment supply value.
96 - Project Indirect Costs - Overhead Expenditures/ Monetary Fees Input	Insurance	An allowance of 0.45% for direct costs to cover cost of insurance
98 - Project Indirect Costs - Project Indirect Costs - Common Distributables	Freight	Land Freight is based on 4.5% of the equipment supply value.
	Vendor's Rep.	Vendor's representative is based on 1% of the equipment supply value.

21.1.14 Contingency

The contingency was calculated with a probabilistic analysis using a Monte Carlo simulation that was based on the level of accuracy of each line item of the estimate (quantities, labour, material, and equipment, graded on specific scales separately). This simulation was run on the entire estimate and provides a global contingency that accounts for the varying inaccuracies of each individual component feeding into the estimate. The contingency percentage is based on the simulation results at P_{80} as instructed by Wesdome.

The analysis yields a contingency of C$22.2M at P_{80} or 26.3% of direct and indirect costs, excluding the effluent treatment plant rental, the mine development, the mining mobile equipment, and closure costs. The blended contingency across the project and LOM is 10.6%.

No risk amount was included in the capital cost estimate; the overall project risk allocation will be included in Wesdome's corporate costs. However, several mitigation measures were identified in the risk workshops and an amount of C$335k for risk mitigation measures is included in the estimate as part of the contingency in the first year.

21.1.15 Summary

Table 21-14 presents a summary of all capital costs by year and WBS.

Table 21-14: Capital costs summary (in thousands C$)

Description	2021	2022	2023	2024	2025	2026	2027	2028	Grand Total
Infrastructure - Site Preparation	47								47
Infrastructure - Electrical	1,029	2,593							3,621
Administration and Services - Permanent Buildings	104								104
Mine - Infrastructure	6,207	15,801	1,948	1,017	1,711	279	12	0	26,976
Mine - Mining Fleet	1,124	15,893	4,227	2,738	0	0	0	0	23,983
Mine - Mine Development	14,196	15,254	13,461	13,161	9,946	3,494	435	0	69,945
Mine - Telecom	485	407	453	446	297	103	14	0	2,206
Mine - Electrical	2,712	927	1,206	1,206	754	344	0	0	7,150
Processing Plant - General	1,242								1,242
Processing Plant - Grinding	423								423
Processing Plant - Leaching	50								50
Processing Plant - Tailing Detox	204								204
Processing Plant - Process Water and Reagents	500								500
Tailings Storage Facility & Water Management - Tailings Pond and Dykes	18,107		1,002			1,320			20,429
Tailings Storage Facility & Water Management - Effluent Treatment	2,249	3,120	3,120	3,120	3,120	3,120	3,120	2,340	23,309
Tailings Storage Facility & Water Management - Backfill Plant	115	9,565							9,680
Total Direct Costs	**48,795**	**63,560**	**25,416**	**21,689**	**15,828**	**8,660**	**3,581**	**2,340**	**189,869**

Description	2021	2022	2023	2024	2025	2026	2027	2028	Grand Total
Owner's Costs - General	1,133	946	161	93	97	71	1	0	2,502
Owner's Costs - Project Development	50								50
Owner's Costs - Field Programs	130								130
Owner's Costs - Reclamation and Closure	1,975								1,975
EPCM Services	5,996	3,376	738	461	546	200			11,317
Construction - Facilities and Utilities	223								223
Construction - Operation and Maintenance	84								84
Commissioning Services	550	637	116	68	93	10	0	0	1,474
Overhead Expenditures/ Monetary Fees Input	321								321
Project Indirect Costs - Common Distributables	413	643	106	62	85	9	0	0	1,318
Escalation / Risk / Contingency	10,290	9,018	1,189	788	649	604	6	0	22,544
Total Indirect Costs	**21,164**	**14,620**	**2,310**	**1,472**	**1,470**	**894**	**7**	**0**	**41,937**
Grand Total	**69,959**	**78,180**	**27,726**	**23,160**	**17,298**	**9,554**	**3,588**	**2,340**	**231,806**

21.2 Operating Costs

21.2.1 Summary

The operating cost estimates are presented in Q4 2020, Canadian dollars (CAD). The estimates were developed to have an accuracy of ±25%.

The operating cost estimates include mining, processing, water treatment, G&A, and paste backfill costs.

The total operating cost estimates for the LOM are provided in Table 21-15. The overall LOM operating cost is C$295.5M over 7.1 years, or C$187.71/t of ore mined.

Table 21-15: Operating costs summary (in C$)

Item	Total LOM ($M)	Average LOM ($/ore tonne mined)	OPEX (%)
Mining	179.2	113.79	60.6
Processing, Lab & Tailings Management	44.5	28.25	15.0
Water Treatment	5.6	3.55	1.9
General and Administration	57.5	36.53	19.5
Paste Plant	8.8	5.58	3.0
Total	**295.5**	**187.71**	**100.0**

21.2.2 Basis of Operating Cost Estimate

Common to all operating costs estimates are the following assumptions:

- Cost estimates are based on Q4 2020 pricing with no allowances for inflation;
- For material sourced in US dollars an exchange rate of $1.32 CAD per US dollar was assumed;
- The power costs were calculated using a unit price of C$0.06/kWh – this rate was calculated based on Hydro Québec L rate, assuming 10,000 kW of billing demand and a load factor of 0.61;
- The natural gas costs were calculated using a unit price of C$0.45/$m^3$ – this rate was calculated as the average site unit cost billed over the years 2019 and 2020 from Energir;
- A minimum building temperature of 7°C was used to compute heating costs;
- Labour is assumed to come from the local Val d'Or area and there are no accommodation and transport costs included.

Estimate Responsibilities

The overall operating cost estimate combined inputs from several sources including ASDR, BBA, Soutex, and Mine Paste as summarized in Table 21-16.

Table 21-16: OPEX estimate responsibilities

Scope	Responsible entity
Mining	BBA (WP1)
Processing, Lab & Tailings Management	Soutex (WP2)
Water Treatment	Soutex (WP2)
General and Administration	BBA (WP4)
Paste Plant	Mine Paste (WP5)

21.2.3 Mining

The mining operating costs reflect Wesdome being the owner/operator of the Kiena Mine Complex. The 2021 mining operating budget for the Kiena Mine Complex established by Wesdome is reflected as the mining operating cost for the first year of mine production in the life of mine plan. When accounting for contractor fees greatly influencing the 2021 mining operating budget, that year's budget is in alignment with the mining operating costs established for subsequent production years. Mining operating costs effective 2022 until 2028, the expected end of the life of mine, were established from first principles using firm pricing provided by Wesdome Kiena Mine Complex, budgetary pricing provided by suppliers and in-house pricing gained from previous project experience. The mining operating costs account for the following elements:

- Labour;
- Material: Consumables and Mobile Equipment Spare Parts;
- Utilities;
- Contractor Fees, predominantly in 2021 and for mine development and production of Martin Zone;
- Diamond Drilling – Definition;
- Other.

Mining labour headcount, identified in Section 21.2.8, was established to achieve the mine schedule representing the life of mine plan presented in Chapter 16 – Mining Methods. The mining labour headcount was reviewed and approved by Wesdome. Salaries established by Wesdome for the roles expected at the Kiena Mine Complex were considered.

Material consumable costs were provided by the Kiena Mine Complex as firm pricing negotiated for routine purchasing and as budgetary pricing from the respective suppliers. Mobile equipment costs for spare parts and frequency for replacement were obtained from the respective manufacturers. As such, the primary mining operating activities accounting for ore development and production mining were developed from first principles.

Utilities, including the cost of electricity for underground mobile equipment and the cost of natural gas for mine air ventilation heating requirements, were established considering the established unit rates.

Contractor fees were considered in 2021 prior to Wesdome establishing their own proprietary workforce and to account for the development and production mining of the Martin Zone. In addition, underground mobile equipment rentals and surface waste management fees were included. The contractor and rental fees are based on firm and in-house pricing.

Diamond drilling operating costs for definition purposes were established considering the annual expected drill length and applying a contractor unit cost. Both the annual drill length and unit cost were provided by Wesdome. The unit cost was scaled from firm pricing realized by Wesdome for diamond drilling conducted in 2020. Diamond drilling for exploration is deemed a capital expense not allocated to the PFS.

Other mining operating costs capture elements such as:

- Drift rehabilitation;
- Road and sump maintenance;
- Underground workshops;
- Underground crusher.

Table 21-17 summarizes the average mining operating costs established for the life of mine.

Table 21-17: Mining Operating Costs (in C$)

Item	Total LOM ($M)	Average LOM ($/tonne mined)	OPEX (%)
Labour	82.8	52.58	46.2
Material	33.0	20.98	18.4
Utilities	21.3	13.56	11.9
Contractor Fees	13.0	8.27	7.3
Diamond Drilling - Definition	6.6	4.19	3.7
Other	22.4	14.22	12.5
Total	**179.2**	**113.79**	**100.0**

21.2.4 Processing

Processing costs encompass the operation and maintenance of the processing plant, the operation of the assay lab, and the pumping costs to and from the tailings management facility. The processing plant will be operated intermittently at a feed throughput of 1,000 tpd on a weekly schedule in such a way that it will be operating only the time required to process the mine production. During shut down periods, all process mixing tanks will stay full of slurry and agitated; both thickeners will be put in recirculation mode while the tailings and water lines located outdoors will be kept running, especially during low temperature periods to prevent freezing. The manpower and electricity cost are thus based on this strategy meaning that some equipment will consume electric power and need operators' attention all year round.

The major costs broken down by item within the processing category are shown in Table 21-18. The processing costs are estimated to be $44.5M over the LOM. The processing cost on a per tonne mined basis is $28.25/t mined (LOM).

The greatest cost within the processing category is labour, representing approximately 40%, while reagents represent the second greatest cost accounting for 29% of the total. Mill grinding consumables as well as electricity each represent approximately 10% of the processing costs.

Table 21-18: Processing, lab and tailings management operating costs

Item	Total LOM ($M)	Average LOM ($/tonne mined)	OPEX (%)
Manpower	17.5	11.12	39.4
Electricity	4.5	2.84	10.0
Services	1.7	1.07	3.8
Reagents	12.9	8.18	29.0
Grinding Media and Liners	4.9	3.13	11.1
Maintenance Supplies	2.0	1.28	4.5
Operation Supplies	0.2	0.11	0.4
Other Costs	0.8	0.51	1.8
Total	**44.5**	**28.25**	**100.0**

The labour component of the processing costs is based on the labour plan for the life of mine that can be found in Section 21.2.8, and includes the staff from the processing department. The employee total varies from 21 to 31 over the life of mine, depending on the mill schedule. The number of employees allows having the necessary work force during weekly shut-down periods.

The electricity cost was estimated from the equipment list using all connected running equipment power. The power for the two grinding mills was based on the effective power demand that is related to the feed throughput. The grinding power consumed during the bulk sample processing allows obtaining an accurate estimate. The power required during shut-down periods is based on the equipment that is required to be kept in operation.

The reagents cost is based on consumption and current prices. The consumption is based on metallurgical testwork combined with the operation strategy. The price of the reagents is based on what was paid during the bulk sample processing for most of the reagents. Since there was no cyanide destruction during this period, the price of the reagents related to cyanide destruction was based on budget prices obtained from suppliers. The services include only the natural gas that is used for process building heating and for the carbon regeneration furnace. The consumption is based on historical data. The grinding media and liners consumption is based on metallurgical testing; the bulk sample processing and the prices are based on quotations received for the bulk sample processing needs. The maintenance and operation supplies costs are based on historical values adjusted to current prices. The other costs mainly include contractor services required for special maintenance activities such as mill relining. The estimation is based on historical values.

21.2.5 Water Treatment

Water treatment costs encompass the operation and maintenance of the effluent treatment plant (ETP). The effluent treatment plant is leased for the LOM and annual leasing costs have been capitalized. The calculation of the operating costs for the effluent treatment plant is divided into two components:

1. The first component includes electricity and reagents. It is based on a calculated unit cost of C$1.05 per m^3 of treated water and a yearly average treated volume of 621,000 m^3.
2. The second component includes the labour for the operation and maintenance of the ETP and is based on the labour plan from Section 21.2.8.

The major costs, broken down by item within the water treatment category, are shown in Table 21-19. The water treatment costs are estimated to be $5.6M over the LOM. The processing cost on a per tonne mined basis is $3.55/t mined (LOM).

The greatest cost within the water treatment category is reagents, representing approximately 71% of the operating costs.

Table 21-19: Water Treatment operating costs

Item	Total LOM ($M)	Average LOM ($/tonne mined)	OPEX (%)
Electricity	0.6	0.38	10.8
Reagents	3.9	2.51	70.7
Labour	1.0	0.66	18.6
Total	**5.6**	**3.55**	**100.0**

21.2.6 Paste Plant

The paste plant operating costs encompass the operation and maintenance of the new paste plant as well as the supplies for the underground paste distribution system. The labour required for backfilling operations underground is included with the mining operating costs in Section 21.2.3.

The calculation of the operating costs for the paste plant is divided into three components:

1. The first component includes consumables, services and utilities (sand, binder at 2.5%, flocculant, maintenance parts and materials, piping, boreholes and casing, barricades, and power). It is based on a calculated cost per tonne of C$11.02 per tonne of dry paste backfill and varies for each year of the life of mine according to the backfill tonnage requirements;
2. The second component includes costs that are considered fixed for each year of operation of the paste plant and includes the labour for the operation and maintenance of the paste plant as well as heating costs (natural gas) for the paste plant building. The labour costs are based on the labour plan from Section 21.2.8, and as the labour requirements are low (1.30 full time equivalent), all positions for this area are shared with other departments (Processing, Maintenance, and Engineering);
3. The third component is the 1% binder addition to the tailings that are to be deposited in the North Zone (behind the barricades), as described in Chapter 16 – Mining Methods.

The major costs broken down by item within the paste plant cost area are shown in Table 21-20. The paste plant costs are estimated to be $8.8M over the LOM. The paste plant cost on a per tonne mined basis is $5.58/t mined (LOM).

The greatest costs within the paste plant area are binder and sand, representing respectively 48% and 31% of the total.

Table 21-20: Paste backfill operating costs

Item	Total LOM ($M)	Average LOM ($/tonne mined)	OPEX (%)
Sand	2.7	1.75	31.3
Front-end Loader	0.4	0.24	4.3
Binder	3.7	2.32	41.6
Other Consumables and Electricity	0.4	0.25	4.5
Heating	0.3	0.17	3.0
Labour	0.8	0.48	8.7
Binder to North Zone	0.6	0.38	6.7
Total	**8.8**	**5.58**	**100.0**

21.2.7 General and Administration

General and Administrative (G&A) costs encompass all the items not included in the mining, processing, water treatment, and backfill costs of the Project.

The supplies and services component of the G&A costs is based on inflation-adjusted historical data from the Kiena Mine Complex provided by Wesdome for the years 2011 and 2012, which are the last two full calendar years of operation before production stopped. The supplies and services component of the G&A area includes the following items:

- Administration: general insurance, IT, property taxes, office furniture, telephone, refining cost, permits, mobile equipment insurance, memberships and association fees, internet, travel expenses, meals and entertainment fees, warehousing, subscription fees, management, and internships/co-op students;
- Corporate: financing fees, bank fees, long-term debt interest;
- Human Resources: HR services, employee relations, collective agreement, recruitment fees, public relations, donations and sponsorships, retirement committee;
- Security, HSE and Training: security services, safety materials, first aid and medical tests, mine rescue, major events, safety rewards, prevention services, courses and workshops, underground and surface radio, emergency measures, training services, customized training, conferences, health and safety committee;
- Surface Operations: mobile surface equipment, mine - camp utilities, surface structures, wheel loaders, electric compressors, surface workshop, yard maintenance, powerline maintenance, road maintenance, maintenance supervision, transport vehicles, generators, camp sanitation services;
- Technical Services: engineering, surveying, geology, environment.

The labour component of the G&A costs is based on the labour plan for the life of mine that can be found in Section 21.2.8, and includes the staff from the Purchasing, Health & Safety, Management, Geology, Engineering, Human Resources, Finance, and Environment departments. The employee total for the overall General and Administration services is about 40 over the life of mine. The G&A staff will work 40 hours per week on day shift.

The major costs broken down by item within the General and Administration category are shown in Table 21-21. The General and Administration costs are estimated to be $57.5M over the LOM. The G&A cost on a per tonne mined basis is $36.53/t mined (LOM).

The greatest cost within the G&A category is labour, representing approximately 61%, while administrative supplies and services (insurance, IT, property taxes, etc.) is the second greatest cost, accounting for approximately 13%. HSE and Training as well as Surface Ops each represent approximately 10% of the G&A costs.

Table 21-21: General and Administration operating costs

Item	Total LOM ($M)	Average LOM ($/tonne mined)	OPEX (%)
Supplies and Services - Admin	7.5	4.75	13.0
Supplies and Services - Corporate	0.2	0.14	0.4
Supplies and Services - HR	2.8	1.77	4.8
Supplies and Services – Security, HSE and Training	5.9	3.73	10.2
Supplies and Services - Surface Ops	5.8	3.66	10.0
Supplies and Services - Technical Services	0.4	0.25	0.7
Labour	35.0	22.23	60.9
Total	**57.5**	**36.53**	**100.0**

21.2.8 Site Personnel – All Areas

The total site workforce will peak in 2024 with a count of 225. The personnel list for the life of mine is presented in Table 21-22.

Table 21-22: Personnel list – All areas

Department	Job title	Shift	#Shifts	Employee Count							
				2021	2022	2023	2024	2025	2026	2027	2028
Purchasing	Purchaser	5/8	1	1	1	1	1	1	1	1	1
Purchasing	Warehouse Clerk	5/8	1	2	2	2	2	2	2	2	2
Purchasing	Contract Manager	5/8	1	1	1	1	1	1	1	1	1
Purchasing	Warehouse Manager	5/8	1	1	1	1	1	1	1	1	1
Purchasing	Purchasing Superintendent	5/8	1	1	1	1	1	1	1	1	1
Maintenance	Drill specialist	4/10.5	1	1	1	1	1	1	1	1	1
Maintenance	Underground electrician	7/10.5	2	4	4	4	4	4	4	4	4
Maintenance	Surface electrician	4/10.5	1	2	2	2	2	2	2	2	2
Maintenance	Stationary mechanic	5/8	1	1	1	1	1	1	1	1	1
Maintenance	Surface mechanic	5/8	1	1	1	1	1	1	1	1	1
Maintenance	Carpenter	5/8	1	1	1	1	1	1	1	1	1
Maintenance	Hoist operator	7/12	2	4	4	4	4	4	4	4	4
Maintenance	Welder	7/10.5	1	2	2	2	2	2	2	2	2
Maintenance	Instrumentation technician	5/8	1	1	1	1	1	1	1	1	1
Maintenance	Underground mechanic	7/10.5	2	17	19	21	23	23	19	19	9
Maintenance	Surface loader operator	4/10.5	1	2	2	2	2	2	2	2	2
Maintenance	Maintenance superintendent	5/8	1	1	1	1	1	1	1	1	1
Maintenance	Electrical supervisor	5/8	1	1	1	1	1	1	1	1	1
Maintenance	Maintenance supervisor	5/8	1	1	1	1	1	1	1	1	1
Maintenance	Maintenance planner	5/8	1	1	1	1	1	1	1	1	0
Health & Safety	HSE Advisor	5/8	1	1	1	1	1	1	1	1	1
Health & Safety	Instructor	5/8	1	1	1	1	1	1	1	1	1
Health & Safety	Nurse	5/8	1	1	1	1	1	1	1	1	1
Health & Safety	Health & Safety Superintendent	5/8	1	1	1	1	1	1	1	1	1

Department	Job title	Shift	#Shifts	Employee Count							
				2021	2022	2023	2024	2025	2026	2027	2028
Processing	General Foreman Plant Maintenance	4/12	2	1	1	1	1	1	1	1	1
Processing	Plant Worker	4/12	2	2	2	2	3	3	3	4	2
Processing	Mechanic - Plant	4/12	2	2	2	2	3	3	3	3	2
Processing	Mechanic / Welder - Plant	4/12	2	1	1	1	1	1	1	1	1
Processing	Metallurgist	4/12	2	1	1	1	1	1	1	1	1
Processing	Grinding Operator	4/12	2	2	2	2	3	3	3	4	2
Processing	Solution Operator	4/12	2	4	4	4	4	4	4	4	4
Processing	Plant Planner	4/12	2	1	1	1	1	1	1	1	1
Processing	Plant Superintendent	4/12	2	1	1	1	1	1	1	1	1
Processing	Metallurgy Technician	4/12	2	1	1	1	1	1	1	1	1
Processing	Lab Manager	5/8	1	1	1	1	1	1	1	1	1
Processing	Lab Sample Prep Technician	5/8	1	2	2	2	4	4	4	4	2
Processing	Lab Fire Assay Technician	5/8	1	1	1	1	2	2	2	2	1
Processing	Lab Technician	5/8	1	1	1	1	2	2	2	2	1
Processing	Operator - Paste Plant	4/12	2	0	2	2	2	2	2	2	2
Processing	Operator - ETP	7/6	1	0	1	1	1	1	1	1	1
Management	Continuous Improvement Coordinator	5/8	1	1	1	1	1	1	1	1	1
Management	General Manager	5/8	1	1	1	1	1	1	1	1	1
Management	Project Manager	5/8	1	1	1	1	0	0	0	0	0
Management	IT Technician	5/8	1	1	1	1	1	1	1	1	1
Geology - Surface	Chief Surface Exploration Geologist	5/8	1	1	1	1	1	1	1	1	1
Geology - Surface	Core Geologist	5/8	1	1	1	1	1	1	1	1	1
Geology - Surface	Senior Surface Exploration Geologist	5/8	1	1	1	1	1	1	1	1	1
Geology - Surface	Surface Exploration GIT	5/8	1	1	1	1	1	1	1	1	1
Geology - Surface	Geology Technician	5/8	1	1	1	1	1	1	1	1	1
Geology - Resources	Senior Engineer - Mineral Resources	5/8	1	1	1	1	1	1	1	1	1

Department	Job title	Shift	#Shifts	Employee Count							
				2021	2022	2023	2024	2025	2026	2027	2028
Geology - Production	Chief Production Geologist	5/8	1	1	1	1	1	1	1	1	1
Geology - Production	Production Geologist	5/8	1	1	1	1	1	1	1	1	1
Geology - Production	Senior Production Geologist	5/8	1	1	1	1	1	1	1	1	1
Engineering	Engineering Coordinator	5/8	1	1	1	1	1	1	1	1	1
Engineering	Mining Engineer 1	5/8	1	1	1	1	1	1	1	1	1
Engineering	Mining Engineer 2	5/8	1	1	1	1	1	1	1	1	1
Engineering	EIT	5/8	1	1	1	1	1	1	1	1	1
Engineering	Mining Technician 1	5/8	1	1	1	1	1	1	1	1	1
Engineering	Mining Technician 2	5/8	1	1	1	1	1	1	1	1	1
Engineering	Mining Technician Draftsman	5/8	1	1	1	1	1	1	1	1	1
Engineering	Mining Technician Lt	5/8	1	1	1	1	1	1	1	1	1
Human Resources	HR & Administration Agent	5/8	1	1	1	1	1	1	1	1	1
Human Resources	Director, Administration & HR	5/8	1	1	1	1	1	1	1	1	1
Finance	Accounts Payable Agent	5/8	1	1	1	1	1	1	1	1	1
Finance	Accountant	5/8	1	1	1	1	1	1	1	1	1
Finance	Financial Controller	5/8	1	1	1	1	1	1	1	1	1
Finance	Payroll Clerk	5/8	1	1	1	1	1	1	1	1	1
U/G Operations	Truck operators	7/10.5	4	12	16	24	28	28	20	20	4
U/G Operations	Jumbo operators	7/10.5	4	4	8	8	8	8	4	4	0
U/G Operations	LHD operators	7/10.5	4	12	12	12	12	12	8	8	0
U/G Operations	Production drill operators	7/10.5	4	4	8	8	12	12	12	12	4
U/G Operations	Bolter & scissor lift operators	7/10.5	4	12	12	12	12	8	4	4	4
U/G Operations	Emulsion loader operators & blasters	7/10.5	2	4	4	4	4	4	4	4	0
U/G Operations	Personnel carrier & boom truck operator	7/10.5	4	4	4	4	4	4	4	4	4
U/G Operations	Grader operator	7/10.5	4	4	4	4	4	4	4	4	4
U/G Operations	Cement transmixer operator / backfill operator	7/10.5	4	4	4	4	4	4	4	4	4

Department	Job title	Shift	#Shifts	Employee Count							
				2021	2022	2023	2024	2025	2026	2027	2028
U/G Operations	Rehabilitation miner	7/10.5	2	2	2	2	2	2	2	2	0
U/G Operations	Construction - Miner 3	4/10.5	4	4	4	4	4	4	4	4	0
U/G Operations	U/G Superintendent	5/8	4	1	1	1	1	1	1	1	1
U/G Operations	Mine Captain	5/8	4	1	1	1	1	1	1	1	0
U/G Operations	U/G Operations Supervisor	7/10.5	4	4	4	4	4	4	4	4	4
U/G Operations	Cage Attendant - Miner 2	7/10.5	4	4	4	4	4	4	4	4	4
U/G Operations	Deckman - Miner 2	7/10.5	4	2	2	2	2	2	2	2	2
U/G Operations	U/G Operations Clerk	7/10.5	4	1	1	1	1	1	1	1	1
U/G Operations	Dry Attendant	7/10.5	4	2	2	2	2	2	2	2	2
Environment	Environmental Coordinator	5/8	1	1	1	1	1	1	1	1	1
Environment	Environmental Superintendent	5/8	1	1	1	1	1	1	1	1	1
Environment	Environmental Technician	5/8	1	1	1	1	1	1	1	1	1
			Total	**182**	**199**	**209**	**225**	**221**	**197**	**199**	**132**

22. ECONOMIC ANALYSIS

The economic/financial assessment of the Kiena Mine Complex project for Wesdome Gold Mines was carried out using a discounted cash flow approach on a pre-tax and after-tax basis, based on a gold price of $1,600 in United States currency and cost estimates in Canadian currency. An exchange rate of USD 1.00 per CAD 1.32 was assumed to convert USD market price projections and components of the capital cost estimates into Canadian dollars (CAD), based on the three-year trailing average as of Q4 2020. No provisions were made for the effects of inflation.

Future tax liabilities were estimated by using a simplified tax model provided by Wesdome. This model uses current Canadian tax regulations to assess the corporate tax liabilities and the most recent provincial regulations to assess the Québec mining tax liabilities.

The internal rate of return (IRR) on total investment was calculated based on 100% equity financing, even though Wesdome may decide in the future to finance part of the Project with debt. The net present value (NPV) was calculated from the cash flow generated by the Project, based on a discount rate of 5%. The payback period after the start of operations, which is based on the undiscounted annual cash flow of the Project, is also indicated as a financial measure. Furthermore, a sensitivity analysis has been performed for the after-tax base case to assess the impact of variations in the Project capital costs, operating costs, CAD to USD exchange rate, price of gold, average grade, and discount rate.

The economic analysis presented in this section contains forward-looking information about the mineral resource estimates, commodity prices, exchange rates, proposed mine production plan, projected recovery rates, operating costs, construction costs and project schedule. The results of the economic analysis are subject to several known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

22.1 Assumptions and Basis

The economic analysis was performed using the following assumptions and basis:

- The deposits included are the resources located within the Kiena mine area that contain enough value for development and mining costs, as presented in the production plan (see Chapter 16);
- The project execution schedule developed in Chapter 24, taking into consideration key project milestones;
- The mine is currently expected to start commercial production from the third quarter (Q3) of 2021 using existing equipment and facilities; however, the financial analysis assumes that all capital expenditures in 2021 are sustaining and that there is no preproduction period;

- US$ 1,600/oz. gold price was used in the cashflow model;
- The Canadian to United States dollar exchange rate has been assumed to be USD 1.00: CAD 1.32 over the life of mine (CAD:USD exchange rate of 0.76), corresponding to the three-year trailing average as of Q4 2020;
- All cost estimates are in constant Q4 2020 Canadian dollar with no inflation or escalation factors considered;
- All gold is assumed sold in the same year it is produced;
- Project revenue is derived from the sale of gold into the international marketplace.

This financial analysis was performed on both a pre-tax basis and after-tax basis. The general assumptions used for this financial model, LOM plan tonnage and grade estimates are summarized in Table 22-1.

Table 22-1: Financial model parameters

Description	Unit	Value
Long Term Gold Price	USD/oz	1,600
Exchange Rate	USD:CAD	1.32
Discount Rate	%	5.00
Mine Life	year	7.1
Total Mined	Mt	1.6
Gold Grade	g/t	11.89
Process Overall Gold Recovery	%	98.4
Mining	$/t mined	113.79
Processing, Lab & Tailings Management	$/t mined	28.25
Water Treatment	$/t mined	3.55
General and Administration	$/t mined	36.53
Paste Plant	$/t mined	5.58
Royalties	% NSR	0.00
Pre-production Capital Costs	$M	0.0
Sustaining Capital Costs	$M	229.8
Reclamation and Closure Costs	$M	2.0

22.2 Gold Production

Over the life of mine, a total of 592,113 oz of payable gold will be produced (average annual: 83,574 oz). Figure 22-1 provides a summary of the payable gold production by year.

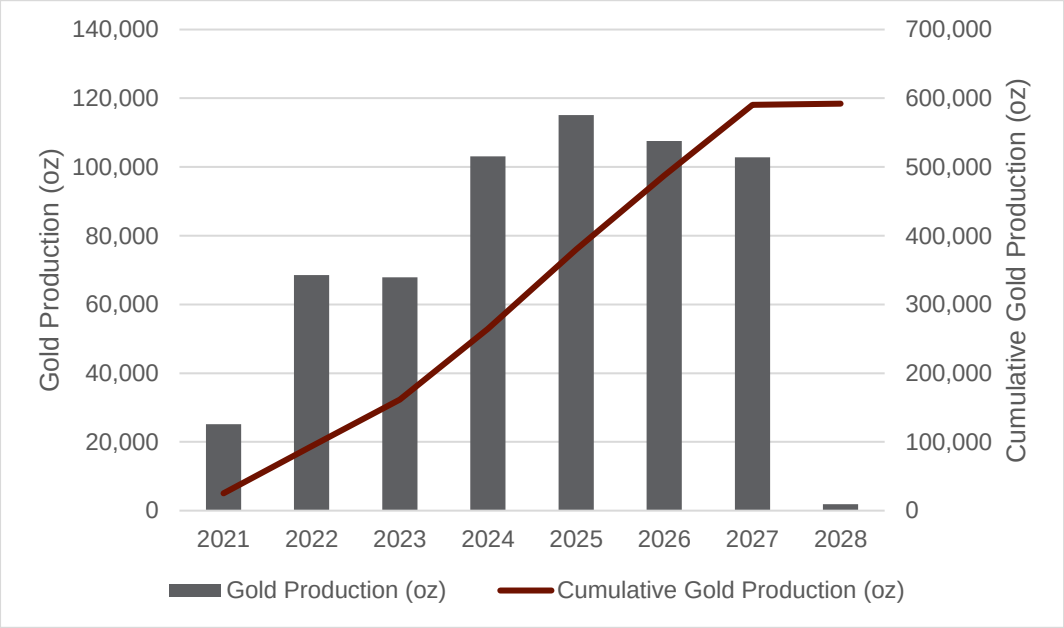


Figure 22-1: Annual payable gold production (oz)

22.3 Capital and Sustaining Costs

All capital costs (pre-production, sustaining, reclamation and closure) for the Project have been distributed against the development schedule to support the economic cash flow model. Figure 22-2 presents the planned annual and cumulative LOM capital cost profile.

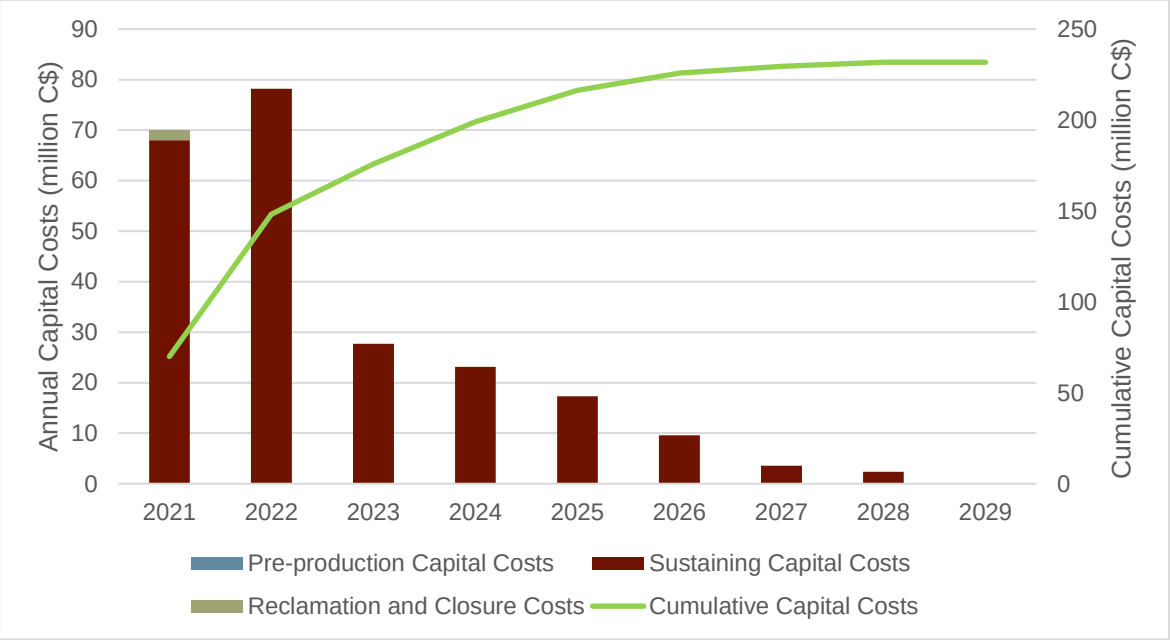


Figure 22-2: Overall Kiena Mine Complex project capital cost profile

22.4 Royalties

Based on information from Wesdome's audited 2020 financial statement, there are no royalties applicable to the Kiena Mine Complex project.

22.5 Taxation

The taxation calculations for the Project were made based on a simplified spreadsheet model that was provided by Wesdome.

At the effective date of the Report, it has been determined that the Kiena Mine Complex project will be subject to the following tax regime:

- The Canadian Corporate Income Tax system consists of the federal income tax (modelled at a rate of 15%) and the provincial (Québec) income tax (modelled at a rate of 11.5%);
- The Québec Mining Tax was modelled using a statutory tax rate of 16.5%.

The tax calculations are underpinned by the following key assumptions:

- The calculations are based on the tax regime at the date of the PFS. Future changes in tax laws could impact the calculations;

- No tax attributes have been considered for the calculation of income taxes based on information provided by Wesdome;
- A total of $98.4M in tax attributes has been considered for the calculation of mining taxes based on information provided by Wesdome;
- The Project is held 100% by a corporate entity and the after-tax analysis does not attempt to reflect any future changes in corporate structure or property ownership;
- Assumes 100% equity financing and therefore does not consider interest and financing expenses;
- Québec Mining Tax is deductible for federal and provincial income tax purposes;
- Actual taxes payable will be affected by corporate activities, and current and future tax benefits, with respect that these activities have not been considered.

The combined effect on the Project of the two levels of taxation, including the elements described above, is an approximate cumulative effective tax rate of 35%, based on project earnings (earnings before interest and tax or EBIT). It is anticipated, based on the Project assumptions, that Wesdome will pay approximately $174M in Canadian Corporate Income Tax and $78M in Québec Mining Tax over the life of the Project.

22.6 Financial Analysis Summary

A 5% discount rate was applied to the cash flow to derive the NPV for the Project on a pre-tax and after-tax basis. Cash flows have been discounted to January 1, 2021 under the assumption that the Project starts in Q1 2021 and that major project financing would be carried out at this time. The summary of the financial evaluation for the base case of the Project is presented in Table 22-2.

Table 22-2: Financial analysis summary (pre-tax and after-tax)

Description		Unit	Base case
Pre-tax	Net Present Value (0% disc)	$M	722.8
	Net Present Value (5% disc)	$M	569.1
	Internal Rate of Return	%	135.2
	Payback Period (after start of operations)	year	2.2
After-tax	Net Present Value (0% disc)	$M	470.6
	Net Present Value (5% disc)	$M	366.6
	Internal Rate of Return	%	98.2
	Payback Period (after start of operations)	year	2.7

The pre-tax base case financial model resulted in an IRR of 135.2% and an NPV of $569.1M with a discount rate of 5%. The pre-tax payback period after the start of operations is 2.2 years. On an after-tax basis, the base case financial model resulted in an IRR of 98.2% and an NPV of $366.6M with a discount rate of 5%. The after-tax payback period after the start of operations is 2.7 years.

The summary of the Kiena Mine Complex project discounted cash flow financial model (pre-tax and after-tax) is presented in Table 22-3.

Table 22-3: Kiena Mine Complex project financial model summary

Year		1	2	3	4	5	6	7	8	9	Total
	Unit	2021	2022	2023	2024	2025	2026	2027	2028	2029	
Production Summary											
Total Tonnes Processed	kt	76.0	161.2	198.1	277.5	257.9	255.6	337.2	11.0	0.0	1,574.4
Head Grade Au	g/t	11.43	13.44	10.82	11.74	14.10	13.30	9.65	5.52	0.00	11.89
Payable Gold	koz	25.2	68.6	67.9	103.1	115.1	107.6	102.8	1.9	0.0	592.1
Revenue											
Gold Price	USD	1600.00	1600.00	1600.00	1600.00	1600.00	1600.00	1600.00	1600.00	1600.00	
Exchange Rate	USD:CAD	1.32	1.32	1.32	1.32	1.32	1.32	1.32	1.32	1.32	
Gross Revenue	$M	53.2	144.8	143.3	217.6	243.0	227.2	217.0	4.0	0.0	1,250.1
Operating Expenditures											
Mining	$M	17.5	22.5	23.9	28.6	27.1	28.3	29.8	1.4	0.0	179.2
Processing, Lab & Tailings Management	$M	3.6	5.3	5.8	7.3	7.0	7.0	8.0	0.5	0.0	44.5
Water Treatment	$M	0.0	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.0	5.6
General and Administration	$M	8.2	8.2	8.2	8.0	8.0	8.0	8.0	0.7	0.0	57.5
Paste Plant	$M	0.0	0.8	1.4	1.4	1.7	1.6	1.7	0.1	0.0	8.8
Total Operating Costs	$M	29.3	37.6	40.1	46.1	44.7	45.7	48.4	3.5	0.0	295.5
Royalty Payments	$M	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Capital Expenditures											
Pre-production	$M	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Sustaining	$M	68.0	78.2	27.7	23.2	17.3	9.6	3.6	2.3	0.0	229.8
Reclamation and Closure	$M	2.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	2.0
Total Capital Costs	$M	70.0	78.2	27.7	23.2	17.3	9.6	3.6	2.3	0.0	231.8
Changes in Working Capital	$M	-0.2	1.4	-0.1	1.2	0.5	-0.3	-0.3	-2.2	0.1	0.0
Pre-tax Cash Flow											
Pre-tax Cash Flow	$M	-45.9	27.6	75.6	147.1	180.5	172.2	165.3	0.4	-0.1	722.8
Cumulative Pre-tax Cash Flow	$M	-45.9	-18.4	57.2	204.4	384.9	557.1	722.4	722.8	722.8	
Taxes and Duties											
Mining Tax	$M	-0.9	-1.0	8.2	16.5	20.1	18.8	18.0	-1.7	0.0	77.9
Income Tax	$M	0.1	17.6	15.0	31.8	39.1	36.4	34.6	-0.3	0.0	174.2
Total Taxes and Duties	$M	-0.8	16.6	23.1	48.3	59.2	55.2	52.6	-2.0	0.0	252.2
After-tax Cash Flow											
After-tax Cash Flow	$M	-45.1	10.9	52.5	98.9	121.3	117.1	112.7	2.5	-0.1	470.6
Cumulative After-tax Cash Flow	$M	-45.1	-34.2	18.3	117.1	238.4	355.5	468.2	470.7	470.6	

Figure 22-3 shows the cumulative cash flows for the Project projected for the LOM on a pre-tax and after-tax basis.

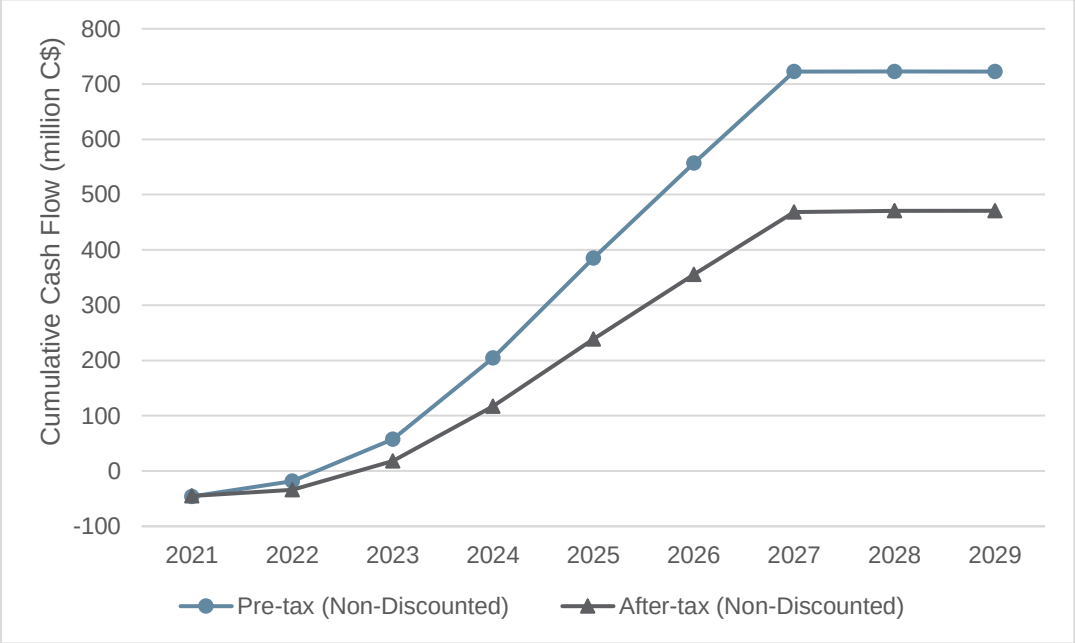


Figure 22-3: Life of mine cash flow projection (cumulative, pre-tax and after-tax)

22.7 Production Costs

A summary of the Project's production costs is provided in Table 22-4. All costs are in USD. Total cash costs are calculated per ounce on a payable basis using the costs of mining, processing, lab, tailings management, water treatment, on-site G&A, paste plant, refining and smelting, and royalties.

The LOM operating cash cost per ounce is USD 380/oz Au (CAD 502/oz Au). The LOM all-in sustaining cost (AISC) per ounce is USD 676/oz Au, derived from the total cash costs plus sustaining capital and closure costs. The operating margin over the LOM has been estimated to be USD 1,220/oz Au based on a gold price of USD 1,600/oz.

Table 22-4: Production cost summary

Description	Unit	LOM
Metal Production		
Gold payable	Moz	0.592
Costs and Royalties		
Mining	US$M	135.6
Processing, Lab & Tailings Management	US$M	33.7
Water Treatment	US$M	4.2
General and Administration	US$M	43.5
Paste Plant	US$M	6.6
Refining and Smelting	US$M	1.3
Royalties	US$M	0.0
Total Operating Cost	US$M	225.0
AISC Costs and Profit Margins (per oz payable)		
Gold Price	US$/oz	1,600
Cash Cost (Operating)	US$/oz	380
Sustaining and Closure Costs	US$M	175.4
Total Costs (Operating and Sustaining)	US$M	400.4
All-in Sustaining Costs	US$/oz	676
Operating Margin	US$/oz	1,220

22.8 Sensitivity Analysis

A financial sensitivity analysis was conducted on the base case after-tax cash flow NPV ($M) and IRR of the Project, using the following variables: capital costs, operating costs, CAD:USD exchange rate, price of gold, average grade, and discount rate. The after-tax results for the Project IRR and NPV ($M) based on the sensitivity analysis are summarized in Table 22-5 through Table 22-10.

Table 22-5: NPV sensitivity results (after-tax) for metal price and exchange rate variations

CAD:USD	Gold Price (USD/ounce)								
	1,200	1,300	1,400	1,500	1600	1,700	1,800	1,900	2,000
0.60	330.2	383.1	435.2	487.8	540.5	593.2	645.9	698.6	751.3
0.65	281.1	330.2	379.1	427.2	475.7	524.3	572.9	621.5	670.2
0.70	237.5	284.6	330.2	375.7	420.3	465.3	510.4	555.5	600.7
0.76	197.2	238.7	282.3	324.5	366.6	408.0	449.5	491.2	532.9
0.80	170.1	209.4	248.9	290.3	330.2	370.0	409.2	448.3	487.8
0.85	141.8	179.6	216.3	253.6	292.6	330.2	367.7	404.6	441.4
0.90	116.6	152.3	187.8	222.4	257.7	294.7	330.2	365.6	400.5

Table 22-6: NPV sensitivity results (after-tax) for capital (LOM) and operating costs variations

OPEX	CAPEX						
	-30%	-20%	-10%	0%	10%	20%	30%
-30%	477.2	456.2	435.1	414.1	393.0	372.0	350.9
-20%	461.4	440.4	419.3	398.3	377.2	356.2	335.1
-10%	445.6	424.6	403.5	382.5	361.4	340.4	319.3
0%	429.8	408.7	387.7	366.6	345.6	324.5	303.5
10%	413.5	392.4	371.4	350.3	329.3	308.2	287.2
20%	395.7	374.6	353.6	332.5	311.5	290.4	269.4
30%	379.4	358.3	337.3	316.2	295.2	274.1	253.1

Table 22-7: IRR sensitivity results (after-tax) for metal price and exchange rate variations

CAD:USD	Gold Price (USD/ounce)								
	1,200	1,300	1,400	1,500	1,600	1,700	1,800	1,900	2,000
0.60	86.0%	103.5%	121.5%	142.4%	166.8%	195.8%	230.7%	273.7%	328.3%
0.65	71.3%	86.0%	102.2%	118.6%	137.3%	158.9%	184.0%	213.7%	249.4%
0.70	59.9%	72.3%	86.0%	101.1%	116.1%	133.1%	152.4%	174.6%	200.3%
0.76	49.8%	60.1%	71.6%	84.2%	98.2%	111.8%	126.8%	143.9%	163.1%
0.80	43.4%	52.8%	62.8%	73.9%	86.0%	99.3%	112.2%	126.4%	142.4%
0.85	36.8%	45.7%	54.5%	64.1%	74.6%	86.0%	98.5%	110.7%	123.7%
0.90	31.0%	39.2%	47.6%	56.0%	65.2%	75.2%	86.0%	97.8%	109.3%

Table 22-8: IRR sensitivity results (after-tax) for capital (LOM) and operating costs variations

OPEX	CAPEX						
	-30%	-20%	-10%	0%	10%	20%	30%
-30%	283.6%	197.7%	150.5%	120.6%	99.8%	84.4%	72.5%
-20%	250.9%	180.0%	139.2%	112.5%	93.7%	79.5%	68.4%
-10%	224.2%	164.7%	129.0%	105.1%	87.9%	74.8%	64.5%
0%	201.4%	151.0%	119.6%	98.2%	82.4%	70.3%	60.6%
10%	177.8%	136.4%	109.6%	90.6%	76.5%	65.4%	56.5%
20%	158.3%	123.7%	100.5%	83.7%	70.9%	60.8%	52.5%
30%	142.0%	112.7%	92.4%	77.4%	65.8%	56.5%	48.8%

Table 22-9: NPV sensitivity results (after-tax) for average grade over ±15% range

	Average Grade (g/t)						
	10.10	10.70	11.29	11.89	12.48	13.07	13.67
NPV	264.0	299.2	332.9	366.6	399.7	432.7	466.0

Table 22-10: NPV sensitivity results (after-tax) for discount rate

	Discount Rate				
	0%	3%	5%	8%	10%
NPV	470.6	404.5	366.6	317.4	288.9

The graphical representations of the financial sensitivity analysis are depicted below in Figure 22-4 for the Project's NPV and Figure 22-5 for the Project's IRR.

The sensitivity analysis reveals that the price of gold, the average gold grade and the exchange rate have the most significant influence on the NPV compared to the other parameters, based on the range of values evaluated. By comparison, variations in the capital and operating costs have a limited influence on the NPV.

The IRR is most impacted by variations in the capital costs and the price of gold. Variations in the exchange rate and average gold grade also impact the IRR but to a lesser extent, while the effect of changes to the operating costs are limited.

Overall, it should be noted that the economic viability of the Project will not be significantly impacted by variations in the capital or operating costs, and that the NPV and IRR of the Project are positive over the range of values used for the sensitivity analysis when analyzed individually.

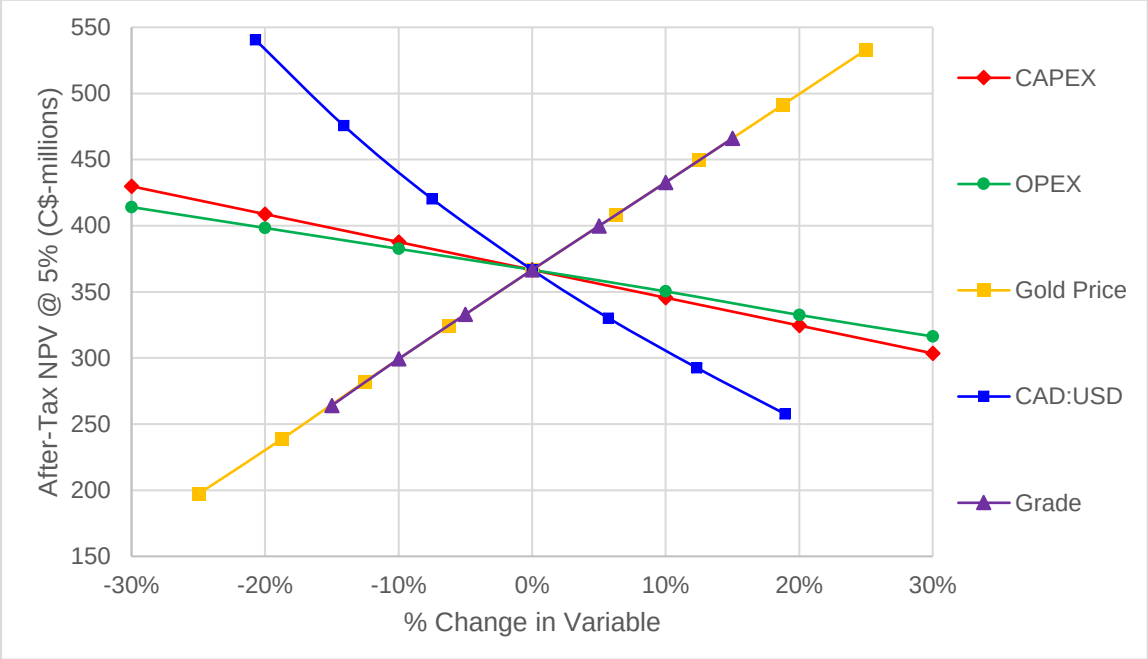


Figure 22-4: Sensitivity of the net present value (after-tax) to financial variables

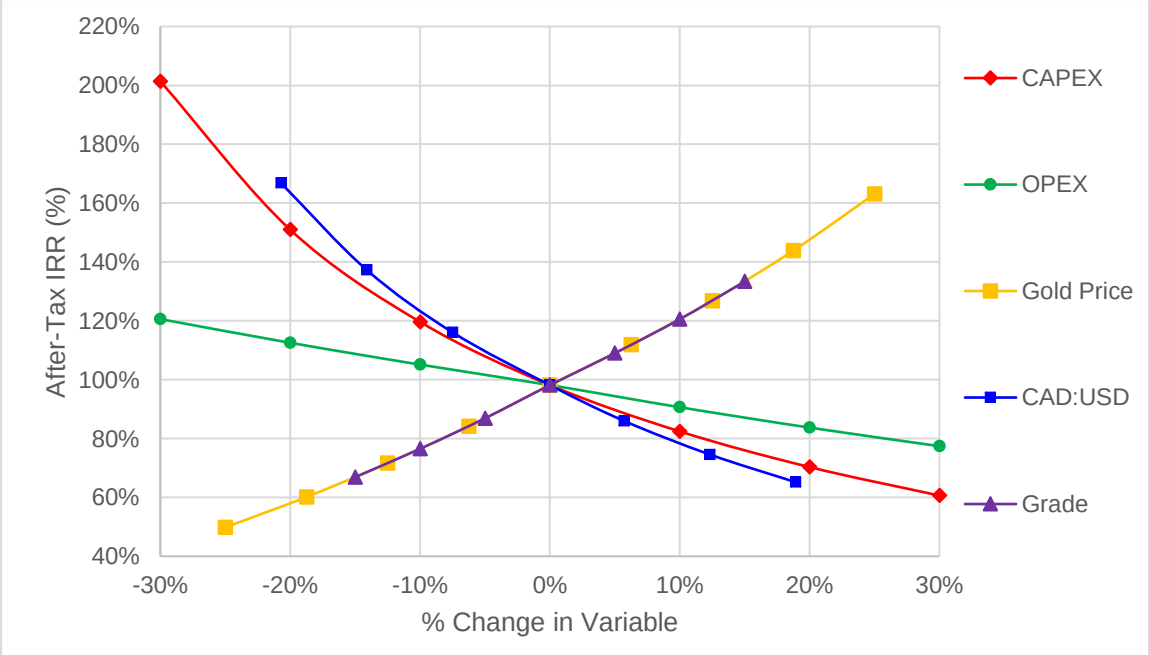


Figure 22-5: Sensitivity of the internal rate of return (after-tax) to financial variables

23. ADJACENT PROPERTIES

The Kiena Mine Complex is located in the Val-d'Or mining camp as illustrated in Figure 23-1. Several mining and junior exploration companies are active in the area of the Kiena Mine Complex. Pierre-Luc Richard, P. Geo., has not been able to verify some of the information presented below and the information is not necessarily indicative of the mineralization on the Kiena Mine Complex Property area (the subject of this Report). Some information was also taken from the previous technical report of Beausoleil et al. (2019).

23.1 Canadian Malartic Property

In June 2014, Yamana Gold and Agnico Eagle Mines bought all the common shares from Osisko. They created the Canadian Malartic Partnership. It is located approximately 6 km west of the Project. As of December 2019, proven and probable mineral reserves are estimated at 4.78 Moz Au (Agnico Eagle Mines-Canadian Malartic, 2020).

Pierre-Luc Richard, QP, has not been able to verify the information presented above and the information is not necessarily indicative of the mineralization on the Kiena Mine Complex Property area.

The Canadian Malartic Property lies at the southern margin of the eastern portion of the Archean Abitibi volcanic belt, mainly within the Pontiac Group of metasedimentary rocks. The property covers a 16-km-long section of the Larder Lake Cadillac Fault Zone (LLCFZ) and is underlain by mafic and ultramafic metavolcanic rocks of the Piché Group cut by intrusions, as well as metasediments of the Cadillac Group north of the fault zone.

The Canadian Malartic mine is a large-tonnage, low-grade Archean gold system, consisting of a broad shell of disseminated gold-bearing pyrite mineralization hosted by porphyritic felsic to intermediate intrusions and altered metasediments. The system is open to the west and to the south at depth. Mineralization in the Canadian Malartic Extension (Barnat deposit) is largely along the southern edge of the LLCFZ. The two deposits contain the bulk of the current reserves and are part of one large pit. The Jeffrey and Gouldie deposits, a few hundred metres east and south of the pit, respectively, contain some of the mineral resources (Agnico Eagle Mines-Canadian Malartic, 2019).

23.2 Dubuisson JV Property

The Dubuisson Property is a joint venture between Probe Metals Inc. and Agnico Eagle Mines. It is located in Dubuisson Township to the south of the Project and consists of 31 contiguous claims covering a surface area of approximately 748 ha.

The joint venture was announced in July 2010. Under the terms of the agreement, Agnico Eagle Mines acquired 51% of the rights, title and interest in the property for $100,000 in cash and by issuing 15,000 shares of Agnico Eagle Mines for a total value of approximately $1 million.

As part of the agreement and following the exercise of the option, Agnico Eagle Mines may acquire an additional interest, which would bring Agnico Eagle Mines share up to 70% and Probe Metals to 30% interest in the Property, by completing a bankable feasibility study. Agnico Eagle Mines will act as the operator for all exploration work carried out on the Property during the option period (Probe Metals-Dubuisson, 2019).

23.3 Agnico Eagle Mines Ltd. Properties

Agnico Eagle Mines Ltd. holds several properties and parts of historical properties northwest and south of the Project: Goldex, Joubi, Bigué, Callahan and School Mine.

The Goldex Property, adjacent to the southeast limit of the Project, has proven and probable reserves of 18.925 Mt at 1.58 g/t Au for 962,000 oz Au (Agnico Eagle Mines-Goldex 2019). It also encompasses the Quebec Explorers Gold Deposit. The property straddles a 5-km segment of the prolific LLCFZ. The gold system locally exceeds 20 m in thickness and can be traced for more than 800 m along strike. The mineralization is known to a depth of more than 1,500 m (Deep Zone 3).

Pierre-Luc Richard, QP, has not been able to verify the information presented above and the information is not necessarily indicative of the mineralization on the Kiena Mine Complex Property area.

The Goldex Property is located in the Dubuisson Township of Quebec in the southern Abitibi Greenstone Belt. The intermediate to mafic and ultramafic volcanic sequence that underlies the property dips steeply to the northeast. It is intruded by a large tabular-shaped quartz-diorite body known as the Goldex Granodiorite that also dips steeply northeast.

Goldex is a large, relatively low-grade body defined by the intensity of stockwork veins and gold grades rather than by individual veins. Most of the gold occurs as microscopic particles associated with pyrite, while the rest occurs as coarse native gold grains. There are several zones of gold mineralization with isolated mineralized intercepts over mineable widths on the property, and all of them, except the South Zone, are hosted by the Goldex Granodiorite. The M and E Zones and the Deep 1 Zone contain gold-bearing quart-tourmaline- pyrite veins and veinlets. The South Zone consists of quartz veins that have higher grades than those in the primary mineralized zones at Goldex (Agnico Eagle Mines-Goldex, 2019).

Past producers on the Goldex property include the Joubi Mine and some exploration shafts (Goldex Shaft 1 and the School Mine).

The Callahan Property, adjacent to the northwest limit of the Project, hosts the Callahan deposit in the Dubuisson Formation. In 1987, Falconbridge prepared a resource estimate. The latest major work was in 2010 when Kinross Gold Corp., as owner of the claims, conducted a diamond drilling campaign of 10,722 m (Beauregard and Gaudreault, 2010).

23.4 Tarmac Project Property

Globex Mining Enterprise holds six claims in the middle of the Project. Those claims are located west of the closed Siscoe Mine. The claims have an area of 94.1 ha.

23.5 O3 Mining Property

O3 Mining Inc. acquired Harricana River Mining Corporation as per press release August 23, 2019. O3 Mining Inc. holds 113 claims in the vicinity of the closed Sullivan Mine. Those claims are located approximately 6 km east of the Project.

During the period between 1934 and 1968, the Sullivan mine produced a total of 1,134,342 oz of gold and 293,857 oz of silver from 4,613,500 t of mineralized material grading an average 7.65 g/t Au and 1.98 g/t Ag. The veins mined at the Sullivan mine are found within the narrow west end of the Bourlamaque granodiorite batholith. The granodiorite is considerably altered, but chemically it is similar to the quartz-albite facies in the Siscoe stock (Sauvé et al., 1993).

23.6 East West Property

Emgold Mining Corporation holds seven claims located 4 km west of the Project covering an area of 184.2 ha.

In December 2019, Emgold announced it had signed a Claim Purchase and Option Agreement giving the Company to acquire up to a 50% interest in the East-West Property from a private individual. The remaining 50% interest in the Property is owned by Knick Exploration Inc. Emgold has also acquired, as part of the transaction, a second option to increase its ownership in the Property to 55%. The property is an early to advanced exploration project (Emgold news release of December 11, 2019).

23.7 Bonterra Property

Bonterra Resources Inc. holds 21 claims located approximately 6 km southeast of the Project. The claims contain some showings and deposits (Nouvelle Zone Aurifère and Zone No. 5).

23.8 Marban Block Project

The Marban Block Property held 100% by NioGold Mining Corporation (now O3 Mining is located about 15 km west of the town of Val-d'Or. The property consists of 30 claims and three mining concessions. The Marban Block has three past producing mines: Marban, Norlartic and Kierens. The Marban Project has a NI 43-101 resource estimates effective as September 3, 2020 (Raponi et al., 2020) totalling 54,1 Mt at 1.1 g/t Au for 1,914,249 oz of gold in measured + indicated resources (Raponi et al., 2020).

Pierre-Luc Richard, QP, has not been able to verify the information presented above and the information is not necessarily indicative of the mineralization on the Kiena Mine Complex Property area.

The following description of the deposits is mostly modified and summarized from Trudel and Sauvé (1992), Gustin and Ronning (2013), and references therein.

During the period between 1961 and 1974, the Marban mine produced a total of 330,000 oz of gold from 1,983,000 t of mineralized material grading an average 5.27 g/t Au (Ducharme et al., 2009).

During the periods between 1959 and 1966 and between 1990 and 1992, the Norlartic mine produced a total of 188,000 oz of gold from 1.435 Mt of mineralized material at an average of 4.07 g/t Au (Ducharme et al., 2009).

During the period between 1965 and 1966, the Norlartic North Zone produced a total of 11,000 oz of gold from 81,000 t of mineralized material grading an average 4.35 g/t Au. The North-North Zone is located 500 m northeast of the Norbenite Fault. It is a near-surface intrusive-hosted deposit with mineralized quartz-tourmaline stockwork. Gold mineralization is confined to a conformable quartz-albite-carbonate-pyrite alteration envelope with a quartz-tourmaline-carbonate vein stockwork localized in the central to lower portions of a 60-m-wide granodiorite sill (Ducharme et al., 2009).

During the periods between 1965 and 1966 and between 1988 and 1992, the Kierens Mine produced a total of 52,000 oz of gold from 251,000 t of mineralized material from the Kierens Zone with an average grade of 6.30 g/t Au (Ducharme et al., 2009).

Three styles of gold mineralization were identified within the Kierens Zone by Aur Resources geologists: 1) high-grade single-vein structures; 2) sill stockworks; and 3) laminated veins in recrystallized mafic volcanic rocks.

23.9 Siscoe East Property

The Siscoe East Property is located 3 km northwest of Val-d'Or between the former Siscoe and Sullivan mines. It hosts the former Stabell mine (Shaft No.1) that produced 13,629 oz of gold at an average grade of 0.21 oz/t and Shaft No.2 that produced 1,782 oz of gold at an average grade of 0.27 oz/t (Khobi and Frechette, 1987). The property is held 50% by NioGold Mining Corporation (now Osisko Mining Inc.) and 50% by Alexandria Minerals Corporation (now O3 Mining).

The Siscoe East Property is situated within the Malartic Group of mafic to ultramafic volcanic rocks north of the LLCFZ. The property is characterized by the contact between mafic to ultramafic volcanic rocks of the Dubuisson Formation (Lower Malartic Group) and the multi-phased granodiorite-diorite of the Bourlamaque Batholith. In the summer of 2008, Alexandria Minerals entered into a joint venture agreement with NioGold Mining allowing NioGold Mining to earn a 50% interest in the property by issuing 650,000 shares (issued) and completing $750,000 of exploration work on the property.

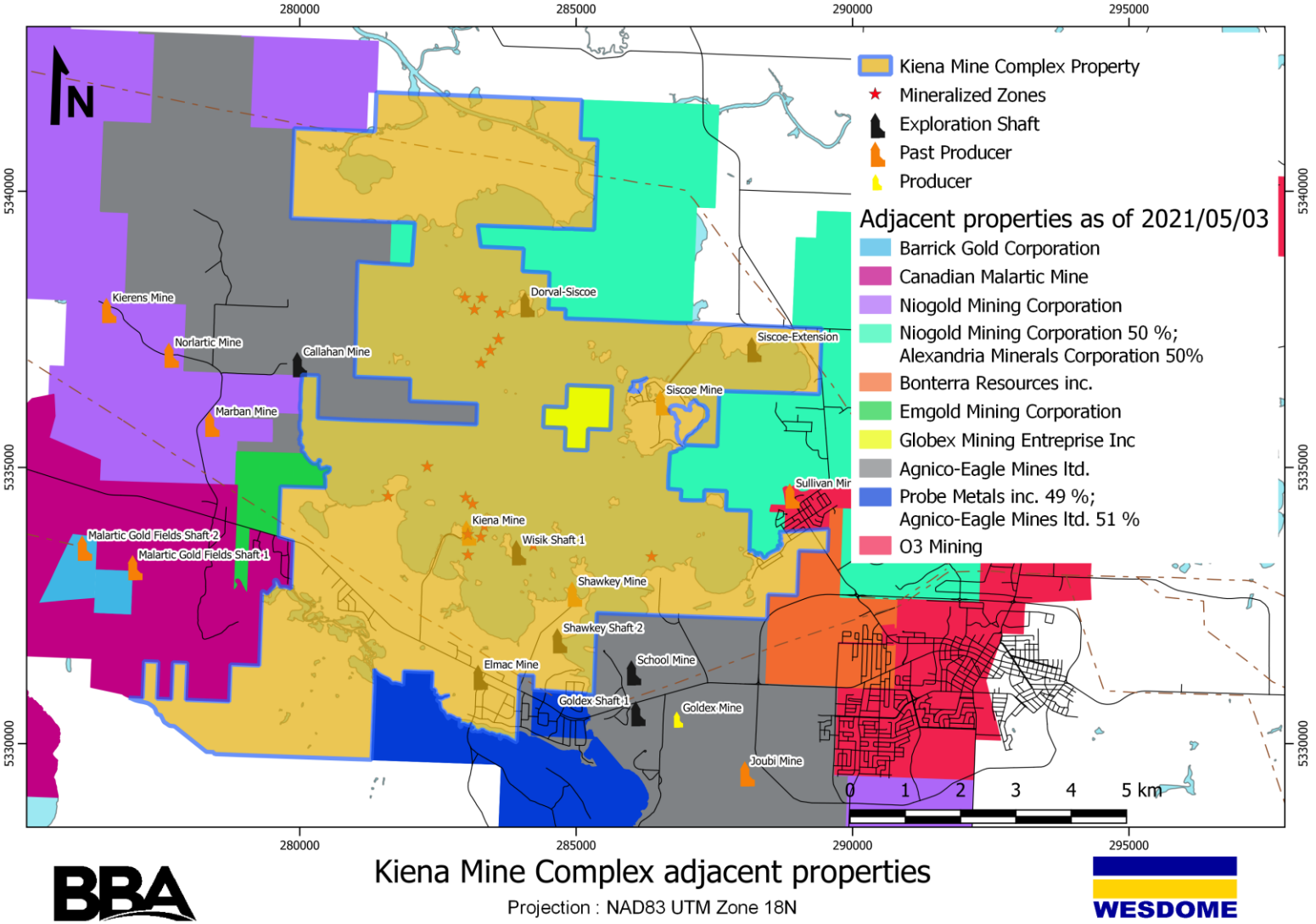


Figure 23-1: Kiena Mine Complex adjacent properties

24. OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution and Organization

Following the completion of the current prefeasibility study, the Project is expected to proceed directly to the detailed engineering and execution phase, with no plans for a feasibility study. This is made possible by the significant portion of mining, processing and site infrastructure required for gold production currently in operation at the site. This includes the mine hoist, ore crushing facilities, mill, power infrastructure, road infrastructure, administrative buildings, security and tailings facilities. Several of these items require upgrades; however, they can be used for certain periods of time as outlined in Chapters 16, 17 and 18 until the upgrades occur, as many of the upgrades are for long term operational needs. The completion dates required for the upgrades have been considered and used to develop the project schedule in Section 24.2. Some work such as the tailings management facility reinforcement and the on-site power upgrades have already started and are represented as such in the schedule.

The execution plan presented here is conceptual in nature and is currently being developed in detail by Wesdome. The proposed execution is based on completing the work using an Owner integrated team strategy in a manner similar to the breakdown of responsibilities employed in the PFS, with the additional construction responsibilities undertaken by Wesdome.

Based on the above considerations, an organizational chart for the execution of the Project has been developed with a simplified version presented in Figure 24-1.

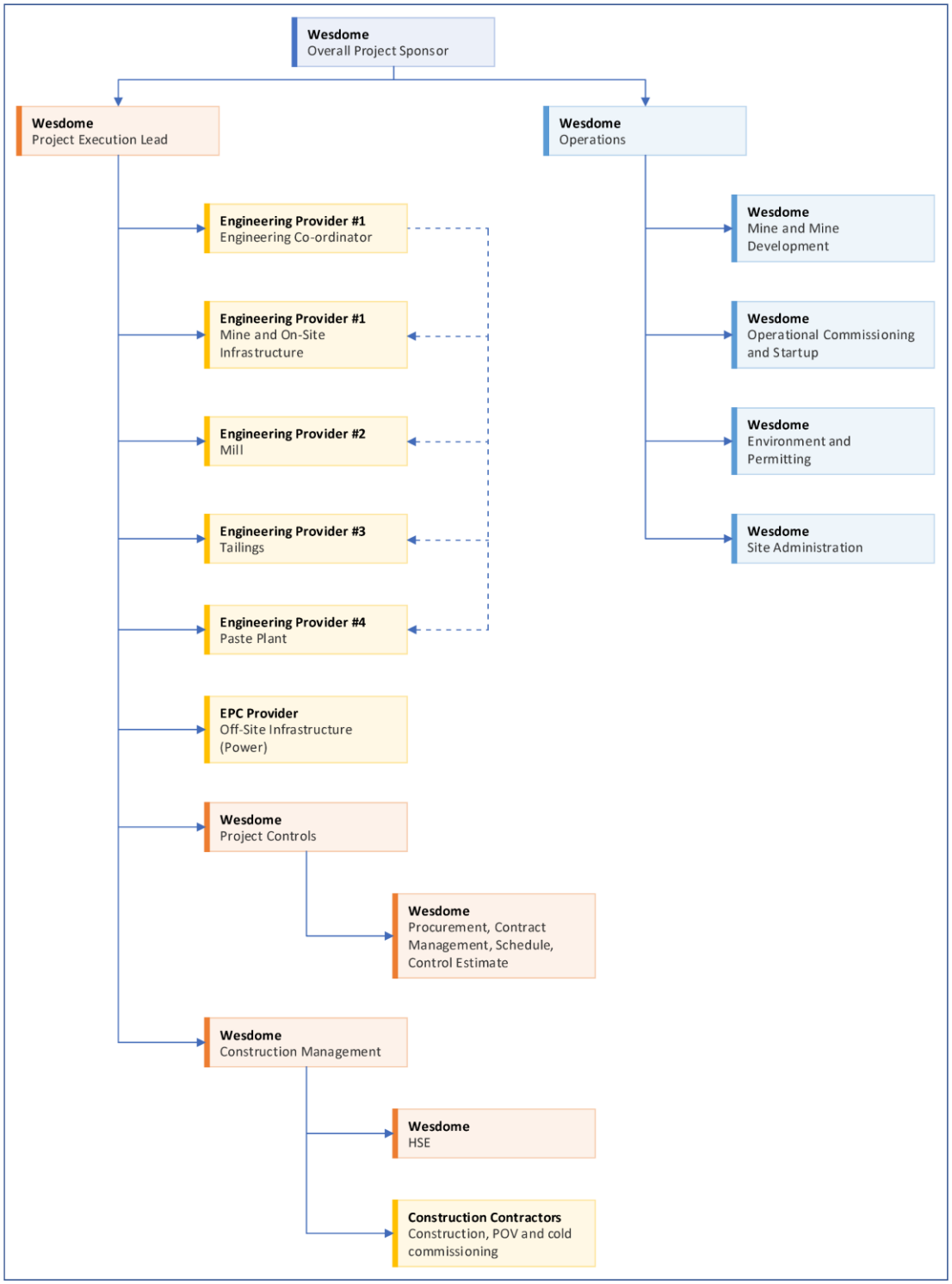


Figure 24-1: Wesdome Kiena Mine Project organizational chart

The overall management of the project activities will be the responsibility of Wesdome. To accomplish this, an execution team to supplement the existing operational team will be assembled using the services of a specialized management firm(s) (in orange) to complement the Wesdome staff roles (in blue). It should be noted that some of these roles already exist and are staffed for the purposes of planning the execution works.

The Project Execution Leader will report directly to Wesdome Project Sponsor. The Project Execution Leader will have single point accountability for the performance of the engineering and construction activities for the Project. This leader will also interface with the Operations Leader to ensure smooth and timely handovers.

The off-site infrastructure identified in Chapter 18 is expected to be completed as an EPC package contracted to the local power provider, i.e., Hydro-Québec.

The engineering providers will in general be responsible for providing all engineering and deliverables required to allow for the Wesdome execution team to purchase equipment and contract construction activities required for the delivery of the Project. Engineering providers #2, #3 and #4 will functionally and contractually report to the Wesdome Project Execution Lead but will, however, have an indirect reporting relationship to Engineering Provider #1 in order to ensure that site wide infrastructure is coordinated and can be executed in an efficient manner. In general, the scopes of the engineering providers (EPs) are presented in Table 24-1.

Table 24-1: Scope by engineer provider

Engineering Provider	Major Scope Items
EP1 – General & Infrastructure	▪ Site wide power distribution; ▪ Roads and Civil (including access and clearing for all works); ▪ Tailings pipelines and pumping; ▪ Surface ventilation upgrades; ▪ Underground dewatering stations; ▪ Underground power distribution including BEV charging; ▪ Underground services; ▪ Site wide automation.
EP2 – Mill	▪ Cyanide Destruction Circuit integration; ▪ Reagents preparation and distribution systems additions; ▪ Effluent Treatment Plant.
EP3 – Tailings	▪ Tailings Storage Facility reinforcement; ▪ Tailings Storage Facility dykes raises.
EP4 – Paste Plant	▪ Paste plant; ▪ Paste borehole placement and design.

Due to the nature of the already existing site and staff, Wesdome operations will be responsible for the mine development, operational commissioning, environment and permitting, and site administration (including payroll, IT, security, medical, etc.). All of these roles, except operational commissioning, currently exist on site. The operational commissioning will be executed by the future site operators currently being recruited and accounted for in the staffing plan in Section 21.2.8.

24.2 Project Execution Schedule

A Level 2 Detailed Project Execution Schedule has been developed based on information and options developed as part of the Kiena Mine Complex Prefeasibility Study. The high-level schedule is shown in Figure 24-2. The overall project schedule will inform key project decisions and ensure that targets for all areas of the Project are met throughout each phase.

The schedule was developed using information and expertise provided by the various Qualified Persons that supported the overall PFS Project. Durations and required scope to be completed was entirely provided by the various QPs and the Owner's team.

The final schedule was resource loaded, and Full-Time Equivalent (FTE) counts were derived based on the resources required for each scope. The schedule was also reviewed by the various QPs and the Owner's team. The Project features a peak staffing of just over 100 FTEs and has three peaks due to the various work fronts and timing. The first peak is 105 FTEs due primarily to the tailings and borehole development. The second peak is 101 FTEs and is due to the construction of the backfill plant. The third and final peak is a lower 74 FTEs and is due to the remaining infrastructure and headframe activities. The FTE histogram can be seen in Figure 24-3.

The following assumptions were made as part of the schedule development process:

- All engineering, procurement, construction, and commissioning activities are based on a 5 day week, 8 hours per day working calendar. The only exception is the borehole activities, which are based on a 7-day per week, 10 hours per day schedule;
- All durations and activities were based on the project estimate and information provided by each QP;
- All permitting targets were provided by the Owner's team representatives;
- Constrained targets for the completion of the upgraded powerline, paste plant, and cyanide destruction system were provided by the Owner's team;
- Care was taken to limit the number of constraints and to ensure the schedule is logically driven with as little lag as possible.

The critical path of the project schedule flows through the procurement of engineering services, process plant and mining equipment and its installation. Specifically featured on the critical path is the cyanide destruction system, the tailings detox system, and the mine dewatering equipment.

The completion of the cyanide destruction system is planned for October 2021, and the paste plant completion is June 3, 2022. For a full list of completion milestones refer to Figure 24-2, which includes the project milestone table. The overall program length is approximately 10 months.

It is important to note that as the project progresses the schedule will be further refined and detailed.

Activity ID	Activity Name	Remaining Duration	Start	Finish	Total Float
Kiena Mine - Execution Schedule		412	1-Feb-21 A	9-Nov-22	0
Project Milestones		373	1-Feb-21 A	9-Nov-22	0
A3880	Permitting Activities Start	0	1-Feb-21 A		
A3710	Limited Notice to Proceed	0	22-Feb-21 A		
A2970	Upgraded Powerline Requested from Hydro-Quebec	0	7-Jun-21		42
A3700	Wesdome Notice to Proceed	0	1-Jul-21*		0
A4250	Boreholes Complete	0		4-Oct-21*	280
A3900	Cyanide Destruction Commissioning Complete	0		29-Oct-21*	0
A3040	Tailings Lift 1 Complete	0		9-Mar-22	175
A3000	Effluent Treatment Plant Commissioning Complete	0		22-Mar-22	166
A2990	Paste Plant Commissioning Complete	0		3-Jun-22*	20
A3720	Mobile Equipment Procurement Complete	0		27-Jul-22	75
A2960	Upgraded Powerline Complete	0		1-Sep-22*	42
A3010	Process Plant Commissioning Complete	0		5-Oct-22	25
A3030	Ventilation Upgrades Complete	0		5-Oct-22	25
A2910	Construction Complete	0		9-Nov-22	0
A3440	Commissioning Complete	0		9-Nov-22*	0
Permitting		231	1-Feb-21 A	1-Mar-22	181
Detailed Engineering		307	5-Apr-21 A	15-Jun-22	85
Procurement		342	23-Feb-21 A	3-Aug-22	70
Construction		412	23-Feb-21 A	9-Nov-22	0
Commissioning		263	4-Oct-21	5-Oct-22	25
Mining		109	1-Sep-21	1-Feb-22	201

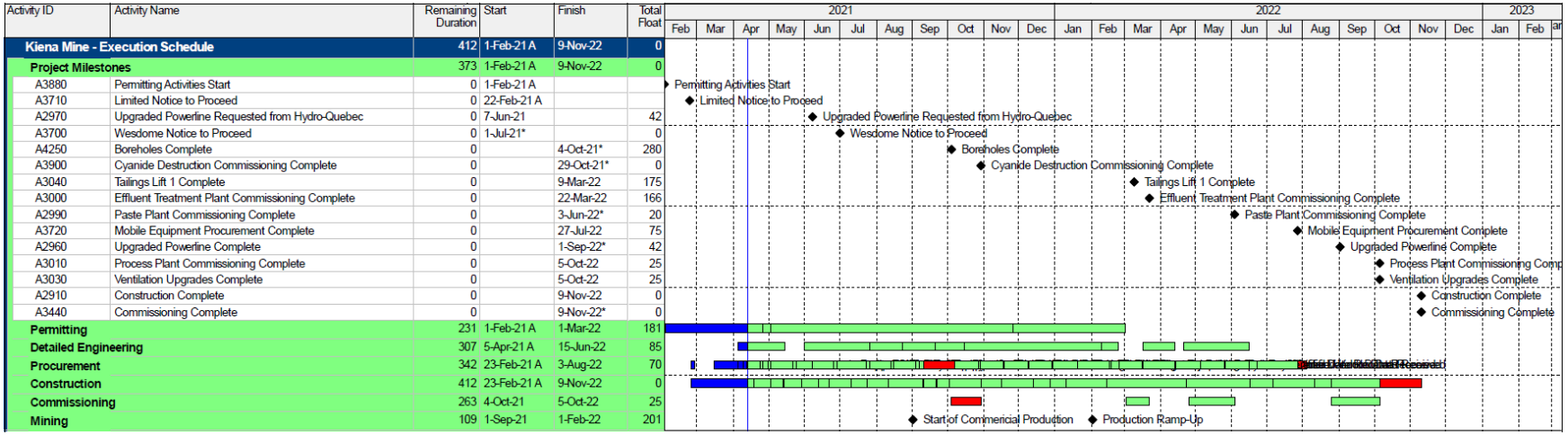


Figure 24-2: Level 1 Schedule roll-up

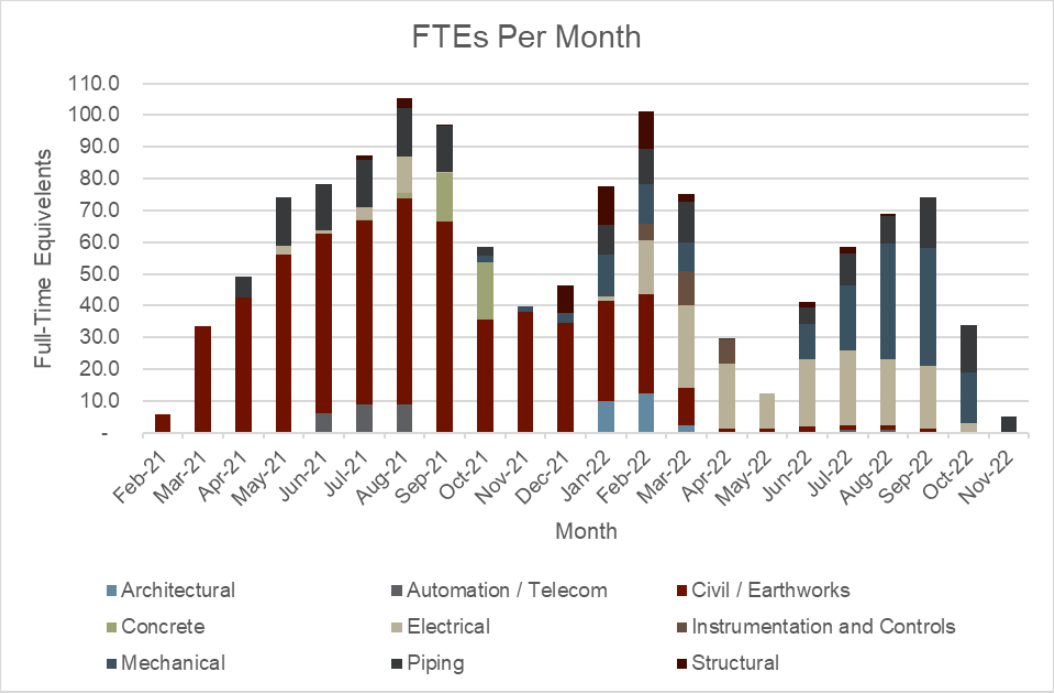


Figure 24-3: FTE Histogram

24.3 Value Improving Practices

During the course of the prefeasibility study, opportunities to improve the profitability of the Project through value improving practices (VIPs) have been sought. The main outcomes are listed below:

1. The duration of the life of mine plan has been compressed as much as possible in order to minimize the fixed component of the operating costs. The duration of the mine plan has been reduced by 22 months when compared to the first iteration.
2. The mill operating schedule has been optimized to best match the yearly tonnages, and varies between 4, 5, and 7 days per week depending on the year to minimize operating costs.
3. The initial plan of using diesel mining equipment exclusively called for a cooling plant once the mine reached a certain depth. Switching a portion of the fleet to battery electric vehicles eliminated the need for the cooling plant, resulting in cost savings.
4. The effluent treatment plant will be leased over the life of mine, and not purchased as originally planned. This improves the discounted cashflow profile and adds flexibility.
5. Whenever possible, the existing electrical infrastructure is reused to reduce new equipment requirements. For example, the paste plant will be powered by the existing hydraulic fill plant transformer, while the mine intake fans are expected to be powered by the existing underground distribution transformer.

24.4 Project Risks and Opportunities

Three risk workshops have been conducted as part of this study to identify the risks and opportunities that apply to each area of the Project, as well as potential mitigation measures.

As with most mining projects, there are risks that could affect the economic viability of the Project. Many of these risks are based on a lack of detailed knowledge and can be managed as more sampling, testing, design, and engineering are conducted during the next stages. Table 24-2 identifies what are currently deemed to be the most significant internal project risks, potential impacts, and possible mitigation approaches that could affect the technical feasibility and economic outcome of the Project.

External risks are, to a certain extent, beyond the control of the Project proponents and are much more difficult to anticipate and mitigate. Although, in many instances, some risk reduction can be achieved. External risks are things such as the political situation in the Project region, metal prices, exchange rates and government legislation. These external risks are generally applicable to all mining projects. Negative variance to these items from the assumptions made in the economic model would reduce the profitability of the mine and the mineral resource estimates. The main external risks to the Project are linked to the ongoing COVID-19 crisis, which is causing disruptions to the supply chains and labour, cost increases, delayed deliveries and lower productivities. The risks associated with COVID-19 have been deemed business risks and as such have not been considered in this study.

Internal and external risks have not been included in the calculation of the contingency.

There are opportunities that could improve the economics or timing of the Project. The major opportunities that have been identified at this time are summarized in Table 24-3 excluding those typical to all mining projects, such as changes in metal prices, exchange rates, etc. Further information and assessments are needed before these opportunities should be included in the Project economics.

Table 24-2: Project risks

Area	Risk Description and Potential Impact	Mitigation Approach
Geology, Mining & Infrastructure	*Arc flash.* The arc flash study is incomplete and arc flash risks may be present.	Complete the arc flash study.
	Reduced reliability of UG power at depth. The voltage of the current transformer for the mine electrical distribution is 4.16 kV, which is not suitable beyond a certain depth. Delays in the procurement of the new 13.8 kV transformer may delay the mine development required for start-up.	Engineer and purchase a dry type transformer (lower lead time compared to oil type).
	Traffic issues at mine road entrance. Highway 117 turn to access the mine is known to be risky and accident prone.	Add a right turn lane (by the city), increase lighting (not included in estimate). New speedometer signs have been installed.
	Loss of lake bottom. Failure of crown pillar under the lake could result in the flooding of the mine through the old mine workings.	Complete the construction of barricades to seal off the existing mine workings under the crown pillar. Perform daily visual monitoring. Backfill the stopes near the crown pillar.
	Higher dilution in komatiite contact zone. The ground is untested with the selected mining method; this may increase dilution.	Additional definition diamond drilling of bulk sampling area (not included in estimate).
	Delays in power line. The completion of the power supply capacity increase could take longer than expected and constrain or delay certain activities on site.	Expedite the process with Hydro-Québec.
	Failure of ventilation fans. The vendor of the existing fans no longer exists, and spare parts are unavailable. A failure would reduce the operating level. The fans will be replaced in 2022.	Expedite the ventilation system upgrade.
	Increase in electrical refurbishing work. The electrical rehab assessment that was conducted on the underground infrastructure identified several deficiencies; more deficiencies may be found on the surface infrastructure.	Conduct a detailed inspection of the electrical installation for the surface infrastructure.

Area	Risk Description and Potential Impact	Mitigation Approach
Metallurgy & Water Treatment Plant	*Difficulty maintaining production rate.* Unskilled operations crews result in a reduced production rate or an increase in operating costs.	Develop the training and hiring plan.
	Incomplete gold leach. Design does not allow for sufficient variability in ore, resulting in a reduction in recovery.	Add reagents, replace air by oxygen, or perform metallurgical testwork for variability (not included in estimate).
	Increased human resources costs. Shortage of skilled labour in the Val-d'Or area increases operating costs.	The mill operates on a 4/3 schedule for the first three years.
Tailings & Water Management	*Delays in start of operations due to permit delays.* The MELCC may refuse to permit operations of TSF during construction.	Add second shift or extend work hours to expedite construction.
	Ice build-up at tailings beach. On/off milling plan and tailings deposition at the TSF may cause reduced stability and safety of the upstream berm raise at the north cell.	Conduct subsurface investigation.
	Tailings dam failure. A failure of the existing dams may contaminate the environment.	The design criteria of the dykes reinforcements and raises takes the historical dams into account.
	Spill of tailings into the lake. Break or failure of tailings line may cause tailings to spill in the lake.	Install concrete protection to prevent collisions where possible. The tailings line uses double wall pipe.
Environment	*Liquefaction of foundation soil units.* The design of the TSF assumed foundation soil units would not liquefy during the design earthquake event based on preliminary screening level analysis. Larger toe buttress and ground improvement would otherwise be required.	The TSF is designed for the 1 in 5,000 years event.
	Tailings and waste rock could be acid generating. The assumption in the PFS is that tailings and waste rock are non acid generating.	Complete ongoing testing.

Area	Risk Description and Potential Impact	Mitigation Approach
	Opposition of local residents to the Project because of excessive noise. The mine ventilation exhaust fans create a lot of noise.	Silencer and permanent decibel metres have been installed. Meetings with residents, sound surveys, local stakeholder committee periodic follow-up, semi-annual open house / town hall style presentations.
	Opposition of local residents to the Project because of excessive dust. The entrance of the mine will see higher traffic at shift changes; sand transport (for backfill) and dry tailings at the TSF may also cause dust issues.	Use truck covers and wind protection around the sand stockpile. Revegetate the outer banks of the TSF.
Project execution	*Poor health and safety performance.* The contractors could be improperly trained due to incomplete company procedures, causing safety incidents and/or equipment damage.	Conduct contractor pre-qualification and site specific induction.
	Late equipment delivery to site. Fabrication durations are unconfirmed, the procurement plan is yet to be prepared (expediting requirements), and the pandemic may impact supplier operations; this could cause schedule delays.	Include expediting and inspection at key suppliers facilities. Complete definition of expediting and inspection levels.
	Contractor non-compliance. Weak terms and conditions could cause issues with contractors that result in cost increases.	Conduct a review of existing clauses and prepare new general terms and conditions that are site specific.
	Substandard equipment delivered to site. Insufficient QA/QC checks could result in poor performance for some equipment, thus lowering production.	Conduct inspection at the shop prior to shipping release.
	Poor change management control. Lack of appropriate policies and procedures for project management may result in scope creep.	Prepare project control plan and project execution plan.

Table 24-3: Project opportunities

Area	Opportunity Explanation	Benefit
Tailings & Water Management	*Lower actual cost for the effluent treatment plant.* The effluent treatment design criteria is conservative as the water data is not sufficiently well defined. As a result, the ETP may be overdesigned. Following initial tests conducted in 2022 with leased treatment units, some steps in the treatment may prove unnecessary.	The overall rental cost of the ETP would decrease as units are removed.
	Additional tailings storage. A preliminary design for a filter plant and dry stack storage facility has been completed as part of this study to expand the tailings storage capacity. It is not required for the Project with the current LOM plan.	Should the life of mine be extended, there is the possibility to store an additional 0.55 Mt of tailings at the mine site.
Paste Plant	*Reduced sand consumption.* Sand addition may not be used for the last sequential stope backfills.	Reduction in operating costs and lower capacity utilization in the TSF as more tailings would be used for backfill.

25. INTERPRETATION AND CONCLUSIONS

25.1 Overview

This Report was prepared by a group of independent consultants (QPs) to demonstrate the economic viability of an underground mine and process plant complex based on the reserves estimated for the Kiena Mine Complex Project. This Report provides a summary of the results and findings from each major area of investigation to a level that is considered to be equivalent and normally expected for a PFS of a resource development project. Standard industry practices, equipment and processes were used in this study.

25.2 Data Verification and Mineral Resources

The October 2020 Kiena Mine Complex Mineral Resource Estimate (2020 MRE) was prepared by Karine Brousseau, P. Eng., Wesdome, and audited by Pierre-Luc Richard, P. Geo., and Charlotte Athurion, P. Geo., both from BBA, using all available information including historical and recent diamond drillholes (DDH). Pierre-Luc Richard, P. Geo. and Charlotte Athurion, P. Geo. are the Independent and Qualified Persons (QPs) for the 2020 MRE and all activities related to it. The effective date of the estimate is October 31, 2020 based on the compilation status and resource cut-off grade parameters.

The resource database for the Project, as of September 18, 2020, consisted of 6,823 DDH (951,602.6 m), including 213 DDH from the 2019-2020 drilling program (60,865 m). As of April 12, 2021, since the close out date of the 2020 MRE, Wesdome has completed an additional 143 DDH for 41,780.8 m that are not included in the 2020 MRE. The QPs are of the opinion that while the addition of these new diamond drillholes would increase knowledge and confidence on the Project and have a positive impact on the resource estimate, it does not have a material impact on the current study presented in this Report.

Pierre-Luc Richard and Charlotte Athurion, both QPs, reviewed the drilling, sample preparation, analytical and security procedures, as well as insertion rates and the performance of blanks, standards and duplicates for the 2019-2020 drilling programs, and concluded that the observed failure rates are within expected ranges and that no significant assay biases are present. The QPs are of the opinion that the protocols in place are adequate and have been followed. The database for the Project is of good overall quality and adequate to industry standards. The QPs are also of the opinion that the database is appropriate for the purpose of the 2020 MRE and that the sample density allows for a reliable estimate to be made of the size, tonnage and grade of the mineralization in accordance with the level of confidence established by the Mineral Resource categories in the CIM Standards.

After completing the 2020 MRE and a detailed review of all pertinent information, Pierre-Luc Richard and Charlotte Athurion, QPs, concluded the following:

- The 2020 MRE is constrained within 3D wireframes of 46 mineralized lenses;
- Using a resource cut-off grade of 2.8 g/t Au (>40 degree dip) and 3.6 g/t Au (<40 degree dip, Wesdome Zone only), the Inferred Resources amount to 3,420,500 t grading 5.98 g/t Au containing approximately 657,600 ounces of gold and the Indicated Resources (inclusive of mineral reserves) amount to 1,790,600 t grading 13.83 g/t Au containing approximately 796,000 ounces of gold;
- No Measured Mineral Resources have been defined in the 2020 MRE;
- It is likely that further diamond drilling would upgrade most of the Inferred Mineral Resources to Indicated Mineral Resources.

25.3 Metallurgy and Processing

The metallurgical testwork program and bulk sample processing showed the ore is relatively soft and that high gold recoveries are obtained at a grind size of 80 µm, 75 g/t of lead nitrate and 500 ppm of NaCN. No significant difference in gold recovery was observed between GTL and WOL after 48 hours of leaching time. As a result, the installation of a gravity circuit in the plant is not recommended. The cyanide destruction tests did not reach the target of 1 ppm CN_{WAD} in the tails, therefore additional testwork is recommended to determine the proper reagents addition requirements. It is nonetheless expected that the reagents dosage will be in the same order of magnitude as the ones used for this testing thus the impact on operation costs will be low.

The existing equipment, along with the addition of SMBS and copper sulfate systems, allows reaching the throughput and performance objectives. The grinding circuit's available power is higher than required and it allows reaching the targeted size reduction without using the existing surface crushing circuit. The process plant operating schedule will need to vary over time to operate in the most efficient manner to match with the mine production schedule. The hydraulic backfill plant is not required as is replaced by a paste backfill plant.

25.4 Mineral Reserves

The Project Probable Mineral Reserves are estimated at 1.574 M ore tonnes at an average gold head grade of 11.89 g/t (0.601 M oz contained gold). The mineral reserve estimate is dependent on Indicated Mineral Resources represented in the 2020 MRE and converted based on technical feasibility and economic viability. Three mineralized zones were considered in the mine schedule representing the LOM plan for the PFS. Most of the ore tonnes as well as gold ounces to be mined are sourced from the Kiena Deep A Zone supplemented by the S50 and Martin Zones. Five mineralized zones were evaluated at the onset of the study.

The mineral reserve estimate is deemed prudent based on the detail allocated both technically in design and economically in costing. The overall mine operating cost for the Project was established and refined to reflect the progression of the PFS mandate. The cut-off grades for the respective mineralized zones were established according to the overall mine operating costs established for the respective mineralized zones. Incremental cut-off grades excluding mining costs were not considered for lateral drift development required through mineralization and would present opportunity to increase the mineral reserve estimate. The capital waste rock development costs were considered in the economic analysis by mineralized zone to ensure profitability in accessing the prospective mining zones. Furthermore, the capital waste rock development costs were used on an incremental basis to remove any stopes above cut-off grades that were unprofitable based on development access constraints.

25.5 Mining Methods

The life of mine plan representing the PFS is deemed prudent based on the productivity rates, durations and factors considered in the mine schedule influenced by the development designs, stope empirical designs, mine sequencing, mining cycle constraints, expected labour headcount, new fleet of mobile equipment, existing and new fixed infrastructure, hoist shaft conveyance and milling throughput capabilities. Production mining utilizing both overhand and underhand long hole stope methods on longitudinal retreat offer efficiencies for enhancing mining recovery, eliminating the requirement for re-sill development and maximizing production in the mining zones. Mining independence in the Kiena Deep A Zone is afforded by the eight mining fronts, further independence to the LOM plan is afforded by three distinct mining zones. Cemented rockfill, rockfill and paste backfill will support the rock mass after extraction. Mining recovery will be maximized and production stope cycle time optimized in the Kiena Deep A Zone by making paste backfill the predominant backfill type.

25.6 Paste Plant

The paste plant using 65% sand and 35% tailings at 100 t/h solids and ~83wt% solids with ~2.5% average binder consumption is expected to ensure high fill rate efficiency and backfill strength to enable 7 and 14 day turnaround times with good fill stability and low dilution from fill. While use of sand creates the need for more tailings disposal on surface, the mining rate is critical to reach good levels of productivity and lower mining costs.

25.7 Infrastructure

25.7.1 Surface Infrastructure

The Kiena Mine Complex surface infrastructure is currently being used to develop the underground mine and is expected to be reused for the Project. Civil works as well as modifications and additions to the power distribution system are required in the areas of the new paste plant, ventilation system upgrade, and effluent treatment plant. The site power needs are expected to increase from 7.8 MVA to 12.7 MVA, which will require a new 25 kV power line from Hydro-Québec.

25.7.2 Tailings Management

Dam safety assessment of the existing TSF facility indicated that most of the perimeter dykes requires upgrade to meet current dam safety guidelines and standards. The existing TSF does not have an emergency spillway, which is required to satisfy current dam safety guidelines and standards. The proposed upgrade will involve construction of downstream toe buttress where possible and ground improvement combined with toe buttress where toe area is limited due to the proximity of the lake shore. Deep Soil Mixing (DSM) was chosen as the preferred ground improvement method. The perimeter ditch will be realigned to allow for the TSF upgrade construction.

The tailings deposition plan was developed to store 1.3 Mt of tailings by slurry deposition. This provides flexibility with the backfill operation by providing additional tailings storage of 0.20 Mt beyond the PFS life of mine (LOM) tailings to TSF of 1.10 Mt. Based on the tailings deposition planning, Years 2021-2025 tailings will be stored in North Cell with the construction of a 3 m high upstream tailings berm over compacted tailings beach. Remaining Years 2026-2028 tailings will be stored at South Cell by raising the South Cell perimeter dykes by 1.0 m. Operating pond and Environmental Design Flood (EDF) will be managed within the South Cell. An emergency spillway will be constructed at the South Cell. An internal spillway will be constructed to allow for passive flow from North to South Cell.

Water will be pumped from South Cell using a pump system to a water treatment plant located at the northeast end of the polishing pond. The pump system will be located within the South Cell with access road connecting to Dyke 10. Treated water will be discharged into the polishing pond, which will then be released to environment via the existing discharge outlet. The reclaim water for mill operation will be obtained from TSF South Cell via the pump system.

25.7.3 Effluent Treatment Plant

The use of a cyanide destruction process produces cyanate ions that evolve to ammonia. Residual concentrations are expected to be high enough to exceed lethality limits of rainbow trout and daphnia magna, which is not acceptable for the effluent of the tailings pond. A water treatment plant is required and will consist in mobile units, based on filtration and reverse osmosis, which will allow the removal of heavy metals cyanide complexes and reject a stream containing contaminants back to the South Cell. No excessive accumulation of contaminants in the South Cell will occur considering its size and the lower reagents consumption in the early years of operation. During that period, it is recommended to perform an extensive sampling plan to determine the design criteria of the permanent water treatment plant if it is needed, or if the mobile plant can be kept in operation.

25.8 Environmental Studies

There were no findings from the environmental studies or review that were in conflict with the proposed design. Wesdome is completing additional studies as needed to support obtaining the necessary environmental approvals which are in progress. The environmental effects expected from the re-opening of the Kiena Mine are typical of mining operations in these settings. Stabilization of the existing TSF will reduce the potential for environmental damage from this older facility. Reclamation of the mine progressively during operations and on eventual closure of the mine will return the site to a more naturalized condition.

25.9 Capital and Operating Costs

The total capital cost (±25%) over the LOM of the Project including indirect costs, contingencies and funding of reclamation activities is estimated to be C$231.8M. This number excludes escalation, risk events, sunk costs, and financing costs. The calculated contingency is C$22.2M corresponding to 10.6% of the overall capital costs, to which an amount of C$0.3M is added for risk mitigation measures, for a total contingency amount of C$22.5M.

The total operating cost (±25%) over the LOM of 7.1 years is estimated to be C$295.5M or C$187.71/t of ore mined. It is anticipated that the Project's workforce requirements will average 205 employees over the LOM, with a peak of 225 employees in 2024.

25.10 Indicative Economic Results

The financial analysis performed as part of this PFS using the base case assumptions results in an after-tax net present value (NPV) of C$366.6M at a 5% discount rate and an internal rate of return (IRR) of 98% after mining and income taxes. The payback period is 2.7 years. All-in sustaining cost of production is US$676/oz over the LOM.

The results of the PFS indicate that the proposed Kiena Mine Complex Project is economically viable at the base case gold price of US$1,600/oz. In addition, the sensitivity analysis indicates that the economic viability of the Project will not be significantly impacted by variations in the capital or operating costs, and that the NPV and IRR of the Project are positive over the range of values used for the sensitivity analysis when analyzed individually.

25.11 Project Risks and Opportunities

Internal and external risks that could affect the economic viability of the Project have been identified as part of this study. Measures have been proposed to mitigate internal risks; notable internal risks are mostly schedule risks deriving from the aggressive execution timeline. The main external risks to the Project are linked to the ongoing COVID-19 crisis, which is causing disruptions to the supply chains and labour, cost increases, delayed deliveries and lower productivities. The risks associated with COVID-19 have been deemed business risks and as such have not been considered in this study.

26. RECOMMENDATIONS

26.1 Summary

A decision has been made by Wesdome to proceed directly to the execution phase of the Project from the completion of the PFS. While the Project shows positive returns over the range of input variables used in the sensitivity analysis, the reader is advised that the typical risks of executing a project based on PFS level engineering and planning still apply.

It is recommended to proceed with the next steps as per the execution plan presented in Chapter 24, namely detailed engineering and procurement. To that end, it is recommended that Wesdome assembles the execution team, engages the detail engineering firms, and collects firm bids for critical items to meet the schedule.

The following sections summarize the key recommendations arising from this study.

26.2 Drilling and Geology

In order to add and renew mineral resources on a yearly basis it is recommended to continue all ongoing exploration and definition diamond drilling programs of the zones included in the 2020 MRE. Exploration drilling programs should be done to continue investigating any lateral and down plunge of the currently identified mineralization.

A compilation of historic geological data of the area adjacent to the Kiena Mine Complex and across the entire Property is recommended to identify additional targets.

26.3 Mining Methods

26.3.1 Mineral Reserves

Further diamond drilling and mineral resource block modelling may introduce more prospective mineral reserves to increase the PFS mining zones as well as introduce prospective mining zones not currently represented by the LOM plan. Mine designs and economic analyses should progress with suitably classified mineral resources.

26.3.2 Geomechanics

Future geomechanical activities should be pursued and directed at strengthening design considerations and regulating risk. Geomechanical activities are grouped into four predominant categories that follow.

26.3.2.1 Geotechnical Data

Perform detailed geotechnical mapping in lateral sill development focusing on development in Schist Komatiite rock units.

- Establish rock mass characterization and variability.
- Establish joint and foliation trends of primary rock units including orientation, persistence and typical joint / foliation spacing.

Revise the Schist Zone geological model with the identification of distinct lithologies (Komatiite and Schist) and faulting including regional named faults (example: Marbenite Fault).

Develop geotechnical models (rock mass characterization and structural detail) from all available Kiena Deep A Zone diamond drillhole data with focus on lower elevations.

26.3.2.2 Stress Considerations

Confirm stress considerations at depth. Field observations of excavation stability as well as additional in situ stress testing at depth are recommended.

- Stress testing should employ triaxial stress measurement tools (example: CSIRO or USBM stress cell).
- Data interpretation from regional stress measurements suggest lower K-ratios at depth. As a result, current stress consideration may be overestimating in situ stress magnitudes (and severity of stress generated damage) at increased depth.

26.3.2.3 Excavation Design

Initial lateral development and production stopes in the A1 and A2 lenses of the Kiena Deep A Zone provide opportunity to assess rock mass characteristics, rock mass response to stress (damage mechanisms) and optimize excavation designs and ground support practices. Assessment would be done via visual observations, instrumentation and monitoring, ground support testing and modelling back analysis.

Monitor the stability of lateral development in Komatiite and Schist to examine surface displacement and the depth (extent) of displacement. Suggested instrumentation includes:

- Grouted borehole extensometers (MPBX) installed into the drift walls.
- Monitor near-surface convergence using fixed points and a Tape Extensometer.
- Periodic surface surveys, via 3D scan.

Evaluate the stability of mined production stopes via 3D scan.

Numerical modelling.

- Perform modelling back-analysis of observed damages and stability performance to verify numerical modelling confidence.

26.3.2.4 Ground Support

Lateral development in the Schist Zone rock (in particular, Schist) is anticipated to experience progressive deformation with intensity potentially increasing at depth. Wall control requires suitable surface cover, along with a tendon with appropriate load and deformation capacity.

- Lateral development in Schist and Komatiite lithologies provides an opportunity to evaluate alternative ground support products or support systems.
- Testing would include: viability of product installation; pull testing (load and deformation capacity) and assessment of long-term effectiveness of the support products.

26.3.3 Mine Air Ventilation

Mine air ventilation monitoring should be pursued to establish underground temperatures as the Kiena Deep A Zone extends at depth while mobile equipment is operated. Most of the underground mobile equipment will be diesel powered with three battery powered haulage trucks integrated into the fleet. Field monitoring will provide pertinent data for consideration in future mine air ventilation modelling and cooling studies. Detailed studies will further establish the benefits of utilizing a mixed fleet of diesel and battery powered haulage trucks and establish the possible integration of more than battery powered haulage trucks into the underground mobile fleet.

26.3.4 Battery Electric Vehicles

A carbon emission inventory assessment should be conducted to further establish the impact of operating battery electric vehicles (BEV) compared to operating only diesel powered underground mobile equipment. Currently, BEV haulage trucks only are intended for integration into the underground mobile equipment fleet. This assessment would account for both diesel fuel and electricity requirements for BEV.

The noise level underground is expected to be reduced with BEV haulage trucks, a benefit to underground mine personnel exposed to the sounds of underground mobile equipment. A noise assessment should also be conducted to establish the anticipated noise reduction underground with the integration of the BEV haulage trucks.

26.4 Metallurgical Testwork

The metallurgical testwork program allowed to determine most of the operating conditions. However, the following actions are recommended to better determine some operating conditions:

- The cyanide destruction tests did not reach the target of 1 ppm CN_{WAD} in the tails, even at a SO_2 ratio of 7 SO_2:CN, most likely due to the presence of nickel. Additional testwork is recommended to determine the proper reagents addition requirements. A SO_2 ratio of 8 SO_2:CN and the addition of 50 ppm of copper as copper sulfate is recommended for the additional testwork;

- Martin Zone has not been subjected to testwork nor been processed in the past. It is possible that the recovery associated with this zone is lower than expected or that it contains certain minerals or has certain characteristics that are detrimental to the process. Exploitation of that zone is scheduled to start in 2024 so it is recommended, in the meantime, to perform mineralogical characterization and preliminary leaching tests on material from that zone. The results will require analysis by a metallurgist to determine if processing in the existing plant will give appropriate metallurgical performances and if further testwork is required.
- All tests showed good filtering properties. Should dry stacking be necessary in the future, tailings filtration is expected to perform well but air blowing duration should be the subject of future tests to meet the final moisture requirements.

26.5 Processing

Due to the differences in tonnage from one year to the next, it is recommended to investigate the feasibility of using only the SAG Mill when operating at low throughputs. This would improve the grinding efficiency resulting lower energy costs and would also reduce the grinding media cost.

As there is no need for tailings filtration with the current life of mine, it is recommended to assess the possibility of rearranging the tailings circuit in order to put the tailings thickener before the cyanide destruction circuit. This would allow recovering cyanide bearing water to be reused in the process resulting in lower cyanide consumptions and lower cyanide destruction reagents consumption (SMBS, copper sulfate, lime).

26.6 Paste Plant

The use of sand within the paste system requires a sand resource confirmation having the appropriate tonnage and quality (particle size) distribution. This will require some in-pit screening, rehandling and blending. The cost of the sand was based on a surrogate in the Timmins area; local supply costs need to be confirmed.

Use of tailings only as a paste backfill may be possible with confirmation testing. The offset of increased binder while eliminating sand may have both capital and operating costs trade-offs that include tailings disposal costs on surface. The paste plant capital cost would increase but the operating cost would decrease, while some reduction in the tailings disposal facility capacity requirement can offset the increased paste plant capital cost.

26.7 Infrastructure

26.7.1 Surface Infrastructure

It is recommended that Wesdome proceed with the power capacity increase request to Hydro-Québec to avoid delays.

26.7.2 Tailings Management

The following actions are recommended:

- Advanced geotechnical laboratory testing on intact Shelby tube tailings sample and reconstituted tailings samples to determine liquefaction resistance and liquefied shear strength.
- Advanced geotechnical laboratory testing on undisturbed varved silty clay samples (Shelby tube samples) to determine static shear strength, liquefaction resistance and cyclic soften shear strength.
- Perform site-specific seismic hazard assessment and estimate design earthquake parameters (up to 1 in 10,000-year earthquake).
- Develop design earthquake time histories and perform advanced seismic deformation analyses of critical dyke section for the design earthquake.
- Comprehensive geochemistry assessment of tailings and waste rock.

It is to be noted that these recommendations are currently being addressed as part of ongoing detailed design work.

26.7.3 Effluent Treatment Plant

It is recommended that during the operation of the mobile treatment plant, samplings and measurements are taken to get sufficient information on the chemical evolution of the tailings pond water. This information will be used to design the permanent ETP.

26.8 Environment and Permitting

It is recommended that Wesdome continues to communicate with its local neighbours and Indigenous communities to maintain and improve on existing relationships for the long term benefit of the mine. Ongoing studies should be completed to confirm acid generation / metal leaching potential.

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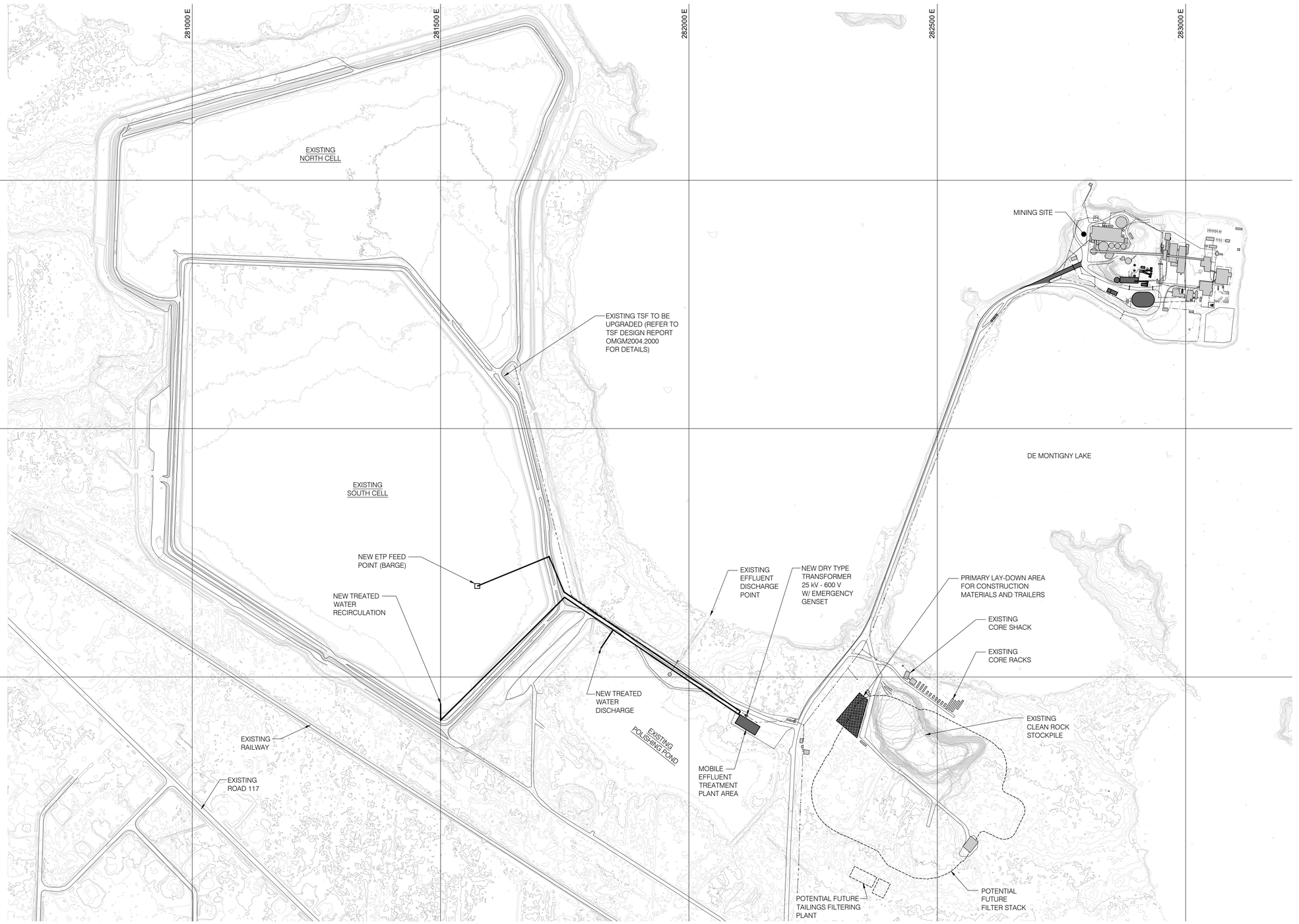
Appendix A: Plot Plans

- Overall
- Mining Site
- Mining Site – Demolition



LEGEND

- NEW
- EXISTING
- FENCE
- POWER LINE (UNDERGROUND)
- POWER LINE (OVERHEAD)

FOR INFORMATION
NOT TO BE USED FOR CONSTRUCTION
DATE: 2021-05-07

BBA

PLAN VIEW

50 0 100 200 300 400 m

SCALE: 1:4000

115	WOOD - TAILINGS FILTER STACK - PLAN, PROFILES AND DETAILS	AC	FOR INFORMATION	H. BON	N. GANGADIN	2021-05-07
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TERRAIN MODEL	FROM MINISTRY OF FORESTS, WILDLIFE & PARKS (QC) 2107 LIDAR DATA	AA	FOR COMMENT	H. BON	N. GANGADIN	2021-02-23
DRAWING No.	DESCRIPTION	REV	DESCRIPTION	VERIFIED BY	APPROVED BY	DATE
REFERENCE DRAWINGS		REVISIONS				

SEAL:

BBA

CLIENT: WESDOME

PROJECT: KIENA MINE COMPLEX PFS

TITLE: OVERALL PLOT PLAN

DESIGNED BY: H. BON	DRAFTED BY: M. VUKOSAV
VERIFIED BY: N. GANGADIN	APPROVED BY: N. GANGADIN
SCALE: 1:4000	DATE: 2020-12-16

DRAWING No.: 3767003-403000-45-D09-0001 | SHEET: 01 | SIZE: A1 | REV.: AC

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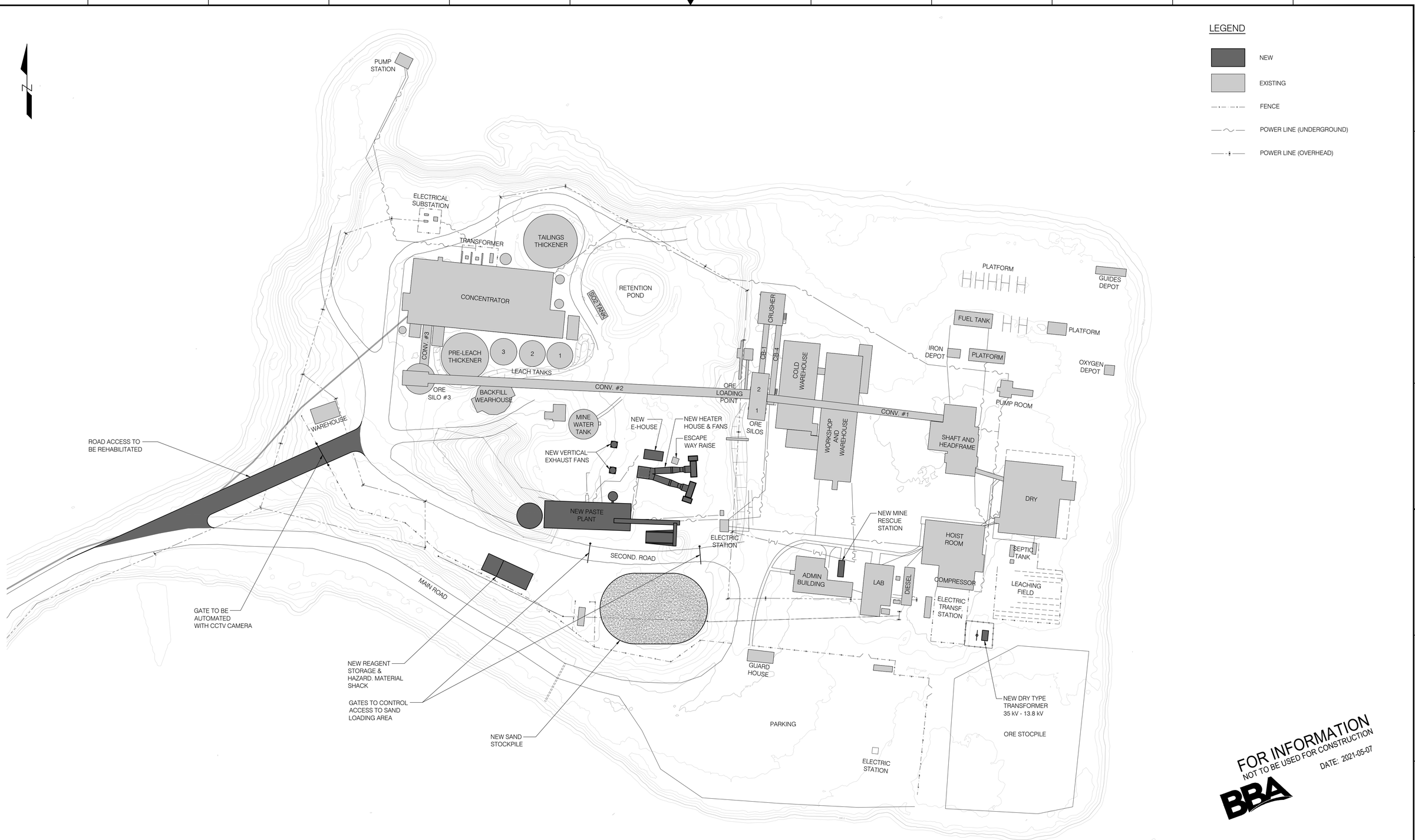

PLAN VIEW

FOR INFORMATION
NOT TO BE USED FOR CONSTRUCTION
DATE: 2021-05-07

BBA

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SCALE: 1:750

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L-1001_1005 RA	MINE PASTE ENG. LTD. - SAND & THICKENED TAILINGS PASTE - GA - PLANS & SECTIONS
PARKER2000V13	WESDOME - SITE GENERAL ARRANGEMENT
TERRAIN MODEL	FROM MINISTRY OF FORESTS, WILDLIFE & PARKS (QC) 2107 LIDAR DATA

REFERENCE DRAWINGS

REV	DESCRIPTION	VERIFIED BY	APPROVED BY	DATE
AC	FOR INFORMATION	H. BON	N. GANGADIN	2021-05-07
AB	FOR INFORMATION	H. BON	N. GANGADIN	2021-03-23
AA	FOR COMMENT	H. BON	N. GANGADIN	2021-02-23

REVISIONS

SEAL:

CLIENT: WESDOME

PROJECT: KIENA MINE COMPLEX PFS

TITLE: MINING SITE PLOT PLAN

DESIGNED BY: H. BON	DRAFTED BY: M. VUKOSAV
VERIFIED BY: N. GANGADIN	APPROVED BY: N. GANGADIN
SCALE: 1:750	DATE: 2020-12-16

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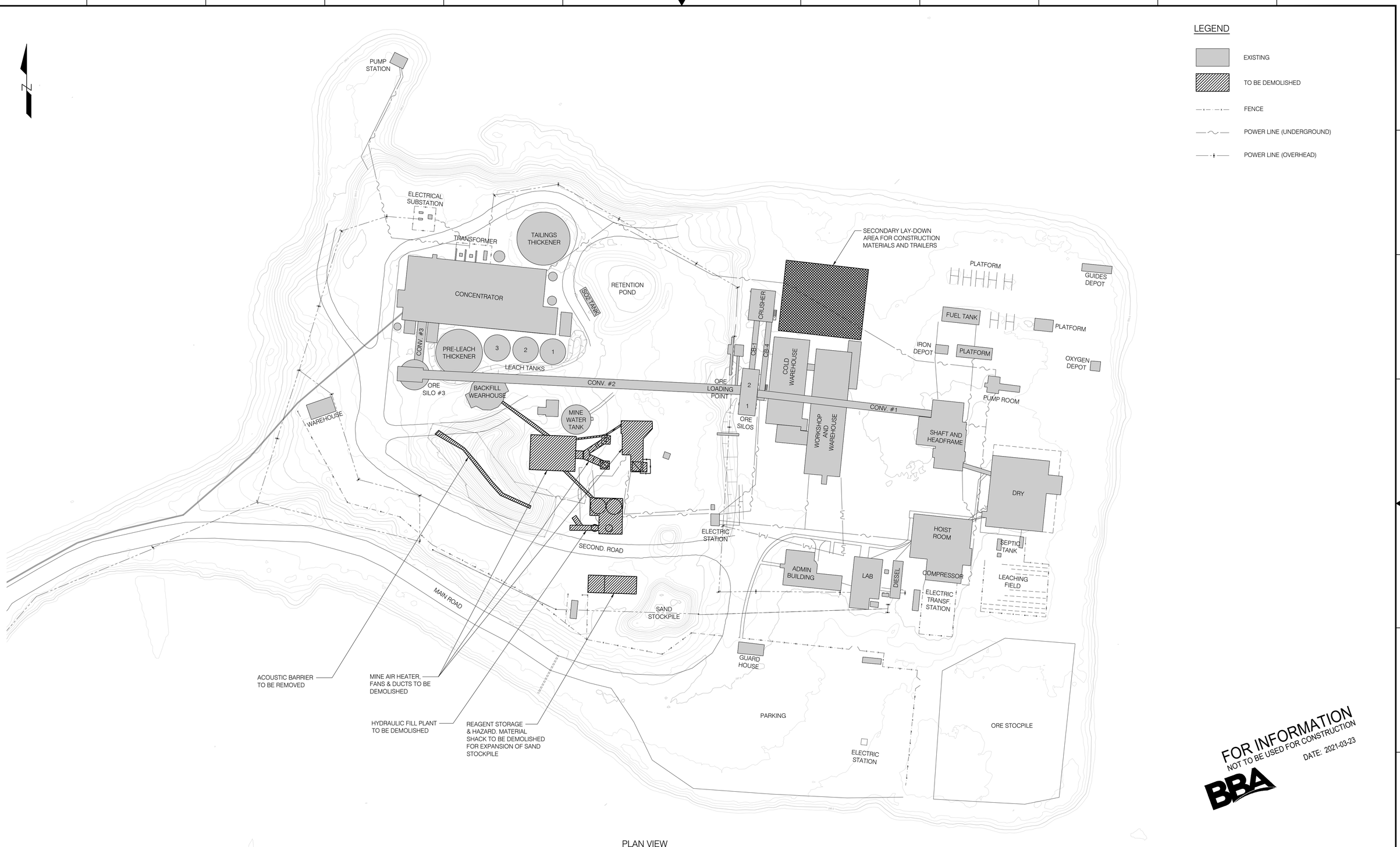

PLAN VIEW

FOR INFORMATION
NOT TO BE USED FOR CONSTRUCTION
DATE: 2021-03-23

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SCALE: 1:750

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TERRAIN MODEL	FROM MINISTRY OF FORESTS, WILDLIFE & PARKS (QC) 2107 LIDAR DATA	AA	FOR COMMENT	H. BON	N. GANGADIN	2021-02-23
DRAWING No.	DESCRIPTION	REV	DESCRIPTION	VERIFIED BY	APPROVED BY	DATE
REFERENCE DRAWINGS		REVISIONS				

SEAL:

CLIENT: WESDOME

PROJECT: KIENA MINE COMPLEX PFS

TITLE: MINING SITE PLOT PLAN DEMOLITION

DESIGNED BY: H. BON	DRAFTED BY: M. VUKOSAV
VERIFIED BY: N. GANGADIN	APPROVED BY: N. GANGADIN
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Appendix B: Detailed list of mineral claims (verified on May 3, 2021)

Mineral claims detailed list (verified on May 3, 2021)

Claim No.	Claim Status	Issue Date	Anniversary Date	Area Ha.	Owner	Claim Name	Land Claim
403542101	Active	1962-11-30		184.07	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	494	CM
403633306	Active	2014-10-14	2022-06-15	1.98	Mines Dynacor inc. (2146) 50%; Wesdome Gold Mines Ltd. (84889) 50% (responsable)	2411926	CDC
403633307	Active	2014-10-14	2022-06-15	16.28	Mines Dynacor inc. (2146) 50%; Wesdome Gold Mines Ltd.. (84889) 50% (responsable)	2411927	CDC
403633308	Active	2014-10-14	2022-06-15	3.02	Mines Dynacor inc. (2146) 50%; Wesdome Gold Mines Ltd. (84889) 50% (responsable)	2411928	CDC
403633309	Active	2014-10-14	2022-06-15	1.09	Mines Dynacor inc. (2146) 50%; Wesdome Gold Mines Ltd. (84889) 50% (responsable)	2411929	CDC
403633310	Active	2014-10-14	2022-06-15	9.6	Mines Dynacor inc. (2146) 50%; Wesdome Gold Mines Ltd. (84889) 50% (responsable)	2411930	CDC
403633311	Active	2014-10-14	2022-06-15	12.9	Mines Dynacor inc. (2146) 50%; Wesdome Gold Mines Ltd. (84889) 50% (responsable)	2411931	CDC
403635183	Active	2014-11-26	2022-06-22	11.06	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2415481	CDC
403645224	Active	2014-11-26	2022-06-22	36.43	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2415482	CDC
403637064	Active	2015-03-13	2022-04-24	32.63	9264-7890 Quebec inc. (95234) 25%; Wesdome Gold Mines Ltd. (84889) 75% (responsable)	2421972	CDC
403637065	Active	2015-03-13	2022-04-24	41.29	9264-7890 Quebec inc. (95234) 25%; Wesdome Gold Mines Ltd. (84889) 75% (responsable)	2421973	CDC
403637066	Active	2015-03-13	2022-04-24	7.95	9264-7890 Quebec inc. (95234) 25%; Wesdome Gold Mines Ltd. (84889) 75% (responsable)	2421974	CDC
403637067	Active	2015-03-13	2022-04-24	9.55	9264-7890 Quebec inc. (95234) 25%; Wesdome Gold Mines Ltd. (84889) 75% (responsable)	2421975	CDC
403637068	Active	2015-03-13	2022-04-24	32.84	9264-7890 Quebec inc. (95234) 25%; Wesdome Gold Mines Ltd. (84889) 75% (responsable)	2421976	CDC
403637069	Active	2015-03-13	2022-04-24	11.93	9264-7890 Quebec inc. (95234) 25%; Wesdome Gold Mines Ltd. (84889) 75% (responsable)	2421977	CDC
403637070	Active	2015-03-13	2022-04-24	9.99	9264-7890 Quebec inc. (95234) 25%; Wesdome Gold Mines Ltd. (84889) 75% (responsable)	2421978	CDC
403637071	Active	2015-03-13	2022-04-24	41.38	9264-7890 Quebec inc. (95234) 25%; Wesdome Gold Mines Ltd. (84889) 75% (responsable)	2421979	CDC

Claim No.	Claim Status	Issue Date	Anniversary Date	Area Ha.	Owner	Claim Name	Land Claim
400083335	Active	2015-07-16	2022-03-05	57.51	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428771	CDC
400083339	Active	2015-07-16	2022-03-05	57.51	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428772	CDC
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400084306	Active	2015-07-16	2022-03-05	57.51	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428775	CDC
400084307	Active	2015-07-16	2022-03-05	57.51	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428776	CDC
400084308	Active	2015-07-16	2022-03-05	57.52	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428777	CDC
400084313	Active	2015-07-16	2022-03-05	57.51	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428778	CDC
400084660	Active	2015-07-16	2022-03-05	57.45	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428779	CDC
400084685	Active	2015-07-16	2022-03-05	57.5	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428780	CDC
400084686	Active	2015-07-16	2022-03-05	57.5	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428781	CDC
400084688	Active	2015-07-16	2022-03-05	57.5	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428782	CDC
400085134	Active	2015-07-16	2022-03-05	57.49	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428783	CDC
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400085146	Active	2015-07-16	2022-03-05	57.47	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428788	CDC
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402977033	Active	2015-07-16	2022-03-05	57.52	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428794	CDC
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403638123	Active	2015-07-16	2022-03-05	4.81	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428796	CDC
403638124	Active	2015-07-16	2022-03-05	54.9	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428797	CDC
403638126	Active	2015-07-16	2022-03-05	57.53	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428799	CDC

Claim No.	Claim Status	Issue Date	Anniversary Date	Area Ha.	Owner	Claim Name	Land Claim
403638129	Active	2015-07-16	2022-03-05	3.54	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428802	CDC
403638130	Active	2015-07-16	2022-03-05	57.43	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428803	CDC
403638131	Active	2015-07-16	2022-03-05	45.47	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428804	CDC
403638132	Active	2015-07-16	2022-03-05	49.58	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428805	CDC
403638133	Active	2015-07-16	2022-03-05	57.08	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428806	CDC
403638134	Active	2015-07-16	2022-03-05	40.82	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428807	CDC
403638135	Active	2015-07-16	2022-03-05	57.51	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428808	CDC
403638136	Active	2015-07-16	2022-03-05	57.45	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428809	CDC
403638137	Active	2015-07-16	2022-03-05	13.5	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428810	CDC
403638138	Active	2015-07-16	2022-03-05	1.58	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428811	CDC
403638140	Active	2015-07-16	2022-03-05	53.96	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428813	CDC
403638141	Active	2015-07-16	2022-03-05	57.52	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428814	CDC
403638142	Active	2015-07-16	2022-03-05	29.94	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428815	CDC
403638143	Active	2015-07-16	2022-03-05	18.37	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428816	CDC
403638144	Active	2015-07-16	2022-03-05	57.22	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428817	CDC
403638145	Active	2015-07-16	2022-03-05	13.96	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428818	CDC
403638146	Active	2015-07-16	2022-03-05	57.44	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428819	CDC
403638147	Active	2015-07-16	2022-03-05	28.11	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428820	CDC
403638148	Active	2015-07-16	2022-03-05	57.53	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428821	CDC
403638149	Active	2015-07-16	2022-03-05	57.52	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428822	CDC
403638150	Active	2015-07-16	2022-03-05	57.53	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428823	CDC
403638151	Active	2015-07-16	2022-03-05	2.23	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428824	CDC
403638152	Active	2015-07-16	2022-03-05	54.7	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428825	CDC
403638153	Active	2015-07-16	2022-03-05	57.52	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428826	CDC
403638154	Active	2015-07-16	2022-03-05	29.97	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428827	CDC
403638155	Active	2015-07-16	2022-03-05	0.14	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428828	CDC
403638156	Active	2015-07-16	2022-03-05	15.78	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428829	CDC

Claim No.	Claim Status	Issue Date	Anniversary Date	Area Ha.	Owner	Claim Name	Land Claim
403638157	Active	2015-07-16	2022-03-05	38.02	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428830	CDC
403638158	Active	2015-07-16	2022-03-05	47.79	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428831	CDC
403638160	Active	2015-07-16	2022-03-05	57.43	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428833	CDC
403638161	Active	2015-07-16	2022-03-05	19.17	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428834	CDC
403638162	Active	2015-07-16	2022-03-05	54.6	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428835	CDC
403638163	Active	2015-07-16	2022-03-05	6.78	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428836	CDC
403638164	Active	2015-07-16	2022-03-05	57.54	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428837	CDC
403638165	Active	2015-07-16	2022-03-05	57.52	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428838	CDC
403638166	Active	2015-07-16	2022-03-05	50.83	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428839	CDC
403645865	Active	2015-07-16	2022-03-05	41.68	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428840	CDC
403638168	Active	2015-07-16	2022-03-05	35.74	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428841	CDC
403638169	Active	2015-07-16	2022-03-05	12.49	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428842	CDC
403638171	Active	2015-07-16	2022-03-05	57.53	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428844	CDC
403638172	Active	2015-07-16	2022-03-05	2.12	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428845	CDC
403638173	Active	2015-07-16	2022-03-05	33.25	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428846	CDC
403638174	Active	2015-07-16	2022-03-05	40.48	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428847	CDC
403638175	Active	2015-07-16	2022-03-05	57.53	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428848	CDC
403638176	Active	2015-07-16	2022-03-05	37.4	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428849	CDC
403638179	Active	2015-07-16	2022-03-05	19.78	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428852	CDC
403638180	Active	2015-07-16	2022-03-05	1.05	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428853	CDC
403638181	Active	2015-07-16	2022-03-05	54.33	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428854	CDC
403638182	Active	2015-07-16	2022-03-05	51.35	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428855	CDC
403638183	Active	2015-07-16	2022-03-05	32.68	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428856	CDC
403638184	Active	2015-07-16	2022-03-05	57.44	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428857	CDC
403638185	Active	2015-07-16	2022-03-05	57.53	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428858	CDC
403638186	Active	2015-07-16	2022-03-05	50.94	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428859	CDC
403638187	Active	2015-07-16	2022-03-05	57.45	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428860	CDC

Claim No.	Claim Status	Issue Date	Anniversary Date	Area Ha.	Owner	Claim Name	Land Claim
403638189	Active	2015-07-16	2022-03-05	15.86	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428862	CDC
403638190	Active	2015-07-16	2022-03-05	5.59	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428863	CDC
403638191	Active	2015-07-16	2022-03-05	23.96	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428864	CDC
403645222	Active	2015-07-16	2022-03-05	56.76	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428865	CDC
403638193	Active	2015-07-16	2022-03-05	57.45	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428866	CDC
403638194	Active	2015-07-16	2022-03-05	57.53	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428867	CDC
403638195	Active	2015-07-16	2022-03-05	28.72	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428868	CDC
403638197	Active	2015-07-16	2022-03-05	8.79	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428870	CDC
403638198	Active	2015-07-16	2022-03-05	22.64	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428871	CDC
403638199	Active	2015-07-16	2022-03-05	4.81	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428872	CDC
403638201	Active	2015-07-16	2022-03-05	44.62	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428874	CDC
403638204	Active	2015-07-16	2022-03-05	14.98	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428877	CDC
403638206	Active	2015-07-16	2022-03-05	57.55	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428879	CDC
403638207	Active	2015-07-16	2022-03-05	57.45	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428880	CDC
403638209	Active	2015-07-16	2022-03-05	57.44	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428882	CDC
403638210	Active	2015-07-16	2022-03-05	25.48	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428883	CDC
403638211	Active	2015-07-16	2022-03-05	14.89	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428884	CDC
403638212	Active	2015-07-16	2022-03-05	57.51	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428885	CDC
403638213	Active	2015-07-16	2022-03-05	50.43	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428886	CDC
403638214	Active	2015-07-16	2022-03-05	29.97	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428887	CDC
403638215	Active	2015-07-16	2022-03-05	7.43	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428888	CDC
403638216	Active	2015-07-16	2022-03-05	33.96	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428889	CDC
403638217	Active	2015-07-16	2022-03-05	33.99	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428890	CDC
403638218	Active	2015-07-16	2022-03-05	57.53	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428891	CDC
403638219	Active	2015-07-16	2022-03-05	20.91	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428892	CDC
403638220	Active	2015-07-16	2022-03-05	29.99	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428893	CDC
403638221	Active	2015-07-16	2022-03-05	33.94	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428894	CDC

Claim No.	Claim Status	Issue Date	Anniversary Date	Area Ha.	Owner	Claim Name	Land Claim
403638224	Active	2015-07-16	2022-03-05	57.45	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428897	CDC
403638225	Active	2015-07-16	2022-03-05	57.52	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428898	CDC
403638226	Active	2015-07-16	2022-03-05	34.56	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428899	CDC
403638228	Active	2015-07-16	2022-03-05	54.26	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428901	CDC
403638229	Active	2015-07-16	2022-03-05	10.55	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428902	CDC
403645227	Active	2015-07-16	2022-03-05	55.6	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428904	CDC
403638232	Active	2015-07-16	2022-03-05	0.22	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428905	CDC
403638233	Active	2015-07-16	2022-03-05	43.02	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428906	CDC
403638234	Active	2015-07-16	2022-03-05	57.44	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428907	CDC
403638235	Active	2015-07-16	2022-03-05	57.44	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428908	CDC
403638237	Active	2015-07-16	2022-03-05	19.55	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428910	CDC
403638238	Active	2015-07-16	2022-03-05	37.4	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428911	CDC
403638239	Active	2015-07-16	2022-03-05	41.22	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428912	CDC
403638240	Active	2015-07-16	2022-03-05	11.53	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428913	CDC
403638242	Active	2015-07-16	2022-03-05	49.74	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428915	CDC
403638243	Active	2015-07-16	2022-03-05	1.98	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428916	CDC
403638244	Active	2015-07-16	2022-03-05	56.54	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428917	CDC
403638245	Active	2015-07-16	2022-03-05	31.08	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428918	CDC
403638247	Active	2015-07-16	2022-03-05	57.55	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428920	CDC
403638248	Active	2015-07-16	2022-03-05	52.79	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428921	CDC
403638249	Active	2015-07-16	2022-03-05	32.74	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428922	CDC
403638250	Active	2015-07-16	2022-03-05	57.52	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428923	CDC
403638251	Active	2015-07-16	2022-03-05	57.55	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428924	CDC
403638252	Active	2015-07-16	2022-03-05	13.47	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428925	CDC
403638253	Active	2015-07-16	2022-03-05	20.41	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428926	CDC
403638254	Active	2015-07-16	2022-03-05	57.54	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428927	CDC
403645873	Active	2015-07-16	2022-03-05	32.99	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428929	CDC

Claim No.	Claim Status	Issue Date	Anniversary Date	Area Ha.	Owner	Claim Name	Land Claim
403638257	Active	2015-07-16	2022-03-05	57.55	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428930	CDC
403638258	Active	2015-07-16	2022-03-05	55.52	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428931	CDC
403638260	Active	2015-07-16	2022-03-05	56.77	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428933	CDC
403638262	Active	2015-07-16	2022-03-05	57.44	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428935	CDC
403638263	Active	2015-07-16	2022-03-05	57.54	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428936	CDC
403638264	Active	2015-07-16	2022-03-05	2.08	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428937	CDC
403638266	Active	2015-07-16	2022-03-05	21.87	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428939	CDC
403638267	Active	2015-07-16	2022-03-05	57.44	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428940	CDC
403638268	Active	2015-07-16	2022-03-05	11.86	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428941	CDC
403645228	Active	2015-07-16	2022-03-05	54.48	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428942	CDC
403638270	Active	2015-07-16	2022-03-05	1.85	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428943	CDC
403638271	Active	2015-07-16	2022-03-05	57.53	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428944	CDC
403638272	Active	2015-07-16	2022-03-05	33.98	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428945	CDC
403638273	Active	2015-07-16	2022-03-05	30.07	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428946	CDC
403638274	Active	2015-07-16	2022-03-05	10.6	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2428947	CDC
403649342	Active	2016-09-20	2023-06-22	12.19	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2459317	CDC
403649343	Active	2016-09-20	2023-06-22	48.76	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2459318	CDC
403649344	Active	2016-09-20	2023-06-22	21.6	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2459319	CDC
403649345	Active	2016-09-20	2023-06-22	11.86	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2459320	CDC
403649346	Active	2016-09-20	2023-06-22	5.29	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2459321	CDC
403649347	Active	2016-09-20	2023-06-22	1.61	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2459322	CDC
403649348	Active	2016-09-20	2023-06-22	5.6	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2459323	CDC
403652165	Active	2016-12-22	2021-12-21	18.92	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2471205	CDC
403645862	Active	2016-12-22	2021-12-21	12.21	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2471206	CDC
403652162	Active	2016-12-22	2021-12-21	50.29	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2471207	CDC
403652166	Active	2016-12-22	2021-12-21	56.42	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2471208	CDC
403652167	Active	2016-12-22	2021-12-21	50.93	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2471209	CDC

Claim No.	Claim Status	Issue Date	Anniversary Date	Area Ha.	Owner	Claim Name	Land Claim
403645864	Active	2016-12-22	2021-12-21	31.43	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2471210	CDC
403645869	Active	2016-12-22	2021-12-21	41.65	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2471211	CDC
403645868	Active	2016-12-22	2021-12-21	41.29	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2471212	CDC
403645867	Active	2016-12-22	2021-12-21	41.49	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2471213	CDC
403645863	Active	2016-12-22	2021-12-21	29.53	Wesdome Gold Mines Ltd. (84889) 100% (responsable)	2471214	CDC

Appendix C: List of drillholes on the Kiena Mine Complex Property

List of drillholes of the 2019-2020 Program on the Kiena Mine Complex Property included in this MRE (UTM NAD 83 Zone 18N)

Survey	Easting	Northing	Elevation	Azimuth (°)	Dip	Length (m)	Status
6486	283172.72	5334034.94	-769.96	356.9	-46.7	100.5	In MRE
6487	283172.95	5334034.66	-769.66	5.0	-37.9	90.5	In MRE
6488	283171.66	5334034.93	-769.54	349.9	-23.7	101.4	In MRE
6489	283171.81	5334034.92	-769.64	323.7	-28.4	80.1	In MRE
6490	283171.76	5334034.98	-769.35	324.4	-15.9	122.6	In MRE
6491	283172.00	5334034.98	-769.86	327.7	-36.7	88.3	In MRE
6492	283172.15	5334034.91	-770.13	331.1	-55.6	77.5	In MRE
6498	283319.40	5333989.34	-766.83	222.5	-48.8	227.3	In MRE
6499	283319.42	5333989.32	-766.72	225.8	-41.3	227.6	In MRE
6502	283320.41	5333989.00	-767.17	197.2	-78.1	272.8	In MRE
6503	283320.48	5333989.17	-767.26	208.1	-72.8	240.6	In MRE
6504	283320.23	5333989.34	-766.89	211.0	-68.1	250.8	In MRE
6505	283480.70	5333856.72	-741.76	337.7	-81.6	518.5	In MRE
6506	283481.46	5333857.43	-739.63	8.1	-83.1	595.1	In MRE
6507	283480.70	5333856.61	-741.95	329.8	-80.5	529.2	In MRE
6508	283480.87	5333856.87	-741.75	337.1	-76.9	450.4	In MRE
6509	283481.11	5333856.87	-741.79	335.5	-76.5	455.0	In MRE
6520	283518.81	5333719.12	-735.24	351.5	-65.6	619.9	In MRE
6525	283172.81	5334034.72	-767.68	354.9	23.4	238.8	In MRE
6526	283172.90	5334034.30	-766.88	353.9	37.1	247.6	In MRE
6528	283174.11	5334033.63	-767.78	34.4	22.4	186.0	In MRE
6530	283172.88	5334034.62	-767.96	4.3	16.7	231.0	In MRE
6534	283172.22	5334034.94	-766.60	341.4	38.6	233.0	In MRE
6535	283172.21	5334034.98	-767.29	343.8	30.5	245.4	In MRE

Survey	Easting	Northing	Elevation	Azimuth (°)	Dip	Length (m)	Status
6536	283172.14	5334035.12	-767.55	343.8	24.5	278.0	In MRE
6537	283172.16	5334035.16	-767.92	344.1	17.5	302.5	In MRE
6538	283331.36	5333919.81	-761.54	359.7	-84.8	321.8	In MRE
6539	283331.82	5333919.95	-761.52	359.1	-73.7	269.3	In MRE
6540	283331.58	5333920.18	-761.63	358.5	-75.1	301.3	In MRE
6541	283331.72	5333920.05	-761.63	19.3	-82.9	287.3	In MRE
6542	283331.94	5333919.64	-761.43	21.5	-75.0	266.0	In MRE
6543	283332.10	5333919.41	-761.38	28.0	-75.5	311.0	In MRE
6544	283332.39	5333919.22	-761.39	41.9	-81.6	310.6	In MRE
6545	283332.64	5333919.28	-761.44	40.9	-74.6	355.0	In MRE
6546	283137.54	5334114.50	-374.84	325.2	-41.1	502.5	In MRE
6547	283137.47	5334114.48	-374.93	334.8	-40.1	527.7	In MRE
6548	283137.05	5334114.30	-374.76	322.1	-36.4	512.5	In MRE
6549	283462.04	5333840.18	-741.99	314.5	-50.8	331.0	In MRE
6550	283462.18	5333840.05	-742.23	313.8	-62.6	339.6	In MRE
6551	283462.25	5333839.87	-742.36	317.4	-65.6	356.0	In MRE
6552	283462.27	5333839.64	-742.17	314.7	-68.6	460.7	In MRE
6553	283462.43	5333839.51	-742.19	319.7	-72.8	383.9	In MRE
6554	283462.60	5333839.64	-742.42	320.5	-81.0	454.0	In MRE
6555	283519.26	5333718.89	-735.06	2.9	-67.2	719.0	In MRE
6556	283137.05	5334114.30	-374.76	320.6	-29.1	517.0	In MRE
6557	283137.40	5334114.46	-374.78	319.5	-34.0	503.7	In MRE
6558	283178.86	5334027.44	-770.68	39.5	-79.1	108.4	In MRE
6559	283179.24	5334027.73	-770.69	40.6	-64.0	135.2	In MRE
6560	283179.44	5334027.86	-770.36	43.9	-51.9	101.4	In MRE
6561	283179.38	5334027.88	-769.91	44.4	-38.6	89.5	In MRE
6562	283179.35	5334027.82	-769.57	43.0	-24.8	81.7	In MRE

Survey	Easting	Northing	Elevation	Azimuth (°)	Dip	Length (m)	Status
6563	283179.40	5334027.83	-769.22	44.5	-11.2	87.0	In MRE
6564	283179.36	5334027.80	-768.41	42.9	11.7	100.1	In MRE
6565	283179.36	5334027.71	-767.40	43.2	29.2	104.5	In MRE
6566	283190.86	5334014.96	-771.08	43.3	-73.6	135.0	In MRE
6567	283191.13	5334014.71	-770.85	40.6	-67.6	121.2	In MRE
6568	283191.00	5334015.05	-770.92	40.1	-59.2	153.9	In MRE
6569	283191.17	5334014.98	-770.88	37.9	-54.7	24.0	In MRE
6570	283191.26	5334014.99	-770.73	42.2	-44.1	90.5	In MRE
6571	283191.18	5334014.92	-770.23	40.1	-32.8	86.4	In MRE
6572	283191.10	5334014.93	-769.90	42.4	-20.6	90.5	In MRE
6573	283191.18	5334015.00	-769.46	43.6	-2.9	82.5	In MRE
6574	283191.11	5334014.94	-768.73	44.6	12.3	92.2	In MRE
6575	283191.11	5334014.91	-768.27	46.3	23.1	101.4	In MRE
6576	283191.03	5334015.16	-767.72	44.0	32.7	116.5	In MRE
6577	283137.32	5334114.34	-375.03	317.3	-45.8	417.8	In MRE
6578	283137.21	5334114.36	-374.95	316.5	-39.6	403.1	In MRE
6579	283519.44	5333718.97	-735.17	8.3	-66.1	555.6	In MRE
6580	283520.01	5333718.58	-735.25	3.7	-69.0	781.8	In MRE
6581	283519.85	5333718.62	-735.28	351.8	-69.8	655.6	In MRE
6582	283342.61	5333771.03	-721.77	37.8	-70.7	579.8	In MRE
6583	283519.69	5333718.75	-735.26	355.9	-67.9	680.4	In MRE
6584	283342.55	5333772.50	-723.87	30.7	-66.5	621.0	In MRE
6585	283519.39	5333718.32	-733.58	13.5	-76.4	667.0	In MRE
6586	283137.20	5334114.38	-374.59	317.3	-22.9	491.0	In MRE
6587	283137.25	5334114.42	-374.45	315.8	-18.0	560.6	In MRE
6588	283137.12	5334114.45	-374.39	316.3	-13.4	526.0	In MRE
6589	283137.10	5334114.47	-374.30	316.5	-7.9	526.1	In MRE

Survey	Easting	Northing	Elevation	Azimuth (°)	Dip	Length (m)	Status
6590	283522.58	5333717.94	-733.48	66.1	2.5	197.8	In MRE
6591	283522.49	5333716.79	-733.40	84.6	3.9	255.8	In MRE
6592	283399.80	5333881.96	-751.56	288.1	-68.2	295.2	In MRE
6593	283398.75	5333880.98	-751.40	280.8	-64.2	290.4	In MRE
6594	283398.75	5333880.98	-751.40	274.6	-59.5	267.0	In MRE
6595	283398.75	5333880.98	-751.40	267.0	-56.4	250.0	In MRE
6596	283398.75	5333880.98	-751.40	273.0	-71.1	337.7	In MRE
6597	283398.75	5333880.98	-751.40	269.4	-65.7	301.0	In MRE
6598	283398.75	5333880.98	-751.40	274.1	-76.0	351.7	In MRE
6599	283398.71	5333880.97	-751.39	267.2	-73.2	314.5	In MRE
6600	283398.76	5333880.71	-751.50	256.7	-67.1	293.9	In MRE
6601	283464.01	5333840.48	-742.11	353.6	-79.2	507.4	In MRE
6602	283463.56	5333840.48	-742.13	354.9	-79.1	20.9	In MRE
6603	283464.13	5333840.69	-742.11	359.8	-74.6	369.0	In MRE
6604	283464.37	5333840.46	-742.10	359.9	-79.4	516.9	In MRE
6605	283464.73	5333840.82	-742.11	2.4	-77.5	23.9	In MRE
6606	283201.44	5334003.94	-771.35	44.5	-68.2	162.7	In MRE
6607	283201.71	5334004.25	-771.03	41.4	-44.7	86.0	In MRE
6608	283201.65	5334004.19	-769.26	42.1	13.5	87.0	In MRE
6609	283201.57	5334004.11	-768.64	43.3	25.8	126.0	In MRE
6610	283201.59	5334004.12	-768.00	46.5	32.5	140.0	In MRE
6611	283224.96	5333990.41	-771.73	44.5	-84.9	141.8	In MRE
6612	283225.22	5333990.19	-771.68	41.9	-77.3	141.8	In MRE
6613	283225.47	5333990.41	-771.58	41.6	-64.0	138.3	In MRE
6614	283225.84	5333990.67	-769.65	44.6	6.1	89.0	In MRE
6615	283225.86	5333990.67	-769.24	45.8	13.6	89.8	In MRE
6616	283137.04	5334114.34	-374.64	312.3	-13.7	445.2	In MRE

Survey	Easting	Northing	Elevation	Azimuth (°)	Dip	Length (m)	Status
6617	283136.97	5334114.37	-374.36	313.1	-10.2	395.4	In MRE
6618	283136.98	5334114.31	-374.63	312.2	-24.0	458.7	In MRE
6619	283137.03	5334114.38	-374.78	313.3	-28.2	468.7	In MRE
6620	283137.01	5334114.41	-374.90	312.8	-33.6	282.6	In MRE
6621	283242.04	5333960.29	-772.65	44.4	-83.6	190.0	In MRE
6622	283241.85	5333960.50	-772.75	41.1	-78.7	173.0	In MRE
6623	283241.83	5333960.75	-772.66	42.6	-73.4	152.8	In MRE
6624	283241.83	5333960.75	-772.66	40.9	-66.8	187.0	In MRE
6625	283241.83	5333960.75	-772.65	44.1	-60.6	164.5	In MRE
6626	283242.45	5333960.57	-772.61	38.8	-54.9	141.4	In MRE
6627	283118.79	5333998.12	-520.45	287.5	1.9	527.3	In MRE
6628	283118.56	5333997.34	-520.45	262.2	2.1	350.3	In MRE
6629	283399.88	5333882.41	-751.38	297.6	-64.0	308.4	In MRE
6630	283399.60	5333882.07	-751.47	286.3	-64.9	280.1	In MRE
6631	283399.54	5333882.03	-753.08	271.9	-57.1	302.4	In MRE
6632	283399.76	5333882.24	-753.08	281.3	-70.5	277.6	In MRE
6633	283398.88	5333881.84	-751.20	276.0	-46.4	282.6	In MRE
6634	283399.28	5333882.13	-751.23	269.5	-47.3	269.9	In MRE
6635	283118.13	5333995.47	-520.37	235.8	2.6	295.5	In MRE
6636	283119.01	5333998.59	-520.75	292.3	2.0	462.0	In MRE
6638	282961.19	5333848.33	4.86	251.2	-7.6	197.6	In MRE
6639	282960.89	5333848.24	5.38	249.8	8.6	150.6	In MRE
6640	282960.79	5333848.27	6.04	252.3	22.9	150.3	In MRE
6641	282960.76	5333848.27	6.59	251.4	32.0	165.7	In MRE
6642	283128.01	5334183.02	-544.32	316.4	-17.8	419.0	In MRE
6643	283127.92	5334183.06	-544.19	314.1	-14.0	425.0	In MRE
6644	283127.89	5334183.06	-544.12	313.7	-9.8	425.0	In MRE

Survey	Easting	Northing	Elevation	Azimuth (°)	Dip	Length (m)	Status
6645	283127.95	5334182.98	-543.95	313.7	-5.2	425.0	In MRE
6646	283242.23	5333960.75	-772.18	42.1	-45.6	110.3	In MRE
6647	283242.18	5333960.77	-771.77	42.5	-29.4	110.0	In MRE
6648	283242.20	5333960.76	-771.41	43.6	-10.4	99.8	In MRE
6649	283242.28	5333960.85	-770.56	44.0	10.0	96.4	In MRE
6650	283242.34	5333960.76	-768.95	48.2	36.1	102.0	In MRE
6651	283143.09	5334293.93	-555.50	303.8	-19.2	349.1	In MRE
6652	283143.19	5334293.71	-555.46	302.9	-16.6	300.1	In MRE
6653	283143.21	5334293.87	-555.24	303.0	-9.8	349.9	In MRE
6654	283143.21	5334293.87	-555.24	303.0	-8.2	349.4	In MRE
6655	283238.95	5333957.66	-772.76	237.9	-84.0	163.6	In MRE
6656	283238.68	5333957.44	-772.73	233.3	-74.6	136.4	In MRE
6657	283233.77	5333967.35	-772.48	45.8	-76.9	160.0	In MRE
6658	283233.99	5333967.56	-772.51	39.6	-68.2	189.5	In MRE
6659	283234.41	5333967.57	-772.51	41.2	-61.2	150.9	In MRE
6660	283400.15	5333882.59	-751.35	309.3	-77.8	349.6	In MRE
6661	283400.11	5333882.15	-751.36	294.3	-75.8	379.1	In MRE
6662	283399.18	5333881.85	-751.44	286.3	-64.7	281.7	In MRE
6663	283398.82	5333881.46	-751.37	275.1	-53.1	309.7	In MRE
6664	283398.91	5333880.76	-751.36	264.4	-62.9	275.0	In MRE
6665	283227.31	5333976.13	-771.67	52.9	-86.8	115.5	In MRE
6667	283227.02	5333976.43	-771.65	42.1	-79.1	117.0	In MRE
6668	283227.29	5333976.70	-771.67	41.0	-72.9	213.8	In MRE
6669	283227.84	5333976.64	-771.80	41.3	-66.1	172.3	In MRE
6670	283234.36	5333967.74	-772.51	39.0	-60.9	139.3	In MRE
6672	283465.02	5333840.51	-741.93	9.5	-79.0	582.7	In MRE
6673	283227.50	5333976.92	-770.53	41.5	-22.2	107.4	In MRE

Survey	Easting	Northing	Elevation	Azimuth (°)	Dip	Length (m)	Status
6674	283227.47	5333976.92	-770.28	39.7	-8.8	100.1	In MRE
6675	283227.47	5333976.90	-769.93	41.0	2.4	99.7	In MRE
6676	283227.52	5333976.78	-769.26	45.0	15.0	105.4	In MRE
6677	283518.58	5333719.28	-735.86	358.0	-69.0	662.0	In MRE
6678	283127.24	5334182.66	-545.01	313.9	-47.3	255.5	In MRE
6679	283127.17	5334182.75	-544.92	313.7	-39.8	227.3	In MRE
6680	283127.12	5334182.83	-544.72	313.8	-31.5	233.3	In MRE
6681	283143.24	5334293.88	-555.97	302.5	-47.8	146.2	In MRE
6682	283143.15	5334293.93	-555.88	303.4	-39.2	174.0	In MRE
6683	283143.09	5334293.96	-555.78	303.3	-31.4	41.5	In MRE
6684	283399.08	5333880.91	-751.44	265.0	-79.9	377.0	In MRE
6685	283399.04	5333880.64	-751.58	257.6	-76.0	336.8	In MRE
6686	283399.05	5333880.50	-751.37	248.5	-71.7	325.4	In MRE
6688	283143.05	5334294.06	-555.53	301.9	-21.5	365.3	In MRE
6691	283143.13	5334294.17	-555.05	302.0	-5.0	343.0	In MRE
6703	283126.95	5334182.84	-544.48	316.7	-22.0	428.3	In MRE
6704	283342.39	5333772.62	-723.85	28.0	-65.0	553.5	In MRE
4810AW1	283528.84	5333697.49	-1072.75	154.9	-56.3	144.0	In MRE
6126AW1	283197.27	5334087.22	-668.74	96.5	-61.8	94.0	In MRE
6153AW1	283313.52	5333986.55	-909.52	26.8	-66.5	108.0	In MRE
6384AW1	283501.77	5333909.10	-1041.95	13.7	-79.3	336.9	In MRE
6479AW1	283546.49	5333880.40	-958.62	320.1	-75.2	100.0	In MRE
6510AW1	283523.97	5333894.62	-1010.90	324.1	-73.3	287.6	In MRE
6520AW1	283509.36	5333850.83	-1026.74	354.3	-65.9	361.7	In MRE
6555AW1	283535.85	5333840.25	-1036.09	7.7	-68.2	270.2	In MRE
6569A	283191.17	5334014.92	-770.91	41.8	-52.5	108.8	In MRE
6580AW1	283548.21	5333875.28	-1156.11	11.3	-69.7	310.4	In MRE

Survey	Easting	Northing	Elevation	Azimuth (°)	Dip	Length (m)	Status
6580BW2	283546.17	5333866.90	-1132.64	11.7	-69.8	347.1	In MRE
6580CW3	283538.56	5333833.19	-1038.79	9.5	-69.6	439.5	In MRE
6581W1	283514.09	5333831.43	-1053.42	356.5	-71.1	356.7	In MRE
6581W2	283514.26	5333824.13	-1032.35	356.5	-70.7	405.4	In MRE
6581W3	283515.53	5333785.89	-923.58	355.4	-70.5	519.9	In MRE
6582W1	283410.01	5333843.65	-1004.62	42.0	-71.1	319.9	In MRE
6582W2	283404.77	5333838.22	-982.78	41.9	-71.0	352.4	In MRE
6583W1	283520.39	5333846.41	-1055.77	359.5	-68.6	367.5	In MRE
6583W2	283520.10	5333836.45	-1030.35	359.5	-68.5	469.3	In MRE
6583W3	283569.48	5333947.11	-1324.26	43.0	-65.9	166.6	In MRE
6583W4	283553.43	5333928.34	-1269.05	33.4	-65.9	204.8	In MRE
6584W1	283424.02	5333887.28	-1055.19	35.5	-67.5	272.2	In MRE
6584W2	283418.67	5333880.39	-1034.12	35.9	-67.6	286.0	In MRE
6584W3	283409.90	5333868.92	-999.19	35.3	-67.6	272.9	In MRE
6602A	283463.77	5333840.46	-742.12	350.5	-79.0	511.4	In MRE
6603A	283464.62	5333840.18	-742.11	359.3	-75.2	496.9	In MRE
6603AW1	283470.22	5333892.76	-956.07	5.7	-76.7	326.1	In MRE
6603AW2	283467.92	5333876.07	-886.89	5.9	-76.3	379.5	In MRE
6605A	283464.28	5333840.13	-742.10	357.7	-78.6	535.6	In MRE
6605AW1	283468.50	5333887.41	-987.85	6.5	-79.3	299.6	In MRE
6605AW2	283467.82	5333882.75	-962.79	5.9	-79.4	381.3	In MRE
6605AW3	283465.63	5333864.74	-865.66	5.3	-79.5	410.6	In MRE
6637A	283118.80	5333998.45	-520.22	292.4	7.2	182.5	In MRE
6666A	283227.17	5333976.32	-771.66	35.7	-82.5	153.6	In MRE
6672W1	283478.96	5333895.67	-1010.98	11.1	-78.1	347.3	In MRE
6677W1	283516.43	5333849.03	-1114.75	359.0	-71.5	275.1	In MRE
6704W1	283413.76	5333881.98	-1032.43	33.2	-67.7	202.6	In MRE

List of drillholes not included in the MRE (September 19, 2020 to April 7, 2021) on the Kiena Mine Complex Property (UTM NAD 83 Zone 18N)

Survey	Easting	Northing	Elevation	Azimuth (°)	Dip	Length (m)	Status
6637	283118.9	5333998	-520.13	291.4	4.7	18	Post MRE
6666	283227.3	5333976	-772.13	41.7	-82	25.1	Post MRE
6671	283234.3	5333968	-772.52	37.1	-54.5	138.6	Post MRE
6687	283399	5333881	-751.59	307	-83	365.3	Post MRE
6689	283143.1	5334294	-555.61	306.3	-23.5	251.5	Post MRE
6690	283519.5	5333719	-735.23	359	-69	26.3	Post MRE
6692	283126.4	5334183	-545.13	306	-13	450	Post MRE
6693	283143.2	5334294	-555.24	302.5	2.8	265.5	Post MRE
6694	283143.2	5334294	-555.24	311	-13	324.5	Post MRE
6695	283143.2	5334294	-555.24	311	-16	350	Post MRE
6696	283142.3	5334293	-555.23	311	-19	351.1	Post MRE
6697	283142.3	5334293	-555.23	311	-22	350	Post MRE
6698	283127.4	5334183	-544.13	318	-4	450	Post MRE
6699	283127.9	5334183	-544.06	316	-6	450.4	Post MRE
6700	283126.8	5334182	-544.13	315	-10.6	452.5	Post MRE
6701	283126.9	5334183	-544.21	316	-13	464.5	Post MRE
6702	283126.8	5334183	-544.08	318.8	-19	444.5	Post MRE
6705	283399.5	5333881	-751.58	276	-79	360.9	Post MRE
6706	283399.7	5333881	-751.57	263	-86	381.8	Post MRE
6707	283399.4	5333880	-751.32	262.3	-82.3	371.1	Post MRE
6708	283398.5	5333880	-751.23	249	-79	24.3	Post MRE
6709	283398.3	5333880	-751.39	255.7	-72.8	281.5	Post MRE
6710	283208.9	5333996	-770.41	45	-30	99.5	Post MRE

Survey	Easting	Northing	Elevation	Azimuth (°)	Dip	Length (m)	Status
6711	283208.5	5333996	-771.46	25	-88	183	Post MRE
6712	283209.9	5333995	-771.63	45.5	-81.9	149.3	Post MRE
6713	283209.9	5333995	-771.63	45	-78	153.7	Post MRE
6714	283203.2	5333995	-769.34	276.2	5.9	93	Post MRE
6715	283209.9	5333995	-771.13	40.9	-19.8	90.8	Post MRE
6716	283209.9	5333995	-771.13	45	-5	84.7	Post MRE
6717	283209	5333996	-769.35	44	9	90.9	Post MRE
6718	283342.2	5333773	-724.13	19.3	-65	434.4	Post MRE
6720	283203.3	5333994	-769.43	260.2	5.8	330	Post MRE
6721	283226.1	5333990	-771.4	86	-60	300	Post MRE
6722	283224.8	5333991	-771.13	87	-63	262.6	Post MRE
6723	283224.8	5333991	-771.13	87	-59	217.5	Post MRE
6724	283224.8	5333991	-771.13	87	-56	240	Post MRE
6725	283224.8	5333991	-771.13	77	-77	250	Post MRE
6726	283224.8	5333991	-771.13	77	-63	275	Post MRE
6727	283224.8	5333991	-771.13	77	-59	230	Post MRE
6728	283224.8	5333991	-771.13	65	-72	230	Post MRE
6729	283397.8	5333880	-751.63	304.3	-89	413.1	Post MRE
6730	283397.8	5333880	-751.63	350	-84	400	Post MRE
6731	283397.8	5333880	-751.63	3	-81	400	Post MRE
6732	283397.8	5333880	-751.63	330	-80	380.6	Post MRE
6733	283397.8	5333880	-751.63	17	-77	420	Post MRE
6734	283400.2	5333882	-751.45	351.6	-77	380	Post MRE
6735	283397.8	5333880	-751.63	0	-74	380	Post MRE
6736	283399.8	5333880	-751.34	299.5	-82	354.8	Post MRE

Survey	Easting	Northing	Elevation	Azimuth (°)	Dip	Length (m)	Status
6737	283342.2	5333773	-724.13	20.4	-67.5	493.4	Post MRE
6738	283119.9	5334000	-521.18	329.6	-30	350	Post MRE
6739	283519.2	5333719	-735.86	2.4	-67.5	670.9	Post MRE
6740	283465.3	5333841	-742.14	13	-77.5	606.8	Post MRE
6741	283342.5	5333772	-723.56	22.9	-68.5	525.9	Post MRE
6742	283342.5	5333773	-723.53	25	-62.6	603.5	Post MRE
6743	283126.4	5334183	-545.13	306	-7	450	Post MRE
6744	283126.4	5334183	-545.13	306	-1	450	Post MRE
6745	283126.4	5334183	-545.13	306	5	450	Post MRE
6746	283126.4	5334183	-545.13	306	11	450	Post MRE
6672W2	283490.6	5333917	-1135.24	33.5	-79.2	229.3	Post MRE
6672W3	283487.6	5333913	-1108.22	32.6	-78.7	235.6	Post MRE
6672W4	283490.7	5333917	-1133.76	41	-79.3	221.4	Post MRE
6690A	283519.5	5333719	-735.23	354	-69	21.3	Post MRE
6690B	283519.1	5333719	-735.86	358.4	-68.7	718.7	Post MRE
6690BW1	283525.1	5333900	-1222.81	0.4	-70.2	191	Post MRE
6690BW2	283525.8	5333905	-1235.7	0.7	-70.2	163.3	Post MRE
6690BW3	283524.5	5333885	-1180.38	0	-70.2	226.2	Post MRE
6690BW4	283524.2	5333876	-1156.49	0.4	-70.2	262.6	Post MRE
6704W2	283411.5	5333879	-1023.09	33.4	-67.7	218.6	Post MRE
6704W3	283406	5333871	-999.76	33.4	-67.8	303.7	Post MRE
6704W4	283400.5	5333863	-976.93	32.7	-67.5	359.1	Post MRE
6708A	283398.5	5333880	-751.27	249.4	-79.1	354.6	Post MRE
6714A	283203.3	5333995	-769.21	272.5	10.5	76	Post MRE
6714B	283202.6	5333995	-769.65	277.7	0.4	353	Post MRE

Survey	Easting	Northing	Elevation	Azimuth (°)	Dip	Length (m)	Status
6719B	283206.3	5333992	-771.63	284	0	85.6	Post MRE
6719C	283206.3	5333992	-771.63	284	-5	80	Post MRE
6737W1	283383	5333861	-955.71	24.3	-67.6	242.8	Post MRE
6738A	283120	5334000	-521.22	329.6	-34	383	Post MRE
6738C	283119	5333999	-521.13	330	-30	700	Post MRE
6739W1	283536.7	5333907	-1176.82	3.2	-66.3	193	Post MRE
6739W2	283534.4	5333886	-1128.63	4.5	-66.4	241.5	Post MRE
6739W3	283532.1	5333865	-1080.5	3.5	-66.6	400	Post MRE
6739W4	283530.4	5333847	-1038.08	3	-66.7	300	Post MRE
6740W1	283483.7	5333896	-1033.21	17.3	-79.8	340.5	Post MRE
6740W2	283490.9	5333908	-1108.08	34.5	-79	280.1	Post MRE
6740W3	283513.4	5333929	-1251.26	53.7	-77.7	400	Post MRE
6742W1	283409	5333905	-1012.03	24.8	-62.8	271.8	Post MRE
6742W2	283398.8	5333885	-968.3	24.8	-62.8	500	Post MRE
6740W4	283481.7	5333891	-1002.03	18.5	-79.5	382.2	Post MRE
6742W3	283393.6	5333875	-946.05	24.9	-62.9	425.9	Post MRE
6747	283171.7	5334035	-769.876	330	-36	13.7	Post MRE
6748	283171.5	5334035	-770.292	330	-44	85.2	Post MRE
6749	283171.3	5334035	-769.866	336	-36	115.1	Post MRE
6750	283171.8	5334035	-769.83	342	-36	95.5	Post MRE
6751	283171.8	5334035	-769.83	342	-49	111	Post MRE
6738CW1	283029.6	5334169	-604.93	331.6	-19.1	378.5	Post MRE
6739W5	283528.4	5333825	-986.91	2.7	-66.9	289	Post MRE
6739W6	283527.5	5333814	-960.28	3	-66.7	590.6	Post MRE
6742W4	283418	5333916	-1034.4	31.1	-60.9	239.4	Post MRE

Survey	Easting	Northing	Elevation	Azimuth (°)	Dip	Length (m)	Status
6740W5	283497.4	5333921	-1158.79	32	-77.5	250.6	Post MRE
6742W5	283424.8	5333926	-1055.73	34.7	-60.9	2.5	Post MRE
6742W6	283422.2	5333922	-1047.69	33.8	-61	351.8	Post MRE
6738CW2	283044	5334140	-593.22	331.1	-21.1	418.8	Post MRE
6740W6	283507.3	5333932	-1226.33	44.6	-77.9	147.5	Post MRE
6760	283519.2	5333719	-736.13	358.5	-67.5	688	Post MRE
6747B	283171.5	5334035	-769.965	320	-36	114	Post MRE
6761	283120.7	5333706	-72.13	48	-1.5	676.2	Post MRE
6742W7	283368.1	5333824	-835.44	24.7	-62.8	631.1	Post MRE
6762	283465.7	5333841	-742.13	358	-74	516.7	Post MRE
6763	283120	5334000	-521.13	335	-30	554.2	Post MRE
6760W1	283521.1	5333784	-892.81	360	-67.5	732.2	Post MRE
6757	283171.8	5334035	-769.83	342	-17	121.7	Post MRE
6758	283171.8	5334035	-769.83	342	-13	121.8	Post MRE
6759	283171.8	5334035	-769.83	336	-16	130.6	Post MRE
6764	283171.8	5334035	-769.83	337	-11.5	114.4	Post MRE
6765	283171.8	5334035	-769.83	330	-39	24.5	Post MRE
6766	283171.8	5334035	-769.83	328	-23.7	106.7	Post MRE
6767	283171.8	5334035	-769.83	329.5	-16	130.5	Post MRE
6768	283171.8	5334035	-769.83	324	-24	126.6	Post MRE
6769	283171.8	5334035	-769.83	324	-35	130	Post MRE
6762W1	283467.9	5333901	-959.15	2.5	-74.7	377.7	Post MRE
6742W8	283413.2	5333895	-989.57	37.5	-61.5	255.8	Post MRE
6770	283125.8	5334180	-545.13	256	3	521	Post MRE
6771	283521.3	5333715	-734.13	82	-15	235.5	Post MRE

Survey	Easting	Northing	Elevation	Azimuth (°)	Dip	Length (m)	Status
6772	283521.3	5333715	-734.13	73.6	-25	258.1	Post MRE
6773	283120.7	5333706	-72.13	70	-2	600	Post MRE
6742W9	283405.5	5333885	-966.82	35.45	-61.5	291.5	Post MRE
6762W2	283467.4	5333894	-935.35	0.6	-74.5	427.1	Post MRE
6763W1	283053.3	5334143	-599.24	332.8	-24.3	405	Post MRE
N098-K001	283170.1	5333795	-687.4	254	22	50.5	Post MRE
6774	283556.5	5333690	-732.13	237	-87	575	Post MRE
N098-K002	283169.7	5333796	-688.759	276	-10	41.1	Post MRE
N098-K003	283169.6	5333796	-689.171	274	-25	41.3	Post MRE
N098-K004	283169.9	5333794	-689.446	212	-27	34.8	Post MRE
N098-K005	283220.3	5333773	-689.442	232	-20	49.6	Post MRE
N098-K006	283220.8	5333773	-689.369	221	-20	45	Post MRE
6742W10	283398	5333875	-943.71	33.3	-61.5	480.8	Post MRE
N098-K007	283170.1	5333794	-687.861	221	15	47.4	Post MRE
N098-K005A	283222.9	5333774	-688.81	234	-20	14.1	Post MRE
6765A	283171.8	5334035	-769.63	328.5	-39	91.1	Post MRE
6762W3	283469.5	5333907	-982.85	8.9	-74.5	500	Post MRE
6742W11	283373.3	5333835	-856.68	25.5	-61.2	500	Post MRE
6775	283125.8	5334180	-545.13	262	3	500	Post MRE
6763W2	269369.2	5321724	-2743.13	0	0	600	Post MRE

Appendix D: Electrical Single Line Diagram

FROM HQ 25 kV

3x1c AAAC, 300 MCM

3x1c AAAC, 300 MCM

MAIN BREAKER
1200 A
52

1200 A

630 A
52
50 51N
SEL-351A

1x3c-1/0 AWG
TECK 90, 25 kV,
100%, CU

25 kV-600 V
1.5/2.0 MVA
ANN/AFN
80 DEG RISE

SE-330

NGR
5 A

9x1c-750 MCM,
TECK 90, 1 kV,
CU

ETP AREA

52
52
600 V

NEW
EFFLUENT TREATMENT PLANT
ESTIMATED LOAD: 0.95 MVA

GEN

EMERGENCY GENSET
250 kW, 600 V
3 PHASES, 60 Hz

VER

21000 V
27000 V
600 A
65 A
T1
5000 kVA
25000/4160 V

CONCENTRATOR
REF. 1110001

21000 V
27000 V
600 A
150 A
T7
5400 kVA
25000/2400 V

400 A

NEW PASTE
PLANT

21000 V
27000 V
600 A
150 A
T11
5400 kVA
25000/4160 V

1200 A

NEW SURFACE MINE
VENTILATION E-HOUSE

VER

21000 V
27000 V
600 A
65 A
T10
1500 kVA
25000/600 V

EMERGENCY GENERATOR BUILDING 2.4 kV

2400 V

GEN

EMERGENCY GENSET
1000 kW, 2400 V
3 PHASES, 60 Hz

400 A

MILL FEEDER
REF: 111001

400 A

MINE FEEDER

40 A

2400 V

TRANSFORMER T-8606
75 kVA

600 V

STATION SERVICE
TRANSFORMER

3x1c-AAAC

CUT-OUT FUSE

3x1c-AAAC

VER

2.4 kV - 600 V
3x 75 kVA

1x3c-350 MCM,
TECK 90, 1 kV,
CU

NEW PASTE PLANT
ESTIMATED LOAD:
100 kVA

1200 A

630 A
52
50 51N
SEL-351A

1x3c-1/0 AWG
TECK 90, 25 kV
100%, Cu

VER

25 kV-13,8 kV
7.5/10 MVA
ANN/ANF
150°C RISE

SE-330

NGR
50 A

1x3c-350 MCM
CU, 15 kV, 133%

1200 A

ACSR

50 51N
SEL-351A

630 A
52

1x3c-350 MCM
RISER TECK
CU, 15 kV, 133%

UNDERGROUND
EXPANSION
TO KIENA DEEP ZONE (LEVEL 111)
AND S50 ZONE (LEVEL 96)

FOR INFORMATION
NOT TO BE USED FOR CONSTRUCTION
DATE: 2021-06-04

BBA

DRAWING No.	DESCRIPTION	REV	DESCRIPTION	VERIFIED BY	APPROVED BY	DATE
REFERENCE DRAWINGS		REVISIONS				
		AA	FOR INFORMATION	S. GOHIER-ST-JEAN, P.Eng. O/Q No. 5068350	M.-A. SIOUI, P.Eng. O/Q No. 143597	2021-06-04

SEAL:

CLIENT:

BBA

PROJECT:
KIENA MINE COMPLEX

TITLE:
SURFACE DISTRIBUTION
25 kV / 13.8 kV / 600 V
SINGLE LINE DIAGRAM

DESIGNED BY: S. GOHIER-ST-JEAN, P.Eng. O/Q No. 5068350	DRAFTED BY: A. BERNATCHEZ
VERIFIED BY: M.-A. SIOUI, P.Eng. O/Q No. 143597	APPROVED BY: M.-A. SIOUI, P.Eng. O/Q No. 143597
SCALE: NTS	DATE: 2020-11-30

DRAWING No.: 3767003-403000-47-D21-0002 | SHEET: 01 | SIZE: D | REV. AA

0 1 2 3 4 5 6 7 8 9 10 cm

G:\3767\003\40_ING_ENG\47_ELEC\DESS_DWGS\PROJET_PROJECT\3767003-403000-47-D21-0002.DWG

BY:PROVOST, ROXANE
PRINTED:2021-06-04 13:49:33

BBA