TECHNICAL REPORT AND PRELIMINARY ECONOMIC ASSESSMENT FOR GRAVITY MILLING AND HEAP LEACH PROCESSING AT THE NORTH BULLFROG PROJECT,

BULLFROG MINING DISTRICT,

NYE COUNTY, NEVADA

DATED: NOVEMBER 21, 2020

EFFECTIVE DATE: OCTOBER 7 , 2020

PREPARED FOR:

CORVUS GOLD INC.

BY

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Corvus Gold Inc.

Technical Report and Preliminary Economic Assessment for Gravity Milling and Heap Leach Processing at the North Bullfrog Property, Bullfrog Mining District, Nye County, Nevada

Technical Report Effective Date: October 7, 2020

Dated this 21st Day of November 2020.

(signed/sealed) *Scott E. Wilson*
Scott E. Wilson, SME-RM, CPG
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(signed) *Michael R. Young*
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AUTHOR'S CERTIFICATE

Scott E. Wilson

I, Scott E. Wilson, CPG, SME-RM, of Highlands Ranch, Colorado, as the lead author of the technical report entitled "Technical Report and Preliminary Economic Assessment of Gravity Milling and Heap Leach Processing at the North Bullfrog Project, Bullfrog Mining District, Nye County, Nevada" (the "Technical Report") with an effective date of October 7, 2020, prepared for Corvus Gold Inc. (the "Issuer"), do hereby certify:

1. I am currently employed as President by Resource Development Associates, Inc., 10262 Willowbridge Way, Highlands Ranch, Colorado 80126, USA.
2. I graduated with a Bachelor of Arts degree in Geology from the California State University, Sacramento in 1989.
3. I am a Certified Professional Geologist and member of the American Institute of Professional Geologists (CPG #10965) and a Registered Member (#4025107) of the Society for Mining, Metallurgy and Exploration, Inc.
4. I have been employed as both a geologist and a mining engineer continuously for a total of 31 years. My experience includes resource estimation, mine planning, geological modeling, geostatistical evaluations, project development, and authorship of numerous technical reports and preliminary economic assessments of various projects throughout North America, South America and Europe. I have employed and mentored mining engineers and geologists continuously since 2003.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 – *Standards for Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I made personal inspections of the North Bullfrog Project site on January 30 and 31, 2012, March 24, 2014, November 2 and 3, 2015, June 6-8, 2017 and most recently on January 16, 2018.
7. I am responsible for Sections 1 through 12, Section 14, 23, 24, 25, 26, and 27 of the Technical Report.
8. I am independent of the Issuer as independence is described in Section 1.5 of NI 43-101.
9. Prior to being retained by the Issuer, I have not had prior involvement with the property that is the subject of the Technical Report. I have been an author of previous NBP Technical Reports prepared for the property in 2012, 2013, 2014, 2015, 2017 and 2018.
10. I have read NI 43-101 and Form 43-101F1, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated: November 21, 2020

(signed/sealed) Scott *Wilson*

Scott E. Wilson, CPG, SME-RM

AUTHOR'S CERTIFICATE

Michael R. Young

I, Michael Young, Mining Engineer, SME-RM of Bayfield, Colorado , as an author of the technical report entitled "Technical Report and Preliminary Economic Assessment of Gravity Milling and Heap Leach Processing at the North Bullfrog Project, Bullfrog Mining District, Nye County, Nevada" (the "Technical Report") with an effective date of October 7, 2020 prepared for Corvus Gold Inc. (the "Issuer"), do hereby certify:

1. I am currently employed as the managing member of MinerMike LLC, 2006 County Road, Bayfield, Colorado 81122, USA.
2. I graduated with a Bachelor of Science degree in Mining Engineering from Texas A&M University in 1984.
3. I am a Registered Member (RM#03594500) of the Society for Mining, Metallurgy and Exploration, Inc.
4. I have been employed in the mining industry and consulting industry continuously for a total of 37 years. My experience includes resource estimation, mine planning, and pit optimizations of numerous technical reports and preliminary economic assessments of various projects throughout North America and South America. I have been involved with technical and mine supervision in several open pit mines, including one greenfield start-up and one brownfield start-up.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 – *Standards for Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I made a personal inspection of the North Bullfrog Project site on March 3-6, 2020.
7. I am responsible for Sections 15, 16, and relevant portions of Sections 21 and 25 of the Technical Report**.**
8. I am independent of the Issuer as independence is described in Section 1.5 of NI 43-101.
9. Prior to being retained by the Issuer, I have not had prior involvement with the property that is the subject of the Technical Report nor with any of the previous Technical Reports.
10. I have read NI 43-101 and Form 43-101F1, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated: November 21, 2020

(signed/sealed) *Michael R. Young*
Michael R. Young, SME-RM

AUTHOR'S CERTIFICATE

Adam R. House

I, Adam R. House, Sr. Metallurgical Engineer, MMSA Qualified Professional, of Helena, Montana, as an author of the technical report entitled "Technical Report and Preliminary Economic Assessment of Gravity Milling and Heap Leach Processing at the North Bullfrog Project, Bullfrog Mining District, Nye County, Nevada" (the "Technical Report") with an effective date of October 7, 2020 prepared for Corvus Gold Inc. (the "Issuer"), do hereby certify:

1. I am currently employed as a Sr. Metallurgical Engineer at Forte Dynamics, 314 N. Last Chance Gulch, Ste. 214, Helena, MT 59601, USA.
2. I graduated with a Bachelor of Science degree in Metallurgical Engineering in 2002 and a Master of Science Degree in Project Engineering and Management in 2011, both from Montana Tech of the University of Montana.
3. I am a Qualified Professional Member (#01498QP) of the Mining and Metallurgical Society of America (MMSA).
4. I have been employed an engineer continuously for over 17 years. My experience includes mineral processing and extractive metallurgy, process operations, process and infrastructure design, project management, and safety and environmental management at gold production operations in Nevada, USA. I have worked continuously as a consultant to mining operations globally since 2015.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 – *Standards for Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I made a personal inspection of the North Bullfrog Project site on September 3, 2020.
7. I am responsible for Sections 17, 18**,** 21**, 22** and 25 of the Technical Report.
8. I am independent of the Issuer as independence is described in Section 1.5 of NI 43-101.
9. Prior to being retained by the Issuer, I have not had prior involvement with the property that is the subject of the Technical Report, nor any of the previous NBP Technical Reports.
10. I have read NI 43-101 and Form 43-101F1, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated: November 21, 2020

(signed/sealed) *Adam R. House*
Adam R. House, MMSA QP

AUTHOR'S CERTIFICATE

Richard Delong

I, Richard Delong, Environmental Geologist, RM-SME, of Reno, Nevada as an author of the technical report entitled "Technical Report and Preliminary Economic Assessment of Gravity Milling and Heap Leach Processing at the North Bullfrog Project, Bullfrog Mining District, Nye County, Nevada" (the "Technical Report") with an effective date of October 7, 2020 prepared for Corvus Gold Inc. (the "Issuer"), do hereby certify:

1. I am employed as an Environmental Geologist and President at EM Strategies Inc., located at 1650 Meadow Wood Lane, Reno, Nevada 89502.
2. I am a graduate of the University of Idaho with a Master of Science degree in Geology in 1986 and a Master of Science degree in Resource Management in 1984**.**
3. I am a Qualified Professional Member (#01471QP) with the Mining and Metallurgical Society of America (MMSA) and SME Registered Member No. 4045773.
4. I have worked in the mineral industry for a total of 34 years after attending the University of Idaho. Mr. Delong has over 30 years of permit acquisition and environmental review experience for mining and exploration operations. His work has been focused on Nevada; however, he has worked on numerous projects throughout the western United States.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 – *Standards for Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of education, past relevant work experience and affiliation with a professional association (as defined in NI 43-101), I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I made a personal inspection of the North Bullfrog Project site on April 3, 2017.
7. I am responsible for the preparation of Section 20 of the Technical Report.
8. I am independent of the Issuer as independence is described in Section 1.5 of NI 43-101.
9. Prior to being retained by the Issuer, I have not had prior involvement with the property that is the subject of the Technical Report. I have provided technical support in the area of Environmental Management for NBP since 2013 and was an author of the previous technical report prepared for the property in 2018.
10. I have read NI 43-101 and Form 43-101F1, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated November 21, 2020

(signed/sealed) *Richard Delong*

Richard Delong, QP-MMSA

AUTHOR'S CERTIFICATE

Deepak Malhotra, Ph.D.

I, Deepak Malhotra, Ph.D., of Lakewood, CO. as an author of the technical report entitled "Technical Report and Preliminary Economic Assessment of Gravity Milling and Heap Leach Processing at the North Bullfrog Project, Bullfrog Mining District, Nye County, Nevada" (the "Technical Report") with an effective date of October 7, 2020 prepared for Corvus Gold Inc. (the "Issuer"), do hereby certify that:

1. I am currently employed as President of Pro Solv, LLC with an office at 15450 W. Asbury Avenue, Lakewood, Colorado 80228.
2. I am a graduate of Colorado School of Mines in Colorado, USA (Masters of Metallurgical Engineering in 1973 and Ph. D. in Mineral Economics in 1978).
3. I am a registered member in a good standing of the Society for Mining, Metallurgy and Exploration (RM #2006420) and a member of the Canadian Institute of Mining and Metallurgy (CIM).
4. I have 48 years of experience in the area of metallurgy and mineral economics. I have managed projects in research, process development for new properties, plan troubleshooting, plant audits, detailed plant engineering, due diligence for acquisitions and overall business management. I have authored over 80 technical papers and several books. I also participated in dozens of NI 43-101 Technical Reports.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 – *Standards for Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of education, past relevant work experience and affiliation with a professional association (as defined in NI 43-101), I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I have not visited the North Bullfrog site due to health and travel restrictions associated with COVID 19 in 2020.
7. I am responsible for Sections 13, 21 **and** 25 of **the** Technical Report**.**
8. I am independent of the Issuer as independence is described in Section 1.5 of NI 43-101.
9. Prior to being retained by the Issuer, I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and Form 43-101F1, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated: November 21, 2020

(signed/sealed) *Deepak Malhotra*
Deepak Malhotra, Ph.D., SME-RM

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1. SUMMARY

This report entitled "Technical Report and Preliminary Economic Assessment for Gravity Milling and Heap Leach Processing at the North Bullfrog Project, Bullfrog Mining District, Nye County, with an effective date of October 7, 2020 and dated November 21, 2020 (the "Technical Report"), describes conceptual mining and processing operations at the North Bullfrog Project ("NBP" or the "Project") located near the community of Beatty in Nye County, Nevada. The NBP is 100% controlled by Corvus Gold Nevada Inc. (CGNI), a wholly owned subsidiary of Corvus Gold Inc. ("Corvus"), through federal mining claims (Public Lands) and historic patented mining claims (Private Lands).

This report considers a revised processing approach at NBP where the higher grade gold/silver mineralization from the YellowJacket deposit at the NBP would be processed by a gravity mill located at NBP, with the gravity tail then blended with run-of-mine lower grade heap leach mineralization for final gold recovery on a heap leach pad. With this processing approach, NBP would be a stand-alone mining operation eliminating any advantage of a central mill facility located at the Mother Lode Project, which was the previously the basis of the 2018 Technical Report.

This report contains forward-looking statements by Corvus (the "Company") and the Authors, which are not guarantees of future performance. Actual results are likely to differ, and may differ materially, from those expressed or implied by forward looking statements contained in this report. Such statements are based on a number of assumptions which may prove incorrect, including, but not limited to, assumptions about the level and volatility of the price of gold, the timing of the receipt of regulatory and governmental approvals, permits and authorizations necessary to implement and carry on the Company's planned exploration and potential development programs; the Company's ability to attract and retain key staff, the timing of the ability to commence and complete the planned work at the Company's projects, and the ongoing relations of the Company with its underlying property lessors and the applicable regulatory agencies.

At NBP, the mining and processing would address the predominantly oxide mineralization. Minor amounts of on mineralization would also be mined and would be stockpiled for consideration in future processing plans. In the first 7 years of mining, high-grade mineralization from the YellowJacket vein and vein stockwork deposit would be scheduled for processing using a gravity mill circuit. Additional high-grade mineralization would be processed in the gravity mill in Years 10 and 11. The resulting gravity concentrate would be further processed in an intensive cyanide leach system and a pregnant solution would be transferred to electrowinning cells. The gravity tail would then be blended with low grade Run-of-Mine (ROM) mineralization produced from the YellowJacket deposit, the Sierra Blanca deposit, the Savage Valley deposit, and the Mayflower deposit for final recovery on a heap leach pad. After year 7 of the production plan, mining would continue predominantly in disseminated mineralization with processing on the heap leach pad and mining extended to the Jolly Jane deposit. Pregnant solution from the heap leach pad would feed two pair of vertical carbon-in-columns (VCIC). Gold and silver would be extracted from activated carbon and further processed via a desorption and recovery circuit to produce a final doré product.

Highlights of the NBP, including a preliminary economic assessment, are listed in Table 1-2 and 1-3. Table 1-1 lists the categorized Mineral Resources for the NBP. Mineral Resources are reported according to the CIM Definition Standards of May 10, 2014 ("CIM"). The guidance and definitions of CIM are incorporated by reference in Canadian National Instrument 43-101 - *Standards of Disclosure for Mineral Projects* ("NI 43-101"). The Mineral Resources are pit constrained in order to estimate the portion of the NBP Mineral Resource that demonstrates reasonable prospects of eventual economic extraction. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources will be converted into Mineral Reserves.

1.1 RESOURCE ESTIMATES

The basis for the Mineral Resource estimate is several geologic models interpreted by Corvus geologists and constructed in Leapfrog® Software. Geostatistics and estimates of mineralization were prepared by Resource Development Associates Inc. (RDA) of Highlands Ranch, CO. For the purposes of this Technical Report, the previous Mineral Resource Estimates for Sierra Blanca-YellowJacket-Savage Valley and for Mayflower (Wilson, et al, 2018) were updated to include new drilling performed in 2019 and 2020, and a new interpretation of geology at the Mayflower deposit.

Industry accepted grade estimation techniques were used to develop global mineralization block models. The NBP Mineral Resource estimate considered a new conceptual processing method which includes gravity concentration of YellowJacket vein and vein stockwork mineralization, intensive cyanide leaching of the gravity concentrate followed by heap leach processing of the gravity circuit tail product mixed with of Run-of-Mine (ROM) oxide gold and silver mineralization. The gold and silver in solution would be recovered on activated carbon processed through an ADR plant to produce a doré on-site for sale at spot market prices.

There are four mineral deposits which make up the NBP mineralization: (1) the YellowJacket vein and vein stockwork, (2) the Sierra Blanca/Savage Valley deposit, (3) the Mayflower deposit and (4) the Jolly Jane deposit. The total Mineral Resource estimate at NBP is listed in Table 1-1, subdivided by processing approach. The applicable cut off grades are also indicated for the individual processing approaches in Table 1-1.

Mineralization within the YellowJacket vein and surrounding stockwork is situated within a well-defined zone that holds together at higher cutoff grades within the resource constraining pit. The YellowJacket deposit contains the highest grades.

Table 1-1 North Bullfrog Pit Constrained Measured, Indicated and Inferred Mineral Resource Estimate[1] at a Gold Selling Price of $1,500 per Ounce

Mineral Resource Classification	Milling Sulphide & Oxide COG 0.204 and 0.400 g/t			Heap Leach Oxide COG 0.060 g/t			Total k-ounces		Mill k-oz		Heap Leach k-oz	
Units	K-tonnes	Au g/t	Ag g/t	k-tonnes	Au g/t	Ag g/t	Au koz	Ag koz	Au koz	Ag koz	Au koz	Ag koz
Measured	9,539	1.46	10.18	27,601	0.25	0.78	669	3,816	447	3,121	222	695
Indicated	15,130	1.21	7.61	139,867	0.19	0.62	1,438	6,490	590	3,702	848	2,788
M & I Total	24,669	1.31	8.60	167,469	0.20	0.65	2,107	10,306	1,037	6,823	1,070	3,483
Inferred	418	0.97	7.96	67,254	0.19	0.55	414	1,292	13	107	401	1,185

(1) The qualified person of the above estimate is Scott Wilson, C.P.G., SME.

(2) The Mineral Resources are classified as Measured, Indicated and Inferred Mineral Resources, and are based on the 2014 CIM Definition Standards.

(3) Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

(4) Mineral Resources are estimated using a gold price of $1,500/oz.

(6) Cut-off grades for mill processing were 0.204 g/t for oxide gold mineralization and 0.400 g/t for sulphide gold mineralization

(7) Cut-ff grade for heap leach processing was 0.06 g/t

(8) Numbers may not add up due to rounding.

(9) The effective date of this Mineral Resource estimate is October 7, 2020.

(10) The quantity and grade of reported inferred Mineral Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these inferred Mineral Resources as indicated or measured Mineral Resources.

(11) The qualified person knows of no environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that may materially affect the Mineral Resource estimates in this Technical Report.

Coordinates shown on plans and sections in this report are based on the Universal Transverse Mercator (UTM) North American Datum (NAD) 27 Zone 11 projection in meters unless otherwise specified. An additional 44 exploration drill holes were added to Sierra Blanca data since production of the 2018 Technical Report (as defined herein), with most holes focusing on resource definition drilling along the West side of Sierra Blanca. Geologic boundaries and fault blocks have been redefined. Definition of geologic domains have been updated based on additional information identified by drilling. Mineral Resource estimate parameters were updated from 2018 parameters based upon the reinterpretation of fault locations near West Sierra Blanca and YellowJacket. Declustering parameters were re-evaluated and modified slightly for YellowJacket.

The Mayflower deposit Mineral Resource estimate was updated based on re-interpretation of geological controls and domains, which indicated a high-grade corridor of mineralization through a portion of the Mayflower deposit.

Mineralization transitions to lower grades in proximity to the high-grade mineralization. Stratigraphy does not change through these two mineralogically separate zones within the deposit. An indicator kriging strategy was used to implicitly model and separate the high-grade corridor from low grade mineralization. Historic underground mining drifts, stopes and raises have been modelled in 3D. These previously mined volumes were depleted from the Mineral Resource estimate.

Mineral Resources are not Mineral Reserves and do not demonstrate economic viability. There is no certainty that all or any part of the Mineral Resource will be converted to Mineral Reserves. Quantity and grade are estimates and are rounded to reflect the fact that the Mineral Resource Estimate is an approximation.

1.2 PRELIMINARY ECONOMIC ASSESSMENT

A summary of the current projected financial performance of the Phase I Mining Study for NBP is listed in Table 1-2 through 1-6. Sensitivities are summarized in Figures 1-1 and 1-2.

The preliminary economic assessment is preliminary in nature, and there is no certainty that the reported results will be realized. The Mineral Resource estimate used for the PEA includes Inferred Mineral Resources which are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the projected economic performance will be realized. The purpose of the PEA is to demonstrate the economic viability of the North Bullfrog Project, and the results are only intended as an initial, first-pass review of the Project economics based on preliminary information.

Table 1-2 - NBP Mining Production

Production Schedule		-1	1	2	3	4	5	6	7	8	9	10	11	12	13	Total
Total Production	Units															
Mill Mineralized Material	Tonnes (000's)	635	1,106	1,698	1,755	1,718	1,698	1,034	1,034	0	0	872	800	0	0	12,350
	Contained Au gpt	0.93	1.37	2.06	2.33	1.92	1.45	0.83	0.83	0.00	0.00	0.58	0.58	0.00	0.00	1.47
	Au Oz (000's)	19.0	48.7	112.3	131.7	106.0	78.9	27.6	27.6	0.0	0.0	16.3	14.9	0.0	0.0	583
ROM Mineralized Material	Tonnes (000's)	2,303	10,092	9,573	12,802	13,559	16,772	20,990	21,400	21,400	21,402	16,000	16,000	18,000	8,817	209,110
	Contained Au gpt	0.21	0.23	0.25	0.30	0.32	0.26	0.19	0.19	0.19	0.19	0.14	0.14	0.13	0.16	0.20
	Au Oz (000's)	15.5	74.6	78.2	121.7	138.8	137.8	127.9	130.8	130.8	130.8	72.0	72.0	76.5	44.2	1,352
Total Mineralized Material	Tonnes (000's)	2,938	11,198	11,272	14,556	15,277	18,470	22,024	22,434	21,400	21,402	16,872	16,800	18,000	8,817	221,460
	Contained Au gpt	0.37	0.34	0.53	0.54	0.50	0.36	0.22	0.22	0.19	0.19	0.16	0.16	0.13	0.16	0.27
	Au Oz (000's)	34.5	123.3	190.5	253.4	244.9	216.7	155.5	158.4	130.8	130.8	88.3	87.0	76.5	44.2	1,935
Waste	Tonnes (000's)	18,406	23,764	17,163	22,794	16,819	14,589	12,497	10,967	10,000	12,000	16,214	15,214	13,000	7,545	211,972
Total	**Tonnes (000's)**	21,344	34,962	28,434	37,351	32,096	33,059	34,521	33,401	31,400	33,402	33,086	33,014	31,000	16,362	433,432
Strip Ratio		6.26	2.12	1.52	1.57	1.10	0.79	0.57	0.49	0.47	0.56	0.96	0.97	0.72	0.86	0.96
Contained Au Oz.	Oz (000's)	34.5	123.3	190.5	253.4	244.9	216.7	155.5	158.4	130.8	130.8	88.3	87.0	76.5	44.2	1,934.7

Table 1-3 - NBP Economic Highlights

Parameter	Year 1-7 Data Value	LOM Data Value[4]
Measured & Indicated Mill Feed (contained Au)	-	12.1M t at 1.47 g/t Au for 575.8k oz
Inferred Mill Feed (contained Au)	-	0.2M t at 1.09 g/t Au for 7k oz
Measured & Indicated Heap Leach Feed (contained Au)	-	149.1M t at 0.21 g/t Au for 983.9k oz
Inferred Heap Leach Feed (contained Au)	-	59.9M t at 0.19 g/t Au for 369k oz
Post-Tax and Royalty NPV at 5%	-	$452M
Post Tax and Royalty IRR	-	47 %
Pre-tax cashflow; IRR	-	$763M; 55%
Overall Strip Ratio (mining only)	1.16:1 (overburden:mineralized material)	0.96:1 (overburden:mineralized material)
Average Annual Payable Gold Production	147 kozs/year	112 kozs/year
Total Payable Gold Produced	1,029 kozs	1,467 kozs
Average Gold Recovery - mill	85%	85%
Average Gold Recovery- heap leach	72%	72%
Average Cash Cost[1]	$ 589/Au Oz	$ 751/Au Oz
All-in Sustaining Cost (AISC) [2]	$ 727/Au Oz	$ 885/Au Oz
Average Silver Recovery – mill	63%	63%
Average Silver Recovery – heap leach	13%	13%
Average Total Mining Rate[3]	89 k tonne/day	85 k tonne/day
Average Mineralized Material Mining Rate[3]	43 k tonne/day	43 k tonne/day

1-Cash Cost includes mining, processing, site G&A, refining, and royalties.

2-AISC is a non-GAAP metric and includes mining, processing, site G&A, refining, royalties, sustaining capital (not initial), and reclamation costs, Corvus's calculation methodology is listed in Table 22-2 and may differ from that used by other organizations.

3-Includes mill tails rehandle as well as mining.

4-Values through Year 15, including 2-year draindown

In Table 1-3, Average Cash Cost includes mining and processing costs, plus site general and administrative, refining/transport costs, and royalties, along with a credit for the co-product, silver. AISC is estimated and includes all Cash Costs, plus sustaining capital and reclamation costs. **AISC is a non-GAAP metric and other companies may calculate it differently. Corvus believes that it can aid understanding and comparison of the anticipated production costs of potential mining at NBP. Table 2-2 list the details of Corvus's calculational methodology.**

Table 1-4 - Initial Capital Cost Summary

Area	Initial Capital Cost ($M)
Initial Direct Capital Cost	$137.9
EPCM	$8.0
Contingency	$18.5
Owner's Cost	$3.0
Total Initial Capital	**$167.4**

Table 1-5 - Initial Direct Capital Costs

Area	Initial Capital Cost ($M)
Mill	$19.3
Heap Leach	$49.4
Mobile Equipment	$8.0
Infrastructure & Facilities	$22.2
Capitalized Mining	$38.9
Total Initial Direct Capital	**$137.9**

Table 1-6 - Sustaining Capital Costs

Area	Sustaining Capital Cost ($M)
Sustaining Capital	$101.4
EPCM	$8.1
Contingency	$22.7
Total Initial Indirect Capital	**$132.3**

Figure 1-1 - Sensitivity of Estimated NPV @ 5% (after-Royalty and after-Tax) for Changes in Cost, Gold Recovery or Gold Price as a Percent of the Base Case at a Gold Price of $1,500 per Ounce, Gold:Silver Price Ratio of 80.0, 75.8% Gold Recovery

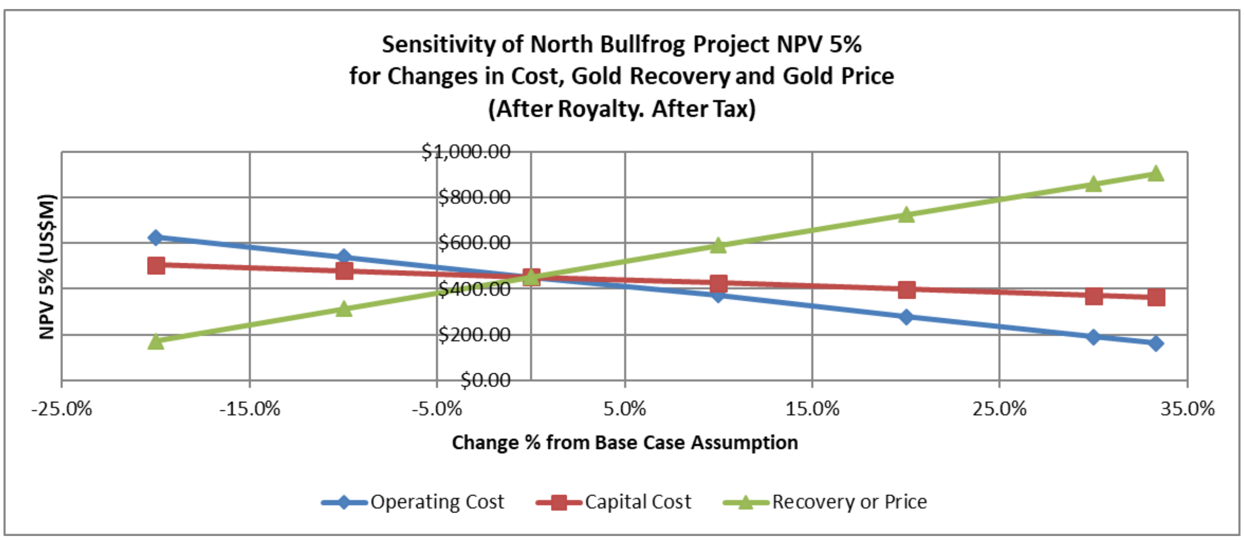
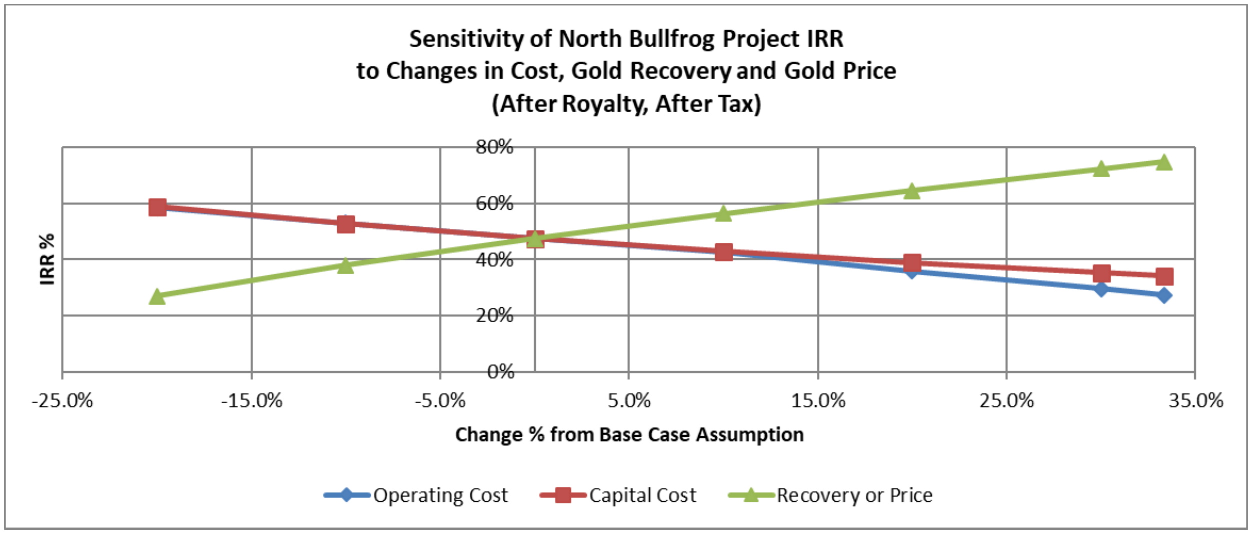


Figure 1-2 - Sensitivity of Estimated IRR (after-Royalty and after-Tax) for Changes in Cost, Gold Recovery or Gold Price as a Percent of the Base Case at a Gold Price of $1,500 per Ounce, Gold:Silver Price Ratio of 80.0, and 75.8% Gold Recovery



1.3 PROPERTY DESCRIPTIONS AND OWNERSHIP

The NBP is located in the Bullfrog Hills of northwestern Nye County, Nevada. Figure 2-1 shows Corvus' property holdings around the community of Beatty, NV and shows the extensions to the south from the eastern edge of NBP to include the Mother Lode deposit and from the western edge of NBP towards the historic Bullfrog Mine.

The NBP (as separate from the other Corvus property holdings) covers approximately 7,223 hectares (17,250 acres) of private land derived from historical patented claims and public lands derived from federal lode mining claims in Sections 20, 21, 25, 26, 27, 28, 29, 32, 33, 34, 35, and 36 of T10S, R46E; sections 1, 2, 11, 12, 13, and 14 of T11S, R46E; section 31 of T10S, R47E; and section 6, 9, 15, 16 and 17 of T11S, R47E, MDBM. The public land is administered by the US Department of the Interior, Bureau of Land Management (BLM).

The Project is accessible by a two- and one-half hour (260 kilometers) drive north of Las Vegas, Nevada along US Highway 95. US Highway 95 is the major transportation route between Las Vegas NV, Reno NV, and Boise, ID. Las Vegas is serviced by a major international airport. NBP lies immediately to the west of the US Highway 95 and approximately 10 kilometers north of Beatty, NV. Beatty is the closest town to the Project with a population of about 1,100 and contains most basic services. Access around the Project area is by a series of reasonably good graveled and dirt roads that extend from US Highway 95 to the exploration areas.

Corvus controls the NBP through a number of private land leases and federal lode mining claims listed in Table 4-1 and 4-2 of Section 4. Corvus owns and leases private land (historical patented lode mining claims) and maintains a large contiguous block of federal lode mining claims. In 2014, Corvus purchased 162 hectares of surface lands in Sarcobatus Flats, approximately 26 kilometers north of the NBP, which included 1,600-acre feet per year of water rights. In 2018, the Nevada Division of Water Resources granted temporary change of the water right usage to mining application for use at NBP.

1.4 GEOLOGY AND MINERALIZATION

The Project lays within the Walker Lane mineral belt and the Southwestern Nevada Volcanic Field (SWNVF). The regional stratigraphy includes a basement of Late Proterozoic to Late Paleozoic metamorphic and sedimentary rocks. Basement rocks are overlain by a thick pile of Miocene volcanic and lesser sedimentary rocks of the SWNVF, ranging in age from ~15-7.5 Ma (Figure 7-1). The pre-Tertiary rocks exhibit large-scale folding and thrust faulting, having been subjected to compressional deformation associated with multiple pre-Tertiary orogenic events. The stratigraphy of the SWNVF is dominated by ash flow tuff sheets erupted from a cluster of nested calderas known as the Timber Mountain Caldera Complex. The southwestern edge of the caldera complex lies approximately ten kilometers east of the NBP (Figure 7-1). The stratigraphy of the SWNVF includes voluminous ash flow tuff sheets, smaller volume lava flows, shallow intrusive bodies, and lesser sedimentary rocks. Many of the volcanic units exposed around the Project include ash flow tuffs that originated from the caldera complex. Other volcanic units are locally sourced outside of the caldera complex, particularly at the NBP.

Gold mineralization in the NBP is primarily hosted in the middle Miocene Sierra Blanca Tuff. Gold mineralization is also hosted to a lesser extent in monolithic and heterolithic debris-flow breccias, as well as in felsic dikes and plugs. Two district-scale north striking normal faults are the dominant structural features in the Project area, but several smaller-scale faults between them are important controls for distribution of hydrothermal alteration and gold mineralization.

Two styles of precious metal epithermal mineralization are present at the NBP: 1) high-grade, structurally controlled fissure veins and associated stockwork zones; and 2) low-grade disseminated or replacement deposits within altered volcanic rocks. Historic drilling on the NBP outlined areas of important mineralization. Drilling by International Tower Hill Mines Ltd. (“ITH”), a predecessor-in-interest of Corvus, was used to develop initial resource estimates, to better understand precious metal mineralization at Air Track Hill and as initial tests at the Sierra Blanca, Pioneer, Savage and YellowJacket targets.

1.5 METALLURGICAL TESTING

During 2012-2013, metallurgical testing was performed using composite samples developed from PQ core materials produced at Mayflower, Sierra Blanca, Savage Valley and Jolly Jane. Column leach testing on up to P80 -19 millimeters ("mm") indicated relatively high gold recoveries in the range of 80% and confirmed the suitability of heap leach processing on disseminated mineralization. In 2014-2015, composite samples of PQ core materials were developed from YellowJacket vein and stockwork mineralization. Those tests indicated high solubility of contained gold in cyanide leach testing at P80 -150 microns, but reduced gold recoveries at heap leach sized particles. The tests indicated that mill processing would be required on YellowJacket mineralization. Further testing of composite samples, using gravity concentration, intense cyanide leaching of gravity concentrates and cyanide leaching of the gravity tails, indicated gold recoveries in the range of 90% and silver recoveries in the range of 70%. These historical data are discussed in Section 13 and in Wilson et al., 2018 and indicated that gravity concentration followed by cyanide leaching of the gravity tailing was a potential processing option for high-grade mineralization.

In 2020 metallurgical tests were performed to investigate the potential for gravity concentration of high-grade mineralization from the YellowJacket vein and vein stockwork to be followed by intense CN leaching of the gravity concentrate. The gravity tail would then be thickened, disk filtered and sent to a heap leach pad that would simultaneously process ROM low-grade mineralization from the YellowJacket, Sierra Blanca, Savage Valley, Jolly Jane and Mayflower deposits. The relevant milling and column leach test work was performed at Resource Development Inc. (RDi) in 2019-2020 and is discussed in Section 13.

Comminution test work consisting of Crushability Work Indices (CWi), Abrasion Indices, Bond's Ball Mill Work Indices (BMWi) and Bond's Rod Mill Work Index (RMWi) was performed on YellowJacket vein and vein stockwork samples. The crushability work index ranged from a low of 4.87 kwh/t to a high of 16.68 kwh/t. No CWi was obtained for Savage Valley mineralization. The average CWi for the four mineralization's from different deposits was 11.50 kwh/t. The abrasion index ranged from a low of 0.06 to a high of 0.71. The average abrasion index was 0.411. This value indicates that the mineralized materials are generally abrasive. The average BMWi for the YellowJacket mineralization was determined to be 22.2 Kwh/t and the RMWi to be 15.9 Kwh/t.

One-kilogram gravity separation testing was conducted with a high-grade YellowJacket (YJ) composite sample at particle sizes of P80 of 28 mesh, 48 mesh, 65 mesh and 100 mesh. The samples were processed utilizing a Knelson concentrator to produce a gravity concentrate followed by a Gemini table treating the rougher gravity concentrate to produce a cleaner gravity concentrate. The combined gravity separation and gravity tail leaching test results indicated that +84% of the contained gold could be recovered using a grind size of P80 -48mesh or finer. Thickening tests were conducted with the YJ High-Grade composite at 28 mesh, 48 mesh, 65 mesh and 100 mesh. All of the samples settled very quickly. Lime addition to adjust the pH to 10.5 improved the turbidity of the overflow.

Final settled densities ranging from 55% to 60% solids were achieved utilizing a high molecular weight anionic polymer. A thickener with a unit area of between 0.001 m^2/mt/day and 0.041 m^2/mt/day would be needed to achieve an underflow density of 55% solids, depending upon the selected particle size.

The leach residue filtration results indicated that a maximum percent solid of 80.3% was achieved during testing. A filtration rate of over 500 dry lb/ft^2/hr was achieved with vacuum filtration for all samples except the 100-mesh material. The filtration rate dropped by over 50% at the grind size of P80 of 100 mesh.

Thickening and filtration tests were repeated on the gravity tailing sample from a larger test. The results indicated that:

- The gravity tails at pH of 10.5 thickened to 55% solids. The unit area required was calculated to be 0.001 m^2/mt/day.
- The filtration test on the thickened slurry resulted in a cake having a moisture content of 23.5% and calculated capacity of 1369.1 dry lbs/ ft^2/hr.
- The tests confirmed the viability of placing a gravity tail product of -48 mesh or finer directly on a heap leach pad, then simple blending of the tail product with ROM heap leach mineralization.

Six column leach tests were performed to simulate the heap leach gold recovery and reagent consumptions. The four tests were run in 30.5 cm (12-inch) diameter by 1.8 m (6 ft.) height columns. The material was tested as follows:

- Column 1 and 2 has run-of-mine (ROM) low-grade mineralization from Sierra Blanca. The mineralization was essentially minus 4 inch. These tests were run in duplicate.
- Column 3 and 5 were constructed of 15% mill gravity circuit tails and 85% of ROM Sierra Blanca mineralization.
- Column 4 and 6 were constructed of 25% mill gravity circuit tails and 75% of ROM Sierra Blanca mineralization.

The test results are given in Table 13-32 and Figure 13-1 to 13-4. The test results indicate the following:

- The ROM mineralization had average gold and silver extractions of 70% and 65% respectively. These recoveries were impacted by the very small amounts of fine material and would be increased in a well graded material.
- The average lime and cyanide consumptions were 5.38 kg/t and 1.16 kg/t, respectively.

- The gold extraction for the blend of 15% gravity tails and 85% ROM mineralization was 71% at 64 days of leaching (Column No. 3). The gold extraction increased to 76.4% when the blend was 25% gravity tails and 75% ROM mineralization (Column No. 4).
- The NaCN consumption for the blends of gravity tail and ROM mineralization was much lower than for ROM mineralization alone (0.59 to 0.68 kg/t versus 1.1 kg/t).
- These gold extractions are exclusive of the gravity gold concentrate which was projected to recover 46.9 % of the gold contained in the YellowJacket mill mineralization.

1.6 MINING METHOD

The NBP contain mineralization at or near the surface that is suitable for open pit mining methods. The method of material transport evaluated for this PEA was open pit mining using drill and blast techniques and 20 m^3 Front End Loaders (FEL) paired with 133 tonne rigid frame haul trucks. Rotary percussion drills would be used to create blast-holes and support equipment includes track dozers, motor graders, and water trucks.

Waste Rock Management Facilities would be located near each producing open pit (YellowJacket, Sierra Blanca, Savage Valley, Jolly Jane and Mayflower). Mill mineralization would be hauled to a mill stockpile while run-of-mine (ROM) mineralization would be hauled and placed directly on the heap leach pad. Mill tails would be hauled and placed directly on the heap leach pad by plug dumping and then mixed into the ROM mineralization by dozing.

The proposed mining operation would use owner operated equipment. The general site layout, including pits, waste dumps, mill site, ponds, and heap leach pad, are shown in graphically in Section 16. It has been assumed that the mineralized material would be blasted with the Ultra-High-Intensity-Blasting (UHIB) method to create a highly fractured material at nominally 100% -102mm (-4 inch) [P_{80} -84mm (-3.3 inch)].

Mill resource production is planned at a nominal rate of 4,700 t/d, equivalent to 1.7 Mt/y with an approximate 7-year mine life. Run of Mine (ROM) heap leach resource is planned at a nominal rate of 32,300 t/d, equivalent to 11.8 Mt/y for the first 5 years followed by a nominal rate of 51,200 t/d, equivalent to 18.7 Mt/y for years 6 through 13. Mining is planned on a 7 day per week 24-hour per day schedule, 360 days per annum. The average mineralized material and waste production is approximately 91,000 t/d. The average LOM stripping ratio is 0.96:1 waste-to-mineralized material, using a 0.13 Au gpt cut-off for ROM resources and a 0.23 Au gpt cut-off for mill resources in production years 1-5. The cutoff grade for years 6-13 is 0.06 Au g/t cut-off for ROM mineralized material and a 0.23 Au g/t cut-off for mill mineralized material.

Pit slope designs have been based on an overall 50-degree inter-ramp angle. Preliminary pit slope geotechnical evaluations at YellowJacket (Knight Piésold, 2020) indicated a 58-degree inter-ramp angle with 80-degree bench faces would be stable. Preliminary pit slope geotechnical evaluations have been performed for Sierra Blanca/YellowJacket, for Mayflower (Engineering Analytics, 2013b) and for Jolly Jane (Engineering Analytics, 2013a).

The majority of the mining production would be conducted above the water table. At YellowJacket, mining below the 1,200 m amsl would be expected to have water inflow. Very preliminary hydrogeologic data (HydroGeoLogica, 2020) suggests a large range in predicted water inflow. Current design assumptions allow for a maximum inflow of 23.8 m^3/hr (105 gpm).

1.7 PROCESSING

Process facilities would be located in the northwest corner of the NBP at the edge of Sarcobatus Flat, a large sedimentary basin with predominantly flat terrain. The location has a substantial area that is topographically favorable for construction with a gentle gradient to the northwest to facilitate storm water drainage. Location of the process facilities is shown in Figure 17-1, and would consist of the mill, heap leach pad, access roads, ponds, adsorption, desorption and recovery (ADR) plant, mobile equipment yard, offices and warehouse. Access to the site from Highway 95 would be along the existing Strozzi Ranch road, which would be improved for the mine operation.

The gravity mill would consist of primary and secondary crushers, a fine crushed product storage bin, rod mill, screens and Falcon concentrators and is based on a design supplied by Sepro Mineral Systems of Vancouver, BC. The gravity concentrate would be treated in an intensive cyanide leach reactor with solutions transferred to the electrowinning circuit in the ADR plant for gold recovery. The gravity tail product would be thickened, filtered and placed on a loading pile, where it would be loaded by FEL into 136 tonne haul trucks carrying ROM mineralization and then transported to the heap leach pad. The comingled gravity tail material and ROM mineralization would be interspersed with end-dumped ROM mineralization, then pushed and ripped by dozer to further mix with the coarser ROM mineralization. Higher concentrations of gravity tail material (>25%) may produce increased gold recovery in the heap leach, so the dumping geometry will be controlled at the heap leach pad to maximize gold recovery.

Process water would be produced from ground water wells which would be developed in the north west corner of NBP. An existing ground water well, NB-WW-14 has been drilled in this location and produces drill water for exploration of the site. Corvus is currently permitted to withdraw 1,277 acre-feet per year of water resources through permit 65756 (SoN Land and Water LLC, a wholly owned subsidiary of Corvus). The point of diversion within the Basin 146 (Sarcobartus Flat) would be temporarily transferred immediately northwest of the heap leach pad for the time of operations. The water is currently permitted for Mining and Milling applications by Nevada Division of Water Resources (NDWR).

1.8 PROJECT PERMITTING

A mining Plan of Operations would be required to be submitted to BLM and the Nevada Department of Environmental Protection (NDEP) Bureau of Mining Regulation and Reclamation (BMRR) and would need to be supported by development of an Environmental Impact Statement (EIS). BLM has recently issued updated guidance (Memorandum to Assistant Secretaries, Heads of Bureaus and Offices, NEPA Practitioners, Additional Direction for Implementing Secretary's Order 3355, From the Deputy Secretary of the Interior, April 27, 2018) on the requirements for preparation of the EIS which controls the time line for issuance of the Record of Decision which represents the delivery of the mining permit. The new guidance specifies a series of 7 steps that must be completed prior to the initiation of a 1-1.5-year EIS Process Timeline.

Corvus anticipates that 12 months will be required to initiate the project, generate the project definition, define the baseline requirements, complete the baseline characterization, develop the plan of operations, develop the plan of development, specify the level of NEPA required for the project and advance the project design elements to sufficient levels of detail. At that point, the NEPA process would commence to develop the Environmental Impact Statement (EIS).

Corvus anticipates that the EIS process would require an additional 12-18 months, based on the recently issued Department of Interior guidance. That process requires public comment on the defined project and review by the BLM and BMRR, development of a draft EIS, publication of the draft EIS and presentation at public hearings, address and respond to received comments from the public and other agencies, development and public review of the Final EIS. After the acceptance of the Final EIS, BLM would issue a Record of Decision.

At that point, the project permits could be issued, and construction activities could begin. Corvus anticipates portions of the operation could begin within 6 months of receipt of the permits, with mill start-up and preparation of the first phase of the heap leach pad.

2. INTRODUCTION

2.1 General Statement

The North Bullfrog Project (“NBP” or the “Project”) is an advanced stage surface exploration project consisting of private land derived from historic patented claims and public land consisting of federal lode claims. NBP is 100% controlled by Corvus Gold Nevada Inc., a wholly owned subsidiary of Corvus Gold Inc. (NASDAQ:KOR, TSX:KOR, “Corvus” or the “Company”). The mix of public and private lands forming NBP covers approximately 72 square kilometers. A location of the NBP property is shown in Figure 2-1, which also shows the extent of Corvus controlled property in the Beatty, Nevada area and the mineral property holdings of other exploration and mining companies in the area.

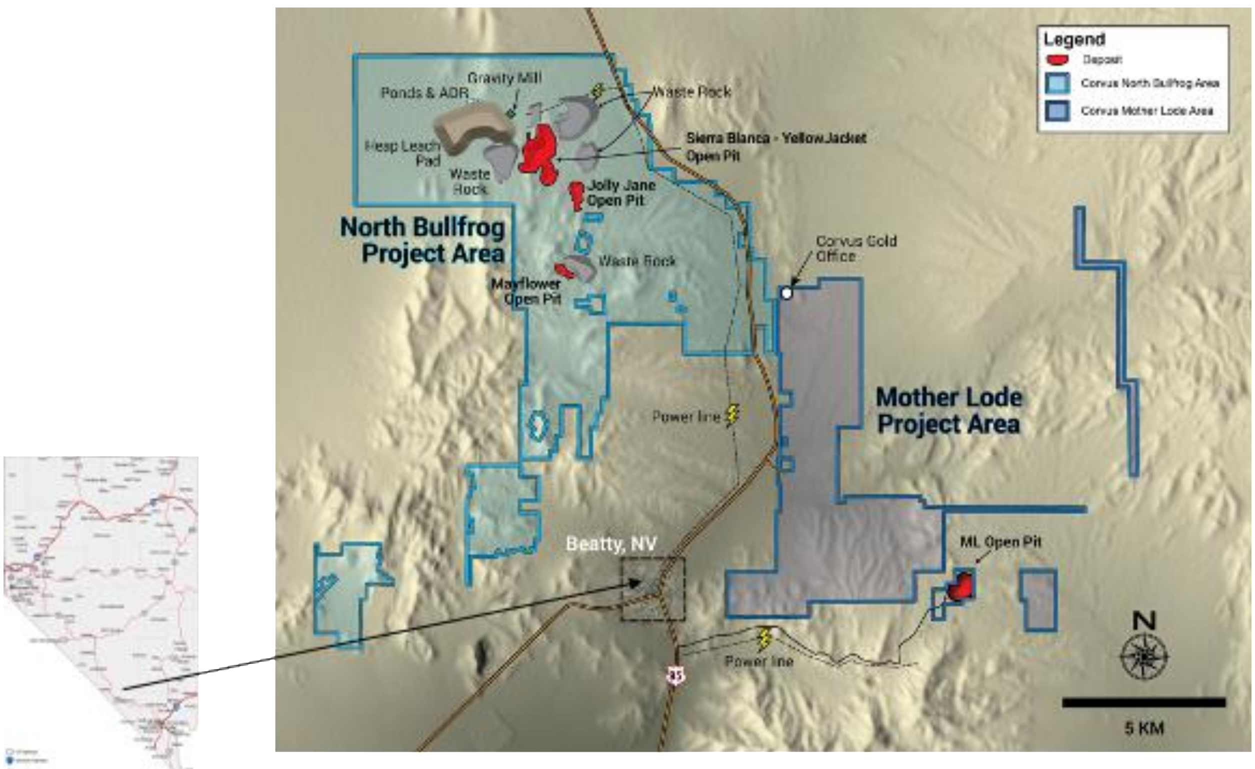
This Technical Report describes the potential mining operations at the NBP. The mill and heap leach processing facilities to be located at the NBP are designed to treat oxide gold mineralization. The higher-grade mineralization from the YellowJacket vein and vein stockwork zone would be processed with a simple gravity mill with heap leaching of blended ROM and gravity mill tail for final gold recovery. This report incorporates recent drilling, geological interpretation and metallurgical data for NBP, developed in 2019 and 2020, in addition to the data developed between 2008 – 2018, which were discussed in the technical report entitled “Technical Report and Preliminary Economic Assessment for the Integrated Mother Lode and North Bullfrog Projects, Bullfrog Mining District, Nye County, Nevada” dated effective September 18, 2018 (the “2018 Technical Report”).

Metallurgical testing in 2020 indicated that high gold recoveries could be obtained from milling to produce a gravity concentrate from higher grade vein and vein stockwork mineralization in the YellowJacket zone at North Bullfrog and then by heap leaching the blended gravity tail material and lower grade disseminated mineralization for final gold/silver recovery. The gravity milling would be at a relatively coarse grind (48 mesh), which would allow blending the gravity tail product with lower grade disseminated mineralization at Run-of-Mine size distribution. This processing approach would allow a lower plant capital investment and a simpler process plant than had been previously considered. A more detailed mining study was therefore performed that focused on a Starter Phase, with mining of the YellowJacket vein and vein stockwork mineralization and higher grade disseminated mineralization for the heap leach. The mining production would then address progressively lower grade disseminated mineralization in the Expansion Phase of the operation.

Mineralization at the NBP is related to a large, low-sulphidation epithermal gold system hosted predominantly in volcanic units and derived sedimentary units. Gold and silver were discovered in the Bullfrog district in 1904. Production records indicate that more than 110,000 ounces of gold and more than 800,000 ounces of silver were produced through 1921. Modern exploration at North Bullfrog began in 1974. In 2009, Corvus Gold Inc. was formed as a spin out from International Tower Hill Mines Ltd. (“ITH”), where 100% percent of the Project was transferred to Corvus. Corvus has been actively exploring at NBP since 2009.

The NBP is located in northwestern Nye County, Nevada, in the Northern Bullfrog Hills and Bare Mountains near the community of Beatty (Figure 2-1). The Project lies within the Walker Lane structural terrain which also hosted the historic Bullfrog Mine where Barrick Gold Corp. (and predecessor companies) produced about 2.3 million ounces of gold and 3.0 million ounces of silver from 1989 through 1999 (NBMG MI-2000, page 34). The NBP contains numerous other epithermal low-sulphidation volcanic rock-hosted gold showings that have had limited historic production, as well as the recent and major mining operations at the Bullfrog, Mother Lode and Daisy mining projects.

Figure 2-1 - Map Showing Corvus Controlled Property in the Beatty Area, North Bullfrog Infrastructure and Mother Lode Project



2.2 Terms of Reference

Corvus requested that this Technical Report be prepared to provide the definition of the mining and processing of gold mineralization at NBP as a stand-alone project. Metallurgical testing on YellowJacket mineralization during 2020 has indicated a simple processing alternative with lower capital and operating cost for NBP which would eliminate the requirement to transport the YellowJacket mineralization to a central mill facility that was to be located at Mother Lode. This Technical Report is based on technical information produced by the Corvus directed exploration drilling program and site characterization information developed in pre-permitting baseline characterization studies conducted between 2012 – 2020. Mr. Scott E. Wilson, C.P.G., SME, Mr. Michael R. Young, SME, Mr. Adam R. House, MMSA-QP, Mr. Richard Delong, MMSA-QP and SME, and Mr. Deepak Malhotra, SME, were commissioned by Corvus to prepare this Technical Report.

Mr. Scott Wilson, (CPG #10965, SME #4025107RM), an independent qualified person, was the principal author responsible for the overall preparation of this Technical Report, and specifically for Sections 1 through 12, Section 14, and Sections 18 through 27 of this Technical Report. Mr. Wilson visited the NBP site on January 30 and 31, 2012, March 24, 2014, November 2-3, 2015, June 6-8, 2017 and on January 16, 2018. Mr. Wilson is independent of Corvus applying all of the tests in Section 1.5 of NI 43-101.

Mr. Michael R. Young, (SME # 03594500RM), an independent qualified person, was responsible for the preparation of Sections 15, 16, 21 and 25 of this Technical Report**.** Mr. Young visited the North Bullfrog Project site on March 3-6, 2020. Mr. Young is independent of Corvus applying all of the tests in Section 1.5 of NI 43-101.

Mr. Adam R House, (MMSA-QP #01498QP), an independent qualified person, was responsible for the preparation of Sections 17, 18, 21 and 25 of this Technical Report. Mr. House visited the North Bullfrog Project site on September 3, 2020. Mr. House is independent of Corvus applying all of the tests in Section 1.5 of NI 43-101.

Mr. Richard Delong, (MMSA-QP: #01471QP; SME # 4045773RM), as an independent qualified person, was responsible for the preparation of Section 20 of this Technical Report. Mr. Delong visited the North Bullfrog Project site on April 3, 2017. Mr. Delong is independent of Corvus applying all of the tests in Section 1.5 of NI 43-101.

Mr. Deepak Malhotra, Ph.D. (SME # 2006420RM) as an independent qualified person, was responsible for the preparation of Section 13, 17, 21 and 25 of this Technical Report. Mr. Malhotra has not visited the North Bullfrog Project site due to health and travel restrictions in 2020 related to COVID 19. Mr. Malhotra is independent of Corvus applying all of the tests in Section 1.5 of NI 43-101.

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

3. RELIANCE ON OTHER EXPERTS

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

4. PROPERTY DESCRIPTION AND LOCATION

Between 2017 and 2020, Corvus significantly expanded its land position in the Bullfrog district by acquiring the historic Mother Lode property from Goldcorp USA, Inc., staking additional claims in the Mother Lode vicinity, and staking claims between Mother Lode and North Bullfrog, staking additional claims south and west of North Bullfrog, and staking claims in gaps between claims staked by other explorers east and north of Mother Lode. Corvus' land position in the Bullfrog district is shown in Figure 2-1.

4.1 NBP Area and Location

The NBP is located in the Bullfrog Hills of northwestern Nye County, Nevada (Figure 2-1). The Project covers approximately 7,223 hectares of patented and unpatented lode mining claims in Sections 19, 20, 21, 22, 23, 24, 25, 26, 27, 28, 29, 30, 31, 32, 33, 34, 35, and 36 of T10S, R46E; Sections 1, 2, 3, 10, 11, 12, 13, 14 and 15 of T11S, R46E; Section 19, 30, 31, 32 of T10S, R47E; and Sections 4, 5, 6, 7, 8, 9, 16, 17 and 18 of T11S, R47E, Mount Diablo Base and Meridian ("MDBM"). Corvus has a total of nine option/lease agreements in place that give it control of 303 hectares (748 acres) of private land derived from 51 historic patented lode mining claims which are listed in Table 4-1. Corvus has also purchased surface rights to an additional 37 hectares (91 acres) of private land derived from 5 historical patented claims in the Mayflower area.

The mining claims and lease agreements give Corvus the right to explore the property subject to required regulatory permits which are described in Section 4.1.3. Corvus currently has permits for exploration on the public and private land from the BLM and Nevada Department of Environmental Protection ("NDEP") Bureau of Mining Regulation and Reclamation ("BMRR"). The permits allow non-exclusive access by Corvus and its contractors for surface disturbance associated with exploration drilling, engineering characterization, and baseline environmental data collection as defined in the permits. The claims and lease agreements also give Corvus the rights to conduct mining operations to extract the Mineral Resources subject to future permits described in Section 20.

4.1.1 Redstar Option/Joint Venture/ITH Purchase of Land

Redstar Gold Corporation (RGC) originally staked 213 public land mining claims and optioned private land related to 21 historic patented lode mining claims from six private parties in 2005-2006 at the NBP. International Tower Hill Mines (ITH) optioned the original NBP land package from RGC in 2006, creating the North Bullfrog Property Joint Venture (NBPJV). In 2007, ITH added the Mayflower Property related to 11 historic patented lode mining claims to the NBPJV under the Greenspun lease agreement. In 2008, RGC added further private land related to 12 historic patented lode mining claims (Connection and adjacent properties) to the NBPJV under the first lease agreement with Lunar Landing LLC. In August 2009, ITH purchased a 100% interest in the NBPJV from RGC by paying RGC C$250,000 and issuing 200,000 ITH common shares. The land holdings were then transferred to Corvus during the spin out from ITH. In March 2011, Corvus completed the Sussman option agreement on private land related to two historic patented lode mining claims in the Jolly Jane area. In May 2014, Corvus amended its existing lease agreement with Kolo Corp. to add the historic Yellowrose and Yellowrose No. 1 patented claims. In March 2015, Corvus added a second option agreement with Lunar Landing LLC, to lease the Sunflower, Sunflower No. 1 and Sunflower No. 2 historic patented claims. Table 4-1 summarizes the obligations of the nine private land lease agreements which are part of Corvus' responsibilities on the Project and lists the historic patented claim names and Patent Numbers associated with the nine lease obligations. Table 4-2 lists the individual owners, claim names and historic patent numbers for the original claims making up the private land leases. Applicable NSR Royalties are also listed in the table. The principal author, Scott Wilson, C.P.G, SME, has verified that all lease obligations of Corvus have been met and are paid in full as of the date of this Technical Report.

Table 4-1 - Summary of Lease Obligations at NBP (All Funds USD)

Party	Area	Claims/Acres	Next Payment	Property Taxes	NSR Royalty.	Signing Date	Term (yrs)	Term Extension[3]	Option to Purchase Property	Option to Purchase Royalty	NSR Option Term
Gregory	North Pioneer	1/8.2	$3,600	na	2%	6/16/2006	10	annual	no	$1 M/%	na
Wylie[1]	Savage	3/45.7	$8,600	na	2%	5/22/2006	5	annual	no	$1 M/%	na
Kolo Corp	Jolly Jane & Yellowrose	4/81.7	$6,000	$258	3%	5/8/2006	10	annual	no	$0.85/%	na
Milliken	Pioneer	3/24.5	$5,400	na	2%	5/8/2006	10	annual	no	$1M/%	na
Pritchard	Pioneer	12/203.0	$20,000	na	4%	5/16/2006	10	annual	no	$1M/%	na
Lunar Landing LLC	Connection	12/195.0	$16,200	$207	4%	10/27/2008 Amended 5/28/2014	10	annual	$1 M	$1M/%	35 yrs
Lunar Landing LLC	Sunflower	3/59.2	$5,000	$180	4%	3/30/2015	4	annual	$0.3 M	$0.5/%	35 yrs
Greenspun[2]	Mayflower	11/183.05	$10,000	$214	4%	08/25/2017	10	3 yrs	$7.5 M	No	3 yrs
Sussman	Jolly Jane	2/37.4	$30,000	$113	2%	3/14/2011	10	10 yrs	Inclusive in Royalty Purchase	$1M/%	na
Total	-	**51/748.7**	**$104,800**	**$972**	-	-	-	-		-	-

[1] Original title transferred from Hall to Wylie due to death in the family

[2] Plus 50,000 ITH shares and 25,000 Corvus shares

[3]leases extended annually beyond the original term

4.1.2 Mayflower Area

ITH, through its subsidiary Talon Gold Nevada Inc. (now Corvus Gold Nevada, Inc.), entered into a mining lease with the Greenspun Group with an option to purchase 74 hectares (183 acres) of patented lode mining claims that cover much of the Mayflower prospect. The Mayflower lease requires Corvus to make payments and complete work programs as outlined in Table 4-3. During the term of the lease, any production from the Mayflower property is subject to a sliding scale royalty, also outlined in Table 4-4. Corvus has the right to purchase a 100% interest in the Mayflower property for $7.5 million plus a 0.5% NSR (if gold is less than $500) or 1.0% (if gold is above $500) at any time during the term of the lease (subject to escalation for inflation if the option is exercised after the 10th year of the lease). The annual property taxes to be paid by Corvus for the Mayflower property are $214. On February 11, 2015, the Mayflower mining lease with option to purchase was amended with the addition of an anti-dilution clause applying to the ITH shares and with an increase in the annual payment to include 25,000 Corvus shares. On November 22, 2017, the Mayflower lease was extended until 2027 with the original terms and an annual payment of $10,000, and 50,000 ITH shares and 25,000 Corvus shares.

On February 21, 2013, Corvus signed a purchase agreement for the surface rights to 37 hectares (91 acres) of private land related to five historic patented lode mining claims owned by Mr. and Mrs. Gordon Millman. The claims are located immediately east of the Mayflower deposit. The purchase was completed in December 2015.

Table 4-2 - List of Historic Patented Claims in the Nine NBP Private Land Lease Agreements

Owner/Lessor	Claim Name	Patent Number
Gregory	Jim Dandy	448055
Wylie	Gold Basin	330227
Wylie	Savage	330227
Wylie	Savage 2	330227
KoloCorp.	Black Jack	163170
Kolo Corp	ZuZu	261838
Kolo Corp	Yellowrose	369130
Kolo Corp	Yellowrose No. 1	369130
Milliken	Indiana 1	245488
Milliken	Indiana 2	245488
Milliken	Indiana 3	245488
Pritchard	Banker's Life	493623
Pritchard	Bimettalic 1	46204
Pritchard	Bimettalic 2	46204
Pritchard	Bimettalic 3	46205
Pritchard	Bluff	493623
Pritchard	Conservative	611953
Pritchard	KK1	504301
Pritchard	Mutual	493623
Pritchard	Penn Mutual	493623
Pritchard	Prudential	493623
Pritchard	Sunrise 1	114544
Pritchard	Sunrise 2	114544
Lunar Landing LLC	Dewey Bailey	269019
Lunar Landing LLC	Four Aces	269019
Lunar Landing LLC	Parson Haskins	269019
Lunar Landing LLC	Bull Con	269019
Lunar Landing LLC	Ugly	296019
Lunar Landing LLC	Hardtack	341527
Lunar Landing LLC	Connection Mine	342533
Lunar Landing LLC	Equity	342533
Lunar Landing LLC	Geraldine 3	342533
Lunar Landing LLC	Grey Eagle 2	342533
Lunar Landing LLC	Grey Eagle 4	342533
Lunar Landing LLC	Vinegarroan	342533
Lunar Landing LLC	Sunflower	369130
Lunar Landing LLC	Sunflower No.1	369130
Lunar Landing LLC	Sunflower No. 2	369130
Greenspun	Mayflower	2548
Greenspun	Mayflower No. 1	2548
Greenspun	Mayflower No. 2	2548
Greenspun	Mayflower No. 3	2548
Greenspun	Moonlight	2640
Greenspun	Moonlight No. 1	2640
Greenspun	Moonlight No. 2	2640
Greenspun	Starlight No. 4	2640
Greenspun	Starlight No. 5	2640
Greenspun	Starlight No. 6	2640
Greenspun	Starlight No. 7	2640
Sussman	Jolly Jane	402672
Sussman	Valley View	402672

4.1.3 Other NBP Property Considerations

Corvus staked 652 additional public land federal mining claims between 2012 and 2015, including 595 claims staked in 2012, and 57 claims staked in 2015. Table 4-4 lists the total of 865 mining claims on federal land at the NBP. All of these properties are held through Corvus Gold Nevada, Inc. ("CGNI")**,** which is a wholly owned subsidiary of Corvus Gold, Inc. The Corvus companies were created as a spin-out from ITH on August 26, 2010. These claims are subject to annual fees paid to BLM and to Nye County Nevada.

All of the mining claims on public land are administered by the BLM. The mining claims require payment of yearly maintenance fees to the BLM and Nye County (recording fees) of an aggregate of $153,717 (estimated for 2019). Annual property taxes to be paid by Corvus for some of the properties subject to the original six RGC leases and subsequent leases are tabulated in Table 4-1. These claims give Corvus the right to explore for and mine minerals, including gold and silver, subject to the necessary permits described in Section 20.

The current exploration permits from BLM and NDEP allow Corvus surface access, maintenance of roads, and exploration drilling with a defined amount of accompanying surface disturbance. Current exploration activities are covered by a Plan of Operations (NVN-083002) with the BLM. Two Plans of Operations are in place with the Nevada Department of Environmental Protection ("NDEP") (NDEP #0280 and #0290) that fulfill the State of Nevada permitting obligations on the NBP private and public lands, respectively. Reclamation bonds, related to environmental liabilities to which the NBP is subject, are in place to cover activities on the property. Corvus' reclamation liabilities are covered by surety bonds issued by Lexon Insurance Company in the amount of $313,704 for up to 120 acres of disturbance on public land with the BLM and $209,070 for 20.3 acres of disturbance on private land with NDEP.

Two Notice-level operating permits have been obtained for disturbance in areas outside the Plan of Operations area. These include the North Bullfrog Baseline Collection Project and the East Bullfrog Exploration Project. Corvus' reclamation liabilities for these permits are covered by surety bonds issued by Lexon Insurance Company in the amounts of $46,851 and $31,715, respectively. Additional permits and bonding may be required for the expanded exploration program outlined in the Recommendation Section of this Technical Report.

In December 2013, the Company completed the purchase of 170 hectares (420 acres) parcel of private land 16 kilometers north of the NBP, which includes 1,600 acre-feet of irrigation water rights within the Sarcobatus Flat water basin. The cost of the land was $1,000,000. The Company has registered the purchase of water rights with the Nevada State Engineer ("NSE") and has received Permit 87214 which allows a consumptive use of 1,277 acre-feet of water per year for the purpose of mining. The Company has also received Permit 87745T which designates the use of the water for mining and milling and the transfer of the extraction point of a portion of the water right to NBP and allows the consumption of 67 acre-feet of water per year to support its drilling programs.

Table 4-3 Summary of the Original Terms for the Mayflower/Greenspun Group Lease

Term: 5 Years Beginning December 1, 2007
Five additional years to December 7, 2017, plus an additional 3-year period or so long thereafter as commercial production continues Ten Additional years to December 7, 2027, plus an additional 3-year period or so long thereafter as commercial production continues
Lease Payments: Due on Each Anniversary Date of the Lease
On regulatory acceptance - $5,000 and 25,000 ITH shares Each of first – fourth anniversaries, $5,000 and 20,000 ITH shares Each of fifth – ninth anniversaries, $10,000, 50,000 ITH shares and 25,000 Corvus Common Shares Each tenth to 20th anniversaries, $10,000, 50,000 ITH shares and 25,000 Corvus Common Shares
Work Commitments: Excess Expenditures in Any Year Can Be Carried Forward, or if under Spent the Unspent Portion Paid to Greenspun Group
Years 1-3 $100,000 each year the lease is in effect Years 4-6 $200,000 each year the lease is in effect Years 7-10 $300,000 each year the lease is in effect Years 11-20 $300,000 each year the lease is in effect
Retained Royalty: Production Sliding Scale Net Smelter Return Based on Price of Gold Each Quarter
2% if gold is less than $300 per ounce 3% if gold is between $300 and $500 per ounce 4% if gold is more than $500 per ounce
Advance Minimum Royalty Payments (if not in commercial production by the twentieth anniversary, in order to extend lease for an additional three years)
Years 21-23 $100,000 each year the lease is in effect and commercial production has not been achieved
Purchase Option:
During first 10 years property can be purchased for $ 7.5 million plus an 0.5% NSR (if gold is less than $ 500) or 1.0% (if gold is above $ 500) After the tenth anniversary the $ 7.5 million purchase price escalates by the Consumer Price Index, using the CPI immediately prior to the tenth anniversary as a base

Table 4-4 Summary of the NBP Federal Lode Mining Claims

Land Holder	Claim Name	US Bureau of Land Management Serial Number
Corvus Gold Nevada, Inc.	NB 1 – NB 149	922928 – 923076
Corvus Gold Nevada, Inc.	NB 150	943108
Corvus Gold Nevada, Inc.	NB-151A	1078379
Corvus Gold Nevada, Inc.	NB 152 – NB 154	943110 – 943112
Corvus Gold Nevada, Inc.	NB-155A	1078381
Corvus Gold Nevada, Inc.	NB 156 – NB 161	943114 – 943119
Corvus Gold Nevada, Inc.	NB 162 – NB 213	989863 – 989914
Corvus Gold Nevada, Inc.	NB 214 – NB 510	1069332 – 1069628
Corvus Gold Nevada, Inc.	NB 511	1078379
Corvus Gold Nevada, Inc.	NB 512 – NB 808	1085130 – 1085426
Corvus Gold Nevada, Inc.	NB 809 – NB 865	1109343 – 1109399

4.2 ENVIRONMENTAL LIABILITIES

Corvus currently has permits to conduct exploration activities at NBP with both the Nevada Division of Environmental Protection (NDEP)-Bureau of Mining Regulation and Reclamation (BMRR) and the Bureau of Land Management (BLM). Those permits allow 20 acres and 120 acres of surface disturbance on the private and public land, respectively. The permits for activities on both the public lands and private lands are based on Environmental Assessments that contains environmental baseline data on wildlife, climate and local physical characteristics.

Reclamation bonds are determined assuming the full extent of the disturbance allowed under the two NBP permits and the two NBP Notices of Intent, which currently total $601,330. The currently reclamation liability based on the extent of actual disturbance is estimated to be $246,950.

None of the authors know of any significant factors and risks that may affect access or title to the NBP, or the right or ability to perform work on the Project. To the extent known, the authors know of no other royalties, back-in rights, payments or other agreements and encumbrances to which the property is subject or environmental liabilities, permits or any other significant factors and risks that may affect access, title, or the right or ability to perform work on the Project.

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

The NBP is accessible from Beatty, Nevada, approximately 2.5 hours' drive (193 kilometers, 120 miles) north of Las Vegas, Nevada via US Highway 95. US Highway 95 is the major transportation route between Las Vegas, Nevada, Reno, Nevada and Boise, Idaho. Las Vegas is serviced by a major international airport. Beatty is the closest town to the Project having a population of about 1,100 and providing most basic services. NBP lies 16 kilometers north of Beatty and is accessed via several dirt roads to the west from US Highway 95 between mile markers 66 and 72 (6-12 road miles north of Beatty on US Highway 95). A network of typically well-maintained dirt roads provides access to most of the important exploration areas.

NBP is in Western Nevada's high desert which receives about 15 centimeters ("cm") of precipitation per year, mostly as modest snowfall in the winter and thunderstorms in the summer. The average daily temperature varies from a low of 5°C (40.8°F) in January to a high of 27 °C (80.8 °F) in July, peak temperatures can reach 43°C (110°F). Due to the mild climate at NBP, the operating season is year-round, though occasional thunderstorms may prohibit operations for short periods due to safety concerns regarding lightning strikes.

The hills at NBP are covered with sparse low brush including creosote, four-wing saltbush, rabbit brush and ephedra. The Project is in the Basin and Range province. Topographic relief is several hundred feet. Topography varies from low hills and desert plains to locally very steep, rocky and rugged hills. The elevation of the Project ranges from 1,100m (3,600 feet) to 1,500m (4,800 feet). Most of the Project is characterized by low hills separated by modest width valleys (Figure 5-1).

As described in Section 4, Corvus maintains sufficient surface rights to support mining operations; including waste disposal areas, tailings storage areas, heap leach pads and mill sites, subject to necessary permits which are defined in Section 20. The private and public lands included in the NBP are contiguous and Corvus has access to all areas of the property which are underlain by the gold mineralization and are the basis of this report. Electrical power is available from the Valley Electric Association (VEA) located in Pahrump NV. VEA has a main powerline that runs along the east side of US Highway 95 through NBP which has sufficient available capacity to provide the NBP as currently planned. Corvus owns ground water permits with sufficient annual volume to meet the processing and mining needs of the project. The extraction point for the water can be temporarily relocated at NBP. These infrastructure elements are described in Section 18. The towns of Beatty (10 km south of NBP), Tonopah (140 km north of NBP) and Pahrump (119 km south of NBP) are connected by paved highways and support an ample population for mining personnel.

Figure 5-1 - Mayflower Ridge Looking to the Northwest

6. HISTORY

6.1 BULLFROG DISTRICT

The Bullfrog district is informally divided into three subdistricts: 1) Main Bullfrog; 2) North Bullfrog; and 3) Bare Mountain. Figure 2-1 shows the Corvus controlled property which extends into all three of the subdistricts. Gold was first discovered in the main Bullfrog district by Frank “Shorty” Harris and Ernest Cross on August 9, 1904 at the location of the Original Bullfrog mine (Elliot, 1966). The discovery sparked a rush of prospectors and within a few weeks the district was staked “for nine miles in all directions from the sagebrush flats, including part of the desert to the tip of every summit in sight” (Elliot, 1966). Lincoln (1923) reported that 111,805 ounces of gold and 868,749 ounces of silver were produced in the district between 1905 and 1921. According to Lincoln (1923), the Montgomery-Shoshone mine was the most important mine in the district, operating between 1907 and 1910. A number of other small mines in the Main Bullfrog and North Bullfrog subdistricts contributed to the total reported production.

Modern exploration for precious metals in the main Bullfrog subdistrict began as early as 1982, when geologists from St. Joe Minerals Corporation became interested in the Montgomery-Shoshone area. St. Joe Minerals conducted extensive exploration in the area of the Montgomery-Shoshone and Senator Stewart mines, resulting in the discovery of the Bullfrog vein deposit in 1987 (Jorgenson et al, 1989). Several company acquisitions resulted in Barrick Gold Corporation being the final owner and operator of the mine. The Bullfrog mine produced gold and silver from three deposits including: 1) Bullfrog (open pit and underground); 2) Montgomery-Shoshone (open pit); and 3) Bonanza Mountain (open pit). Between 1989 and 1999, the Bullfrog mine produced 2.31 million ounces (“Moz”) of gold and 3.0 Moz of silver (NBMG MI-2000, page 34). The Gold Bar open pit mine, located 4 kilometers northeast of the Original Bullfrog mine, produced a small (unreported) amount of gold in the late 1980’s.

Modern exploration for precious metals in the North Bullfrog subdistrict began as early as 1974 (see Section 6.2, NBP History). There has been no reported modern production from the North Bullfrog subdistrict.

Resurgent interest in the Bullfrog District began in 2017, and major gold mining companies, including Coeur Mining, Kinross and AngloGold Ashanti have developed large property positions adjacent to the Corvus properties.

6.2 NBP HISTORY

Also known as the Pioneer district, the early history of the NBP property is comingled with the main Bullfrog district. A rush of prospectors came to the Bullfrog Hills soon after the original Bullfrog discovery, which resulted in discoveries at the Mayflower mine in 1906 and the Pioneer mine in 1907. The Pioneer and Mayflower mines were the principal mines at the NBP. The Mayflower mine was intermittently active between 1906 and 1940 (Kral, 1951). The Pioneer mine was intermittently active between 1908 and 1920 with 1909-1910 being the most productive years (Kral, 1951). There are no accurate production figures, but limited records suggest that head grades were between approximately 0.5 to 1 ounce of gold per ton at both the Mayflower and Pioneer mines. Underground development at Sierra Blanca, Jolly Jane, Savage Valley and YellowJacket deposits between 1910 and 1914 had no reported production.

Modern exploration for precious metals at NBP started with Cordex at the Connection prospect in 1974. Table 6-1 outlines a number of companies that worked in various target areas up to 1996. These programs consisted of a variety of activities including surface mapping and sampling, underground mapping and sampling, and drilling. Through the 1996 Barrick program, approximately 249 rotary and reverse-circulation drill holes were drilled on the Project by eight different operators. Barrick had dropped all interest in the NBP area by 1998.

Table 6-1 - Summary of Other Companies That Have Explored NBP

Company	Years of Activity	Principal Target
Cordex Exploration Co.	1974-1982	Connection, Pioneer
US Borax Incorporated	1982	Mayflower
GEXA/Galli Exploration	1984-1991	Pioneer, Connection
CR Exploration	1984-1985	Mayflower
Western States	1987	West Mayflower
Bond Gold/Sunshine JV	1988-1994	Sierra Blanca, YellowJacket
Pathfinder Minerals	1991, 1992	Pioneer
Barrick Gold Corporation	1995-1996	Jolly Jane, Sierra Blanca, Mayflower, etc.

With the downturn in gold price around the start of the 2000s, interest in the NBP area was essentially nonexistent. Redstar Gold Corp. (RGC) began staking unpatented claims and acquiring leases on patented claims in 2005-2006. In March 2007, RGC granted ITH the right to earn an interest in the NBP and thereafter formed the (North Bullfrog Project Joint Venture (NBPJV). In December 2007, ITH completed a lease of the Mayflower property, which was included in the NBPJV. Following the execution of the NBPJV option/joint venture agreement, ITH commenced active exploration on the NBP. In October 2008, RGC completed a lease of the Connection property, which was also included in the NBPJV. On August 4, 2009, ITH purchased RGC’s interests in the property and continued the exploration program as sole owner/lessor. On August 26, 2010, ITH spun out Corvus as a separate public company in a transaction that resulted in Corvus acquiring all of the interest and responsibilities in the NBP.

7. GEOLOGICAL SETTING AND MINERALIZATION

The geology of the Sierra Blanca-Savage Valley, YellowJacket, Mayflower and Jolly Jane deposits, which contain the mineralization at the NBP is described in this section of the report.

7.1 REGIONAL GEOLOGICAL SETTING

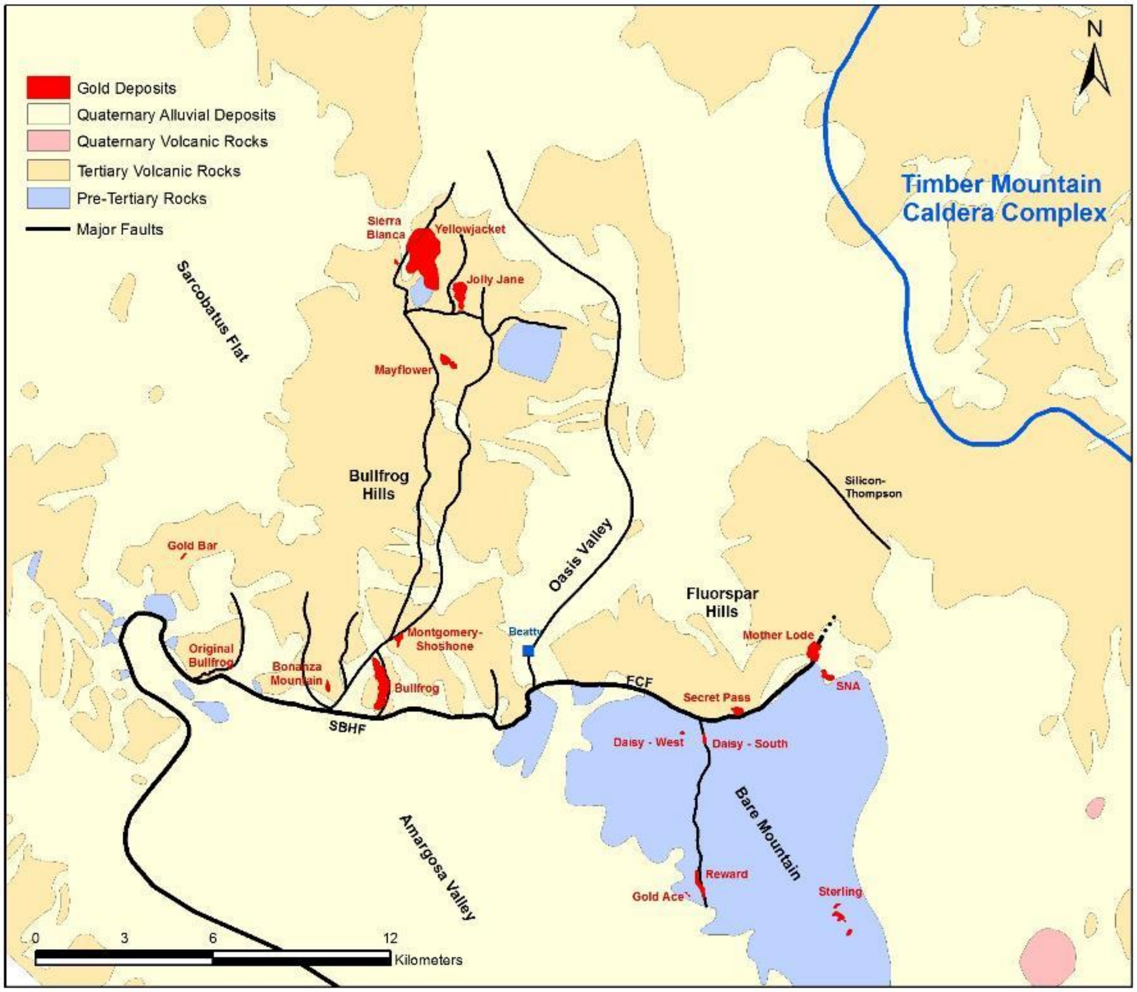
North Bullfrog lies within the Walker Lane mineral belt and the Southwestern Nevada Volcanic Field (SWNVF). The regional stratigraphy includes a basement of Late Proterozoic to Late Paleozoic metamorphic and sedimentary rocks. Basement rocks are overlain by a thick pile of Miocene volcanic and lesser sedimentary rocks of the SWNVF, ranging in age from ~15-7.5 Ma (Figure 7-1). The pre-Tertiary rocks exhibit large-scale folding and thrust faulting, having been subjected to compressional deformation associated with multiple pre-Tertiary orogenic events. The stratigraphy of the SWNVF is dominated by ash flow tuff sheets erupted from a cluster of nested calderas known as the Timber Mountain Caldera Complex. The southwestern edge of the caldera complex lies approximately ten kilometers east of the NBP (Figure 7-1). The stratigraphy of the SWNVF includes voluminous ash flow tuff sheets, smaller volume lava flows, shallow intrusive bodies, and lesser sedimentary rocks. Many of the volcanic units exposed around NBP include ash flow tuffs that originated from the caldera complex. Other volcanic units are locally sourced outside of the caldera complex, particularly at the NBP.

The Bullfrog and Fluorspar Hills comprise a somewhat isolated structural domain within the Walker Lane, where both pre-Tertiary and Miocene rocks have been subjected to large-scale, W- to WNW-directed, syn-volcanic extension (i.e. down-to-the-west normal faulting and east-tilting of stratigraphy). Extensional faulting was coincident with magmatism and volcanic activity between ~15-9.4 Ma. Hydrothermal alteration and gold mineralization were also episodic through this time period. If present, through-going right-lateral faults of the Walker Lane are poorly exposed in the SWNVF. One possible example of a NW-trending, through-going Walker Lane structure cuts through the historic Silicon and Thompson mine areas to the east of NBP (Figure 7-1). Despite the dominance of caldera volcanism in the region, little or no mineralization is associated with any caldera ring fracture system. Rather, mineralization is typically associated with extensional faults outside of the caldera complex.

Extension is accommodated by the Bullfrog Hills Fault System (BHFS); a complex group of kinematically linked faults that facilitate WNW-directed extension and east-tilted block rotation. The primary structure of the BHFS is the Southern Bullfrog Hills Fault (SBHF). The SBHF is an east-west-trending, north-dipping, district-scale, low-angle detachment fault (Eng et al. 1996). West of Beatty NV., in the main Bullfrog district, the SBHF separates Proterozoic metamorphic rocks in the footwall from weakly metamorphosed Paleozoic sedimentary and Tertiary volcanic rocks in the hanging wall. East of Beatty, in the Bare Mountain sub-district, the same fault is called the Fluorspar Canyon Fault (FCF, Figure 7-1). The FCF cuts up-section from west to east such that in Fluorspar Canyon it separates Paleozoic sedimentary rocks (footwall) from brittle Tertiary volcanic rocks (hanging-wall). The FCF continues to cut up-section to the east until it eventually separates brittle Tertiary rocks (footwall) from brittle Tertiary rocks (hanging-wall) at Mother Lode. The magnitude of displacement along the SBHF-FCF appears to decrease from west to east across the greater Bullfrog district. The northward dip of the SBHF-FCF generally increases from west to east including: ~20° at the Bullfrog mine, 25-30° north in lower Fluorspar Canyon, 45° north at the Secret Pass mine, and 55-65° northwest at Mother Lode.

Secondary structures of the BHFS include large-displacement, NNW- to NNE-trending, moderately to steeply west-dipping, down-to-the-west, normal faults. These faults accommodate the east-tilting of the Tertiary units throughout the Bullfrog and Fluorspar Hills. Hydrothermal alteration and gold mineralization are often spatially associated with these large-displacement faults. Such faults are expected to have listric shapes at depth. The MP fault, which hosts the Bullfrog vein at the Bullfrog mine, is an example of a listric fault. Another example is the Contact Fault, which truncates the north side of the Montgomery-Shoshone deposit. The Contact fault also hosts low-grade mineralization under Rhyolite Valley. The Road Fault at the NBP is the northern continuation of the Contact Fault. Such faults are interpreted to sole into the SBHF-FCF detachment fault at depth.

Figure 7-1 - Regional Geology of the Greater Bullfrog District



The long period of syn-volcanic extension in the western SWNVF between ~15-9.4 Ma (depending on location) includes at least two periods of accelerated extension documented in the Bullfrog and Fluorspar Hills. Connors, et al. (1998) have identified a major period of extension between ~12.7 Ma and 11.6 Ma culminating with the eruption of the Timber Mountain Group ash-flows between 11.6 and 11.45 Ma. Evidence for this period of extension lies west of Mother Lode where the 12.7 Ma Tiva Canyon Tuff is tilted up to 45° east, but the nearby 11.6 Ma Rainier Mesa Tuff is essentially horizontal. Block rotation of up to 45° is documented between 12.7 and 11.6 Ma in the Fluorspar Hills. Mineralization at Mother Lode is coincident with the onset of this period of accelerated extension. Connors et al. (1998) postulate a second period of accelerated extension between 11.4 Ma and 10.5 Ma, which resulted in major block rotation, rapid erosion and the deposition of the Rainbow Mountain Debris Flow Sequence. In the Mayflower area, the base of the Rainbow Mountain Debris Flow Sequence is tilted as much as 55° east, whereas the top of the sequence is tilted only 25° east. Block rotation of up to 30° is documented in this area while the Mayflower basin was filling with debris. In the southern Bullfrog Hills east of Rainbow Mountain, the 11.45 Ma Ammonia Tanks tuff is tilted as much as 70° east, whereas the base of the overlying Rainbow Mountain sequence is tilted only 35° east. The block rotation of 35° is documented in this area between 11.4 and 10.5 Ma. Extensional faulting continued through ~9.5 Ma, with an additional 25° of eastward rotation of the Rainbow Mountain Sequence. Extensional faulting ceased in the Bullfrog Hills prior to the eruption of the relatively flat-lying 9.4 Ma Pahute Mesa Tuff (Connors, et al., 1998). The 10 Ma age of Bullfrog and Mayflower mineralization and the 9.5-10.2 Ma age of the Eastern Steam-heated Zone at NBP coincide with the culmination of extensional tectonism in the Bullfrog Hills.

Extensional faulting in the Bullfrog and Fluorspar Hills created fault-bounded sedimentary basins which filled with basement- and volcanic-derived sediments (e.g. the Sedimentary Rocks of Joshua Hollow, the Jolly Jane Formation, and the Rainbow Mountain Debris Flow Sequence). During younger periods of extension, older normal faults in the hanging-walls of large-displacement listric faults may have experienced significant reactivation and subsequent eastward rotation, such that they may exhibit relative reverse displacement.

7.2 NBP GEOLOGY

7.2.1 STRATIGRAPHY

The stratigraphy of the northern Bullfrog Hills was most recently described in published mapping by Connors et al. (1998). Where possible, the terminology of Connors et al. (1998) has been preserved in the unit assignments at NBP. New geochronology has shown that some units were incorrectly correlated and required new names. The most significant examples are the Sierra Blanca Tuff and the Pioneer Formation, which were previously included in the Crater Flat Group. Based on drilling and geochronology studies conducted by Corvus (see Section 7.2.2.1), the local stratigraphy has been significantly refined to warrant the specification of the North Bullfrog Hills Volcanic Complex (NBHVC, Table 7-1). Brief descriptions of the stratigraphy at the NBP area are given in the following sections. Figure 7-2 is a compiled geologic map of the NBP and surrounding areas. An explanation of the map units on Figure 7-2 is shown in Figure 7-3.

7.2.1.1 PALEOZOIC BASEMENT

7.2.1.1.1 WOOD CANYON FORMATION – PZW

The Wood Canyon Formation is Lower Cambrian in age and made up of variably calcareous shale, siltstone, and sandstone, with occasional beds of massive to finely laminated limestone. Lenses of massive siliceous quartzite are also present. The minimum thickness is 350 meters (Connors, et al., 1998).

7.2.1.1.2 ZABRISKIE QUARTZITE – PZZ

The Zabriskie Quartzite is Lower Cambrian in age and generally consists of massive, fine- to medium-grained ortho-quartzite with poorly preserved bedding. The minimum thickness is around 370 meters (Connors, et al., 1998).

7.2.1.1.3 CARRARA FORMATION – PZC

The Carrara Formation is Middle to Lower Cambrian in age and consists largely of thin- to medium-bedded limestone. The lower parts of the Carrara Formation contain cherty, argillaceous and silty limestone interbeds. The minimum thickness is around 280 meters (Connors, et al., 1998)

7.2.1.2 JOLLY JANE FORMATION – TJJ

The Jolly Jane Formation consists of a basal Tertiary conglomerate overlain by heterogeneous sediments including hematitic mudstone, siltstone, sandstone and conglomeratic sandstone. The conglomerate characteristically contains abundant clasts of pre-Tertiary basement rocks. The type locality is in drill holes at Jolly Jane. It consists of up to 50 meters of heterogeneous sediments that appear to have accumulated in isolated structural basins prior to and during the onset of volcanism. The sediments are typically siliceous and hematitic but may include minor calcareous or carbonaceous intervals. The thickness is highly variable, having been deposited on a Tertiary erosional unconformity of significant relief. The Jolly Jane Formation may be time transgressive along the basal Tertiary unconformity. It is considered the litho-stratigraphic equivalent of the Titus Canyon Formation of Connors et al. (1998).

7.2.1.3 NORTH BULLFROG HILLS VOLCANIC COMPLEX

The North Bullfrog Hills Volcanic Complex (NBHVC) is a name that has been given to a sequence of largely locally sourced lavas and pyroclastic rocks exposed in the Western Resource Area of the NBP (Figure 7-2). Some of these rocks were incorrectly correlated with the Crater Flat Group by Connors, et al. (1998). The subdivisions identified in Table 7-1 are based on age-dating and geologic logging, which have clarified both the stratigraphic order and aerial distribution of the units. With ages ranging from 15-14 Ma, the NBHVC correlates with some of the oldest volcanic rocks in the SWNVF. The NBHVC occupies the Tr1 time-stratigraphic position of Eng et al. (1996).

Table 7-1 - Overview of the Stratigraphy of the North Bullfrog Hills

Au min		Major Unit Name	Symbol	Formation	Lithodeme	Major Unit Description
		Quaternary Cover	Qc			Quaternary alluvium, colluvium, talus, and mine dumps
		Gravels of Sober-up Gulch	Tgs	Gravels of Sober-up Gulch		Semi-consolidated cobble and boulder gravels
		Pumiceous Sediments	Tps			Light colored tuffaceous sandstone, conglomerate with pumice clasts
		Rainbow Mountain Sequence	Trl	Donovan Mountain Latite		Latite and quartz latite lava flows and flow breccias.
			Trt	Tuffs and Lavas of Rainbow Mountain		Non-welded crystal-lithic rhyolite ash-flow tuff and aphanitic flow-banded rhyolite flows and domes with minor sedimentary interbeds.
Connection	Mayflower		Tdf	Rainbow Mountain Debris Flow Sequence		A sequence of intercalated heterolithic and monolithic debris flow breccias derived from local stratigraphy. Heterolithic sequences

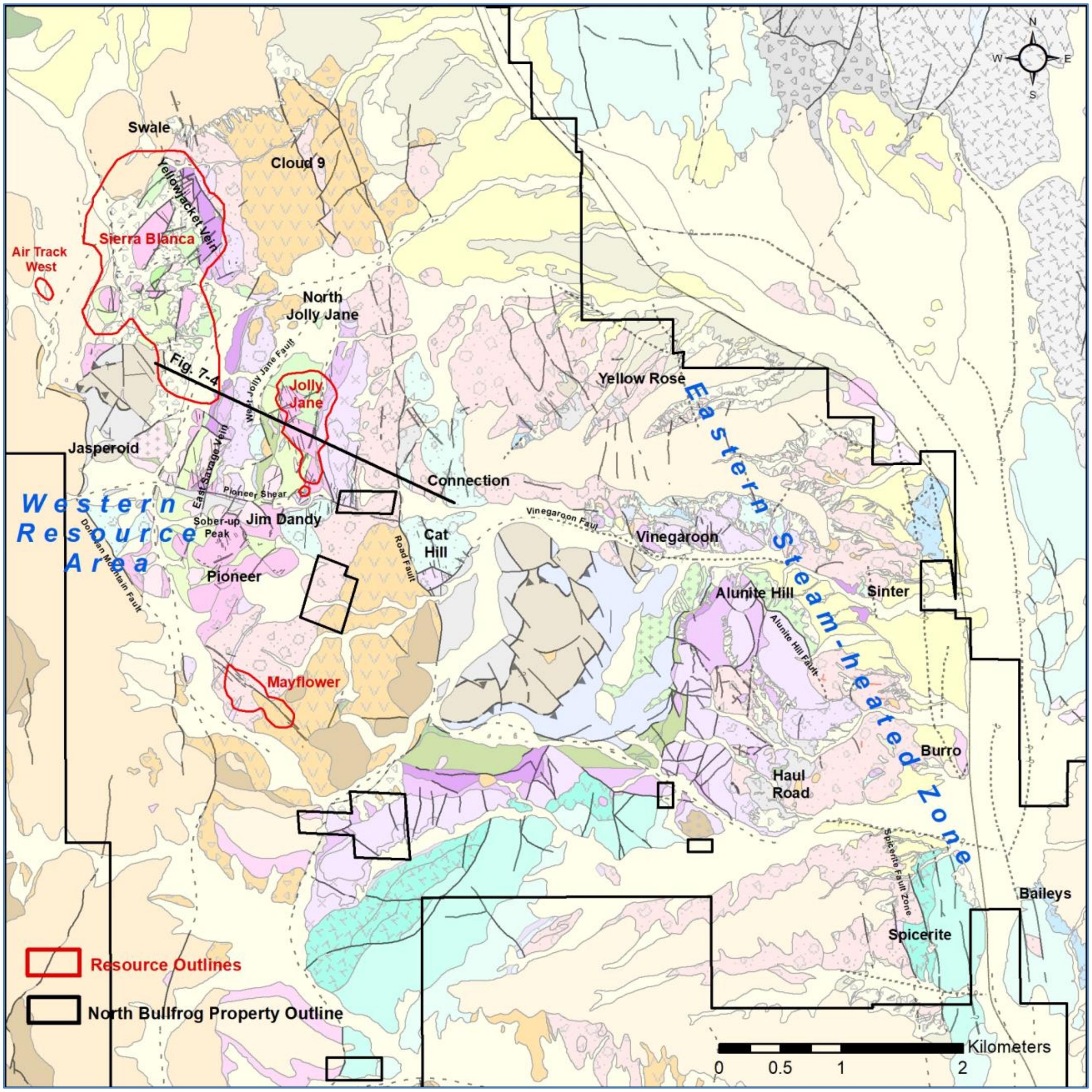
						are poorly sorted, consisting of sand- to boulder-size clasts of volcanic and Pz sedimentary rocks. Monolithic breccias are interpreted as landslide megabreccia deposits shed off local fault scarps.
		Timber Mountain Group	Tma	Ammonia Tank Tuff		Moderate to densely welded crystal-rich rhyolite ash-flow tuff.
			Tmr	Rainier Mesa Tuff		Moderate to densely welded crystal-rich rhyolite ash-flow tuff.
			Tprr			Variably flow-banded rhyolite and rhyolite flow breccia
			Tprt			Light-colored, non-welded, locally bedded, crystal-lithic ash-flow tuff
		Paintbrush Group	Tp	Paintbrush Tuff		Aphanitic phenocryst-poor welded rhyolite ash-flow tuff.
		Crater Flat Group	Tcb	Bullfrog Tuff		Variably welded crystal-lithic rhyolite ash-flow tuffs. Probable equivalent of Bullfrog Member of the Crater Flat Group.
	Sierra Blanca, Jolly Jane and Yellowjacket	Lithic Ridge Tuff	Tlr	Lithic Ridge Tuff		Variably welded, lithic-rich "dacitic" tuff
		North Bullfrog Hills Volcanic Complex	Td	Savage Valley Dacite	**North Bullfrog Suite:** rhyolitic to dacitic dikes, sills and domes of ambiguous origin	Upper Member consists of intercalated lava flows, breccias and pyroclastics of dacitic composition. Probable stratigraphic correlation to Tr1g quartz latite unit in Southern Bullfrog Hills
						Lower Member consists of complex mixture of rhyolite lava flows, tuffs, breccias and sedimentary rocks covering post-Sierra Blanca erosional surface
			Tsb	Sierra Blanca Tuff		Large compound cooling unit of variably welded crystal-lithic rhyolite ash-flow tuff. Lies stratigraphically within the NBHVC, but likely sourced from an unknown older caldera related to the Timber Mountain caldera complex. May be correlative to Upper Tuff of Sawtooth Mountain.
			Tpf	Pioneer Formation		Upper Epiclastic Member: mixed bedded to non-bedded heterogeneous epiclastics: poorly sorted silty, sandy, pebbly and cobbly sediments
						"Green tuff" of Sierra Blanca; Heterogeneous non-welded to semi-welded, lithic-poor to lithic-rich crystal ash-flow tuffs with scattered intervals of bedded tuff and epiclastics. May be correlative to Lower Tuff of Sawtooth Mountain.
			Tnb			
			Tsf	Savage Formation		Sequence of intercalated lava flows, intrusives and epiclastic debris of dacitic to rhyolitic composition.
	SB-JJ	Jolly Jane Formation	Tjj	Jolly Jane Formation		Heterogeneous sedimentary sequence consisting of mudstone, siltstone, sandstone and conglomerate accumulated in localized structural basins.

		Paleozoic Basement	PzC	Carrara Limestone		Micritic and argillaceous carbonaceous limestone
			PzZ	Zabriskie Quartzite		Massive quartzite
			PzW	Wood Canyon Formation		Quartz-rich calcareous siltstone, sandstone, quartzite

7.2.1.3.1 SAVAGE FORMATION – TSF

The Savage Formation consists of locally sourced lava domes, flows, pyroclastics and associated intrusive rocks of generally dacitic composition. The Savage Formation is recognized in drill holes under south Savage Valley, Air Track Hill and Jolly Jane, where it overlies and inter-fingers with sediments of the Jolly Jane Formation. The Savage Formation may also include variably carbonaceous epiclastic intervals of re-worked dacite. The thickness of the Savage Formation varies greatly from 0-100 meters, possibly reflecting both fault-bounded basins and the areal distribution of individual domes, flows and associated pyroclastic or epiclastic aprons. The Savage Formation is correlative to the lower portion of the Trl unit as described in the Southern Bullfrog Hills by Eng et.al. (1996). The Savage Formation represents the onset of SWNVF volcanism in the northern Bullfrog Hills. It is locally mineralized in the Sierra Blanca and Jolly Jane areas.

Figure 7-2 - Geologic Map of NBP Showing Target Areas, Resource Outlines and Property Outline



See Figure 7-3 below for the explanation of map units.

Figure 7-3 - Map Units Legend for Figure 7-2

7.2.1.3.2 PIONEER FORMATION – TPF

The Pioneer Formation consists of a rather monotonous sequence of light green, variably welded, lithic-lapilli ash-flow and air-fall tuffs. The tuffs show marked variations in clast size ranging from coarse tuffaceous sedimentary breccias to fine lithic-lapilli tuffs. Bedded epiclastic intervals have also been observed throughout the Pioneer Formation. The type locality for the Pioneer Formation is west of the Pioneer Mine. In the subsurface north of Sierra Blanca, the Pioneer Formation is interbedded with rhyolite bodies assigned to the North Bullfrog Intrusive Suite. The rhyolite bodies are interpreted as lava flows and shallow intrusive bodies that are genetically related to, and comingled with, the pyroclastic deposits of the Pioneer Formation. The lithic content increases significantly near the top of the unit, grading into a coarse heterogeneous tuffaceous epiclastic sequence known as the Upper Epiclastic

Member. The Upper Epiclastic Member forms a semi-continuous marker horizon at the top of the unit. The thickness of the Pioneer Formation varies from just a few meters at south Savage Valley and Jolly Jane, to several hundred meters north of Sierra Blanca. Divergence of compaction foliation directions between the Pioneer Formation and the Sierra Blanca Tuff indicate that some tilting and erosion of the Pioneer Formation took place prior to the eruption of the Sierra Blanca Tuff. The Pioneer Formation is widely mineralized but is generally lower grade than the overlying Sierra Blanca Tuff.

7.2.1.3.3 NORTH BULLFROG INTRUSIVE SUITE – TNB

The North Bullfrog Intrusive Suite consists of plugs, domes, dikes, sills and flows of generally rhyolite composition that are recognized over much of the NBP. Rocks assigned to the intrusive suite intrude Paleozoic sediments and nearly all volcanic units of the North Bullfrog Hills Volcanic Complex. A lithodeme classification has been created in order to deal with the ambiguity of the emplacement mechanism (intrusive or extrusive) and relative age of many of these bodies (Table 7-1). Plug or dome-like bodies of rhyolite are exposed at south Savage Valley, the north flank of Sober-Up Peak, and at south Jolly Jane. Field relationships suggest that these bodies are intrusive into the Pioneer and Savage Formations. There is no significant age difference between the Pioneer Formation and Tnb rhyolites, and for the most part they are geochemically indistinguishable.

There is one suite of rhyolite bodies that have no compositionally similar pyroclastic rocks. These rhyolites are relatively depleted in light rare earths such as cerium and lanthanum compared to the other rhyolites. They occur as intrusive plugs and flow-domes both above and below the Sierra Blanca Tuff in the Sierra Blanca area. These rhyolites are the host rocks for significant higher-grade disseminated and vein mineralization in the northern part of the Sierra Blanca resource area. Their genetic association with the mineralization events at Sierra Blanca has yet to be determined.

7.2.1.3.4 SIERRA BLANCA TUFF – TSB

The Sierra Blanca Tuff is a large cooling unit of rhyolitic ash flow tuff that underlies much of the Western Resource Area of the NBP. It is named for the exposures at Sierra Blanca and North Sierra Blanca ridges. The unit varies in thickness from 70 meters at Jolly Jane to 170 meters at North Sierra Blanca. The caldera source of the Sierra Blanca Tuff is unknown but is likely located outside of the Bullfrog Hills. Based on lithology and age, it is correlated to the Upper Tuff of Sawtooth Mountain mapped by Maldonado and Hausback (1990) in the southern Bullfrog Hills.

The Sierra Blanca Tuff has a 5-15-meter-thick distinctive shard-rich interval (Tsb1) at the base in the Sierra Blanca area. Tsb1 commonly overlies the Upper Epiclastic Member of the Pioneer Formation. Above the shard-rich marker, the tuff is a relatively homogeneous densely welded crystal tuff with well-developed compaction foliation exhibited by flattened pumice. Thickness variations are partly due to filling of paleo-topography at the base, and an erosional unconformity at the top. The brittle nature of the densely welded Sierra Blanca tuff facilitated significant brittle fracturing. The increased permeability from fracturing likely played a significant role in focusing hydrothermal fluids through the unit. The Sierra Blanca Tuff is the most important host at NBP for both disseminated- and vein-style mineralization.

7.2.1.3.5 SAVAGE VALLEY DACITE – TD

The Savage Valley Dacite consists of locally sourced dacite lava domes, flows and associated pyroclastic and epiclastic deposits overlying the Sierra Blanca Tuff. The Lower Member (Tdl) is a heterogeneous sequence of dacitic pyroclastic and epiclastic deposits of highly variable thickness. The Lower Member locally includes a rhyolite flow breccia, which in some drill holes is of sufficient volume to be differentiated as a rhyolite flow of the North Bullfrog Intrusive Suite. The Lower Member grades upward into more homogenous dacite lava flows of the Upper Member. The Upper Member (Td) consists of fine- to medium-grained porphyritic lava flows of dacitic to andesitic composition. Where it is relatively unaltered, the Savage Valley Dacite is strongly magnetic. The Savage Valley Dacite overlies the Sierra Blanca Tuff along an erosional unconformity of significant relief. Locally the Savage Valley Dacite appears to be deposited on the Pioneer Formation. Some or all of the Sierra Blanca stratigraphy may have been eroded or not deposited in these areas. The Savage Valley Dacite is correlative to the Tr1g unit as described in the southern Bullfrog Hills by Eng et.al. (1996). The Savage Valley Dacite is mineralized in the Sierra Blanca resource area.

7.2.1.4 LITHIC RIDGE TUFF – TLR

The Lithic Ridge Tuff is a regional tuff unit that was first recognized at NBP in 2015. Previously thought to be the Tram Tuff of the Crater Flat Group, it occupies the appropriate regional stratigraphic position above the Savage Valley Dacite and below the Bullfrog Tuff. The Lithic Ridge Tuff consists of poorly to moderately welded, generally lithic-rich, crystal-bearing ash-flow tuff. Lithic content varies from <5 to 20%. It is generally biotite-rich and commonly contains green altered clasts of Savage Valley Dacite. It was originally described as “dacite tuff” and assigned to the upper part of the Savage Valley Dacite. It has been differentiated in several drill holes in the YellowJacket area as overlying and locally inter-fingering with the Savage Valley Dacite.

7.2.1.5 CRATER FLAT GROUP – TCB

The regionally extensive Crater Flat Group has been described in detail by Carr et al. (1986), and three tuff members are recognized in the SWNVF by Sawyer et al. (1994). The largest of these members, the Bullfrog Tuff (Tcb), is exposed along the east side of YellowJacket, Savage Valley, and Jolly Jane. The Bullfrog Tuff is a moderately crystal rich, densely welded, ash-flow tuff. The densely welded middle portion grades upward into a weak to moderately welded upper portion. The contact between the Bullfrog Tuff and underlying Lithic Ridge Tuff is exposed along the east side of Savage Valley, where an interval of bedded tuffaceous epiclastic rocks separates the two units. The

Bullfrog Tuff has been dated at 13.25 Ma (Sawyer et al., 1994). The only gold mineralization interpreted to be hosted in the Bullfrog Tuff at NBP is in drill hole NB-11-80 under the Connection area.

7.2.1.6 PAINTBRUSH GROUP – TP

The Paintbrush Group in the Bullfrog Hills includes the 12.8 Ma Tonopah Springs Tuff and the 12.7 Ma Tiva Canyon Tuff (Sawyer et al., 1994). The Paintbrush tuffs are distinctly shard-rich and phenocryst-poor compared to other major ash flow sheets in the SWNVF. The Paintbrush Group varies from 190 to >240 meters in thickness in the northern Bullfrog Hills (Connors et al., 1998). The two subunits have not been differentiated in published mapping. At the NBP, the Paintbrush Group exists primarily as large slide blocks or monolithic breccias within the Rainbow Mountain Debris Flow Sequence. No significant mineralization has been found to date within the Paintbrush Tuff or monolithic Paintbrush breccias. However, Paintbrush breccias are hydrothermally altered in a number of places at NBP and are considered potential host rocks as they were present at the time of YellowJacket and Mayflower mineralization.

7.2.1.7 TIMBER MOUNTAIN GROUP – TMR AND TMA

Regionally there are two large-volume ash flow sheets that make up the bulk of the Timber Mountain Group. These include the 11.6 Ma Rainier Mesa Tuff (Tmr) and the 11.45 Ma Ammonia Tanks Tuff (Tma). The Timber Mountain tuffs are rather distinctive because of their large (2-4 mm) and abundant (20%) phenocrysts of quartz and feldspar. The units can be distinguished from each other by the presence of sphene and bluish chatoyant sanidine phenocrysts in the Ammonia Tanks Tuff. The lower part of the Timber Mountain Group includes smaller volume rhyolitic tuffs and lava flows that are known as the pre-Timber Mountain tuffs (Tprt) and lavas (Tprr) respectively, which underlie the Rainier Mesa Tuff.

In-situ bedrock exposures of the Timber Mountain Group are found only in the southern and eastern portions of the NBP area. The Timber Mountain Group exhibits widespread steam-heated alteration at the Spicerite and Baileys target areas at the Eastern Steam-Heated Zone (Figure 7-2). All sub-units of the Timber Mountain Group have potential to host younger disseminated and vein type mineralization at NBP. Over much of the NBP, the Rainier Mesa and Ammonia Tanks tuffs occur only as large slide blocks or monolithic breccia bodies within the Rainbow Mountain Debris Flow Sequence. Clasts of the Rainier Mesa and Ammonia Tanks tuffs are a significant component of heterolithic portions of the Rainbow Mountain Debris Flow Sequence.

7.2.1.8 RAINBOW MOUNTAIN SEQUENCE

The Rainbow Mountain Sequence consists of a complex group of sedimentary and volcanic deposits that record the onset of a major phase of extensional tectonism and erosion in the Bullfrog Hills between 11.4 and 10.5 Ma (Connors et al., 1998). The sequence includes in ascending order: 1) the Rainbow Mountain Debris Flow Sequence; 2) the Tuffs and Lavas of Rainbow Mountain; and 3) the Donovan Mountain Latite.

7.2.1.8.1 RAINBOW MOUNTAIN DEBRIS FLOW SEQUENCE – TDF

The Rainbow Mountain Debris Flow Sequence is the most heterogeneous unit at NBP. It consists of thick sequences of non-bedded, poorly sorted, heterolithic and monolithic sedimentary breccias, and interbedded fluvial sediments. Heterolithic debris flow breccias contain sand- to large boulder-size clasts of Miocene volcanic and Paleozoic sedimentary rocks. These deposits are largely the result of the re-working of volcanic and basement rocks via gravity sliding and alluvial fan development around fault-bounded structural highs. Relatively intact blocks of monolithic breccias are interpreted as landslide megabreccia deposits that were shed off local fault scarps. The volcanic debris is derived from many of the SWNVF units including the NBHVC, the Crater Flat Group, Paintbrush Group and Timber Mountain Group (Table 7-1). The Debris Flow Sequence lies unconformably on an erosional surface (i.e. angular unconformity) cut on nearly all pre-Rainbow Mountain Sequence map units. The thickness of Debris Flow Sequence exceeds 300 meters in the Mayflower area. Gold mineralization at Mayflower, Connection and Cat Hill is hosted in the Rainbow Mountain Debris Flow Sequence.

7.2.1.8.2 TUFFS AND LAVAS OF THE RAINBOW MOUNTAIN SEQUENCE – TRT AND TRR

Generally overlying and inter-fingering with the Debris Flow Sequence are light-colored, poorly- to non-welded, pumiceous crystal- and lithic-rich tuffs. The Rainbow Mountain Tuff includes three separate units in ascending order: Trt, Trt2, and Trt3. The most volumetrically significant tuff sub-unit is the middle 10.5 Ma Trt2, which overlies the Debris Flow Sequence over much of the NBP. Trt2 is correlative to the Tr11 unit of the Rainbow Mountain Sequence in the southern Bullfrog Hills (Eng et al., 1996). Trt2 is up to 300 meters thick in the Mayflower-Pioneer area. The base of Trt2 is locally mineralized at Mayflower. Trt2 has yielded an Ar-Ar date of 10.1 Ma (Connors et al., 1998), and a Zircon date of 10.5 Ma (Valencia date, Table 7-2). The Trt tuffs are genetically related to and inter-finger with locally sourced, flow-banded rhyolite plugs, domes and flows (Trr).

7.2.1.8.3 DONOVAN MOUNTAIN LATITE – TL

The uppermost sub-unit of the Rainbow Mountain Sequence is the Donovan Mountain Latite. It consists of numerous lava flows and flow breccias of dark-colored, relatively unaltered, porphyritic latite and quartz latite. The latite occurs primarily in the hanging wall (west) of the Donovan Mountain Fault. Light colored tuffs of Trt3 commonly occur as pyroclastic intervals between individual lava flows. Latite flows of similar composition cap the Rainbow Mountain Sequence throughout the Bullfrog Hills. Ar-Ar dates of between 10.0 and 10.7 Ma are reported by Connors et al. (1998).

7.2.1.9 PUMICEOUS SEDIMENTS – TPS

An unnamed unit of heterogeneous pumiceous sediments overlies the various units of the Rainbow Mountain Sequence. The sediments include white, light-grey, greenish-grey and light brown, weakly indurated, bedded tuffaceous sandstone and conglomerate. The beds may be moderately- to poorly- sorted, commonly containing abundant small pumice and other volcanic clasts. The unit overlies and inter-fingers with the Rainbow Mountain Debris Flow Sequence at the Eastern Steam-heated Zone and overlies the Donovan Mountain Latite west of YellowJacket. The thickness varies, ranging from 0 to 40 meters, probably filling paleo-topography.

7.2.1.10 GRAVELS OF SOBER UP GULCH – TGS

Gently dipping, ridge-forming terraces of older alluvial deposits are named for extensive exposures in the Sober-up Gulch area just south of the NBP. The unit consists of semi-consolidated, heterolithic boulder gravels of probable late Miocene to Pliocene age (Maldonado and Hausback, 1990). The gravels unconformably overlie older Miocene units throughout the NBH. Tgs is similar to Tdf but contains abundant conspicuous boulders of Donovan Mountain Latite. Tgs typically forms a gently east-dipping (<5°) pediment surface. The pediment surface has been deeply incised by Quaternary erosion, resulting in a series of gently dipping gravel terraces along the eastern side of the NBP. The gravel terraces overlap and conceal steam-heated alteration at the Eastern Steam-heated Zone. The gravel unit is not known to be mineralized but contains clasts of altered and mineralized rock.

7.2.1.11 QUATERNARY COVER – QC

Quaternary Cover includes unconsolidated deposits of alluvium, colluvium, talus and mine dump material.

7.2.2 GEOCHRONOLOGY

Data on NBP geochronology were developed from samples submitted for analysis at various facilities using Zircon dating and AR-AR dating. The Zircon dates are tabulated in Table 7-2.

Table 7-2 - Summary of Zircon Dates from North Bullfrog Hills Volcanic Complex

Map Unit	Age (Ma)	2s	Locality	Description	Lab	Lab ID	Number of Dates
Trt2	10.5	0.1	Mayflower Mine	Rainbow Mountain Tuff	Valencia	115904	27
Trt2	11.4	0.2	Mayflower Mine	Rainbow Mountain Tuff	AtoZ	1335-003	26
Tp	12.7	0.2	Jolly Jane	Paintbrush Tuff	Valencia	115923	20
Tcb	13.3	0.2	Ladd Mountain	Crater Flat Tuff, Bullfrog Member	AtoZ	1335-001	25
Tcb	13.4	0.2	East Savage Valley	Crater Flat Tuff, Bullfrog Member	Valencia	115902	35
Tcb	13.5	0.2	Ladd Mountain	Crater Flat Tuff, Bullfrog Member	Valencia	115906	47
Tcb	13.5	0.2	Ladd Mountain	Crater Flat Tuff, Bullfrog Member	Valencia	115907	29
Tcb	13.5	0.2	Jolly Jane	Crater Flat Tuff, Bullfrog Member	Valencia	115924	34
Tct	13.7	0.2	East Savage Valley	Crater Flat Tuff, Tram Member	Valencia	115901	19
Tct	13.8	0.2	YellowJacket	Crater Flat Tuff, Tram member	Valencia	M610285A	13
Tct	14.3	0.2	YellowJacket	Crater Flat Tuff, Tram Member	AtoZ	1293-06	28

Map Unit	Age (Ma)	2s	Locality	Description	Lab	Lab ID	Number of Dates
Td2	14.4	0.2	Jolly Jane	Savage Valley Dacite	AtoZ	1293-10	21
Td2	15.6	0.4	YellowJacket	Savage Valley Dacite	AtoZ	1293-01	8
Td1	14.2	0.3	Air Track Hill	Savage Valley Felsic Facies	Valencia	115908	33
Td1	14.8	0.2	YellowJacket	Savage Valley Felsic Facies	Valencia	115905	22
Td1	15	0.3	YellowJacket	Savage Valley Felsic Facies	AtoZ	1293-02	28
Trl	14.9	0.2	Sawtooth Mtn	Sawtooth Mtn Tuff, Upper Unit	AtoZ	1335-005	26
Tsb	14.4	0.2	YellowJacket	Sierra Blanca Tuff	Valencia	P346369	30
Tsb	14.5	0.2	East Jolly Jane	Sierra Blanca Tuff	AtoZ	1293-09	27
Tsb	14.7	0.3	Jolly Jane	Sierra Blanca Tuff	Valencia	115925	34
Tsb	15	0.2	YellowJacket	Sierra Blanca Tuff	AtoZ	1293-07	28
Tsb	15.1	0.2	Pioneer	Sierra Blanca Tuff	AtoZ	1335-002	29
Tpf	14.5	0.2	YellowJacket	Pioneer Formation	AtoZ	1293-05	27
Tpf	14.6	0.2	YellowJacket	Pioneer Formation	AtoZ	1293-08	28
Tnb	14.3	0.3	YellowJacket	Rhyolite - spherulitic	Valencia	NB171608	32
Tnb	14.5	0.2	YellowJacket	Rhyolite - flow banded	Valencia	P346202	20
Tnb	14.7	0.2	Radio Tower Hill	Rhyolite - flow banded	Valencia	115903	26
Tnb	14.7	0.2	YellowJacket	Rhyolite - spherulitic	Valencia	P346250	26
Tnb	14.8	0.2	YellowJacket	Rhyolite - flow banded	AtoZ	1293-04	28
Tnb	15.3	0.3	Gold Pit	Dacite Porphyry in Cambrian Basement	Valencia	P347979	28
Tnb	15.8	0.3	Jolly Jane	Pre-Pioneer Formation Dacite	AtoZ	1293-12	24
Tnb	15.9	0.3	Radio Tower Hill	Rhyolite - flow banded	AtoZ	1335-004	34
Tnb	16.1	0.3	Savage Valley	Pre-Pioneer Formation Dacite	AtoZ	1293-11	26
Td	1599.5	20.7	YellowJacket	Basement Xenocrysts in Dacite	AtoZ	1293-03	3

7.2.2.1 ZIRCON DATING

Samples were submitted for analysis to two different laboratories: Apatite to Zircon, Inc. (A to Z) and Victor Valencia. Duplicate samples were included to confirm the analytical precision of the dates (Table 7-3). In general, the match between the labs is reasonable, however, in many instances the A to Z dates are substantially older with differences far exceeding the analytical precision (Table 7-2). The Valencia dates match the published Ar-Ar ages for the Rainbow Mountain Tuff (Trt2), the Paintbrush Tuff and the Bullfrog Tuff. However, the A to Z dates of the Rainbow Mountain (Trt2) and the Lithic Ridge tuffs are almost 1 Ma older than the published ages. Similarly, the A to Z age on the duplicate rhyolite from Sober-up Peak is also approximately 1 Ma older than the Valencia date (Table 7-2). For this reason, it appears that some of the dates from A to Z are too old.

7.2.2.2 AR-AR DATING

Only four of eight samples submitted to the University of Alaska, Fairbanks for Ar-Ar dating of vein adularia returned statistically valid ages (Benowitz and Layer, 2013). The valid age samples came from four different YellowJacket drill holes. These age dates constrain vein mineralization at YellowJacket between 11.7-11.2 Ma (Table 7-3). The new age dates for YellowJacket vein mineralization confirm an earlier 11.3 Ma date published by Connors et al. (1998). Connors et al. (1998) also published adularia dates of 11.0 Ma at the East Savage Vein (Figure 7-2) and ages of 10.0 Ma and 9.9 Ma from the Mayflower Mine. The Mayflower deposit is the same age as the Bullfrog vein deposit in the southern Bullfrog Hills.

One of two alunite samples submitted to the University of Nevada, Las Vegas for Ar-Ar dating returned a valid age-date. Coarse vein alunite collected from the Alunite Hill area of the Eastern Steam-heated Zone returned an age of 9.5 Ma (Table 7-3). This age is similar to the 10.2 Ma age of alunite obtained from the Bailey's Hot Springs area (Weiss, et al., 1994), and similar to the published adularia age dates at Mayflower and Bullfrog. The alunite age dates highlight the potential for the discovery of a new Bullfrog-age, high-grade vein system under the extensive 14 square kilometer Eastern Steam-heated Zone (Figure 7-2). Adularia samples are all from the YellowJacket Zone. The Alunite sample is from the Eastern Steam-Heated Zone north of Alunite Hill.

Table 7-3 - Adularia and Alunite Ar-Ar Age Determinations

Hole ID	Sample	Mineral	Integrated Age (Ma)	Plateau Age (Ma)	Plateau Information	Isochron Age (Ma)	% Atmospheric ^{40}Ar	Lab
NB-12-127	M610395	Adularia	11.2 ± 0.1	11.2 ± 0.1	5 of 7 fractions 98.4% ^{39}Ar release MSWD = 1.09	—	15.8	University of Alaska Fairbanks
NB-12-139	M612038	Adularia	11.6 ± 0.1	11.6 ± 0.2	4 of 7 fractions 99.1% ^{39}Ar release MSWD = 2.43	—	13.5	University of Alaska Fairbanks
NB-12-126b	M610140	Adularia	15.5 ± 2.1	11.7 ± 0.4	4 of 7 fractions 51.3% ^{39}Ar release MSWD = 1.21	—	93	University of Alaska Fairbanks
NB-12-138	M611584	Adularia	11.7 ± 0.4	11.7 ± 0.4	6 of 7 fractions 97.6% ^{39}Ar release MSWD = 0.51	11.4 ± 0.4	42.7	University of Alaska Fairbanks
Alunite Hill	Alun SW107	Alunite	9.73 ± 0.5	9.52± 0.5	10 of 14 fractions 96% ^{39}Ar release	–	–	University of Nevada Las Vegas

7.2.3 REGIONAL CORRELATION

The geochronological studies have significantly refined the stratigraphy of the NBP. Prior to obtaining the zircon age dates, the Sierra Blanca Tuff was correlated with the Bullfrog Tuff of the Crater Flat Group. The zircon dates confirm the age difference between rocks of the NBHVC and the Bullfrog Tuff and indicate that volcanism of the NBHVC extended from around ~15 Ma to 14 Ma (Table 7-2). With age range of 15-14 Ma, the NBHVC correlates with some of the oldest volcanic rocks in the SWNVF, as well as the Tr1 time-stratigraphic position of Eng et.al. (1996). Zircon dating also appears to confirm the proposed correlation between the Sierra Blanca Tuff and the Upper Tuff of Sawtooth Mountain (Table 7-2).

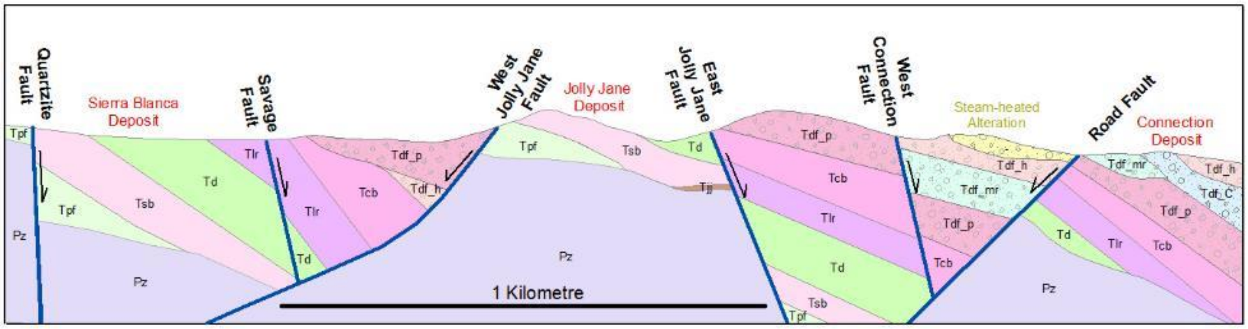
The NBHVC rocks have experienced several significant periods of tectonic reorganization in the Bullfrog Hills extensional domain. Connors, et al. (1998) have identified a major period of extension between 12.7 and 11.6 Ma culminating with the eruption of the Timber Mountain Group ash-flows between 11.6 and 11.45 Ma. These events are coincident with the age of YellowJacket vein mineralization. Connors, et al. (1998) postulate a second major period of extension occurred between 11.4 Ma and 10.5 Ma, which resulted in major block rotation, rapid erosion and the deposition of the Rainbow Mountain Debris Flow Sequence. The 10 Ma Bullfrog vein, the 9.9 Ma Mayflower deposit, and the 9.5 Ma Eastern Steam-heated Zone all coincide with the waning stages of extensional tectonism in the Bullfrog Hills. Extension appears to have ceased by 9.4 Ma (Connors, et al., 1998).

7.2.4 STRUCTURE

Both pre-Tertiary and Miocene rocks have been subjected to large-scale, W- to WNW-directed, syn-volcanic and syn-mineral extension between ~15-9.4 Ma. Such extension is largely exhibited by down-to-the-west normal faulting and east-tilting of stratigraphy. The extensional tectonism was apparently comprised of two periods of intense faulting and tilting (i.e. accelerated extension), within a protracted period of less intense but episodic extension. Extensional faulting events were accompanied by episodic hydrothermal alteration and gold mineralization throughout the Bullfrog district.

Extension at NBP was accommodated by the Bullfrog Hills Fault System (BHFS). The BHFS at NBP consist of two basic fault types: 1) large-displacement NNW- to NNE-trending, moderately to steeply west-dipping, down-to-the-west normal faults; and 2) generally smaller displacement, steeply east-dipping, down-to-the-east, antithetic faults (Figure 7-2 and 7-4). The down-to-the-west faults are interpreted to have listric shapes at depth, similar to the MP fault at the Bullfrog mine. These faults are interpreted to sole into the Southern Bullfrog Hills Fault (detachment) at unknown depth under the NBP. Steeply east-dipping, down-to-the-east faults commonly occur in the hanging walls of the large displacement down-to-the-west faults and are interpreted to be truncated at depth by the down-to-the-west faults (Figure 7-4). Section location for Figure 7-4 is shown on Figure 7-2.

Figure 7-4 - Geologic Cross Section from Savage Valley to Connection Illustrating the Overall Structural Style of the NBP (looking North)



Four major splays of the BHFS cross the NBP including: the Donovan Mountain Fault, the West Jolly Jane Fault, the Road Fault and the Spicerite Fault (Figure 7-2). These faults have apparent down-to-the-west, dip-slip displacements of ~600 to >1000 meters (Figure 7-4). The Road Fault is considered the northern continuation of the Contact Fault from the Bullfrog mine area in the southern Bullfrog Hills.

Some of the oldest faults at the NBP are represented by the down-to-the-east Liberator, East Jolly Jane and Quartzite Faults (Figure 7-4, 7-6, 7-7 and 7-8). Evidence for an older age on these structures is the fact that the Bullfrog Tuff is only preserved on the downthrown side of these faults. It seems likely that down-to-the-east faults developed before or during the 12.7-11.6 Ma deformation event just after the deposition of the Bullfrog Tuff. At Jolly Jane, the Bullfrog Tuff is strongly quartz-adularia altered in the hanging-wall of the East Jolly Jane Fault. The evidence suggests that the major down-to-the-east faults were present during the older disseminated mineralization event at the NBP (see discussion in Section 7.2.5.1) and played a role in the process of mineralization. The Liberator and East Jolly Jane Faults appear to truncate the eastern sides of the Sierra Blanca-YellowJacket and Jolly Jane deposits respectively (Figure 7-4, 7-6, 7-8), suggesting they were also active after the older mineralization event. There are many faults at the NBP with relatively minor displacements, the ages of which are difficult to constrain.

7.2.5 MINERALIZING EVENTS

All of the mineralizing events known to date at the NBP can be classified as low-sulphidation epithermal mineralization. Two general styles of mineralization are present at NBP: 1) pervasive alteration-style disseminated mineralization; and 2) structurally controlled vein and stockwork mineralization. There are at least three distinct periods of mineralization present at the NBP:

- pre-11.7 Ma pervasive alteration-style disseminated (Sierra Blanca)
- 11.7-11.2 Ma structurally controlled alteration-style enrichment and late vein (YellowJacket)
- ~10 Ma structurally controlled alteration-style disseminated and late vein (Mayflower).

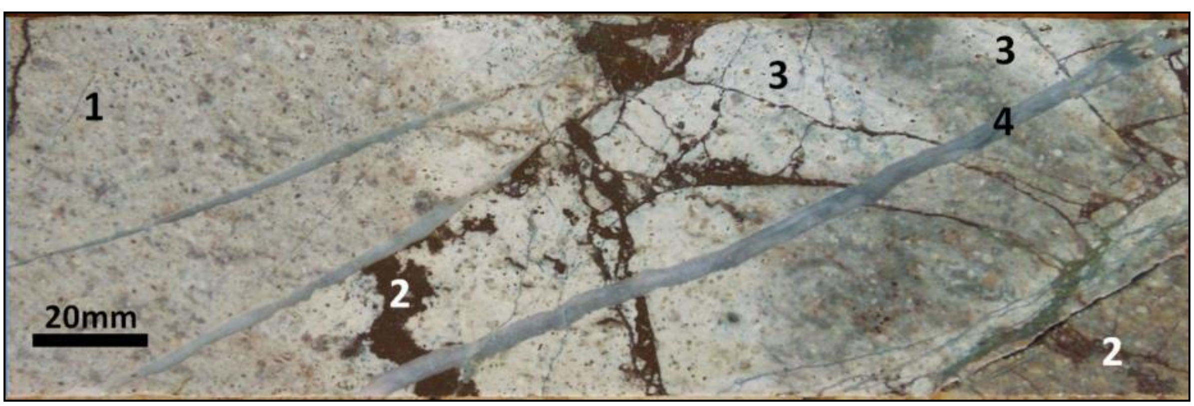
Pre-11.7 Ma pervasive disseminated mineralization is hosted in ~15-14 Ma rocks of the NBHVC. The presence of pervasive quartz-adularia alteration in the relatively barren Bullfrog Tuff east of Jolly Jane suggests the 13.25 Ma Bullfrog Tuff was affected by this early quartz-adularia mineralization, bracketing the earliest event between 13.25 and 11.7 Ma. A barren jasper (quartz-hematite) vein event overprints the pervasive quartz-adularia mineralization at Sierra Blanca-YellowJacket. The jasper vein event was subsequently overprinted by higher grade, structurally controlled, alteration-style enrichment mineralization; which was closely followed by (or contemporaneous with) the YellowJacket vein and stockwork mineralization. Based on overprinting relationships observed in core (Figure 7-5), it is apparent that multiple events have contributed to the gold endowment that had accumulated by 11.2 Ma. Pre-11.2 Ma mineralization will be discussed below as "Older" mineralization and the 10 Ma mineralization at Mayflower will be discussed as "Younger" mineralization.

7.2.5.1 OLDER MINERALIZATION STYLES

7.2.5.1.1 OLDER PERVASIVE ALTERATIONS-STYLE MINERALIZATION

The most widespread mineralization at the NBP is associated with pervasive quartz-adularia alteration and pyritization of iron minerals in the volcanic host rocks. The grade of this mineralization often reflects the intensity of quartz-adularia or illite-adularia alteration, as well as the original iron content of the host rocks. Gold grade in the Pioneer Formation and Sierra Blanca Tuff, which average 1% iron, is on the order of 200-400 ppb gold. In contrast, grades in the Savage Valley Dacite containing 2-5% iron may reach several thousand ppb gold. Pervasive alteration associated with disseminated pyrite mineralization generally shows a progressive change from illite-smectite, to illite-adularia, and to quartz-adularia+illite as the degree of mineralization increases. The silver to gold ratio of the alteration-style mineralization is approximately 1:1.

Figure 7-5 - Cross-cutting Relationships Between Older Mineralization Types at Sierra Blanca



In Figure 7-5 (above) the cross-cutting relationships are as follows: 1) Early pervasive quartz-adularia alteration with gold related to sulphidation of iron; 2) Barren jasper veins filling brittle fractures in adularized tuff; 3) White illite-adularia overprint with an increase in gold; and 4) Grey translucent stockwork veinlets with high-grade gold.

The distribution of pervasive alteration-style mineralization appears to reflect a combination of rock matrix permeability and structural permeability. Although both the Sierra Blanca Tuff and the Pioneer Formation are pervasively altered, it appears that the brittle nature of the densely welded Sierra Blanca Tuff enhanced the permeability relative to the less welded tuffs of the Pioneer Formation. In contrast, alteration-style mineralization in the underlying Savage Formation and the overlying Savage Valley Dacite appears to be controlled largely by fault-related permeability.

Pervasive alteration-style mineralization has been observed locally in the Paleozoic limestone and calcareous sediments. Jasperoid is developed in the Carrara Formation limestone just below the Tertiary unconformity in many drill holes at both Jolly Jane and Savage Valley. The jasperoid is typically anomalous in gold, and locally yields 200-400 ppb values. The jasperoid target area (Figure 7-2) consists of a bedding-parallel replacement jasperoid body in calcareous Wood Canyon Formation sediments. Paleozoic rocks do not appear to be well-mineralized at NBP but cannot be ruled-out as potential host rocks for future exploration.

7.2.5.1.2 OLDER STRUCTURALLY CONTROLLED MINERALIZATION

Structurally controlled mineralization consists of two distinct styles which may represent two periods of mineralization. The first is a structurally controlled alteration-style mineralization and the second is quartz vein-style mineralization. The onset of structurally controlled mineralization is marked by the formation of a distinctive suite of essentially barren jasper (quartz-hematite) veins. The jasper veins crosscut the older pervasive quartz-adularia disseminated mineralization (Figure 7-5). After the formation of the jasper veins, movement on major fault structures (e.g. Liberator, YellowJacket, NE20, NE30, NW10 Faults) apparently resulted in a second stage of more structurally controlled sulphidation and gold enrichment. This structurally controlled mineralization can be distinguished from the older earlier alteration-style event by a higher As/Au ratio and is generally associated with a white to light brown illite or illite-adularia alteration overprint (Figure 7-5). Structurally controlled alteration-style mineralization frequently yields grades >1 g/t, and sometimes >10 g/t if the fluids encounter the higher iron contents of dacitic lithologies. Structurally controlled alteration-style mineralization is clearly crosscut by high-grade, low sulphidation quartz veins of the YellowJacket vein event.

The YellowJacket Vein Zone consists of a massive quartz vein surrounded by hanging wall and footwall quartz stockwork zones. Such quartz vein and stockwork mineralization is found at YellowJacket and along the crest of North Sierra Blanca ridge. Observed textures that are typical of low sulphidation epithermal veins include bladed quartz pseudomorphs after calcite, crustiform banding and milky chalcedonic quartz with distinct but fuzzy banding. Veins with these textures may be relatively barren or have high-grade gold. The most common and best mineralized veins at YellowJacket are grey translucent quartz stockwork veins with little distinctive internal structure (Figure 7-5). Grains of native gold can often be observed in this quartz. There is generally little wall rock alteration associated with grey translucent quartz veins. However, white illite overprint of earlier quartz-adularia is often observed in the general vicinity of grey translucent stockworks. The illite overprint can locally be rather intense creating selvages around jasper veins and destroying all the feldspar in the rock (Figure 7-5).

The primary minerals associated with the vein-style mineralization are gold, electrum, acanthite (Ag2S) and pyrite. Petrographic studies have also documented pyrargyrite (Ag3SbS3), stromeyerite (AgCuS), proustite (Ag3AsS3), chalcopyrite (CuFeS2) and covellite (CuS). Sphalerite has been observed as a late cavity infill. In general, the silver to gold ratio associated with vein mineralization is >6:1 and locally can be 100:1 or more.

7.2.5.2 YOUNGER MINERALIZATION STYLES

A number of deposits in the NBP area formed after the deposition of the 10.5 Ma Rainbow Mountain Tuff (Trt2). These include disseminated and structurally controlled deposits at Mayflower, Connection and Cat Hill. Younger mineralization exhibits similar styles to older mineralization, including alteration-style, disseminated quartz-adularia and quartz-calcite vein mineralization. Steam-heated alteration along the Road Fault and the Eastern Steam-heated Zone are also related to this younger period of mineralization.

7.2.5.2.1 YOUNGER ALTERATION-STYLE MINERALIZATION

At Mayflower, the debris flow sediments and Trt2 tuff have been affected by quartz-adularia alteration developed around a central zone of faulting. The quartz-adularia alteration is directly associated with disseminated gold mineralization. Although the deposit is completely oxidized at present, it appears that the original mineralizing process was sulphidation during the quartz-adularia event. The process is similar to, but significantly younger than the quartz-adularia-pyrite mineralization at Sierra Blanca and Jolly Jane. The distribution of quartz-adularia alteration appears to reflect a combination of structural and stratigraphic permeability. The brittle nature of the quartz-adularia alteration enabled fracturing and development of open space during later faulting events associated with vein mineralization.

7.2.5.2.2 YOUNGER STRUCTURALLY CONTROLLED MINERALIZATION

At Mayflower, quartz veining is only seen in the southeastern part of the deposit on the dumps at the Mayflower Shaft. The quartz can be white, pink and grey translucent, and contains visible gold. It is frequently banded and may have very fine acicular textures indicating replacement of earlier adularia. Bladed pseudomorphs of quartz after calcite have not been observed at Mayflower. Some of the gold at Mayflower is associated with grey manganiferous calcite veining. Observations from both macroscopic and microscopic studies at Mayflower have shown visible gold in the calcite bands rather than in the quartz. Gold grains are often found at the calcite-wallrock contact. In the David Adit, the only cavity infill phase is coarse euhedral manganiferous calcite. Calcite veining is not always mineralized, but historical sampling combined with underground observations shows that the highest-grade areas have calcite cavity linings. The gold-calcite association at Mayflower is very different from YellowJacket, where calcite is generally not associated with high-grade mineralization. Another difference is the Ag:Au ratio in the Mayflower vein zones; it is generally <0.5:1, whereas at YellowJacket it is generally >6:1.

7.2.5.2.3 STEAM-HEATED ALTERATION

Steam-heated alteration is characterized by low-temperature silica replacement, pervasive kaolinite-alunite alteration and late alunite veining. It is the result of wall rocks reacting with steam being generated by hot acidic waters at depth. In modern geothermal systems, steam-heated alteration develops above a groundwater table, which in turn lies above a boiling zone at depth where gold is being deposited. Steam-heated alteration has been identified over an extensive 14 square kilometer area at the "Eastern Steam-heated Zone" (Figure 7-4). Steam-heated alteration also occurs in several places along the Road Fault, particularly in the vicinity of the Connection and Cat Hill target areas. Anomalous gold has been detected in rocks affected by steam-heated alteration at Alunite Hill, Vinegaroon, Cat Hill and Yellow Rose (Figure 7-2). There is a reasonably high probability that this alteration may be associated with a mineralized Bullfrog-Mayflower age (~10 Ma) vein system at depth. Based on drilling in 2015 at the Spicerite target in the Eastern Steam-heated Zone, the target depth is >300 meters below the current topographic surface.

7.3 MINERAL RESOURCE AREAS

Corvus and previous operators exploring in the northern Bullfrog Hills have defined targets in areas of historic mines or prospects, as well as targets associated with high level epithermal alteration. Mineral Resources have been identified in several areas discussed in detail in Section 7 including: Jolly Jane, Sierra Blanca, YellowJacket, Air Track Hill, Air Track West, and Mayflower (Figure 7-2). In 2007-2008, ITH/Redstar (NBPJV) drilled several holes at Air Track Hill and Mayflower, with two holes each at Sierra Blanca, Pioneer and Savage Valley. Between 2010 and 2017, Corvus has drilled 467 core and RC holes at Sierra Blanca, YellowJacket, Jolly Jane, Mayflower and Connection, leading to the Mineral Resource estimates.

7.3.1 SIERRA BLANCA

The greater Sierra Blanca area includes the Savage Valley, Sierra Blanca, North Sierra Blanca, YellowJacket, Air Track Hill and Air Track West areas (Figure 7-6). In 2010-11, Corvus drilled 44 reverse circulation (RC) holes totaling 12,785 meters (41,945 feet) in the Sierra Blanca area. In 2012, Corvus drilled 16 additional holes totaling 3,548 meters (11,640 feet) including: 4 PQ3 holes for metallurgical samples, 6 HQ3 exploration holes and 6 step-out/infill RC holes.

In 2013, Corvus drilled 87 additional holes totaling 19,000 meters (62,340 feet) including: 35 HQ3 core holes, two PQ3 core holes, and 50 RC holes. Fifteen channel sample lines totaling 1,070 meters (3,510 feet) were completed along new road cuts in 2013-14. In 2014, Corvus drilled 48 additional core holes totaling 11,000 meters (36,100 feet) including: 36 HQ3 core holes and 12 PQ3 holes. Between 2015 and 2019, Corvus drilled 100 additional holes (6 core and 94 RC) totaling 26,439 meters (88,719 feet) in the greater Sierra Blanca area. The Sierra Blanca Mineral Resource is based on this drilling.

7.3.1.1 SIERRA BLANCA STRATIGRAPHY

The stratigraphy of the greater Sierra Blanca area includes the following major units in ascending stratigraphic order: 1) Paleozoic basement rocks including the Wood Canyon Formation, Zabriskie Quartzite and Carrara Formation; 2) the Jolly Jane Formation; 3) the Savage Formation; 4) the Pioneer Formation; 5) the Sierra Blanca Tuff; 6) the Savage Valley Dacite; 7) rhyolites of the North Bullfrog Intrusive Suite; 8) the Crater Flat Group; 9) monolithic and heterolithic debris flow breccias of the Rainbow Mountain Sequence; 10) the Trt2 tuff of the Rainbow Mountain Sequence; and 11) the Sober Up Gulch Gravels- (Table 7-1). Figure 7-6 is a geologic map of the Sierra Blanca area updated in 2016.

The Wood Canyon Formation (PzW) and Zabriskie Quartzite (PzZ) crop out along the southwest side of Savage Valley, and both units were penetrated in drill holes under Savage Valley (Figure 7-6). The Wood Canyon Formation consists of calcareous shale, siltstone and sandstone with occasional beds of massive to finely laminated limestone. The Zabriskie consists of light brown, pink or light grey, non-calcareous to weakly calcareous vitreous quartzite. The Carrara Formation (PzC) overlies the Zabriskie and consists of primarily carbonaceous calcareous shale, argillaceous limestone, micritic limestone and sandy limestone. PzC is the primary bedrock unit penetrated under Savage Valley.

The Jolly Jane Formation at Sierra Blanca is only known from drilling under Savage Valley. It includes a heterogeneous sequence of 1) Paleozoic clast-dominated conglomerate; 2) calcareous and carbonaceous lithic-volcanoclastic sediments that appear to be largely re-worked dacite; and 3) rare monolithic debris flow breccias of the Carrara Formation. The Jolly Jane Formation varies dramatically in thickness from 0-35 meters between drill holes and is generally thinner at Sierra Blanca than at Jolly Jane. Overlying the Jolly Jane Formation, the Savage Formation consists of dacitic tuffs, flows and intercalated dacitic sediments. The lava flows thicken to the south where they appear connected with the Savage Plug: a rhyo-dacite porphyry plug-dome that is part of the North Bullfrog Intrusive Suite.

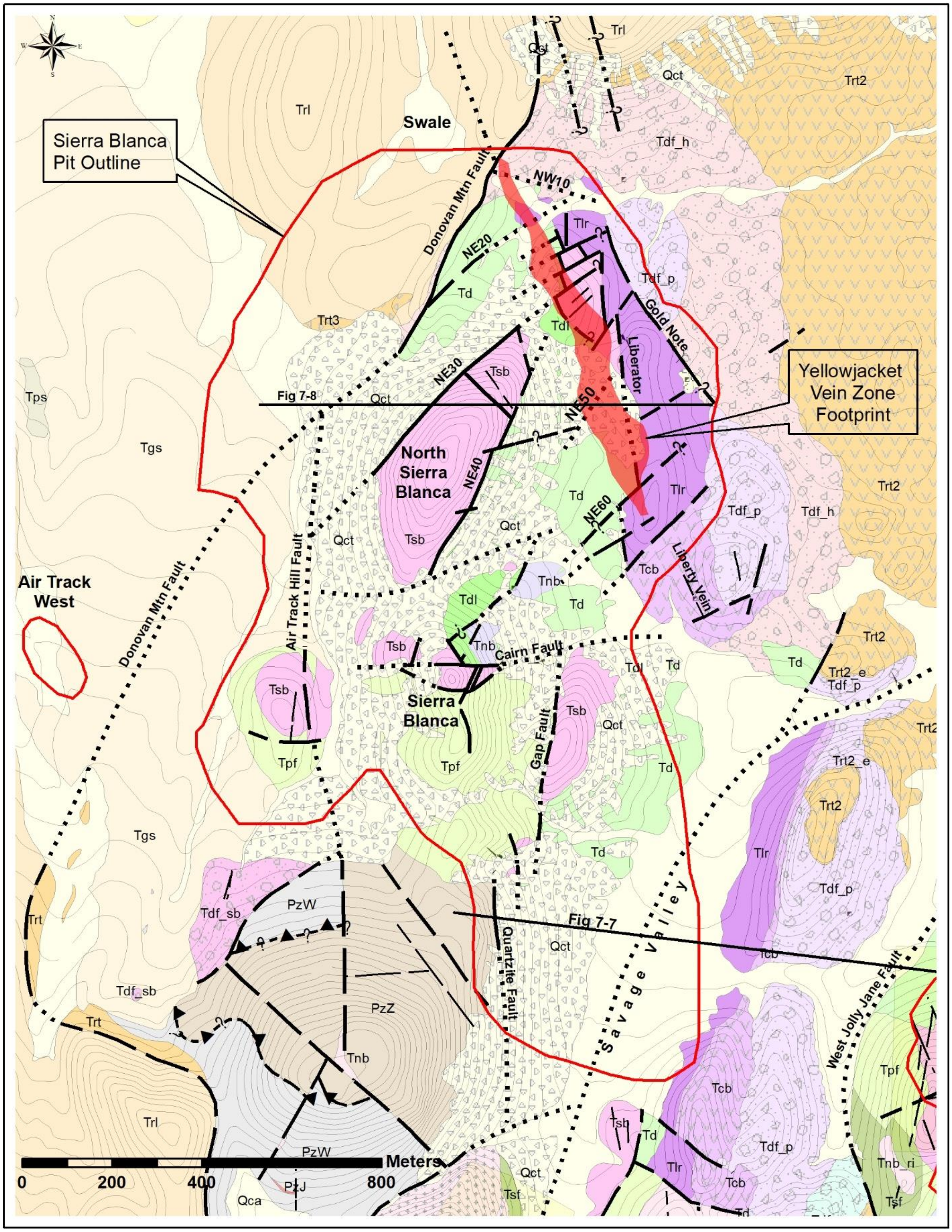
The Pioneer Formation consists of felsic pyroclastic rocks including pale green crystal-lithic tuff and lithic lapilli tuff with common white rhyolite clasts. Near the upper contact, the lithic content increases significantly and grades into a coarse heterogeneous tuffaceous epiclastic sequence known as the Upper Epiclastic Member. The Pioneer Formation ranges in thickness from tens of meters in the south to over 250 meters in the north. The base of the unit has never been drilled in the North Sierra Blanca area. The Pioneer Formation exhibits varying degrees of quartz-adularia alteration and hosts significant gold mineralization.

In the northern and central Sierra Blanca area, rhyolite bodies of the North Bullfrog Intrusive Suite occur stratigraphically below, within and above the Pioneer Formation. The rhyolites exhibit flow-banding, spherulitic textures, and are often highly brecciated. Rhyolite breccias may be intrusive or locally extrusive. Based on zircon age dating and geochemical similarities, Tnb rhyolite volcanism is in part coeval with deposition of the Pioneer Formation. Rhyolite flows are also present within the Savage Valley Dacite stratigraphy above the Sierra Blanca Tuff. The various rhyolite bodies appear to represent episodic intrusive and extrusive phases of rhyolite volcanism contemporaneous with the Pioneer Formation, Sierra Blanca Tuff and Savage Valley Dacite.

The Sierra Blanca Tuff varies in thickness from >160 meters in the north to 30 meters in the south. A 5-15-meter-thick distinctive welded shard-rich interval (Tsb1) at the base of the Sierra Blanca Tuff serves as a marker for the bottom of the unit. Above the shard-rich marker, the tuff is a relatively homogeneous, densely welded, crystal tuff with well-developed compaction foliation exhibited by flattened pumice fragments. The tuff is typically fractured and brecciated and is the primary host rock for gold mineralization.

The Sierra Blanca Tuff is unconformably overlain by the Savage Valley Dacite. The Lower Member is a heterogeneous pyroclastic and epiclastic sequence of dacitic composition. The Lower Member contains a discontinuous rhyolite flow breccia that is assigned to the North Bullfrog Intrusive Suite. The Upper Member is a more uniform sequence of dacitic to andesitic lava flows and breccias. There are dramatic lateral changes in the unit that make it difficult to correlate internal stratigraphy between drill holes. Structurally controlled alteration-style mineralization is common in the Savage Valley Dacite.

Figure 7-6 - Geologic Map of the Greater Sierra Blanca-YellowJacket Area



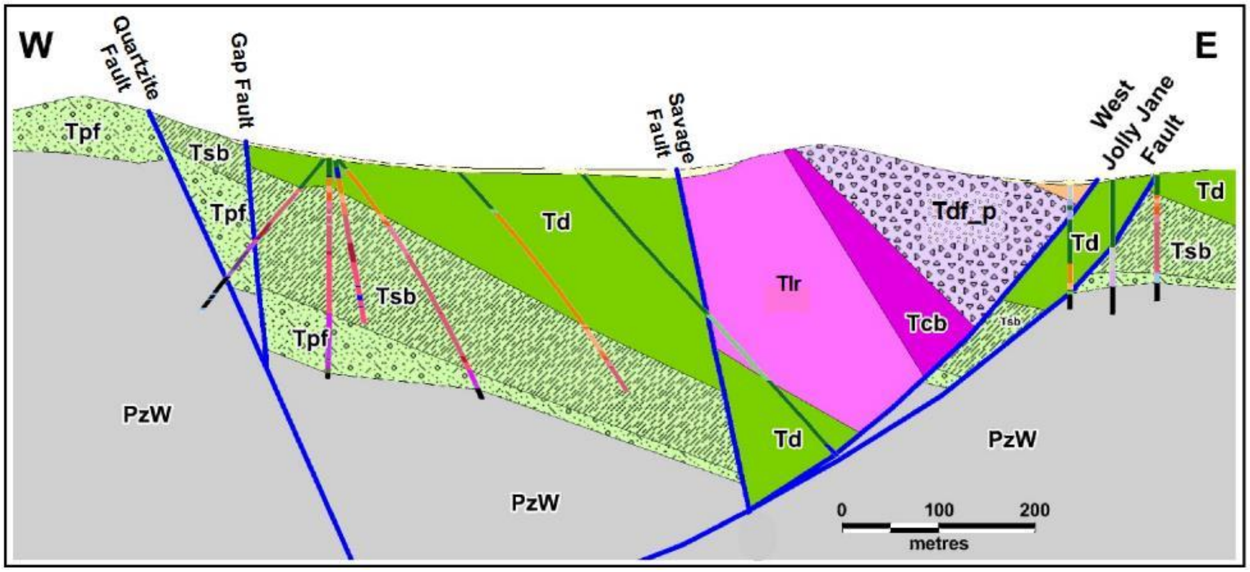
The Lithic Ridge Tuff overlies and inter-fingers with the Savage Valley Dacite. The Lithic Ridge Tuff is a poorly to moderately welded crystal-lithic ash-flow tuff, which typically exhibits abundant chloritized biotite phenocrysts. Lithic clast content varies from 5-20+% lithic clasts. The most common lithic clasts are from the underlying Savage Valley Dacite. The Lithic Ridge is highly altered, locally mineralized near the Liberator Fault, but not a significant volume host to gold mineralization.

The Bullfrog Tuff of the Crater Flat Group is locally exposed in the eastern side of the Sierra Blanca resource area where it overlies and is in fault contact with the Lithic Ridge Tuff (Figure 7-6). There is significant alteration in the Bullfrog Tuff along the Liberty Vein structure. No gold mineralization has been encountered to date in the Bullfrog Tuff at Sierra Blanca.

A significant interval of monolithic debris flow breccia of Paintbrush Tuff unconformably overlies the Bullfrog Tuff east of YellowJacket and Savage Valley. The Paintbrush breccia is not known to be mineralized at Sierra Blanca, but locally exhibits strong hydrothermal alteration and high-level quartz veining, which may be related to the waning YellowJacket vein event. Heterolithic debris flow breccias overlie and locally inter-finger with the Paintbrush breccias. East of YellowJacket and Savage Valley, monolithic and heterolithic debris flow breccias are unconformably overlain by the crystal-lithic Trt2 tuff of the Rainbow Mountain Sequence (Figure 7--6). The base of the Trt2 tuff is marked by bedded epiclastic rocks that locally fill a significant erosional channel cut through the Paintbrush breccia into the Bullfrog Tuff. The 10.5 Ma Trt2 tuff is relatively unaltered and appears to have been deposited after the last known mineralization event affecting the Sierra Blanca area.

The last unit of importance at Sierra Blanca is the Gravels of Sober-Up Gulch. These younger gravels fill the valley to the west of Air Track Hill, directly overlying the Donovan Mountain Latite. The gravel sequence in this area is only known from reverse circulation drilling. The mineralization at Air Track West appears to be a large slide block of quartz-adularia-altered rhyolite (Tnb) hosted within the gravel unit.

Figure 7-7 - Geologic Cross Section Looking North Through Savage Valley Illustrating the Style of Faulting



The location of Figure 7-7 shown in Figure 7-6.

7.3.1.2 SIERRA BLANCA STRUCTURE

The structural setting of the Sierra Blanca area is similar to Jolly Jane, but the area is at least four times larger. The relative timing of the complex structural setting at Sierra Blanca is not completely understood. The Sierra Blanca area is divided in two blocks by the Cairn Fault (Figure 7-6), which separates the Savage Valley block from the North Sierra Blanca block. The Cairn Fault appears to consist of multiple E-W-trending, north-dipping, en echelon splays, comprising a down-to-the-north zone of displacement exhibiting both pre- and post-mineral movement. The Cairn Fault is interpreted to be an accommodation structure similar to the Pioneer Shear.

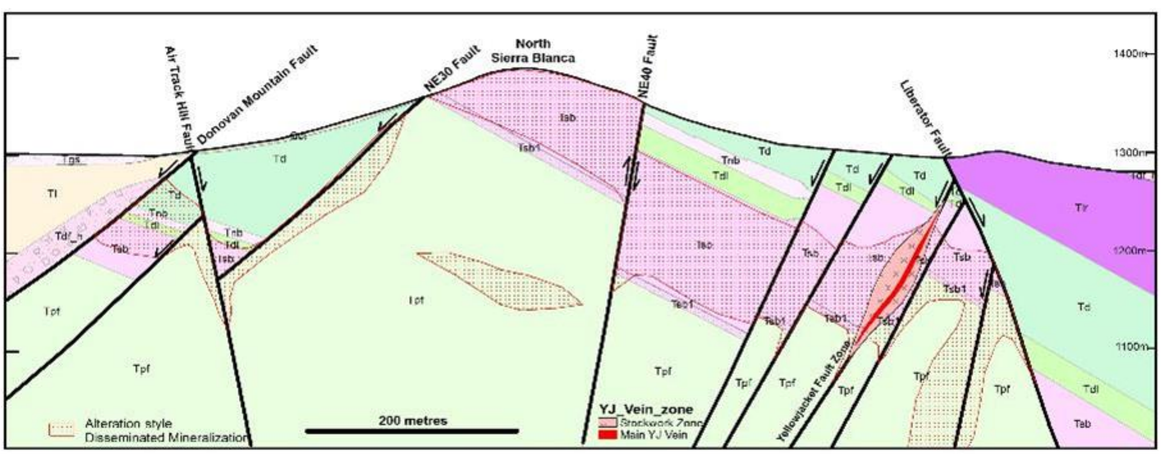
The structure of the Savage Valley block is relatively simple with a series of down-to-the-east faults on the western side of the valley (Quartzite and Gap Faults) and the Savage Fault on the eastern side (Figures 7-6 and 7-7). The stratigraphy dips steeply to the east between the Savage and West Jolly Jane Faults, reflecting dramatic rotation in the hanging-wall of the West Jolly Jane Fault.

The structure in the North Sierra Blanca block north of the Cairn Fault is more complex, with an array of faults that all dip to the west but have mixed apparent normal and reverse displacements (Figure 7-8). The dip of units in this block is quite variable but is generally 20-45° to the east-southeast. Significant down-to-the-west faults at Sierra Blanca include the Donovan Mountain, NE20, NE30 and YellowJacket Faults. The largest fault is the Donovan Mountain Fault (Figure 7-6 and 7-8), which exhibits >1 kilometer of down-to-the-west displacement. Much of this displacement appears to be post-mineral, placing unaltered Donovan Mountain Latite against mineralized Pioneer Formation. The Air Track Hill, NE30, NE20 and NW10 faults all appear to be truncated by post-mineral movement of the Donovan Mountain Fault (Figure 7-6 and 7-8). The Gravels of Sober-up Gulch west of Sierra Blanca overlap and cover the Donovan Mountain Fault.

Significant down-to-the-east faults include the Air Track Hill, NE40, Liberator, Gap and Quartzite faults. The largest down-to-the-east fault is the Liberator. The Liberator locally hosts both vein and alteration-style mineralization, and also exhibits significant post-mineral movement. Some down-to-the-west faults are apparently truncated on the east by the Liberator Fault, including the YellowJacket fault and vein zone (Figure 7-8). The Liberator and YellowJacket faults are also cut by NE-trending cross faults (NE50, NE60 and other smaller faults). The NE-trending cross faults appear to have the youngest movement in the area. The Liberator Fault is apparently truncated on the north by the NW10, and on the south by the Cairn Fault. The NW10 fault is a WNW-trending, down-to-the-north fault at the north end of the YellowJacket Vein Zone. The NW10 fault is mineralized but also interpreted to have post-mineral movement.

The fault pattern at Sierra Blanca consists of generally smaller-scale faulting that has affected the blocks between the larger faults (Figure 7-6 through 7-9). With the onset of accelerated extension during the 11.4-10.5 Ma event, any older faults riding in hanging-walls would be reactivated and rotated to the east. This means that early east-dipping faults have rotated and steepened, and early west-dipping faults would have rotated and flattened. The NE40 fault at Sierra Blanca currently exhibits apparent reverse displacement, probably as a result of eastward rotation of what was a formerly east-dipping normal fault (Figure 7-8).

Figure 7-8 - Simplified Geologic Cross Section Looking North through Sierra Blanca-YellowJacket



The location of the section shown in Figure 7-8 is shown on Figure 7-6.

7.3.1.3 SIERRA BLANCA MINERALIZATION

Mineralization at Sierra Blanca can be classified into the following styles:

- Early, large-volume, pervasive low-grade disseminated gold mineralization associated with quartz-adularia-pyrite alteration.
- Middle, structurally controlled disseminated gold mineralization associated with overprinting illite-adularia-pyrite alteration and gold enrichment (NW10, NE30, NE50, NE60, Liberator Faults).
- Late, high-grade gold associated with quartz veins and quartz stockwork veining (YellowJacket Vein Zone).

The metallurgical characteristics of these styles of mineralization have resulted in the differentiation of three metallurgical classes for the estimate of the Sierra Blanca Mineral Resource: 1) disseminated oxide; 2) disseminated sulphide; and 3) quartz vein and stockwork mineralization.

7.3.1.3.1 PERVASIVE DISSEMINATED MINERALIZATION

The Sierra Blanca Tuff and Pioneer Formation exhibit alteration-style disseminated mineralization over virtually the entire Sierra Blanca area. The Sierra Blanca Tuff is ubiquitously altered to a fine-grained mixture of quartz and adularia with disseminated pyrite. Quartz-adularia alteration is not as widespread or as well developed in the underlying Pioneer Formation, which is more typically altered to a light green smectite-illite+adularia-chlorite assemblage. The tuffs are progressively altered to smectite-illite, illite-adularia and finally to quartz-adularia as the alteration intensity and gold grade increases. The specific controls on alteration development are difficult to constrain with wide-spaced drilling. However, it appears that alteration is controlled by a combination of high-angle structures feeding fluids into permeable stratigraphic intervals.

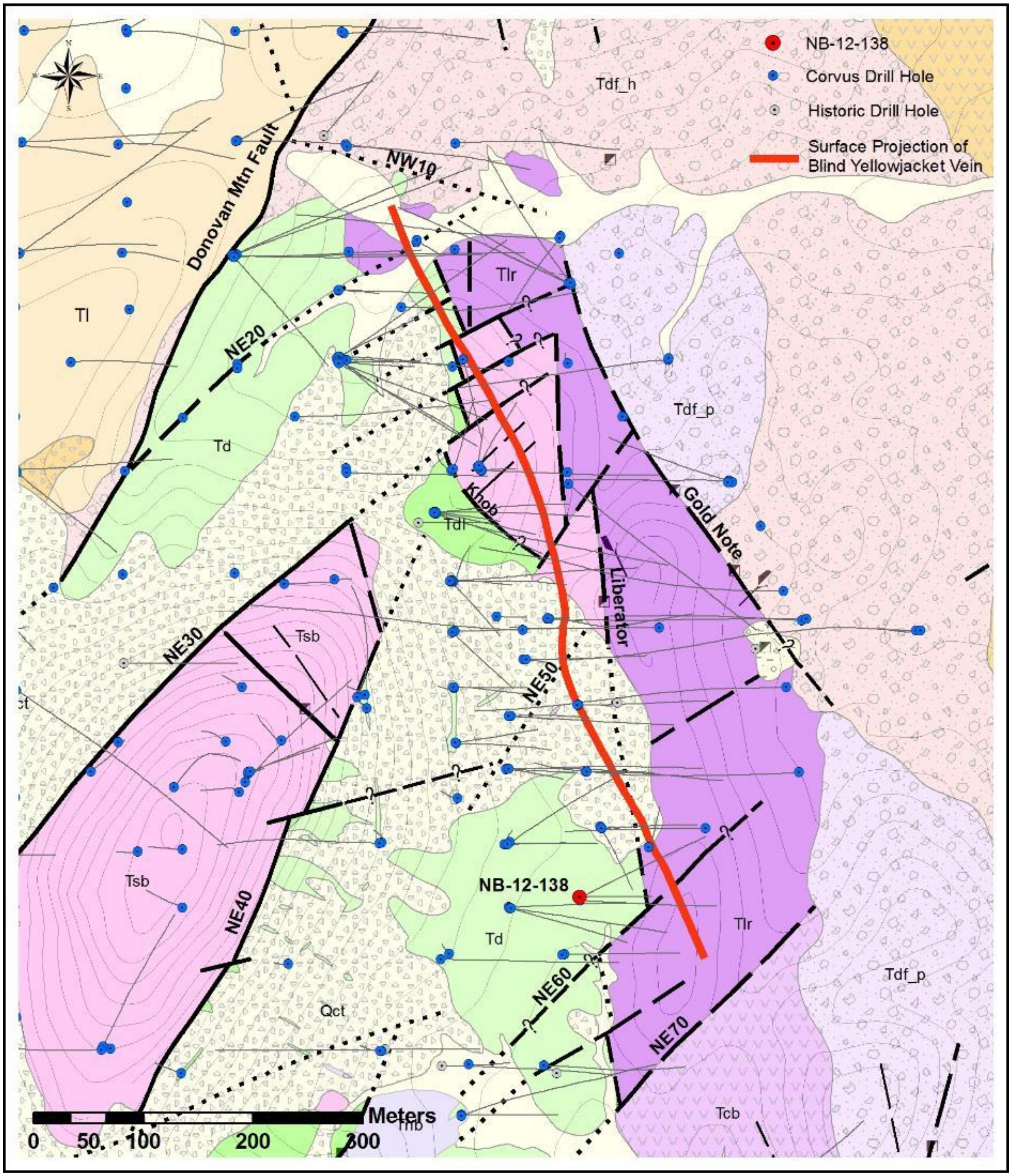
Disseminated mineralization in the un-oxidized portions of the deposit is directly related to the pyrite content, which in turn reflects the original iron content of the rock. Pyrite morphologies in disseminated mineralization include fine to coarse disseminated, biotite-replacement, lithic clast-replacement and veinlet infill; indicating a complex history with multiple generations of pyrite growth. Some pyrite grains show zoned gold and arsenic, but no consistent pattern was observed in the grains studied (AMTEL Report 11/34, August 3, 2011). A gold deportment study carried out on the disseminated mineralization revealed that most of the gold is held in the lattice of the disseminated pyrite. When the pyrite is oxidized, the gold is readily recoverable with simple cyanide (AMTEL Report 11/34, also see Section 13).

7.3.1.3.2 STRUCTURALLY CONTROLLED DISSEMINATED MINERALIZATION

Structurally controlled alteration-style mineralization is characterized by the development of disseminated pyrite within faults and in proximal wallrock directly adjacent to faults. These zones may be over 10 meters wide locally. Many faults host this style of mineralization including the NE20, NE30, NE40, NE50, NE60, NW10, Air Track Hill, and the Liberator Faults. Structurally controlled alteration-style mineralization overprints the earlier pervasive quartz-adularia disseminated mineralization, and results in consistently higher grades (1-17 g/t gold). In the Sierra Blanca Tuff, the illite overprint results in a bleached white illite-adularia-pyrite assemblage (Figure 7-5). Where developed in the Savage Valley Dacite, the alteration is a brown illite-pyrite to illite-adularia-pyrite assemblage, which commonly contains higher grades. This is particularly notable along the Liberator Fault, where gold grades of up to 17 g/t have been encountered in the dacite. This mineralization has a consistently higher Ag/Au ratio than earlier disseminated mineralization.

Disseminated mineralization has been modeled as oxide disseminated and sulphide disseminated. Oxide disseminated is amenable to heap leaching. Sulphide disseminated is amenable to flotation concentration and either ambient air oxidation or pressure oxidation recovery of gold from a sulphide concentrate.

Figure 7-9 - Geologic Map of the YellowJacket Area Showing Major Faults and Drill Holes Related to Discovery of the High-grade Vein System



7.3.1.3.3 QUARTZ VEIN MINERALIZATION

The YellowJacket Vein Zone occurs in the northeast portion of the Sierra Blanca resource area (Figure 7-6 and 7-9). The YellowJacket Vein Zone consists of the massive YellowJacket Vein surrounded by hanging wall and footwall stockwork vein zones. The YellowJacket Vein Zone was discovered with drill hole NB-12-138 and was systematically drilled-out with core in 2013-14. The YellowJacket Vein Zone strikes north-northwest and dips between 65-75° west. The zone varies between 15-35 meters wide and persists over a strike length of ~850 meters. The main YellowJacket Vein is continuous in drill holes along ~700 meters of strike length. The vein zone is entirely blind and not recognizable at the surface. The surface projection of the YellowJacket Vein Zone is shown on Figure 7-9. The continuity of the vein and stockwork zone along strike is remarkably consistent. Quartz vein and stockwork mineralization appears to overprint all earlier alteration-style mineralization.

High-grade quartz veins often exhibit crustiform banding and bladed quartz pseudomorphs after calcite typical of low-sulphidation epithermal veins. The quartz vein mineralogy is very simple and consists of native gold and electrum with varying amounts of acanthite and accessory silver sulphosalts. The high grades of the quartz stockwork zones are typically carried by <1-2 cm grey translucent quartz veinlets containing visible gold and trace amounts of pyrite. Metallurgical testing has shown that the massive quartz vein and quartz stockwork mineralization is free milling. Other YellowJacket-style veins and stockwork zones have been penetrated by drilling in the North Sierra Blanca area outside of the YellowJacket Vein Zone. These are generally small-volume veins or stockwork zones that locally carry high-grades. Most of these subsidiary vein zones appear to be controlled by NE-trending cross faults (i.e. NE50 and the Rhyolite Vein in the vicinity of the NE20). The NE faults are kinematically linked to the YellowJacket Vein structure and served as vein fluid conduits. Potential exists to expand NE-trending or other subsidiary vein zones with additional drilling. The better grade-thickness intercepts are likely associated with intersections between the YellowJacket Fault and NE-trending cross faults.

7.3.2 JOLLY JANE

The pseudo-stratabound nature of disseminated mineralization within the Sierra Blanca Tuff at Jolly Jane was recognized by Barrick Gold in 1995 but was not of sufficient grade to be pursued at that time. This style of mineralization was the main focus of Corvus' drilling program in 2010-11, when twenty-seven reverse circulation (RC) holes totaling 4,128.5 meters (13,545 feet) were drilled at Jolly Jane. In 2012 and 2013, 34 additional holes were drilled at Jolly Jane totaling 4,234 meters (13,891 feet). These included three PQ3 core holes for metallurgical samples, 29 infill RC holes on the ZuZu patented claim, and two step-out RC holes to the north of the Mineral Resource area (Figure 7-8). Eight surface rock chip/channel lines totaling 384 meters (1,260 feet) have been sampled at 5-foot intervals to resemble drill holes. The results of the 2010-13 work, along with data from the Barrick drilling, are the basis for the Mineral Resources presented in this document.

7.3.2.1 JOLLY JANE STRATIGRAPHY

The stratigraphy of Jolly Jane includes the following units in ascending stratigraphic order: 1) the Cambrian Carrara Formation; 2) the Jolly Jane Formation; 3) the Savage Formation; 4) the Pioneer Formation; 5) the rhyolite bodies of the North Bullfrog Intrusive Suite; 6) the Sierra Blanca Tuff; 7) the Savage Valley Dacite; 8) the Lithic Ridge Tuff, 9) the Bullfrog Tuff; and 10) monolithic debris flow breccias of Paintbrush Tuff.

The Carrara Formation consists of primarily micritic and argillaceous limestone, with lesser calcareous shale. Jasperoid is common at or near the basal Tertiary unconformity in many drill holes. The jasperoid is considered hydrothermal in nature as it is anomalous in gold (up to 0.372 ppm) and occurs in proximity to gold mineralization in the overlying Sierra Blanca Tuff throughout the deposit area.

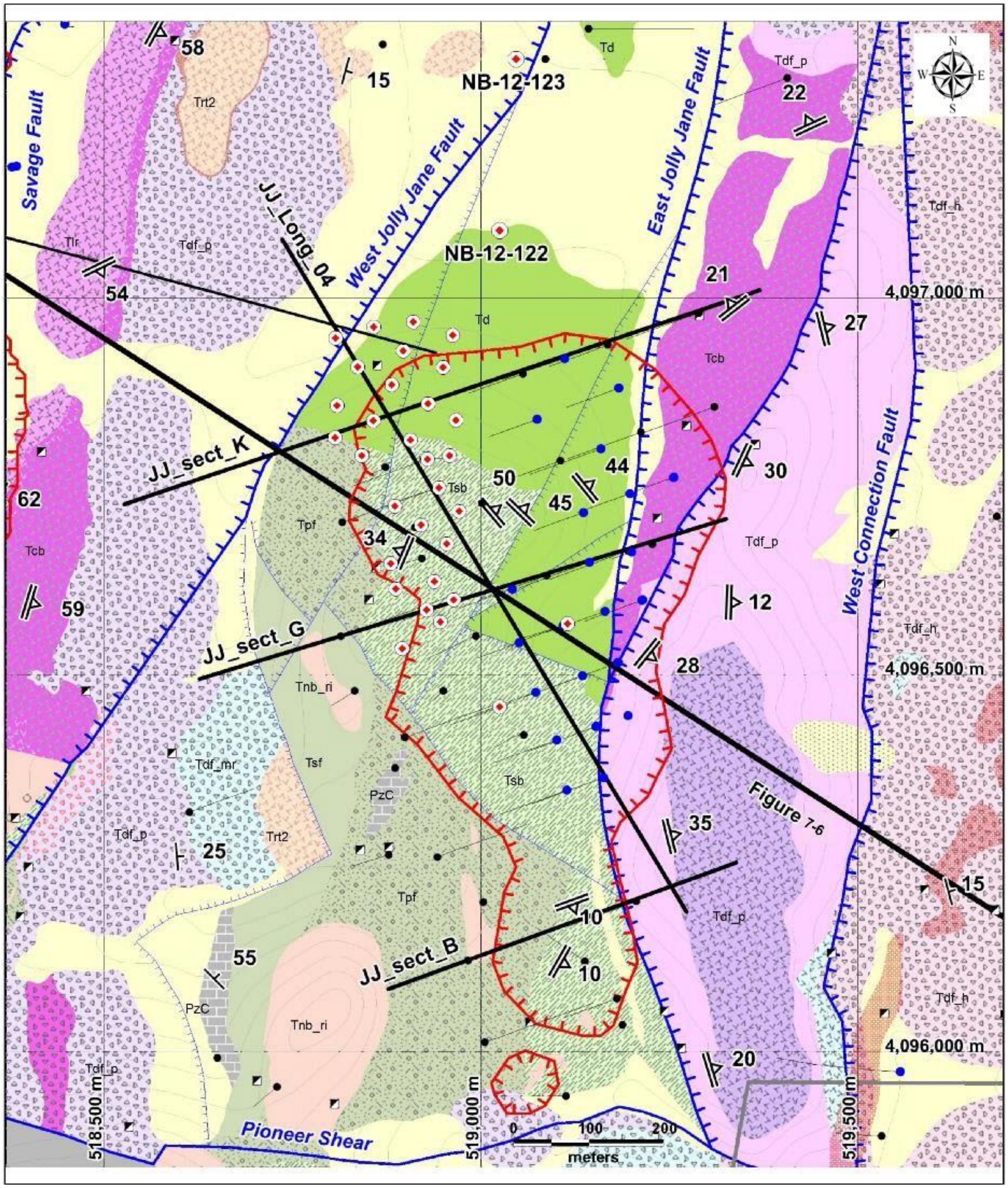
The Jolly Jane Formation was deposited unconformably on the Carrara Formation. It includes a heterogeneous sequence of 1) siliceous hematitic conglomerate, sandstone and siltstone; 2) calcareous and variably carbonaceous lithic-volcanoclastic sediments; and 3) locally intercalated monolithic debris flow breccias of the Carrara Formation limestone. The conglomerate is Paleozoic-clast dominated. It typically occurs directly along the unconformity, grading upward into finer pebbly sandstone and red hematitic siltstone. The thickness of the Jolly Jane unit varies dramatically from 0-50 meters between drill holes. Lithologic variations are also quite dramatic between drill holes.

The Savage Formation consists of aphanitic to porphyritic dacite and associated dacitic pyroclastic and epiclastic rocks. It is interpreted as a dacitic flow-dome complex that pre-dates the Pioneer Formation. The basal flows of the Savage Formation inter-finger with the underlying sedimentary rocks. The Savage Formation varies dramatically in thickness from ~100 meters in the south end of Jolly Jane to zero in the northern portion of the Jolly Jane area.

The Pioneer Formation is only a few tens of meters thick at Jolly Jane. Thickness can vary dramatically across faults, suggesting that there was active tectonism and erosion between the eruptions of the Pioneer and Savage Formations. The preserved portion of the Pioneer Formation coincides with the uppermost intervals found at Sierra Blanca, suggesting that Jolly Jane was a topographic high area during that time. A number of flow-banded aphanitic rhyolite bodies intrude the Savage and Pioneer Formations at southern Jolly Jane. The rhyolites appear to be cross-cutting intrusions and have been assigned to the North Bullfrog Intrusive Suite.

The Sierra Blanca Tuff is the dominant host rock for mineralization at Jolly Jane. The Sierra Blanca Tuff is ubiquitously quartz-adularia-altered. The preserved thickness of the unit at Jolly Jane is approximately 70 meters. At Jolly Jane, there is an interval of Savage Valley Dacite (lava flow or sill?) within the middle of the Sierra Blanca Tuff. While the Sierra Blanca Tuff appears to be a single cooling unit, there may have been time during the eruptive cycle for a simultaneous local dacite eruption to have occurred. The result is a lava flow of Savage Valley Dacite within the Sierra Blanca Tuff.

Figure 7-10 Geologic Map of the Jolly Jane Target Area



The Savage Valley Dacite overlies the Sierra Blanca Tuff, and consists of a heterogeneous sequence of lava flows, pyroclastics and epiclastics of predominantly dacitic composition. There is an angular unconformity between the top of the Sierra Blanca Tuff and the base of the Savage Valley Dacite at Jolly Jane. The Lithic Ridge and Bullfrog tuffs overlie the Savage Valley Dacite east of Jolly Jane and are juxtaposed against the Savage Valley Dacite along the East Jolly Jane Fault (Figure 7-8, Lithic Ridge Tuff not shown). The Lithic Ridge Tuff was not recognized at NBP until late 2015 and was previously lumped into the Crater Flat Group. Drilling to date at Jolly Jane has not identified any significant gold mineralization in the Lithic Ridge or Bullfrog Tuffs.

A large semi-tabular mass of Paintbrush Tuff and monolithic debris flow breccia of Paintbrush Tuff caps the ridge just east of Jolly Jane (Figure 7-8). Relatively consistent compaction foliation in the welded tuff dips 20º to 30º to the east indicating a significant angular discordance with the underlying Bullfrog Tuff. It is possible that the entire section represents a large, relatively intact, slide block of Paintbrush Tuff overlying the Bullfrog Tuff. The monolithic breccia block has been included with the Rainbow Mountain Sequence Debris Flow Sequence (Tdf_p on Figure 7-8). The Paintbrush breccias are not mineralized at Jolly Jane.

7.3.2.2 JOLLY JANE STRUCTURE

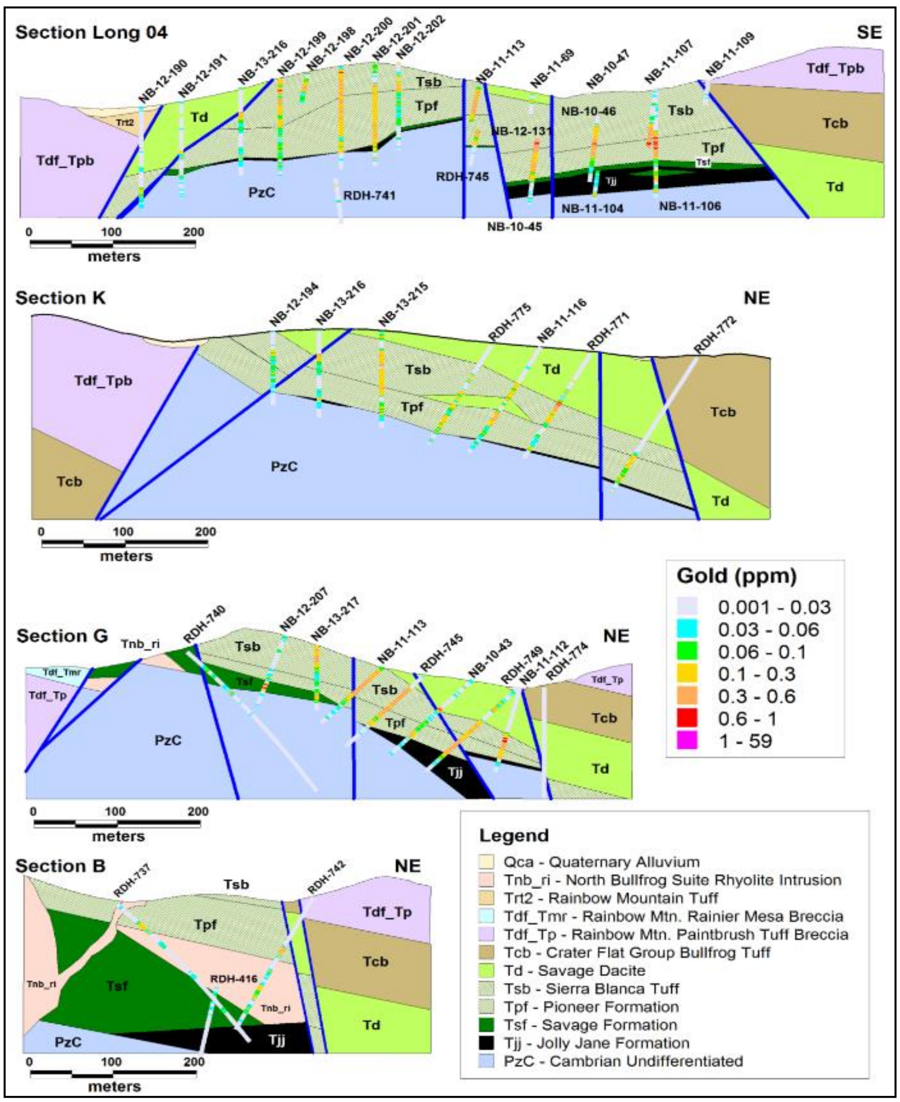
The geology of Jolly Jane is complex, mostly due to the existence of faults and erosional unconformities between each of the major stratigraphic units. The Jolly Jane gold deposit is preserved as a horst between the West Jolly Jane and the East Jolly Jane Faults (Figures 7-6, 7-8 and 7-9). The horst block is cut by steeply dipping NE- and NNW-trending faults that appear to control deposition of the Jolly Jane Formation sediments (Figure 7-9, Sections Long 04 and G). At the north end of Jolly Jane, the structure is relatively simple with large coherent zones of pseudo-stratabound disseminated mineralization between faults (Figure 7-9, Sections G and K). In contrast, the structure in the south becomes increasingly complex with larger volumes of Savage Formation and Tnb rhyolite intrusions present beneath the Sierra Blanca Tuff (Figure 7-9, Section B). The Jolly Jane mineralized zone is truncated on the west side by the West Jolly Jane Fault (Figure 7-8 and 7-9). The West Jolly Jane Fault exhibits >600 meters of down-to-the-west normal displacement, which repeats the Jolly Jane stratigraphy and gold mineralization under the Savage Valley area to the west (Figure 7-6).

7.3.2.3 JOLLY JANE MINERALIZATION

The Mineral Resources at Jolly Jane consist of the older alteration-style quartz-adularia-pyrite disseminated mineralization. The primary host rock is the Sierra Blanca Tuff, and secondary host rocks include the Savage Valley Dacite, Pioneer Formation, Savage Formation and rhyolite of the North Bullfrog Suite. Mineralization is controlled by a combination of small-displacement, high-angle feeder structures and the highly brecciated Sierra Blanca Tuff. The deposit is largely pseudo-stratabound within the Sierra Blanca Tuff. Minor quartz stockwork veining occurs throughout the Jolly Jane area, but no YellowJacket-style gold-silver enrichment event has been identified to date. All of the current Mineral Resources at Jolly Jane are oxide.

Wide spaced drilling at North Jolly Jane, including five additional holes drilled in 2015-2017, has encountered thick intervals of low-grade quartz-adularia-altered Sierra Blanca Tuff. Both low-grade oxide and sulphide mineralization is present in the Sierra Blanca Tuff and Savage Valley Dacite at North Jolly Jane. Low-grade mineralization has been extended ~750 meters north of the Jolly Jane Mineral Resource, but there is insufficient drill density to necessitate an update of the Jolly Jane Mineral Resource in this document.

Figure 7-11 Cross Sections through Jolly Jane target area. See Figure 7-8 for section locations.

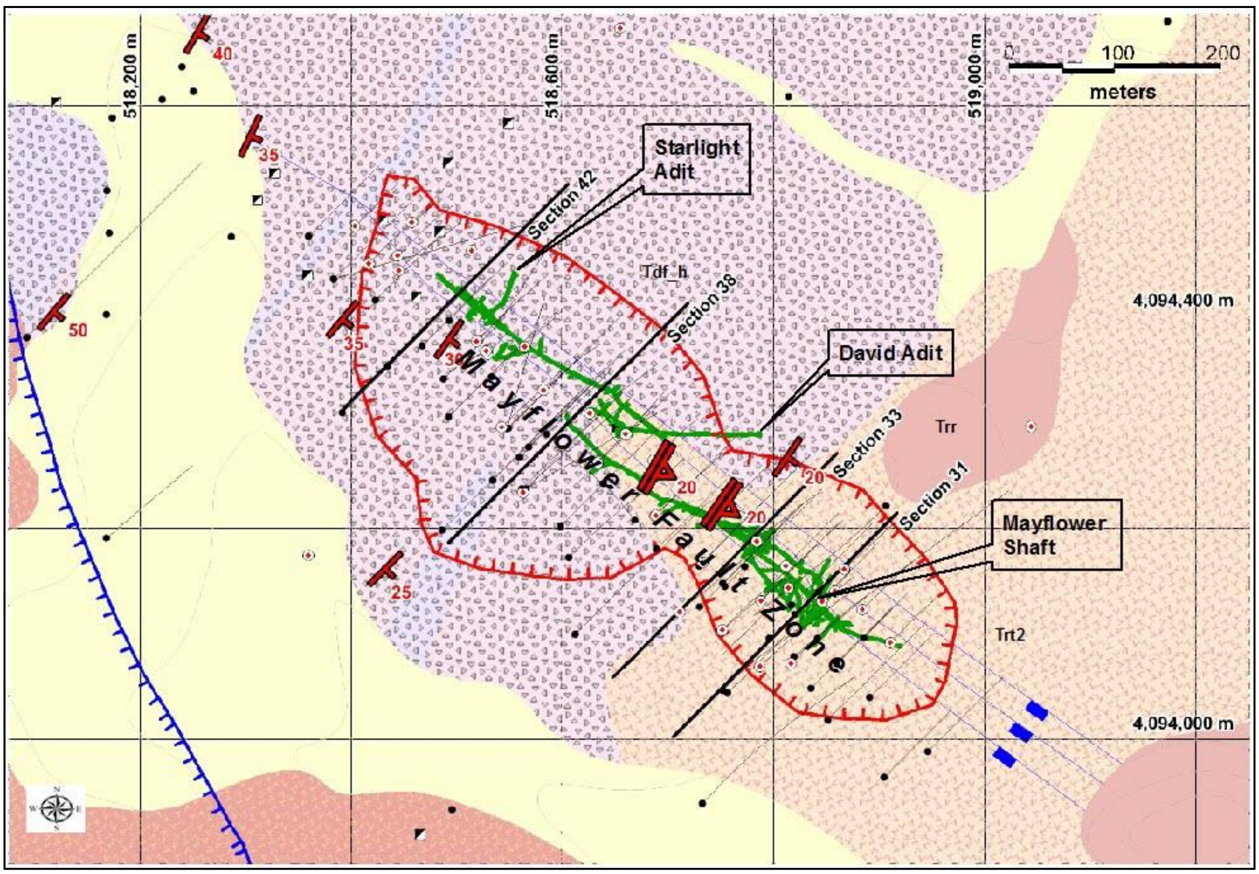


Drill traces are colored by gold assay values in the figure above

7.3.3 MAYFLOWER

The historic Mayflower mine was developed on en-echelon quartz-calcite veins and stockwork zones along the NW-striking, steeply SW-dipping Mayflower Fault Zone (MFZ). The MFZ consists of multiple SW-dipping fault splays with a complex network of fractures linking the main strands together (Figure 7-10). Based on the displacement of the base of Trt2 on Section 31 in Figure 7-11, the total apparent vertical displacement across the MFZ is approximately 60 meters. There appears to be three main fault splays within the MFZ in the southeastern part of the deposit. Two of these splays merge in the vicinity of the David Adit and only two splays remain in the northwestern end of the deposit (Figure 7-11). High-grade mineralization appears to be best developed in the fracture zones along and adjacent to the main fault splays. Dilation caused by differential movement between the fault splays appears to be the main control on disseminated mineralization. The entire Mayflower deposit is oxidized.

Figure 7-12 - Geologic Map of the Mayflower Area Showing Underground Workings, Drill Holes and Cross Section Locations



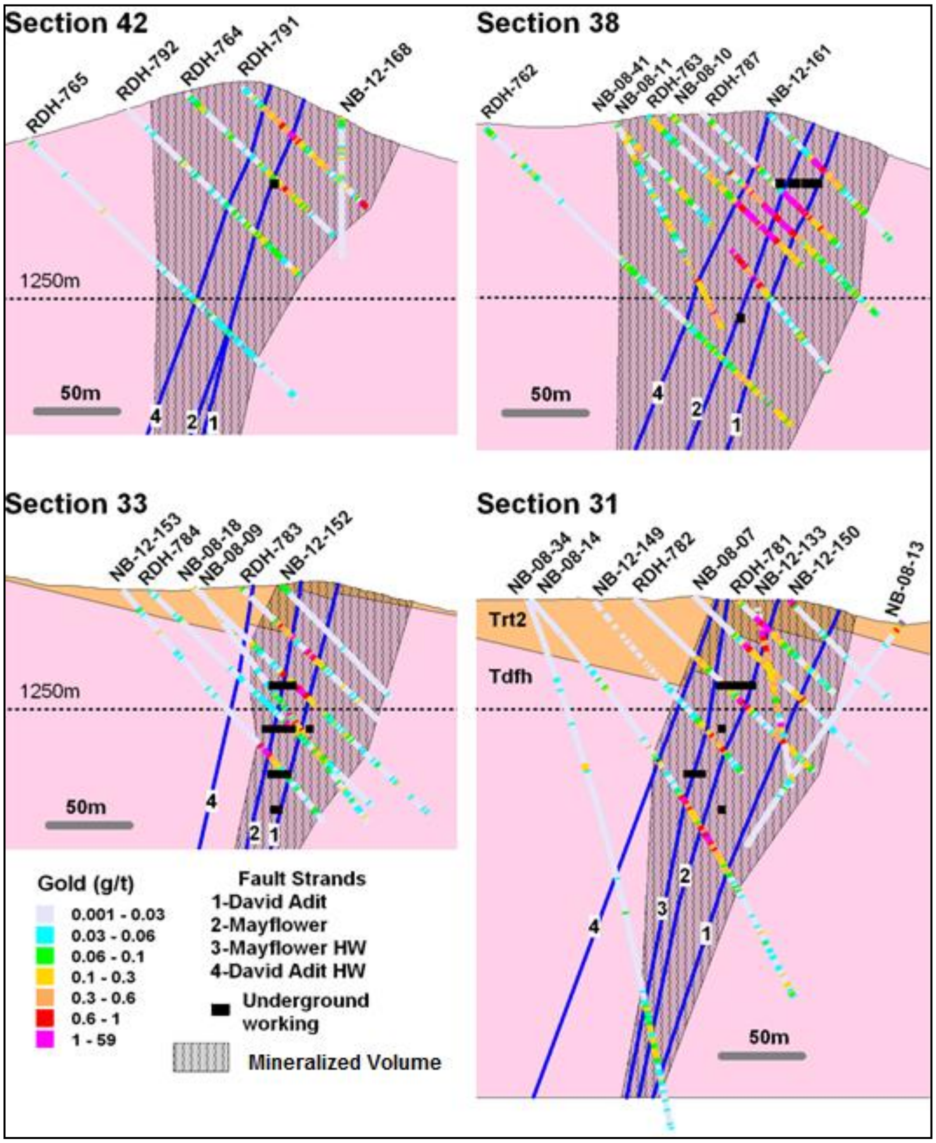
The mineralized zone is traceable for ~900 meters along strike (Figure 7-10). The bulk of the mineralization occurs in debris flow sediments of the Rainbow Mountain Sequence, but locally extends upward into the overlying Rainbow Mountain Tuff (Trt2). Alteration-style quartz-adularia disseminated mineralization surrounds a steeply dipping zone of vein-filled breccias. Quartz-adularia alteration extends out into the debris flow sediments for several tens of meters around the main fault zone. Certain horizons appear more permeable than others and therefore are altered and mineralized for greater distances. Alteration pinches to only a few meters wide along faults in Trt2.

Vein and stockwork mineralization are proximal to individual fault splays. Late stage black manganiferous calcite occurs as veins and breccia fillings along the mineralized structures. This calcite is mineralized and is thought to be a late vein stage of the waning hydrothermal system. Narrow high-grade gold zones (shoots) are known from the historic workings and from Corvus drilling. Overall, the mineralized zone appears to narrow with depth, and has a steeper more planar hanging-wall than footwall (Figure 7-11). There is a clear correlation between higher gold grades and arsenic, both of which are associated with adularization of the host rocks (Myers, 2008).

The Mayflower inclined shaft was initially developed in the early 20th Century on the main Mayflower splay of the MFZ. Historical records indicate the shaft developed four levels, with mining occurring on the 200, 300 and 400 levels. The bulk of the production came from the 300 level (Spencer, 1919). Based on the Spencer (1919) map, it appears that approximately 17,000 tonnes of material had been extracted by that time. The David Adit and Starlight workings were developed on the David Adit splay, which is the footwall to the Mayflower splay. The David Adit was driven to explore the northwest extension of the system.

The Mayflower prospect was the focus of modern exploration and drilling by numerous companies starting in 1982. Drilling results have been collected for most of the drill holes. Original assay certificates are available for the Barrick drilling. In 2008, Corvus (International Tower Hill under the NBPJV) drilled 24 reverse circulation (RC) holes totaling 5,953 meters (19,531 feet) in the Mayflower area. In 2012, Corvus drilled 52 additional holes totaling 7,503 meters (24,615 feet) including: 1) 14 PQ3 core holes totaling 1,922 meters (6,306 feet); 2) 26 in-fill/definition RC holes totaling 3,077 meters (10,095 feet); 3) seven condemnation RC holes totaling 1,218 meters (3,500 feet); 4) four water monitor wells (RC) totaling 981 meters (3,220 feet); and 5) one water pilot RC hole totaling 305 meters (1,000 feet). The PQ3 core holes have been used for additional metallurgical testing including bottle roll and column leach tests. Data for both the Corvus drilling and historic Barrick drilling are used in the Mineral Resource estimate in Section 14.

Figure 7-13 - Cross Sections Looking Northwest Through the Mayflower Deposit



For the cross-section locations of Figure 7-11 see Figure 7-10.

8. DEPOSIT TYPES

Gold mineralization in the main Bullfrog and North Bullfrog sub-districts is characterized as volcanic-hosted low-sulfidation epithermal type. Two styles of epithermal precious metal mineralization are present: 1) disseminated mineralization associated with sulfidation of iron in the volcanic host rocks; and 2) open-space filling quartz and/or carbonate veins, which are controlled by boiling. Pervasive quartz-adularia alteration of volcanic rocks is intimately associated with the disseminated mineralization and is an important ground preparation process for later vein forming events. Epithermal deposits form at shallow depth, from the surface to generally <2 kilometers. Temperatures of formation range between 150 to 300°C. Mineralization at NBP is typical of other low-sulfidation type gold systems in and around the Walker Lane trend such as: Bullfrog, Round Mountain, Rawhide, Aurora, Bodie and Comstock. These deposits commonly contain higher gold grades in vein and stockwork zones surrounded by zones of lower grade disseminated mineralization. This is the accepted exploration model at NBP.

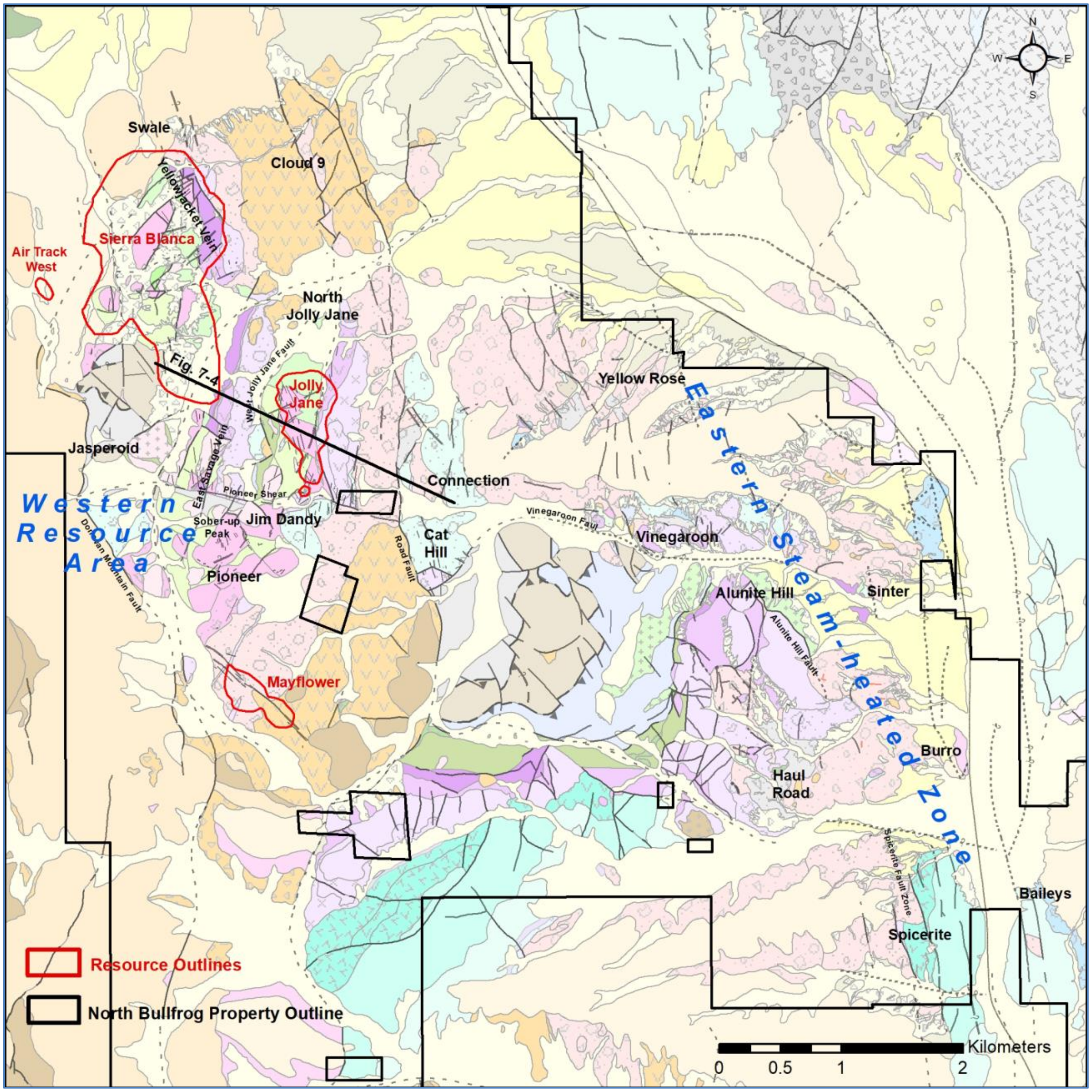
The specific details of these deposit types are in described in Section 7 with respect to the different types of occurrence and associated structures. In Section 7.2.5 the different mineralizing events observed at NBP are also discussed.

9. EXPLORATION

9.1 NBP EXPLORATION

Despite the substantial amount of work that has been done, the exploration potential of the NBP is still significant and the Project remains under-explored. The blind discovery of the Yellowjacket high-grade vein/stockwork deposit in 2012 and the identification of the extensive largely untested Eastern Steam-heated Zone in 2014 indicate significant exploration potential for the discovery of new blind high-grade deposits. Opportunities for expanding the NBP resources include: 1) possible continued expansion of the Yellowjacket vein deposit at depth; 2) new discoveries of blind, high-grade Yellowjacket-style vein systems adjacent to or within current disseminated resources in the Western Resource Area; 3) expanding or identifying new disseminated mineralization at target areas outside of the existing Mineral Resource boundaries; and 4) new discoveries of either high-grade vein or disseminated mineralization under the Bullfrog-age Eastern Steam-heated Zone (Figure 9-1).

Figure 9-1 - NBP Exploration Target Location Map



9.1.1 WESTERN RESOURCE AREA

9.1.1.1 YELLOWJACKET VEIN ZONE

The Yellowjacket Vein Zone has largely been closed off by drilling along strike but remains open at depth on some sections. Potential for the discovery of additional blind Yellowjacket-style high-grade veins exists within and adjacent to the disseminated resource areas. Much of the early resource definition drilling of the disseminated mineralization included vertical holes that were not effective at defining through-going, steeply dipping vein targets. Infill drilling of the low-grade oxide resources may discover new blind high-grade veins, particularly hosted in the Sierra Blanca Tuff.

9.1.1.2 SWALE

The Swale target lies along strike of the north-northwest projection of the Yellowjacket Vein Zone (Figure 9-1). Several east-directed angle holes were drilled in the Swale area looking for the northern extension of the Yellowjacket Vein. Most of the holes drilled at Swale encountered 1+ g/t Au sulfide mineralization, some of which is associated with quartz stockwork veining. Much of this sulfide mineralization appears to be too deep to support open pit mining. The current drilling has not confirmed a northern extension of the Yellowjacket Vein Zone. However, hydrothermal alteration and geochemistry of sulfide mineralization in the Sierra Blanca Tuff, Pioneer Formation and underlying Tnb rhyolite indicate the mineralizing system is still strong in the northernmost drilling at Swale. Additional deep drilling (400-600+ meters below surface) will be required to continue to test the underground vein and disseminated sulfide potential along strike of the Yellowjacket Vein Zone.

9.1.1.3 CAT HILL

Cat Hill lies in the footwall of the Road Fault, just south of the Connection area (Figure 9-1). As at Connection, Cat Hill is underlain by heterolithic and monolithic debris flow breccias of the Rainbow Mountain Sequence. Alteration at Cat Hill is characterized by overlapping assemblages of both steam-heated alteration and silicified ribs with quartz veining. Quartz stockwork veining exhibits boiling textures (quartz pseudomorphs of bladed calcite). Fine vuggy, northeast trending silicified ribs with quartz replacing calcite coexist at the same elevation as an opal-kaolinite-alunite assemblage that locally exhibits alunite veinlets. Surface rock chip sampling has yielded up to 1.4 g/t Au from quartz vein material. The current interpretation of the alteration at Cat Hill suggests a fluctuating paleo-groundwater table, which has resulted in the juxtaposition of contrasting styles of hydrothermal alteration.

Nine east-directed angle holes were drilled at Cat Hill between 2015 and 2019. Hole NB-15-284 encountered multiple 10-40-meter intercepts at >0.15 g/t Au, including two individual samples at >1 g/t Au. Hole NB-19-487 encountered 6.1m at 2.04 g/t Au, as well as significant thicknesses of low-grade oxide mineralization. Two other holes also have significant low-grade oxide intercepts. Most of the mineralization encountered in the drilling at Cat Hill is oxidized. The Corvus drilling suggests that a small oxide resource can be developed at Cat Hill with additional drilling. Additional drilling is planned for 2020 to follow-up the hi-grade interval in NB-19-487 and bring Cat Hill up to Mineral Resource status.

9.1.1.4 CONNECTION

The historic Connection shaft and prospects were developed at Connection in the early 1900's. Connection lies just east of and in the footwall of the Road Fault. Between 1974 and 1982, Cordex drilled a number of shallow holes and delineated a small mineralized zone.

Five general lithologic units have been identified in the Connection drill area including: 1) probable rooted Bullfrog Tuff at depth; 2) a monolithic debris flow breccia unit of Paintbrush Tuff (Tdf_p); 3) a monolithic debris flow unit of Rainier Mesa Tuff (Tdf_mr); 4) a massive slide block of mixed Paleozoic lithologies (Tdf_C); and 5) a quartzite-clast-dominated heterolithic debris flow (Tdf_h) which caps Connection Hill. All of these local units lie within the Rainbow Mountain Debris Flow Sequence. The true thickness of each unit is unknown and is expected to be highly variable in such a debris flow environment.

Corvus has drilled a total of four holes in the Connection area and has not been able to expand the mineralized zone sufficiently to define significant Mineral Resource. Since the host rocks are part of the Rainbow Mountain Debris Flow Sequence, this mineralization is most likely part of the younger ~10 Ma hydrothermal activity. Given the style of sulfide mineralization in the Bullfrog Tuff at depth, the area may have also been affected by older mineralizing events. It is possible that the Tdf_C slide block was mineralized elsewhere and then mass-wasted to the present location during later extensional deformation.

9.1.1.5 LIBERTY VEIN

The Liberty Vein is located along two historic workings that lie along strike of the southeast projection of the Yellowjacket Vein Zone. The workings contain a NNW-trending, steeply west-dipping quartz vein/replacement zone, hosted in strongly clay-altered Bullfrog Tuff. A NE-trending set of cross fractures is also present. The target concept was that the Liberty Vein is a high-level surface expression of the southeast extension of the Yellowjacket Vein Zone. A fence of two angle holes (NB-15-427, 428) was drilled under the historic workings. The Bullfrog Tuff and underlying Lithic Ridge Tuff were not significantly mineralized, and no significant quartz veining was found in either hole. The Savage Valley Dacite in the bottom of NB-15-428 is weakly mineralized (sulfide, up to 0.19 ppm). The Liberty Vein structure exposed at the surface does not appear to be rooted by any significant gold mineralization. However, these holes failed to target the structure at a deep enough elevation to test for a vein zone in the Sierra Blanca Tuff.

9.1.1.6 CLOUD 9

The Cloud 9 target was generated in 2016 from additional mapping along the West Jolly Jane Fault to the north of the North Jolly Jane target area (Figure 9-1). An historic prospect and two small outcrops exhibiting banded manganiferous calcite and quartz veining were discovered along the West Jolly Jane Fault ~1.2 kilometers north of the drilling at North Jolly Jane. The hanging wall unit is Trt2 tuff, and the footwall unit is heterolithic debris flow breccia dominated by Paleozoic clasts. Rock chip sampling has not yielded any significant gold or trace element geochemistry. One east-directed angle hole (NB-16-307) was drilled to test the fault for veining at shallow depth below the vein occurrence. No significant gold was encountered on the fault in hole NB-16-307. The footwall rocks at depth were found to be relatively unaltered, in situ Wood Canyon Formation basement. The drill hole is significant in that it shows that pre-Tertiary basement stratigraphy has returned to near surface in the footwall block of the West Jolly Jane Fault. This implies that the Sierra Blanca Tuff may also return to near surface between North Jolly Jane and Cloud 9. The evidence suggests the existence of a new shallow oxide target in Sierra Blanca Tuff between North Jolly Jane and Cloud 9. Additional drilling is warranted in this area.

9.1.1.7 JIM DANDY

The Jim Dandy target is named for the Jim Dandy patented claim, which lies ~300 meters north-northeast of the Pioneer Mine (Figure 9-1). The Jim Dandy fault is NNE-trending, steeply west-dipping and has yielded anomalous Au in surface rock samples. Historic drill hole P92-9 in this area encountered 1+ g/t Au in the top of the hole along the trace of the Jim Dandy fault. Two east-directed angle holes (NB-17-439, 440) were drilled in 2017 to test the Jim Dandy fault. Both holes encountered intensely altered rocks with quartz veins and stockwork zones. However, the holes intersected only narrow zones of anomalous gold (max. 0.42 ppm). No future work is recommended at Jim Dandy.

9.1.1.8 EAST SAVAGE VEIN

The East Savage Vein lies just east of the south end of Savage Valley (Figure 9-1). The East Savage Vein was identified by previous explorers at NBP but had only one historic drill hole through 2017. The vein has yielded an adularia date of 11.0 Ma (Connors et al., 1998), a similar age to the Yellowjacket Vein. The East Savage Vein is NNE-trending, steeply west dipping, and cuts through Sierra Blanca Tuff, Pioneer Formation, and a large Tnb rhyolite body. There is anomalous Au (max. 0.640 ppm) in surface rock samples. The vein zone persists along strike for nearly 500 meters. Two east-directed angle holes (NB-17-441 and 442) were drilled to test nearly 250 meters of strike length. Both holes drilled thick intervals of intensely quartz-adularia-pyrite-altered Tnb rhyolite, and eventually penetrated into PzC basement. No significant quartz veins were encountered at depth. Anomalous gold as disseminated mineralization (up to 0.130 ppm) was found in the intensely altered rhyolite. No future work is recommended at the East Savage Vein.

9.1.1.9 JASPEROID

The Jasperoid target is located south of the Sierra Blanca Mineral Resource (Figure 9-1). An historic soil grid by the Bond-Sunshine JV in 1990 shows a substantial gold anomaly over the target area, with a number of values >0.200 ppm. The original Barrick target was an occurrence of mineralized bedding-parallel jasperoid in limey beds of the Wood Canyon Formation. The Wood Canyon Formation is the host unit at the Reward deposit south of Beatty.

Barrick drilled hole RDH-767in 1995, which intersected 21 meters at 0.35 g/t oxide mineralization, including 1.52 meters at 1.1 g/t. This intercept had not been followed up since 1995.

Additional mapping in late 2018 identified NNW-trending, hydrothermally altered, recessive quartz-porphyry dikes cutting though the Zabriskie quartzite and Wood Canyon Formations. The mapping also identified discontinuous, crustifom-banded quartz veins up to 0.3 meters wide running sub-parallel to the quartz-porphyry dikes. One such quartz vein has yielded 2 g/t from a prospect on the east side of the target area. The quartz-porphyry dikes are also anomalous in gold, Corvus drilled four angle holes in 2019 targeting the dikes and quartz veins at depth in the Wood Canyon Formation. All four holes intersected widely scattered anomalous gold up to 0.200 ppm, but no significant intercepts. No additional drilling is recommended in this area.

9.1.1.10 ROAD FAULT

The Road Fault is the northern continuation of the Contact Fault from the southern Bullfrog Hills. It is one of the largest displacement faults at NBP. It has gold mineralization in the immediate footwall at Cat Hill and Connection, and steam-heated alteration widely distributed in the hanging wall. The surface characteristics of the fault and its linear extent suggest the Road Fault was a significant structural conduit for hydrothermal fluids. It remains largely untested at depth along the entire strike length across the NBP.

9.1.1.11 WEST CONNECTION

The West Connection vein zone lies ~500 meters west of the Connection area (Figure 9-1). The West Connection vein consists of a zone of high-level chalcedonic quartz and quartz-flooded breccia up to 15 m wide, hosted within silicified monolithic debris flow breccias of the Paintbrush and Rainier Mesa Tuffs. The vein zone strikes N5E, dips 70-80° to the east, and persists along strike for ~250 meters. The vein zone has formed along one or more hanging wall splays of the West Connection Fault, which is antithetic to the Road Fault. The fault and vein geometry suggest that the vein fluids may have ascended from the Road Fault at depth.

In 1992, Pathfinder Exploration drilled a shallow hole (P92-3), which encountered 6.1 meters of 0.243 g/t Au from 48.7 to 54.9 meters at the south end of the vein zone. This intercept has better gold grades than any of the surface rock sampling, indicating that the gold tenor may be increasing with depth. The vein zone has yielded surface trace element values up to 503 ppm As, 33 ppm Sb, and 2.74 ppm Hg. Corvus drilled one hole at West Connection in 2011 (NB-11-77). The hole did not encounter significant quartz veining, but it intersected a zone of anomalous gold (>0.1 ppm) gold between 94 to 107 meters, and several zones with anomalous arsenic and antimony. Well crystallized hydrothermal kaolinite was found in a number of intervals in this hole, possibly linking this structure to the opalite alteration along the Road Fault. There has been no follow-up drilling in this area since 2011.

9.1.1.12 PIONEER

The historic Pioneer workings are located immediately north of the Mayflower Mine (Figure 9-1). A series of underground workings were developed at Pioneer in the early 1900's. Little is known about the production or the nature of the mineralization extracted. Based on maps of historical underground workings, mineralization appears to occur along intersecting northeast and northwest striking faults. Alteration styles from the waste dumps include silicification, adularization, argillization, and minor quartz veining. Fault zones in the Pioneer area also host argillized dacite dikes, which are compositionally similar to the Savage Valley Dacite.

Much of the historic drilling as well as surface and underground sampling demonstrates that the bulk of the unmined mineralization at Pioneer is low-grade (<1 g/t Au). Most of the high-grade gold samples came from the upper levels of the Pioneer mine with grades over a few meters of 1-14 g/t Au. During 2007, the North Bullfrog Project Joint Venture (NBPJV) drilled two holes to investigate the Pioneer mineralization. The first hole targeted the down dip extension of the mineralization. Low grade mineralization was intersected across 130 meters, with a maximum value of 0.26 g/t Au in quartz-adularia-altered Sierra Blanca Tuff. A second hole was designed to drill across known higher-grade mineralization. This hole encountered a total of eight meters of 2 g/t Au, including 17.6 g/t over 0.4 m, on either side of a 3.5-meter-wide stope. The high-grade interval in the core hole is in a clay altered fault zone without visible quartz veining. Bladed pseudomorphs of quartz after calcite are found in outcrop at Pioneer, but such veining lacks significant grade. No new drilling has been undertaken at Pioneer since 2007, and no Mineral Resource has been established.

9.1.2 EASTERN STEAM-HEATED ZONE

Geologic mapping, rock and soil sampling, age dating and a gravity survey was conducted by Corvus over the Eastern Steam-heated Zone ("ESHZ") in 2014 and 2015. In early 2015, a gravity survey was completed in order to define possible structures beneath the extensive alteration zone which is exposed over >14 square kilometers (Figure 9-2). Also, in 2015, an extensive soil grid consisting of 3,672 samples was completed (Figure 9-3). The ESHZ is a broad area characterized by resistive low-temperature opal-chalcedonic silica accumulations (ribs and mounds) surrounded by recessive kaolinite-alunite alteration. Low-temperature residual silica accumulation is interpreted to define a paleo-groundwater table at the ESHZ. Residual silica forms erosional remnants of a flat tabular zone at similar elevations across much of the ESHZ. The steam heated alteration and residual silica are largely barren of metals, as would be expected at this level of erosion over a productive vein system. The target concept at the ESHZ is to test for high-grade veins in a hypothetical boiling zone below the steam-heated alteration and the paleo-groundwater table. The work that has developed this concept has defined a series of exploration targets within and around the ESHZ (Figure 9-1).

9.1.2.1 SPICERITE

The Spicerite area is located at the southeastern corner of the NBP (Figure 9-1). Map units exposed in the Spicerite area include (in ascending order): Pre-Rainier Mesa rhyolite flow breccias (Tprr), the Rainier Mesa Tuff (Tmr), and heterolithic to monolithic debris flow breccias (Tdf). The stratigraphy is cut by at least three through-going NNW-trending, moderate to steeply west-dipping, down-to-the-west normal faults. These faults are collectively known as the Spicerite Fault Zone. A fourth NNW-trending fault (Hematite Fault) is down-to-the-east and forms a graben in the hanging wall of the Spicerite Fault Zone. The graben has been filled with largely non-steam-heated volcanic debris flow breccia, but also contains cobbles and boulders of steam-heated rocks. It is hypothesized that the graben fill may be concealing a primary vein target at depth. The area has yielded an age date of 10.2 Ma from alunite (Weiss, et al., 1994). The evidence observed in the Spicerite area suggests a very dynamic period of extensional faulting, hydrothermal alteration and erosion between 9.5-10.2 Ma, which is equivalent in age to the Bullfrog deposit.

In 2015, Corvus drilled one angled core hole (NB-15-429) and three angled RC holes (NB-15-263, 264 and 265) on an E-W fence across the Spicerite target area. No significant gold or other metal values were encountered at the elevations reached by these holes (~300 meters below surface). All holes bottomed in low-temperature opal-kaolinite-alunite alteration. The stratigraphy encountered in the drill holes demonstrates that the Spicerite Fault Zone has ~700 meters of down-to-the-west displacement. This fault zone is likely the deep structural conduit feeding the hydrothermal fluids at Spicerite. The Spicerite Fault Zone is an analog of the MP fault, which hosts the Bullfrog vein deposit. The Spicerite Fault Zone requires deep drilling down-dip from existing holes. This represents a significant target for a new Bullfrog-age vein system at depth under the ESHZ.

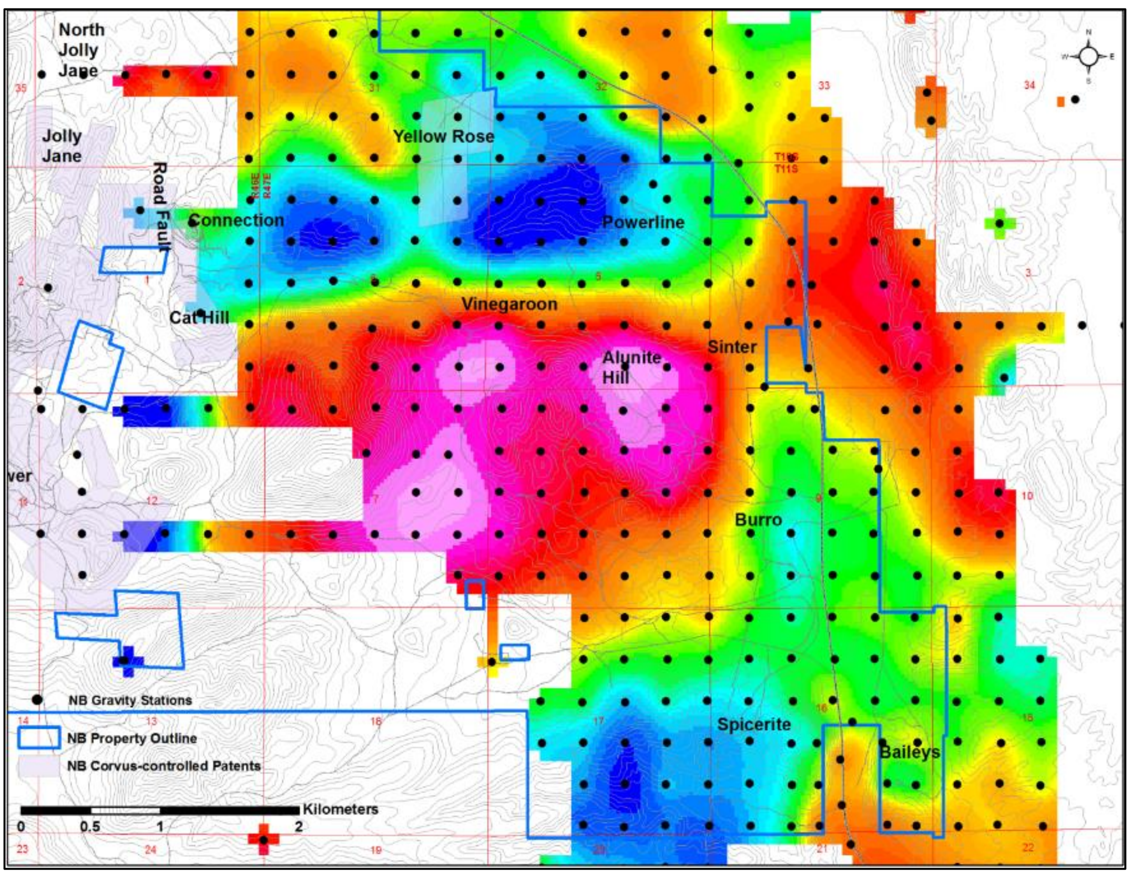
In 2019 Corvus drilled a 232-meter pre-collared RC hole, and subsequently follow with a 774-meter core tail in 2020. The goal was to intersect the Spicerite Fault Zone ~200 meters below previous drilling. Hole NB-19-489CT confirmed the Spicerite Fault Zone at depth and encountered illite-adularia alteration in the Bullfrog Tuff, Sierra Blanca Tuff, and Pioneer Formation. Minor quartz stockwork veinlets are present in these units at depth, but the mineralization appears limited to disseminated sulfide. Assays are pending.

9.1.2.2 ALUNITE HILL

Alunite Hill is located on the western side of the Eastern Steam-heated Zone, in an area of transition from steam-heated alteration to illite and adularia alteration (Figure 9-1). Alunite Hill is named for the abundant hypogene alunite veining in strongly silicified Paintbrush Tuff. The primary structural feature is the NW-trending, moderate to steeply SW-dipping Alunite Hill Fault. The Alunite Hill Fault juxtaposes the Paintbrush and Bullfrog Tuffs in the hanging wall against a dacite porphyry intrusive body. The wall rocks of the Alunite Hill Fault exhibit both steam-heated and non-steam-heated alteration. The fault hosts a discontinuous Au-Ag-bearing quartz vein that locally exhibits spectacular bladed quartz pseudomorphs after calcite. Rock sampling has yielded up to 0.746 ppm Au and 13 ppm Ag. The Alunite Hill and Spicerite No. 1 Faults may be the same structure propagating under the cover of the altered debris flow breccia sequence.

Corvus drilled three holes (NB-15-260, 261 and 262) on the Alunite Hill Fault. Holes 260 and 261 comprise a fence of two angle holes under the best developed portion of the vein at the surface. Both holes intercepted quartz stockwork veining on the fault, each having with 4-6 meters intervals of low-grade Au-Ag mineralization. The drilling indicates a much flatter SW-dip of ~35^{O} in contrasts to the 50-75^{O} dips measured at surface. The initial test of the Alunite Hill Fault was not successful in identifying high-grade gold. The fault still has potential for high-grade mineralization at depth and along strike. Additional drilling is recommended at depth and along strike to fully test the Alunite Hill Fault.

Figure 9-2 - Location of 2015 Gravity Stations on Complete Bouger Anomaly Data with 3rd Order Trend Removed



9.1.2.3 VINEGAROON

The Vinegaroon area is located just north of Alunite Hill in the hanging wall of the Vinegaroon Fault (Figure 9-1). The Vinegaroon Fault is major E-W-trending, moderate to steeply north-dipping, basement-bounding fault that juxtaposes Tertiary volcanic rocks against Paleozoic basement rocks. The Vinegaroon Fault projects eastward into the Eastern Steam-heated Zone from the Road Fault and is truncated by the Road Fault on the west. Much of the Vinegaroon target area is in an alteration transition zone, exhibiting both quartz-adularia and steam-heated alteration assemblages. Numerous NNE- to NNW-trending high angle faults cut the debris flow breccias. Silicified ribs with anomalous gold occur along several of the faults. The interpretation is that hydrothermal fluids have ascended into these hanging wall faults from the Vinegaroon Fault at depth. Hypogene alunite, similar to that of Alunite Hill, is present in this area and has been dated by Corvus at 9.5 Ma (Table 7-2). Anomalous gold is also present associated with apparently stratabound quartz-adularia-pyrite alteration.

Corvus drilled seven holes (NB-15-286 through 292) testing a number of high-angle silicified structures, a stratabound quartz-adularia zone, and the Vinegaroon Fault itself. The holes intersected several scattered narrow low-grade gold zones including 14 meters at 0.20 g/t Au and 1.45 g/t Ag (max. value 0.35 g/t Au), but no quartz veins. This initial wide-spaced drilling of the Vinegaroon area was unsuccessful in finding mineralization of sufficient continuity for resource definition. The area hosts significant alteration and gold mineralization and warrants additional work in the future.

9.1.2.4 HAUL ROAD

The Haul Road target is located along the D & H Mining haul road to the Gold Pit. Haul Road was newly identified in 2016 (Figure 9-1). An irregular quartz vein exposed in a partially reclaimed prospect has yielded gold values up to 0.104 g/t. The vein is NE-trending, steeply NW-dipping, and hosted in monolithic debris flow breccia of Wood Canyon Formation lithologies. The target remains untested by drilling.

9.1.2.5 BURRO

The Burro target area was defined primarily from a gold anomaly in soil sampling (Figures 9-1 and 9-3). There was reason to question the validity of this anomaly given that it lies in the middle of steam-heated alteration, which is generally barren of gold. Clasts of altered Paleozoic lithologies were noted weathering out of generally recessive kaolinite-alunite-altered rocks in the anomalous area. Gold values up to 1.13 ppm were obtained from the Paleozoic clasts. The conclusion is that false anomalies may be present in the Rainbow Mountain Debris Flow Sequence. Regardless of this conclusion, the Burro area exhibits strong alteration in the middle of the ESHZ and has never been drilled. This is an ideal area for a fence of deep holes across the ESHZ to test for gold mineralization at depth below the steam-heated alteration.

9.1.2.6 SINTER

The Sinter target area is located east of Vinegaroon along the eastward projection of the Vinegaroon Fault (Figure 9-1). The Sinter target area was named for outcrops of opalized sediments, which were originally speculated to be sinter. The area hosts intense opaline and chalcedonic silicification in a debris flow sequence with no apparent structural control. The silicification at Sinter is interpreted to represent residual silica at a paleo-groundwater table. Similar to the Burro target, this is an ideal area for a fence of deep holes to test for gold mineralization at depth below the residual paleo-groundwater water table silica.

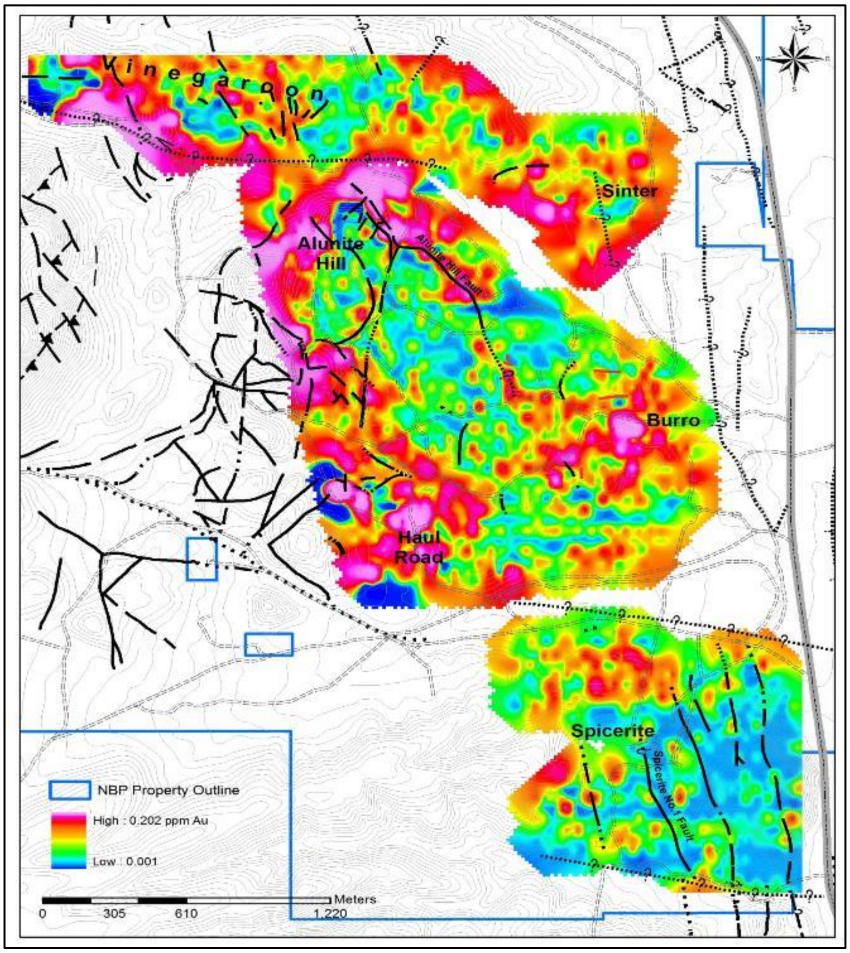
9.1.2.7 YELLOW ROSE

The Yellow Rose target area lies partially on patented claims near the northernmost exposure of the ESHZ (Figure 9-1). Shallow drilling in the surrounding area by Galli Exploration failed to encounter significant gold mineralization. The area is underlain by steam-heated alteration hosted in heterolithic and monolithic debris flow breccias. There is a series of N-S to NNW-trending high angle structures that cut the host rocks. There is anomalous gold in rock samples at the Yellow Rose adit in steam-heated alteration. Gold values up to 0.230 ppm area associated with illite-adularia-alteration ~200 meters west of the adit. The Yellow Rose area exhibits both steam-heated and non-steam heated alteration assemblages at similar elevations, suggesting a fluctuating paleo-groundwater table and the potential for buried gold mineralization. A fence of deep drill holes has been proposed at Yellow Rose but has yet to be drilled.

9.1.2.8 BAILEYS

Limited mapping and rock sampling have been done at the Baileys target area east of US Highway 95 (Figure 9-1). Steam-heated alteration hosted in the Ammonia Tanks Tuff includes resistive residual silica ribs and mounds surrounded by recessive kaolinite-alunite alteration. As expected, there is no significant surface geochemistry found in the current rock sampling. Additional mapping is necessary before targeting deep drilling at Baileys. Additional claims were staked in 2018 which connect the NBP with the MLP, extending the Baileys target area into the Baileys Gap area to the southeast (see Figure 9-1).

Figure 9-3 - Image of Gridded Gold in Soils Over the Eastern Steam Heated Zone Target Areas



10. DRILLING

Between 1974 and 1996 approximately 249 rotary and reverse circulation holes totaling 33,775 meters were drilled on the NBP by several different companies. RGC was able to obtain the assays and geological data for most of these holes and this data were acquired by ITH and then Corvus. Drilling by the NBPJV on many of the same targets encountered similar gold grades and thicknesses as the historic holes, suggesting that the earlier results are reliable. Additionally, much of the historic drilling was conducted by larger companies who mostly conducted sampling and assaying to industry standards at the time. Unfortunately, there is no quality control data available for these historic programs. Therefore, while it seems reasonable to put reliance on the older drill hole results, they must be treated as historic and as guidelines to the location of mineralized areas.

The NBPJV drilled six core holes totaling 1,300 meters in 2007 and 35 reverse circulation holes in 2008 totaling 8,422 meters. All of the core holes were drilled at an angle to intersect the mineralized structures at nearly right angles. Sample intervals in core varied with rock and alteration type and represent nearly true thicknesses. Most of the 2008 holes drilled at Air Track Hill and all of the Mayflower holes were angle drilled nearly perpendicular to the mineralized zones. Reverse circulation drilling above the water table was with a 5 ½ inch hammer bit and, where water became a problem, a 5 ¼ inch tricone bit was used. Samples were collected at 5-foot intervals starting from the top of each hole.

Corvus completed a 75 hole (17,820 meters, 58,465 feet) reverse circulation drilling program between October 2010 and June 2011 using Boart Longyear out of Elko, Nevada. Each five-foot sample was analyzed using a handheld XRF unit at the drill site. The XRF analysis was used to determine the arsenic content of the sample (a direct indicator of mineralization) and the probable stratigraphic correlation of the sample. The drill chips were cursorily logged for lithology and alteration at the drill site, and later logged in greater detail in an office setting using a binocular microscope. Magnetic susceptibility was also measured on the chips for each five-foot interval. The geologic characteristics that were determined routinely on drill chips include: lithology and stratigraphic unit assignment, alteration style and intensity, vein type and percentage, color, sulphide type and percentage, and oxide type and relative intensity. The following five oxide classes were used to quantify the oxidation state of each sample:

- Class 1: Total sulphide, no oxide present
- Class 2: Mostly sulphide with minor oxide present
- Class 3: Mixed oxide/sulphide in generally equal proportions
- Class 4: Mostly oxide with minor fresh sulphide present
- Class 5: Total oxide, no sulphide present

Oxide classes 5, 4 and 3 have consistently yielded favorable gold recoveries in bottle roll tests. Model blocks assigned to classes 5, 4 and 3 comprise the oxide mineralization category. Oxide classes 2 and 1 have consistently yielded un-favorable gold recoveries in bottle roll tests. Model blocks assigned to oxide classes 2 and 1 comprise the sulphide mineralization category.

Between January 2012 and January 2013 additional geological information was collected, including 47 new reverse circulation holes totaling 7,128 meters (23,386 feet) and 18 core holes totaling 3,438 meters (11,279 feet). The 2012 reverse circulation ("RC") drilling included step-out holes in the Sierra Blanca, Jolly Jane and ATW areas; infill holes in the Mayflower and Jolly Jane Mineral Resource areas; and condemnation holes and water monitor wells around the Mayflower area. The 2012 core drilling included PQ3 holes for metallurgical studies in the Mayflower, Jolly Jane and Sierra Blanca Mineral Resource areas; and HQ3 exploration holes in the YellowJacket area. Holes NB-12-117 through -143 were drilled by AK Drilling and holes NB12-144 through -176 were drilled by Boart Longyear. Logging protocols were the same as those employed in 2010-2011, with the exception of the addition of a hydrochloric acid fizz test log which is now done on all RC and core samples.

In 2013, Corvus drilled 87 holes at Sierra Blanca and YellowJacket totaling 19,000 meters (62,340 feet) including 35 HQ3 core holes, 2 PQ3 core holes for metallurgical samples and 50 RC holes. In addition, 13 channel sample profiles were completed along new roadcuts totaling 888 meters. The logging protocol was the same as that carried out in 2012. These new drill results formed the basis for a revised estimate of the Sierra Blanca Zone and first estimate of the YellowJacket Mineral Resource reported in April 2014.

In 2014, Corvus drilled 48 oriented core holes totaling 12,636 meters (41,456 feet). These included 36 HQ3 holes and 12 PQ3 holes for metallurgical samples. The 2014 program was focused on resource definition and metallurgical sampling of the YellowJacket Vein and Stockwork system. Two additional channel sample profiles were completed along new roadcuts totaling 181 meters (595 feet). The logging protocol was refined to improve the logging of vein types and abundances. Corvus uses the Reflex ACT II core orientation tool to orient all core holes and surveys all holes to support the structural interpretation. The YellowJacket vein mineralization is structurally controlled and occurs in distinct quartz veins and stockwork zones, as opposed to the more typical disseminated mineralization at the NBP.

In mid-2015 through early 2017, Corvus drilled 5 oriented core holes totaling 1,449 meters (4,754 feet). All 5 holes were drilled with HQ3 diameter tools. These holes were drilled by First Drilling. Additionally, another 97 RC holes were drilled totaling 26,345 meters (86,434 feet). All of the RC holes were drilled by Boart Longyear. This program was focused on resource definition and expansions of the YellowJacket Vein Zone, as well as testing several new target areas. The logging protocol was the same as the refined procedures used in 2014. The 2015 through 2017 drill results have been incorporated into the revised estimates of mineralized volumes in the Sierra Blanca Disseminated and the YellowJacket Vein Zone reported in this document. A consistent theme in the 2015-2017 drilling was testing for west-dipping veins with east-directed angle holes, primarily targeting the Sierra Blanca Tuff and Tnb rhyolite.

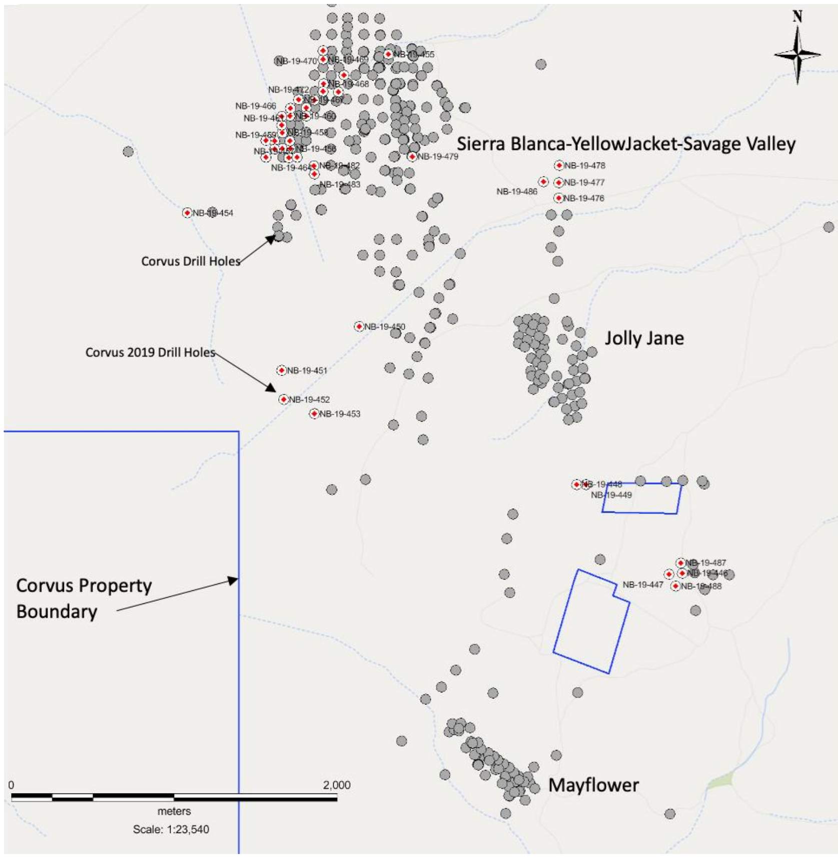
After completing the drill program at NBP in April 2017, an updated Mineral Resource estimate and Technical Report was developed (Wilson, et al, 2017). An additional 44 holes were drilled at North Bullfrog in 2019 with 29 of them located at Sierra Blanca and YellowJacket for an additional 6,972 meters of RC drilling data.

10.1 NORTH BULLFROG DRILLING

Table 10-1 lists the drilling accomplished by Corvus at NBP in the years 2010 through 2018. The table lists the resource area drilled and the number of holes and hole numbers. The drilling has been used to develop the geologic information presented in Section 7 and to develop the resource models described in Section 14.

Two general types of mineralization are shown by the drill data; (1) pervasive alteration-style mineralization (Sierra Blanca-Savage Valley and Jolly Jane) and (2) structurally controlled alteration-style enrichment and late veins (YellowJacket and Mayflower). Geologic cross-sections in Figures 7-7, 7-8 and 7-11 illustrate the gently east dipping orientation of the stratigraphic units and the orientation of drill holes at the Sierra Blanca-Savage Valley and Jolly Jane deposits. The distribution of collar locations throughout NBP are shown in Figure 10-1. Drill holes located to sample the pervasive mineralization have been oriented either vertically or with a west azimuth and 60° dip to optimize the geologic information produced on the Sierra Blanca tuff (Tsb) which hosts the majority of the mineralization.

Figure 10-1 Location of NBP Drill Holes with 2019-2020 Drilling Shown in Red



Drilling to sample the structurally controlled alteration at YellowJacket and Mayflower is designed to intersect the steeply dipping mineralized structure. The structures at both deposits strike NW with dips to the west at YellowJacket where boreholes have been drilled east to west with dips generally 60-80°. The YellowJacket structure is illustrated in Figure 7-8 and 14-2. Collars of holes drilled to sample Mayflower are illustrated in Figures 7-12 with the intersection of the structure which dips to the east as shown in 4 cross sections in Figure 7-13. Mayflower holes have been drilled west to east with dips generally 60-80°.

Table 10-1 - Drilling data from NBP with reported significant intervals identified in previous Technical Reports

Year	Resource Area	No. of Holes	Hole Numbers
2010	Sierra Blanca-YellowJacket	15	NB-10-48 to -60, -62 & 63
2010	Savage Valley	1	NB-10-64
2011	Sierra Blanca-YellowJacket	10	NB-11-68, -96 to -94
2011	Savage Valley	16	NB-11-65, -67, -81 to -85, -94 to -98, -100 to -103

2012	Sierra Blanca-YellowJacket	3	NB-12-117 to -119
2013	YellowJacket	8	NB-13-358 to -363, -367 to -372
2014	YellowJacket	47	NB-14-377 to -387, -389 to -424
2015	Sierra Blanca-YellowJacket	16	NB-15-425 to -426, -266 to 272, -267 to -282
2015	Savage Valley	3	NB-15-273 to -275
2016	Sierra Blanca-YellowJacket	25	NB-16-293 to -300, -303 to -305, -308 to -328
2016	Savage Valley	2	NB-16-301 to 302
2017	Sierra Blanca-YellowJacket	28	NB-17-329 to -340, -430 to -445
2019	Sierra Blanca-YellowJacket	28	NB-19-456 to -475, -479 to 485

10.2 NBP DRILLING

Significant intercepts are listed in Table 10-2 from 2019 drilling to illustrate the distribution of mineralization of the pervasive alteration-style in at NBP. These are representative examples of the pervasive alteration-style mineralization encountered throughout the mineral deposits and include information on some of target areas for exploration as discussed in Section 9. Drilling intercepts are not true widths. There are no known drilling, sampling or recovery factors identified that materially impact the accuracy and reliability of the results

The intercepts in Table 10-2 have been incorporated into the updated Sierra Blanca Resource estimate. No known drilling, sampling or recovery factors have been identified that materially impact the accuracy and reliability of the results. Table 10-2 reported drill intercepts are not true widths and mineralized thickness is calculated at 0.30 g/t cut-off grade.

Table 10-2 - Significant intervals in North Bullfrog holes drilled in 2019 (drilled intercepts not true widths; 0.3 g/t cut-off grade)

Drill Hole #	from (m)	to (m)	Interval (m)	Gold (g/t)	Silver (g/t)	Comment
NB19-446	76.2	91.44	15.24	0.58	n/a	***West Cat Hill***
	158.5	163.07	4.57	0.39	n/a	
	175.26	178.31	3.05	0.35	n/a	

Drill Hole #	from (m)	to (m)	Interval (m)	Gold (g/t)	Silver (g/t)	Comment
	195.07	*198.12*	*3.05*	*0.14*	*n/a*	*Low-grade (0.1 cut)*
	205.74	*220.98*	*15.24*	*0.11*	*n/a*	*Low-grade (0.1 cut)*
	227.08	*240.79*	*13.71*	*0.19*	*n/a*	*Low-grade (0.1 cut)*
AZ 085 dip-60	*248.41*	*252.98*	*4.57*	*0.27*	*n/a*	*Low-grade (0.1 cut)*
NB19-447	128.02	132.59	4.57	0.17	n/a	***West Cat Hill***
	137.16	153.92	16.76	0.19	n/a	*Low-grade (0.1 cut)*
	160.02	175.26	15.24	0.11	n/a	*Low-grade (0.1 cut)*
	234.7	259.08	24.38	0.10	n/a	*Low-grade (0.1 cut)*
	298.7	307.85	9.15	0.13	n/a	*Low-grade (0.1 cut)*
AZ 087 dip-60	315.47	345.95	30.48	0.18	n/a	*Low-grade (0.1 cut)*
NB19-448	13.72	18.29	4.57	0.49	n/a	***South Jolly Jane***
	22.86	41.15	18.29	0.41	n/a	
AZ 270 dip-60	85.34	103.63	18.29	0.13	n/a	*Low-grade (0.1 cut)*
NB19-448	13.72	18.29	4.57	0.49	n/a	***South Jolly Jane***
	22.86	41.15	18.29	0.41	n/a	
AZ 270 dip-60	85.34	103.63	18.29	0.13	n/a	*Low-grade (0.1 cut)*
NB19-450	54.86	59.44	4.58	0.19	n/a	***Jasperoid***
	74.68	79.25	4.57	0.11	n/a	*Low-grade (0.1 cut)*
	94.49	103.63	9.14	0.11	n/a	*Low-grade (0.1 cut)*
	114.3	124.97	10.67	0.11	n/a	*Low-grade (0.1 cut)*
	141.73	147.83	6.10	0.13	n/a	*Low-grade (0.1 cut)*

Drill Hole #	from (m)	to (m)	Interval (m)	Gold (g/t)	Silver (g/t)	Comment
	211.84	216.41	4.57	0.13	n/a	*Low-grade (0.1 cut)*
AZ 240 dip-50	228.6	231.65	3.05	0.10	n/a	*Low-grade (0.1 cut)*
NB19-451 AZ 070 dip-45			no significant results			***Jasperoid***
NB19-452 AZ 070 dip-45	202.69	207.26	4.57	0.20	n/a	***Jasperoid***
NB19-453 AZ 105 dip-45			no significant results			***Jasperoid***
NB19-454 AZ 090 dip-50			no significant results			***West Air Track***
NB19-455	115.82	140.21	24.39	1.21	n/a	***North YellowJacket***
	164.59	172.21	7.62	0.43	n/a	
	233.17	246.89	13.72	0.38	n/a	
	281.94	288.04	6.1	0.30	n/a	
	295.66	300.23	4.57	0.36	n/a	
	306.32	310.9	4.57	0.41	n/a	
	352.04	356.62	4.58	0.14	n/a	*Low-grade (0.1 cut)*
	367.28	374.9	7.62	0.12	n/a	*Low-grade (0.1 cut)*
AZ 000 dip-90	381.00	397.76	16.76	0.16	n/a	*Low-grade (0.1 cut)*
NB19-456	21.34	25.91	4.57	0.63	n/a	***West Sierra Blanca***
	33.53	41.15	7.62	0.38	n/a	
	60.96	76.2	15.24	0.49	n/a	
AZ 090 dip-50	97.54	114.3	16.76	1.37	n/a	
NB19-457	36.58	39.62	3.04	0.17	n/a	***West Sierra Blanca***
	45.72	80.77	35.05	0.20	n/a	*Low-grade (0.1 cut)*
	91.44	94.49	3.05	0.12	n/a	*Low-grade (0.1 cut)*

Drill Hole #	from (m)	to (m)	Interval (m)	Gold (g/t)	Silver (g/t)	Comment
	100.58	105.16	4.58	0.15	n/a	*Low-grade (0.1 cut)*
	117.35	132.59	15.24	0.65	n/a	
AZ 090 dip-55	146.30	182.88	36.58	0.11	n/a	*Low-grade (0.1 cut)*
NB19-458 AZ 090 dip-55	112.78	115.82	3.05	0.16	n/a	***West Sierra Blanca***
NB19-459	111.25	115.82	4.57	0.70	n/a	***West Sierra Blanca***
AZ 090 dip-65	153.92	156.97	3.05	0.42	n/a	
NB19-460	89.92	99.06	9.14	0.49	n/a	
	106.68	135.64	28.96	0.57	n/a	
	140.21	156.97	16.76	0.42	n/a	
AZ 090 dip-65	163.07	166.12	3.05	0.37	n/a	
NB19-461	108.20	111.25	3.05	0.16	n/a	*Low-grade (0.1 cut)*
AZ 090 dip-60	138.68	176.78	38.1	0.37	n/a	
NB19-462	10.67	16.76	6.09	0.78	n/a	
AZ 90 dip-50	73.15	85.34	12.19	0.72	n/a	
NB19-463	19.81	24.38	4.57	0.51	n/a	
AZ 90 dip-55	35.05	38.1	3.05	1.43	n/a	
	32.00	42.67	10.67	0.56	n/a	*Low-grade (0.1 cut)*
	59.44	62.48	3.04	0.20	n/a	*Low-grade (0.1 cut)*
	141.73	149.35	7.62	0.22	n/a	*Low-grade (0.1 cut)*
NB19-464	25.91	42.67	16.76	0.22	n/a	*Low-grade (0.1 cut)*
AZ 90 dip-55	50.29	62.48	12.19	0.11	n/a	*Low-grade (0.1 cut)*
NB19-465	76.20	99.06	22.86	0.19	n/a	*Low-grade (0.1 cut)*
AZ 90 dip-55	126.49	129.54	3.05	0.11	n/a	*Low-grade (0.1 cut)*

Drill Hole #	from (m)	to (m)	Interval (m)	Gold (g/t)	Silver (g/t)	Comment
NB19-466	124.97	128.02	3.05	0.14	n/a	*Low-grade (0.1 cut)*
AZ 090 dip-65	152.40	160.02	7.62	0.19	n/a	*Low-grade (0.1 cut)*
	166.12	179.83	13.71	0.15	n/a	*Low-grade (0.1 cut)*
NB19-467	118.87	131.06	12.19	0.21	n/a	*Low-grade (0.1 cut)*
AZ 090 dip-70	150.88	156.97	6.09	0.39	n/a	
	176.78	188.98	12.19	0.25	176.78	*Low-grade (0.1 cut)*
	204.22	207.26	3.05	0.15	204.22	*Low-grade (0.1 cut)*
NB19-468	73.15	94.49	21.34	0.52	n/a	
AZ 070 dip-45	124.97	128.02	3.05	0.15	n/a	*Low-grade (0.1 cut)*
	140.21	143.26	3.05	0.46	n/a	
	153.92	158.50	4.58	0.15	n/a	*Low-grade (0.1 cut)*
	192.02	199.64	7.62	0.17	n/a	*Low-grade (0.1 cut)*
	208.79	211.84	3.05	0.11	n/a	*Low-grade (0.1 cut)*
NB19-469	114.3	153.92	39.62	0.71	n/a	
AZ 90 dip-55	160.02	169.16	9.14	0.42	n/a	
	181.36	184.40	3.04	0.39	n/a	
	195.07	204.22	9.15	0.72	n/a	
	217.93	220.98	3.05	0.14	n/a	*Low-grade (0.1 cut)*
	248.41	254.51	6.1	0.17	n/a	*Low-grade (0.1 cut)*
	265.18	268.22	3.04	0.19	n/a	*Low-grade (0.1 cut)*
	277.37	283.46	6.09	0.11	n/a	*Low-grade (0.1 cut)*
NB19-470	160.02	164.59	4.57	0.49	n/a	

Drill Hole #	from (m)	to (m)	Interval (m)	Gold (g/t)	Silver (g/t)	Comment
AZ 90 dip-65	227.08	233.17	6.09	1.05	n/a	
	225.55	266.70	41.15	0.30	n/a	*Low-grade (0.1 cut)*
	242.32	246.89	4.57	0.45	n/a	
	306.32	309.37	3.05	0.40	n/a	
NB19-460	89.92	99.06	9.14	0.49	n/a	
	106.68	135.64	28.96	0.57	n/a	
	140.21	156.97	16.76	0.42	n/a	
AZ 090 dip-65	163.07	166.12	3.05	0.37	n/a	
NB19-461	108.20	111.25	3.05	0.16	n/a	*Low-grade (0.1 cut)*
AZ 090 dip-60	138.68	176.78	38.1	0.37	n/a	
NB19-462	10.67	16.76	6.09	0.78	n/a	
AZ 90 dip-50	73.15	85.34	12.19	0.72	n/a	
NB19-463	19.81	24.38	4.57	0.51	n/a	
AZ 90 dip-55	35.05	38.1	3.05	1.43	n/a	
	32.00	42.67	10.67	0.56	n/a	*Low-grade (0.1 cut)*
	59.44	62.48	3.04	0.20	n/a	*Low-grade (0.1 cut)*
	141.73	149.35	7.62	0.22	n/a	*Low-grade (0.1 cut)*
NB19-464	25.91	42.67	16.76	0.22	n/a	*Low-grade (0.1 cut)*
AZ 90 dip-55	50.29	62.48	12.19	0.11	n/a	*Low-grade (0.1 cut)*
NB19-465	76.20	99.06	22.86	0.19	n/a	*Low-grade (0.1 cut)*
AZ 90 dip-55	126.49	129.54	3.05	0.11	n/a	*Low-grade (0.1 cut)*
NB19-466	124.97	128.02	3.05	0.14	n/a	*Low-grade (0.1 cut)*
AZ 090 dip-65	152.40	160.02	7.62	0.19	n/a	*Low-grade (0.1 cut)*
	166.12	179.83	13.71	0.15	n/a	*Low-grade (0.1 cut)*

Drill Hole #	from (m)	to (m)	Interval (m)	Gold (g/t)	Silver (g/t)	Comment
NB19-467	118.87	131.06	12.19	0.21	n/a	*Low-grade (0.1 cut)*
AZ 090 dip-70	150.88	156.97	6.09	0.39	n/a	
	176.78	188.98	12.19	0.25	176.78	*Low-grade (0.1 cut)*
	204.22	207.26	3.05	0.15	204.22	*Low-grade (0.1 cut)*
NB19-468	73.15	94.49	21.34	0.52	n/a	
AZ 070 dip-45	124.97	128.02	3.05	0.15	n/a	*Low-grade (0.1 cut)*
	140.21	143.26	3.05	0.46	n/a	
	153.92	158.50	4.58	0.15	n/a	*Low-grade (0.1 cut)*
	192.02	199.64	7.62	0.17	n/a	*Low-grade (0.1 cut)*
	208.79	211.84	3.05	0.11	n/a	*Low-grade (0.1 cut)*
NB19-469	114.3	153.92	39.62	0.71	n/a	
AZ 90 dip-55	160.02	169.16	9.14	0.42	n/a	
	181.36	184.40	3.04	0.39	n/a	
	195.07	204.22	9.15	0.72	n/a	
	217.93	220.98	3.05	0.14	n/a	*Low-grade (0.1 cut)*
	248.41	254.51	6.1	0.17	n/a	*Low-grade (0.1 cut)*
	265.18	268.22	3.04	0.19	n/a	*Low-grade (0.1 cut)*
	277.37	283.46	6.09	0.11	n/a	*Low-grade (0.1 cut)*
NB19-470	160.02	164.59	4.57	0.49	n/a	
AZ 90 dip-65	227.08	233.17	6.09	1.05	n/a	
	225.55	266.70	41.15	0.30	n/a	*Low-grade (0.1 cut)*
	242.32	246.89	4.57	0.45	n/a	
	306.32	309.37	3.05	0.40	n/a	

Drill Hole #	from (m)	to (m)	Interval (m)	Gold (g/t)	Silver (g/t)	Comment
NB19-471 AZ 90 dip-65	80.77	143.26	62.49	0.49	n/a	
NB19-472	62.48	80.77	18.29	0.79	n/a	
AZ 90 dip-70	102.11	105.16	3.05	0.49	n/a	
	111.25	117.35	6.1	0.36	n/a	
	158.5	169.16	10.66	0.48	n/a	
	173.74	178.31	4.57	0.59	n/a	
NB19-473	71.63	94.49	22.86	0.89	n/a	
AZ 90 dip-60	111.25	114.3	3.05	0.31	n/a	
	143.26	156.97	13.71	0.81	n/a	
NB19-471 AZ 90 dip-65	80.77	143.26	62.49	0.49	n/a	
NB19-472	62.48	80.77	18.29	0.79	n/a	
AZ 90 dip-70	102.11	105.16	3.05	0.49	n/a	
	111.25	117.35	6.1	0.36	n/a	
	158.5	169.16	10.66	0.48	n/a	
	173.74	178.31	4.57	0.59	n/a	
NB19-473	71.63	94.49	22.86	0.89	n/a	
AZ 90 dip-60	111.25	114.3	3.05	0.31	n/a	
	143.26	156.97	13.71	0.81	n/a	
NB19-474 AZ 090 dip-63	54.86	76.20	21.34	0.68	n/a	***WSB Target***
	80.77	88.39	7.62	0.42	n/a	
	141.73	147.83	6.10	1.41	n/a	*NW Structure Target*
inc	*143.26*	*146.30*	*3.04*	*2.23*	*n/a*	*1 g/t cut*
	163.07	169.16	6.10	0.40	n/a	
NB19-475 AZ 090 dip-55	56.39	80.77	24.38	0.31	n/a	***WSB Target***

Drill Hole #	from (m)	to (m)	Interval (m)	Gold (g/t)	Silver (g/t)	Comment
NB19-476 AZ 090 dip-70	153.92	163.07	9.15	0.50	n/a	***North JJ Target***
	175.26	182.88	7.62	0.68	n/a	
	251.46	268.22	16.76	0.51	n/a	
NB19-477 AZ 090 dip-55			Lost hole no results			***North JJ Target***
NB19-478 AZ 090 dip-60			Lost hole no results			***North JJ Target***
NB19-479 AZ 090 dip-70	92.96	96.01	3.05	0.37	n/a	***East SB Extension***
	126.49	138.68	12.19	0.34	n/a	
	178.31	187.45	9.14	0.55	n/a	
	251.46	271.27	19.81	0.40	n/a	
	288.04	292.61	4.57	0.39	n/a	
	310.90	315.47	4.57	0.33	n/a	
	324.61	329.18	4.57	0.37	n/a	
	332.23	335.28	3.05	0.33	n/a	
NB19-480 AZ 090 dip-55	6.10	18.29	12.19	0.37	n/a	***WSB Target***
	24.38	30.48	6.10	0.31	n/a	
	35.05	41.15	6.10	0.31	n/a	
	47.24	51.82	4.58	0.45	n/a	
	56.39	83.82	27.43	0.47	n/a	
NB19-481 AZ 090 dip-50	24.38	27.43	3.05	0.49	n/a	***WSB Target***
	57.91	71.63	13.72	0.93	n/a	
NB19-482 AZ 270 dip-50	33.53	67.06	33.53	0.39	n/a	***WSB Target***
	3.05	111.25	108.20	0.26	n/a	*Low-grade (0.1 cut)*
NB19-483 AZ 270 dip-50	*6.10*	*112.78*	*106.68*	*0.32*	*n/a*	***WSB Target*** *Low-grade (0.1 cut)*
	21.34	33.53	12.19	0.41	n/a	
	60.96	65.53	4.57	0.43	n/a	
	71.63	82.30	10.67	0.55	n/a	

Drill Hole #	from (m)	to (m)	Interval (m)	Gold (g/t)	Silver (g/t)	Comment
	91.44	106.68	15.24	0.46	n/a	
NB19-484 AZ 090 dip-60	*65.53*	*195.07*	*129.54*	*0.46*	*n/a*	***WSB Target*** *Low-grade (0.1 cut)*
	68.58	108.20	39.62	0.61	n/a	
	115.82	147.83	32.01	0.52	n/a	
	155.45	176.78	21.33	0.57	n/a	
NB19-485 AZ 088 dip-55	117.35	141.73	24.38	0.77	n/a	***North SB***
inc	*121.92*	*132.59*	*10.67*	*1.05*	*n/a*	*Deep SB Main Target 1 g/t Cut*
	227.08	233.17	6.09	0.40	n/a	
	252.98	259.08	6.10	0.30	n/a	
NB19-487 AZ 089 dip-60	74.68	96.01	21.33	1.02	n/a	***Cat Hill***
inc	*74.68*	*77.72*	*3.05*	*1.37*	*n/a*	*Stockwork Zone 1 g/t Cut*
inc	*88.39*	*92.96*	*4.57*	*2.47*	*n/a*	*Stockwork Zone 1 g/t Cut*
	292.61	310.90	18.29	0.25	n/a	

11. SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 Introduction

This section summarizes Quality Assurance and Quality Control ("QA/QC") data related to drill hole sample assaying carried out for the Sierra Blanca, YellowJacket, Savage Valley, Jolly Jane and Mayflower deposits by Corvus.

11.2 NBP QA/QC Program

11.2.1 Drilling and Sampling

Corvus drilling between 2007 and 2020 included 327 reverse circulation (RC) holes and 123 core holes with a total length of 99,551 meters.

Reverse circulation samples are collected at continuous 5-foot (1.52 meter) intervals starting from the top of each hole. Two duplicate samples for each interval are captured in large sample bags placed in 5-gallon buckets. The custom-made, heavy duty, sample bags have a white barcode tag for the samples going to the assay lab, and a red barcode tag for the duplicate samples being kept for other purposes (field duplicates, metallurgical testing, etc.). The sample hose and rotary splitter are cleaned thoroughly with a high-pressure water sprayer prior to drilling of each 20-foot rod (6.1 meter). In order to minimize contamination between 5-foot intervals, the splitter is also quickly sprayed out after each interval is drilled, but before the sample bags are pulled, without stopping drill penetration. Individual samples bags are tied-off without pouring off the contained water and placed in orderly rows at the drill site for natural decanting of the excess water. The sampling associated with reverse circulation drilling is supervised by an on-site Corvus rig geologist.

Within 3-5 days the samples are sufficiently dry to allow transport. The samples are loaded into super sacks (bulk bags) and transported to a staging area at Corvus' core shack/field office. Pre-selected blanks and reference standards are placed inside the super sack, and it is sealed with a large, numbered plastic zip-tie. The super sacks are stored in a secure area until they are loaded onto the assay lab truck. The assay lab truck comes to the project for sample pick-ups on an as-needed basis. Chain of custody is transferred to the assay lab personnel at pick-up time.

The drill chips are cursorily logged at the drill site, and later logged in greater detail in an office setting using a binocular microscope. Magnetic susceptibility and an HCL acid "fizz-test" are also measured on the chips for each five-foot interval. The geologic characteristics that are determined routinely on drill chips include: color, lithology and stratigraphic unit assignment, alteration style and intensity, vein type and percentage, sulfide type and percentage, and oxide type and relative intensity. The following five oxide classes are used to quantify the oxidation state of each sample:

Class 1: Unoxidized; total sulfide, no oxide present

Class 2: Mostly unoxidized; sulfide with minor oxide present

Class 3: Mixed oxide/sulfide; generally, both oxide and sulfide in nearly equal proportions

Class 4: Mostly oxidized; oxidized with minor fresh sulfide present

Class 5: Completely oxidized; total oxide; no sulfide present

Oxide classes 5, 4 and 3 have consistently yielded favorable gold recoveries in cyanide solubility tests (see Section 13). Resource model blocks assigned to classes 5, 4 and 3 comprise the oxide mineralization category. Oxide classes 2 and 1 have consistently yielded un-favorable gold recoveries in cyanide solubility tests. Model blocks assigned to oxide classes 2 and 1 comprise the sulfide mineralization category.

Core was drilled and extracted using triple-tube tooling to ensure the best recovery through highly fractured intervals. Triple tube tooling minimizes core separation and rotation within the extraction tube. Selected lengths of each hole were sampled with continuous intervals based on careful logging of geological characteristics. In conjunction with the logging, sample intervals were marked in the core box and assigned unique sample numbers in a sequence that included pre-selected QA/QC samples every tenth sample. Each hole starts with a blank QA/QC sample, and alternates between blanks and reference standards. Once a hole is logged and tagged for sampling, each box is photographed within a fabricated lighting and reference frame. The reference frame allows rectification of the image so that in future applications true lengths can be measured on the core using the photos. Once a hole, or a group of boxes in a hole, are photographed, the photos are reviewed for adequacy and the photo files renamed using hole number and box number. All sample intervals of core are cut in half with a core saw, with one half of the core being placed in a sample bag to be assayed, and the other half returned to the original core box for archive.

From 2007-2017, samples were sent to ALS Minerals Laboratories ("ALS Minerals") and from 2019-2020, samples were sent to American Assay Laboratories (AAL).

11.2.2 Accredited Laboratories

Assaying for the NBP from 2007-2017 was performed by ALS Minerals primarily in Reno, Nevada, with some work performed in Vancouver, British Columbia. Corvus has no relationship with ALS Minerals beyond being a customer for analytical services and it is considered an independent lab. The Reno laboratory is Standards Council of Canada, Ottawa, Ontario, Accredited Laboratory No. 660 and conforms with requirements of CAN-P-1579, CAN-P-4E (ISO/IEC 17025:2005). The North Vancouver, British Columbia laboratory is Standards Council of Canada, Accredited Laboratory No. 579 and conforms with requirements of CAN-P-1579, CAN-P-4E (ISO/IEC 17025:2005).

Assay for the NBP from 2019-2020 was performed by AAL in Sparks, Nevada. Corvus has no relationship with AAL beyond being a customer for analytical services and it is considered an independent lab. The Sparks laboratory is Standards Council of Canada, Ottawa, Ontario, Accredited Laboratory No. 536 and conforms with requirements of CAN-P-1579, CAN-P-4E (ISO/IEC 17025:2005).

Check assaying was performed by a variety of labs; Bureau Veritas Labs, formerly, Inspectorate America Corporation, Sparks, Nevada before 2017; ALS Minerals Reno, Nevada in 2019-2020. Corvus has no relationship with Bureau Veritas Labs (BV) beyond being a customer for analytical services, and it is considered an independent lab. The BV Laboratory is Accredited Laboratory No. 720 and conforms to requirements of CAN-P-1579, CAN-P-4E (ISO/IEC 17025:2005).

11.2.3 Assay Analytical Methods

A variety of assay methods have been utilized throughout the years at the NBP. Table 11-1 shows the gold methods used and its associated limit of detection. Gold assay methods changed as methods improved and provided better accuracy at the lower limits and the understanding of gold concentration of the NBP evolved.

Table 11-1 Gold Analytical methods used on NBP samples

Year	Au Method	Lab	Limit of Detection (ppm)
2007	Au-AA26	ALS Minerals	0.01
2008	Au-AA26	ALS Minerals	0.01
2009	Au-AA26	ALS Minerals	0.01
2010	Au-AA24	ALS Minerals	0.005
2011	Au-AA24	ALS Minerals	0.005
2012	Au-ICP22	ALS Minerals	0.001
2013	Au-ICP22	ALS Minerals	0.001
2014	Au-ICP22	ALS Minerals	0.001
2015	Au-ICP22/Au-ICP21	ALS Minerals	0.001
2016	Au-ICP21	ALS Minerals	0.001
2017	Au-ICP21	ALS Minerals	0.001
2019	FA-PB30-ICP	AAL	0.003

11.2.4 Transport and Security

Individual RC samples were not weighed and were grouped by hole in bulk bags which were sealed with a security tag prior to shipment. Core samples are sawn in half and the halved core is weighed before being grouped by hole in bulk bags that are then sealed with a security tag prior to shipment. Each drill hole was sent to ALS Minerals in Reno, Nevada or AAL in Sparks, Nevada as a separate shipment with a chain of custody document to certify that the seals were intact when the shipment was received.

11.2.5 Duplicates

Duplicates were used to monitor the precision of the assays that were incorporated into the Mineral Resource estimate. Duplicates monitored the three sources of variation: sampling method, preparation and assaying. Preparation duplicates (Prep Duplicates) were used to monitor the sample preparation process, field duplicates were used to document the precision associated with sampling at the drill site, and pulp duplicates were used to monitor the assaying process.

Sample preparation duplicates were created by crushing the sample and then splitting it in half. The two halves were then processed as separate samples. Five Prep Duplicates were created for each drill hole. The selection of preparation duplicates was made by geologists logging the hole, based on their interpretation of lithologies and degree of mineralization.

Figure 11-1 and Figure 11-2 are graphs of the original sample assay versus the duplicate sample assay, for gold and silver, respectively. The data plotted here covers the life of the project from 2007 to present. The ±10% precision lines are shown in the graph to illustrate the trend of the data pairs. The results show that the preparation duplicates reproduced the original assay very well for both gold (coefficient of variation 20%) and silver (coefficient of variation 29%).

Figure 11-1 Preparation Duplicate Gold Assays

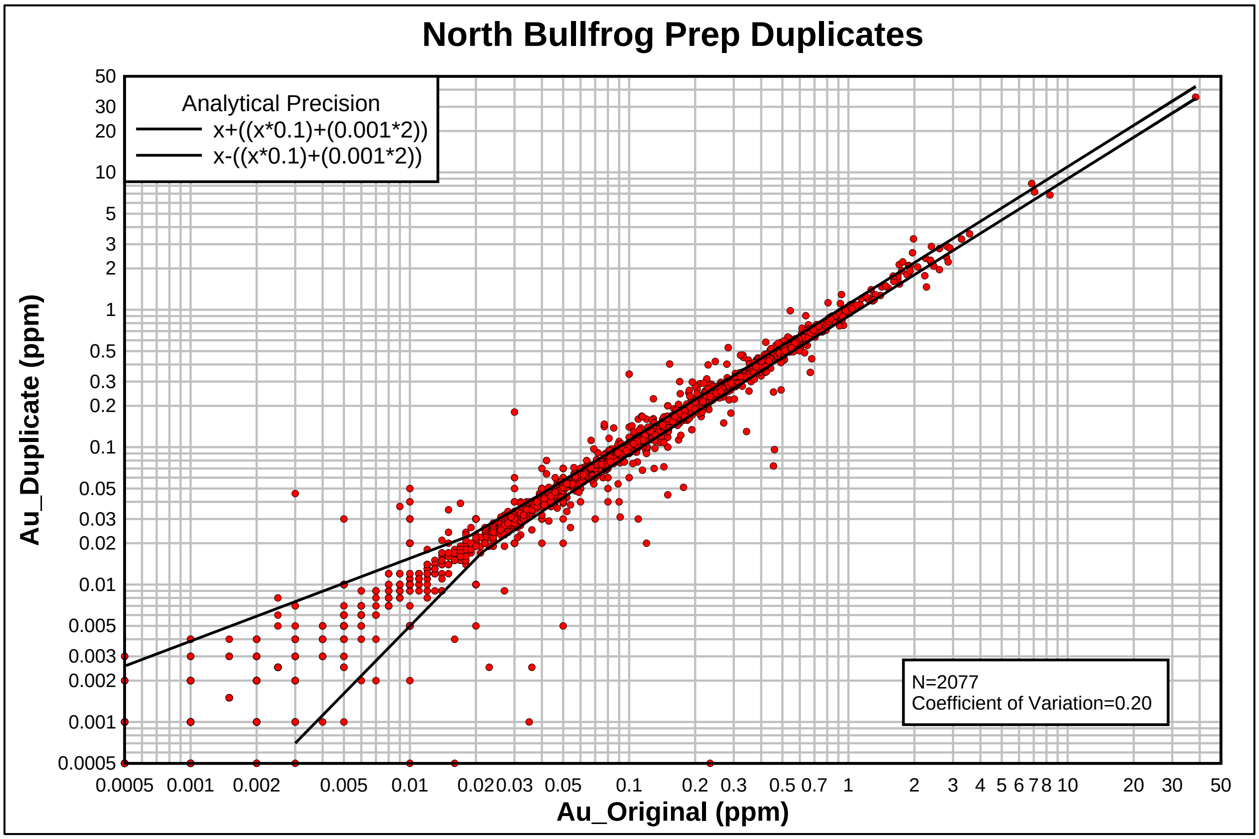
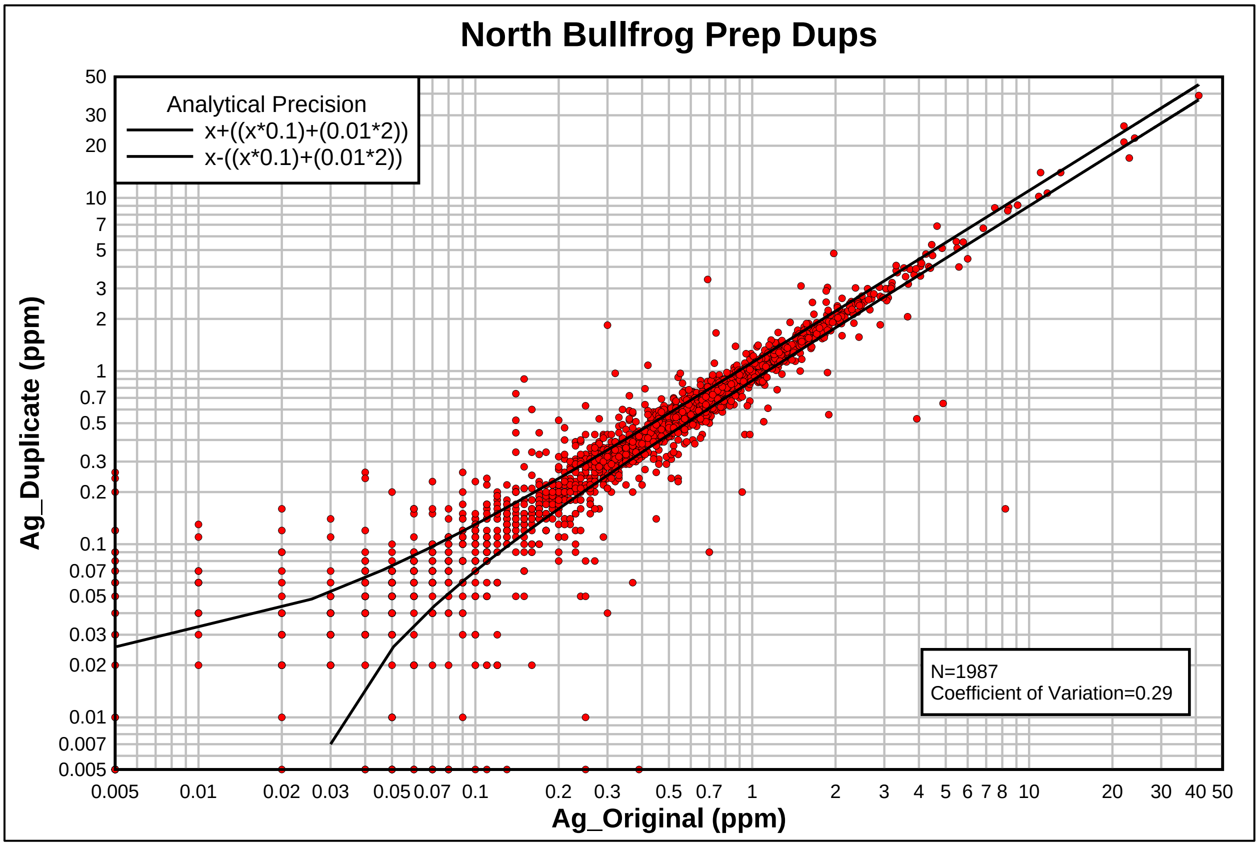


Figure 11-2 Preparation Duplicate Silver Assays



11.2.5.1 Field Duplicates

Field duplicates are selected by the Project Manager after the gold assay results for an RC hole have been received. At the drill site, two samples are taken for every five-foot interval, with the primary sample labeled with a white tag. The secondary sample is labeled with a red tag with an "M" suffix added to the original sample number. The red tag bags are used for the field duplicates. The selected field duplicates would have a gold value of 0.1ppm or higher. Field duplicates underwent the same transport and security procedure as all other RC and rock samples. The field duplicates were used to check the accuracy and precision of the sample splitting at the drill site. The data displayed here cover from 2008 to present. Figure 11-3 and 11-4 are graphs of the original assay versus the duplicate assay and show that the splitting of duplicate samples at the drill site was both accurate and precise for gold (coefficient of variation 19%) and silver (coefficient of variation 21%) content, respectively.

Figure 11-3 Field Duplicate Gold Assays

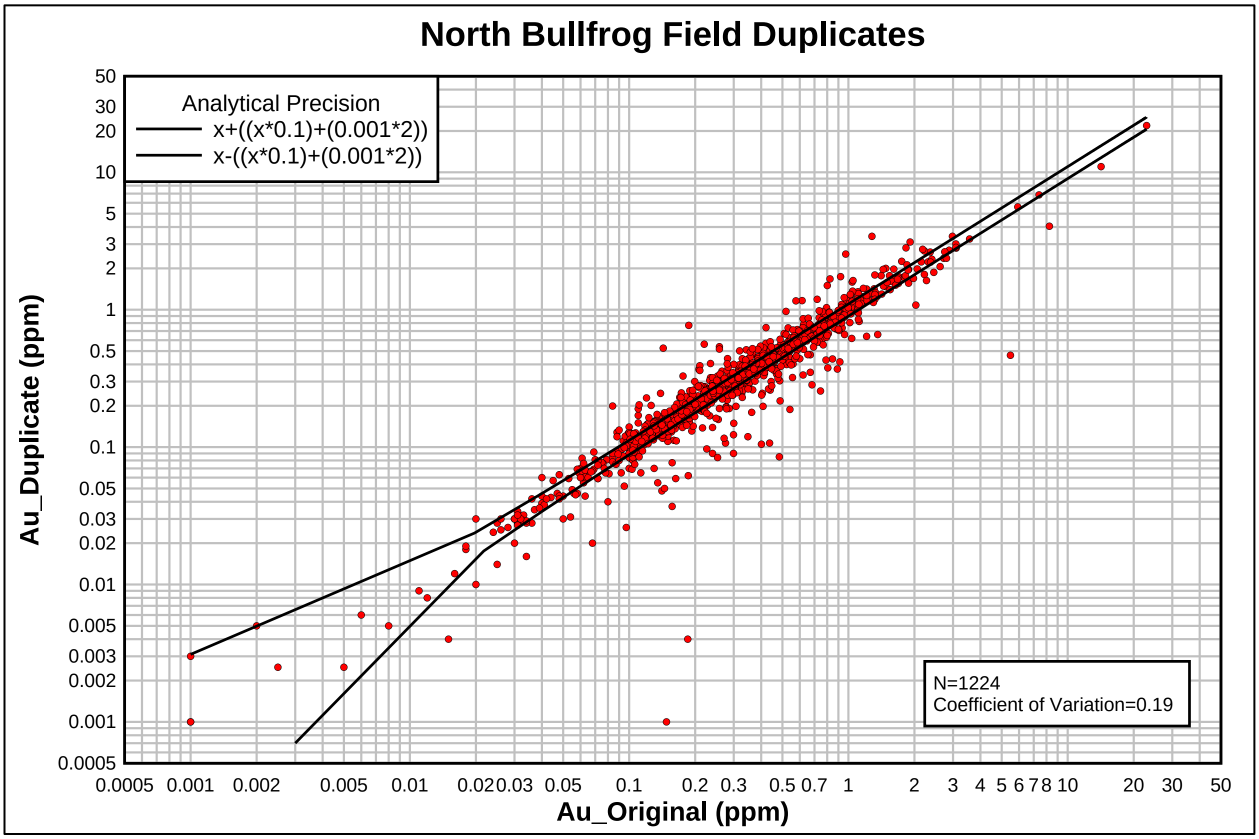
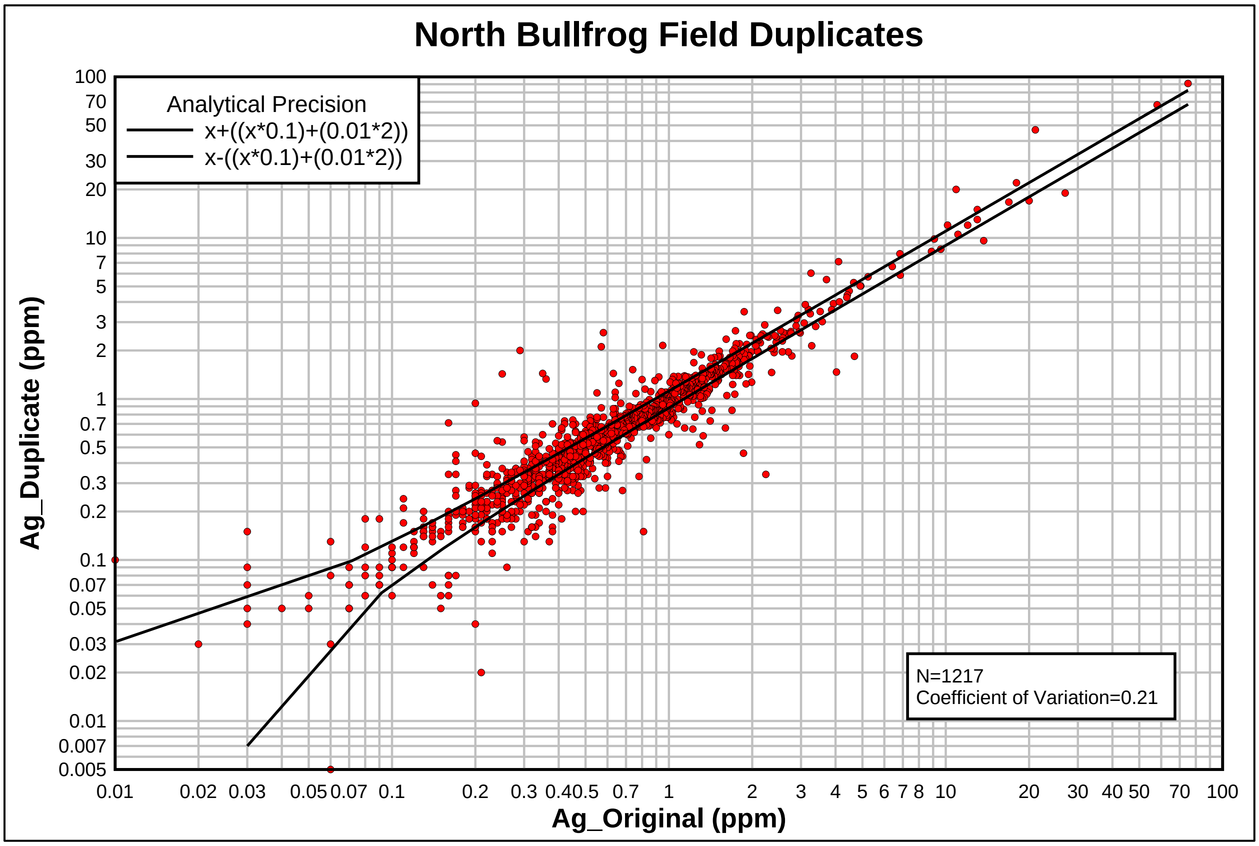


Figure 11-4 Field Duplicate Silver Assays



11.2.5.2 Pulp Duplicates

Pulp duplicates reflect the homogeneity of the pulp material that is subjected to the fire assay and variations generally reflect the nugget effect in gold samples. In this instance ALS Minerals and AAL routinely run pulp duplicates as part of their internal QA/QC program and these assays were reported as part of the assay package.

Figure 11-5 and Figure 11-6 show that the gold and silver assay values for AAL and ALS internal pulp duplicates reproduced accurately for gold (coefficient of variation 18%) and silver (coefficient of variation 13%), respectively. The data displayed in Figure 11-5 for gold covers the life of the project from 2007 to present. The data displayed in Figure 11-6 for silver covers 2014 to present.

Figure 11-5 Pulp Duplicate Gold Assays for NBP

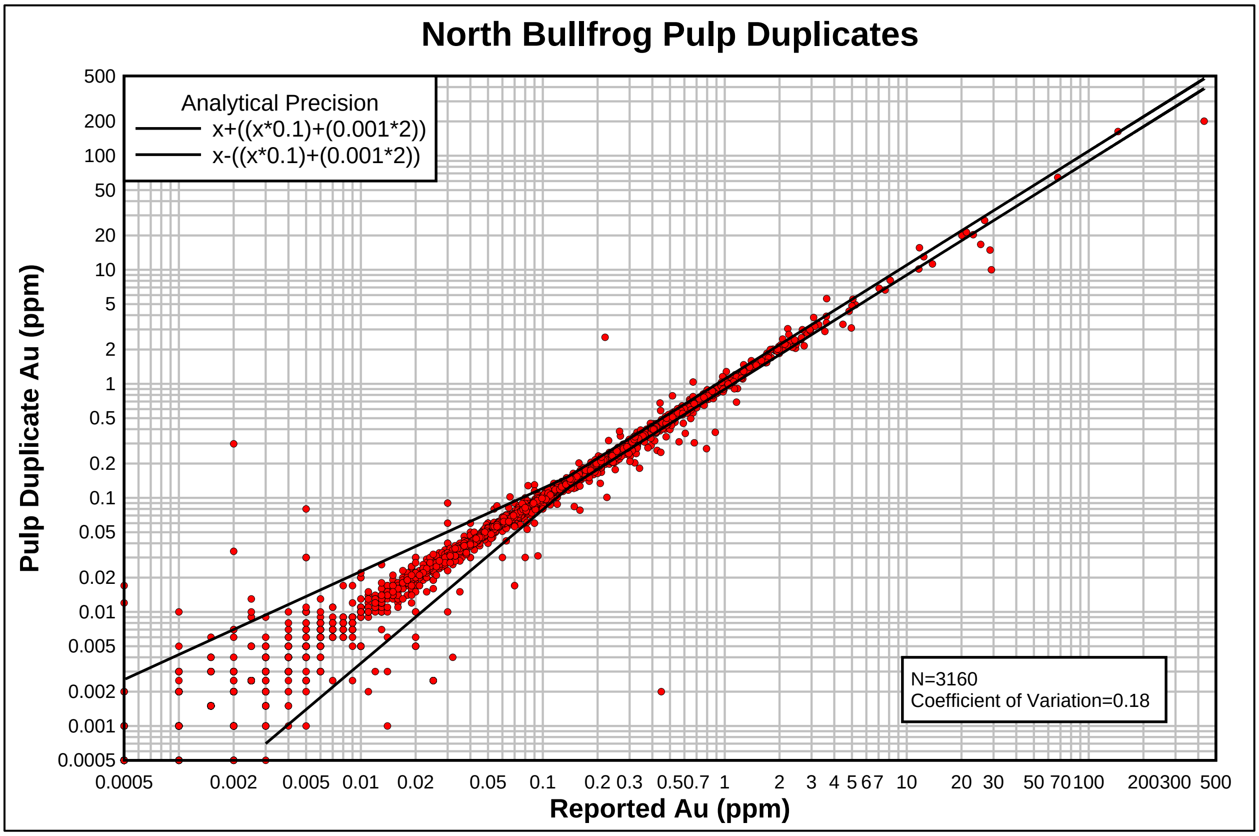
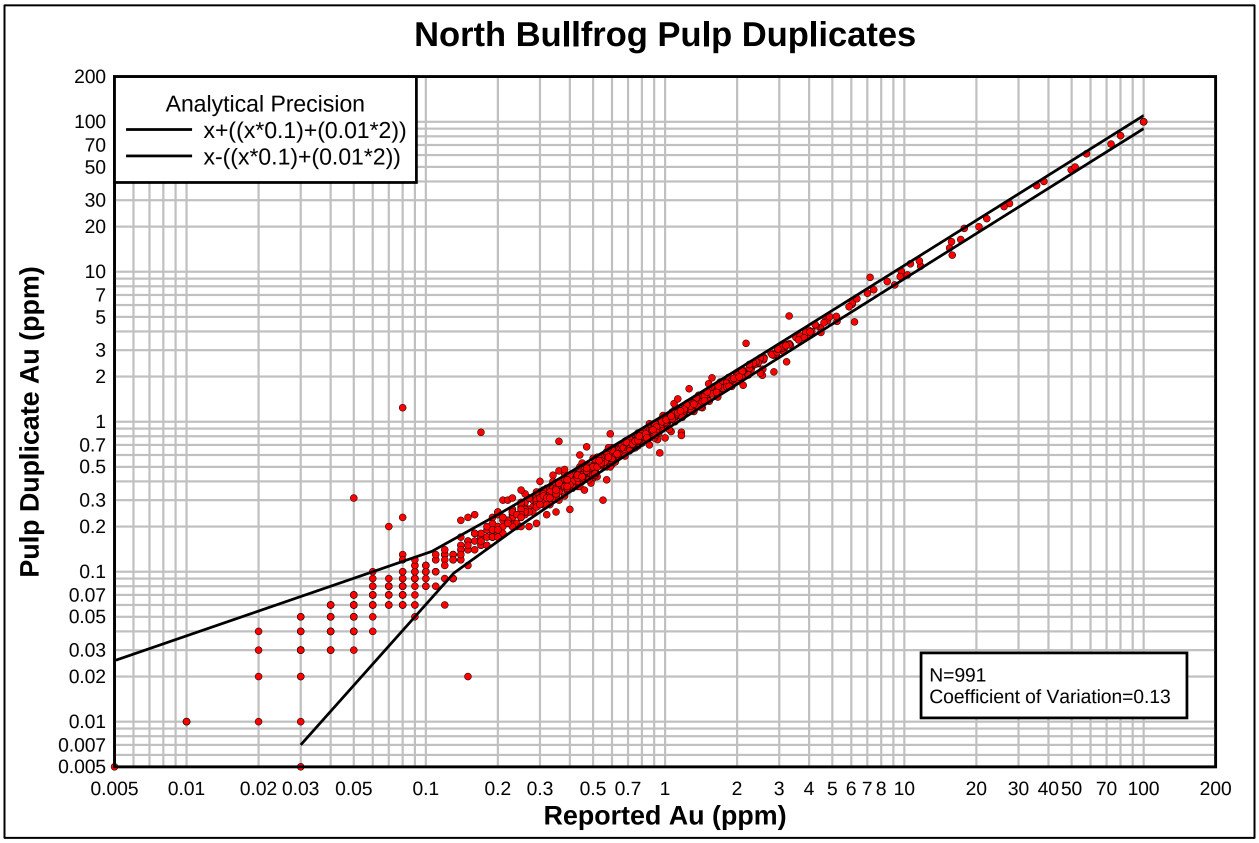


Figure 11-6 Pulp Duplicate Silver Assays for NBP



11.2.6 Check Assays

Three hundred and sixty-four (364) samples were sent to Inspectorate America Corporation to check results from ALS and one hundred and twelve (112) samples were sent to ALS to check results from AAL. Figure 11-7 plotted all the check assay results for gold, and Figure 11-8 plots all the check assays for silver and show that there was very good agreement between the ALS and Inspectorate values and AAL and ALS values, for gold and silver, respectively. The data displayed covers the period 2012 to present.

Figure 11-7 Comparison of Gold Analyses in Duplicate NBP Samples

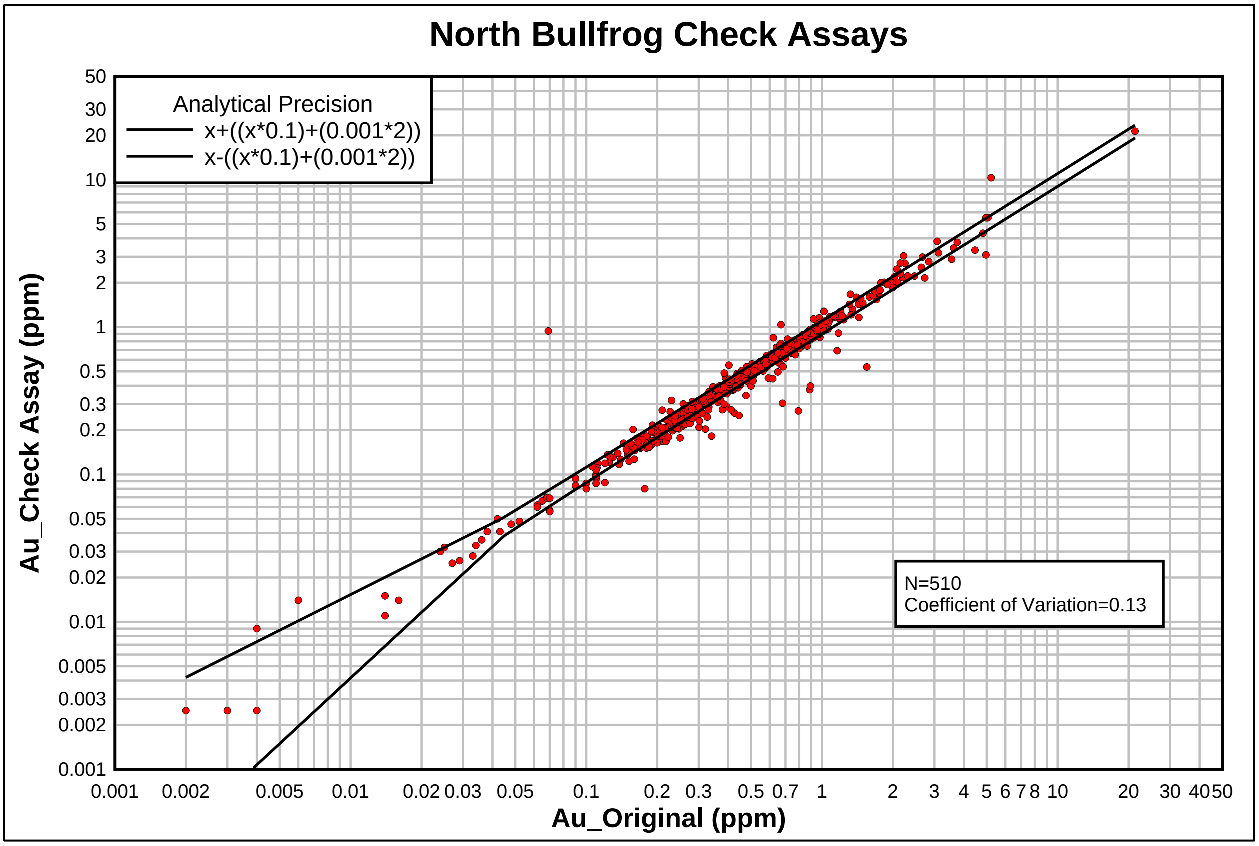
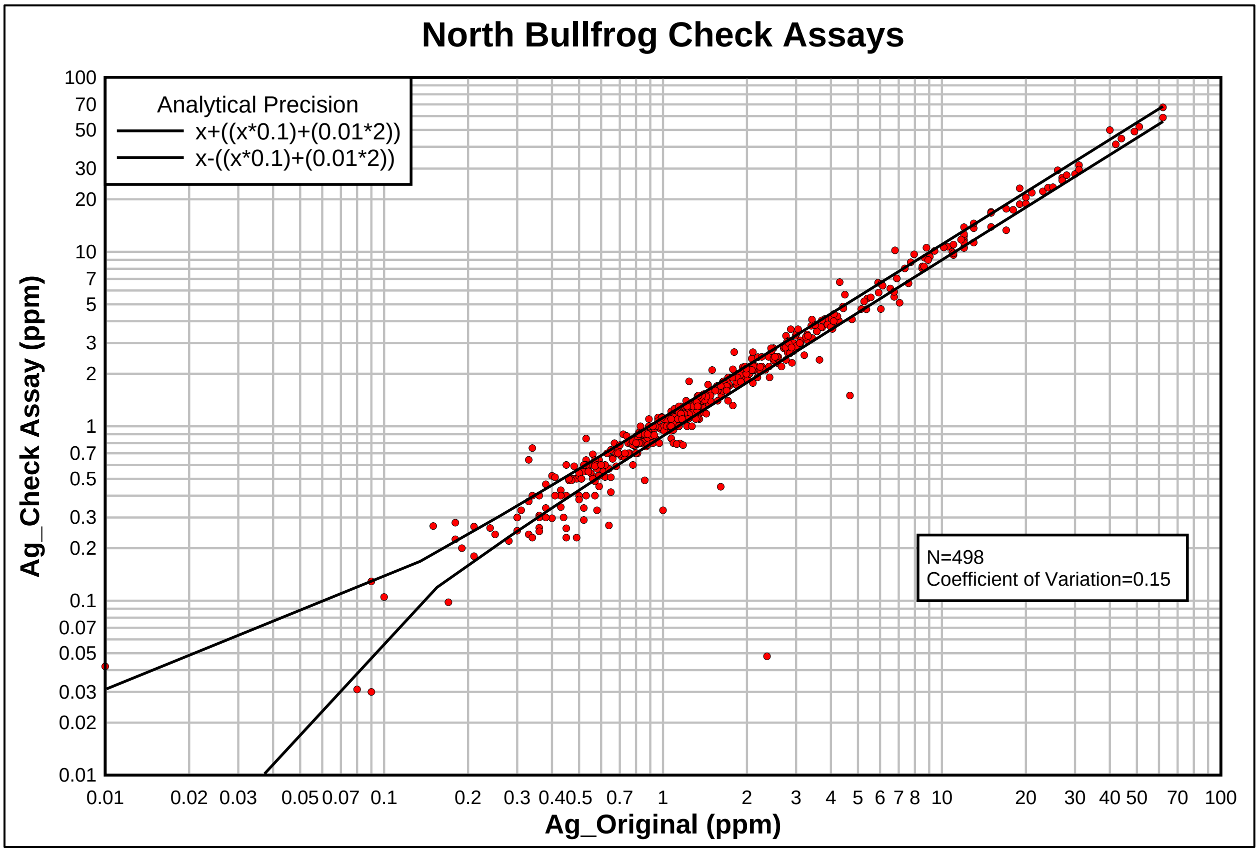


Figure 11-8 Comparison of Silver Analyses in Duplicate NBP Samples

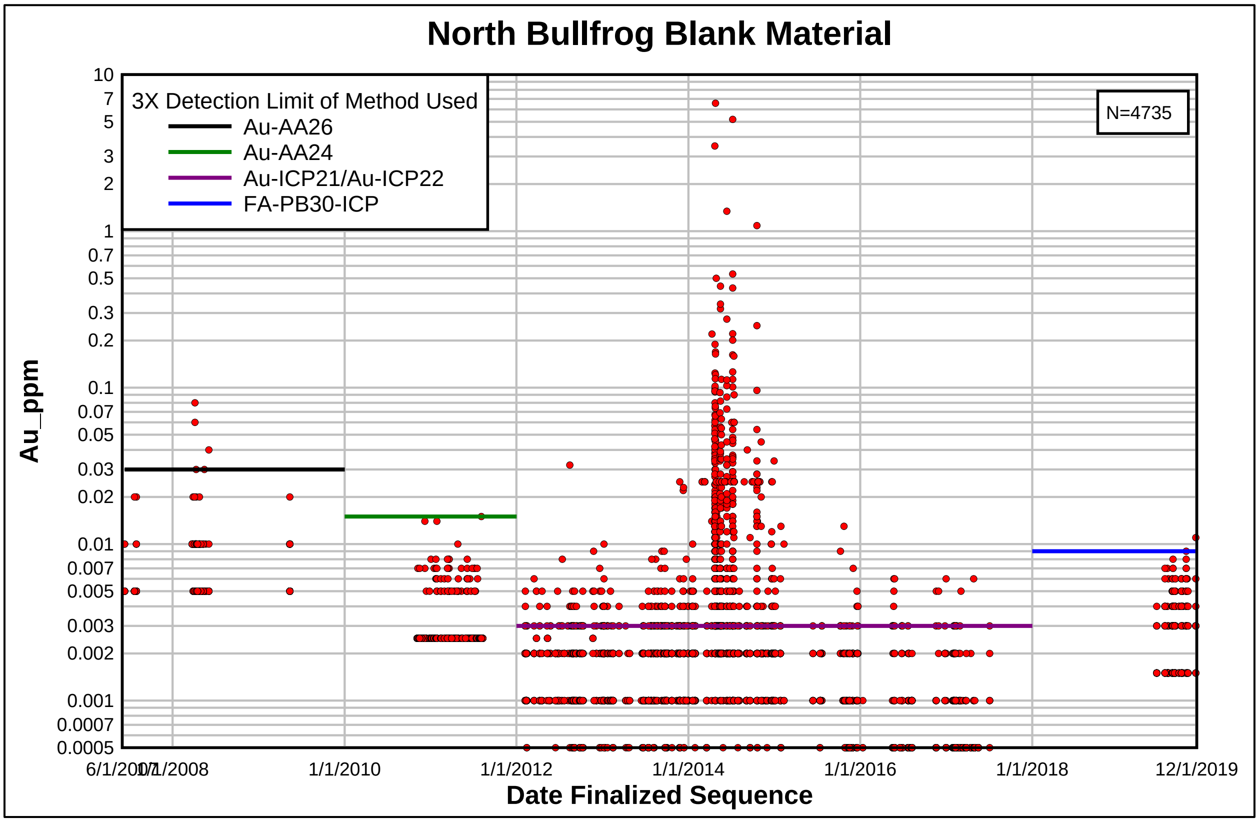


11.2.7 Blanks

Blank samples were inserted into the sample sequence at a ratio of 1:20 to monitor for carryover contamination and to ensure that there is not a high bias in the assay values. Carryover is a process where a small portion of the previous sample contaminates the next sample. ALS Minerals allows a total of 1% carryover from preparation and analytical processes combined. Each blank that assays higher than three times the detection limit was evaluated to see if the value reflected carryover or some other problem. For example, if a blank assayed 0.006ppm Au for the Au-ICP22 method and the previous sample ran 1ppm Au then the blank was not investigated because acceptable carryover could explain up to 0.01ppm. However, if the blank had assayed 0.015ppm Au which was more than could be explained by carryover from a 1ppm previous sample then an investigation was initiated. The investigation included a rerun of the blank and surrounding samples, as well as any documentation that was associated with the work order at ALS Minerals or AAL. There were cases where the investigation did not resolve the reason for the higher than expected value. Figure 11-9 shows the performance of blank samples submitted for the NBP Quality Control program.

In 2014, a number of high-grade samples were analyzed and, in an effort to minimize carryover, quartz blank material was inserted in between these high-grade samples and then analyzed. The spike in 2014 in Figure 11-9 are these quartz blanks that were analyzed. All the blanks were within the allowable 1% carryover.

Figure 11-9 Sierra Blanca Blanks-All Methods



11.2.8 Certified Reference Materials

Certified Reference Materials ("CRMs" or "standards") were used to monitor the accuracy of the assay results reported by ALS Minerals. CRMs were inserted into the sample sequence at a ratio of 1:20 and served to monitor both accuracy and sample sequence errors. A number of different CRMs covering a range of grades and mineral compositions were used at the NBP. Each CRM comes with a certified concentration with a stated uncertainty. However, the precision on the assay is ultimately controlled by the 10% analytical precision reported by ALS Minerals or AAL. Therefore, in the following discussion the performance of the CRMs was discussed relative to the specified ALS Minerals and AAL precision.

CRMs used throughout the drilling campaign were analyzed using the Au-ICP22, Au-ICP21 and FA-PB30-ICP analytical methods. All the CRM values fell within the theoretical analytical precision quoted by ALS Minerals and AAL as shown in Figure 11-10. Figure 11-11 illustrated that a range of concentrations are being monitored with the CRM suite used at the NBP was greater than the generally expected assay values.

When CRMs assay outside of the theoretical analytical precision, an investigation is launched to find a potential cause and make sure surrounding samples were not affected but the cause of the “failure”. Typically, a rerun of samples and the CRM is included in the investigation. One standard, as seen as an outlier in Figure 11-10, was not investigated but comments were made indicating there may have been a conversion factor error in the assay reported.

Figure 11-10 Certified Reference Material Gold Assays

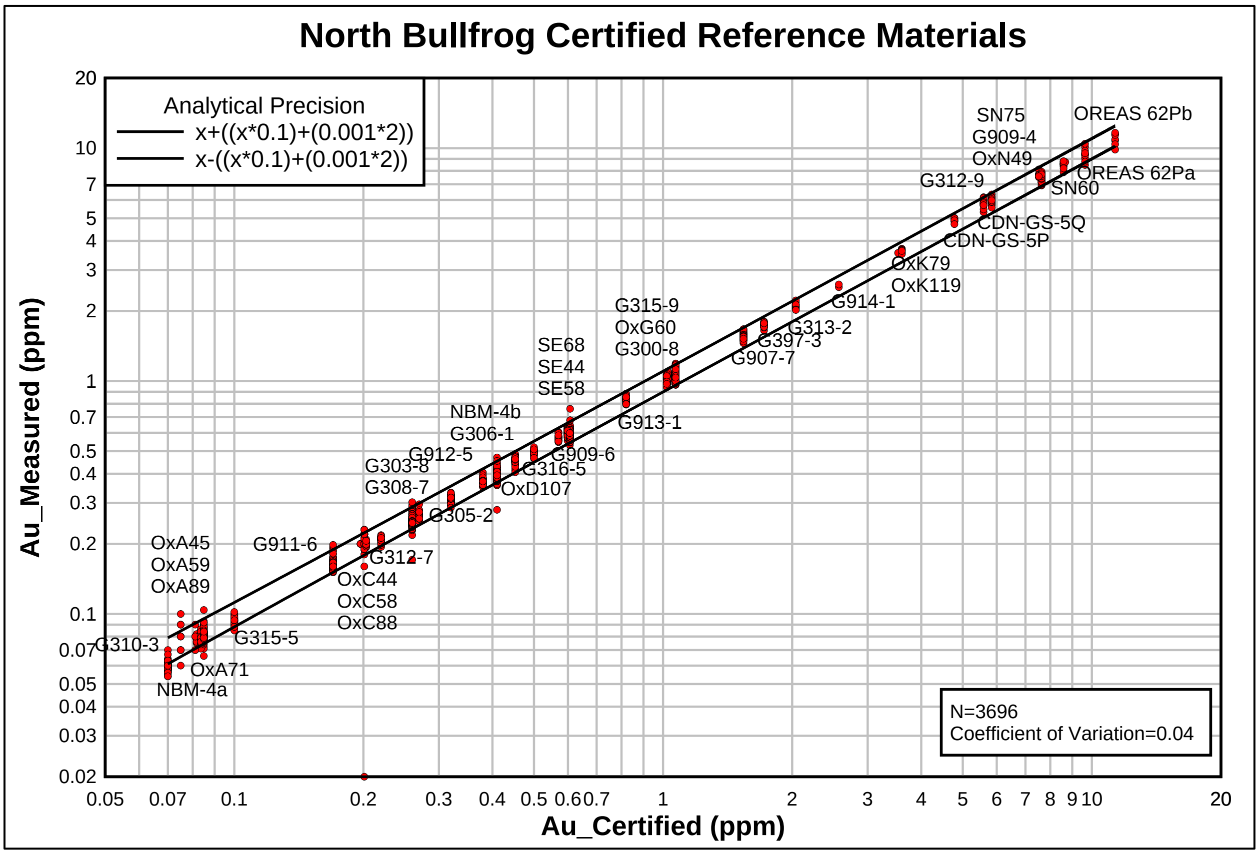
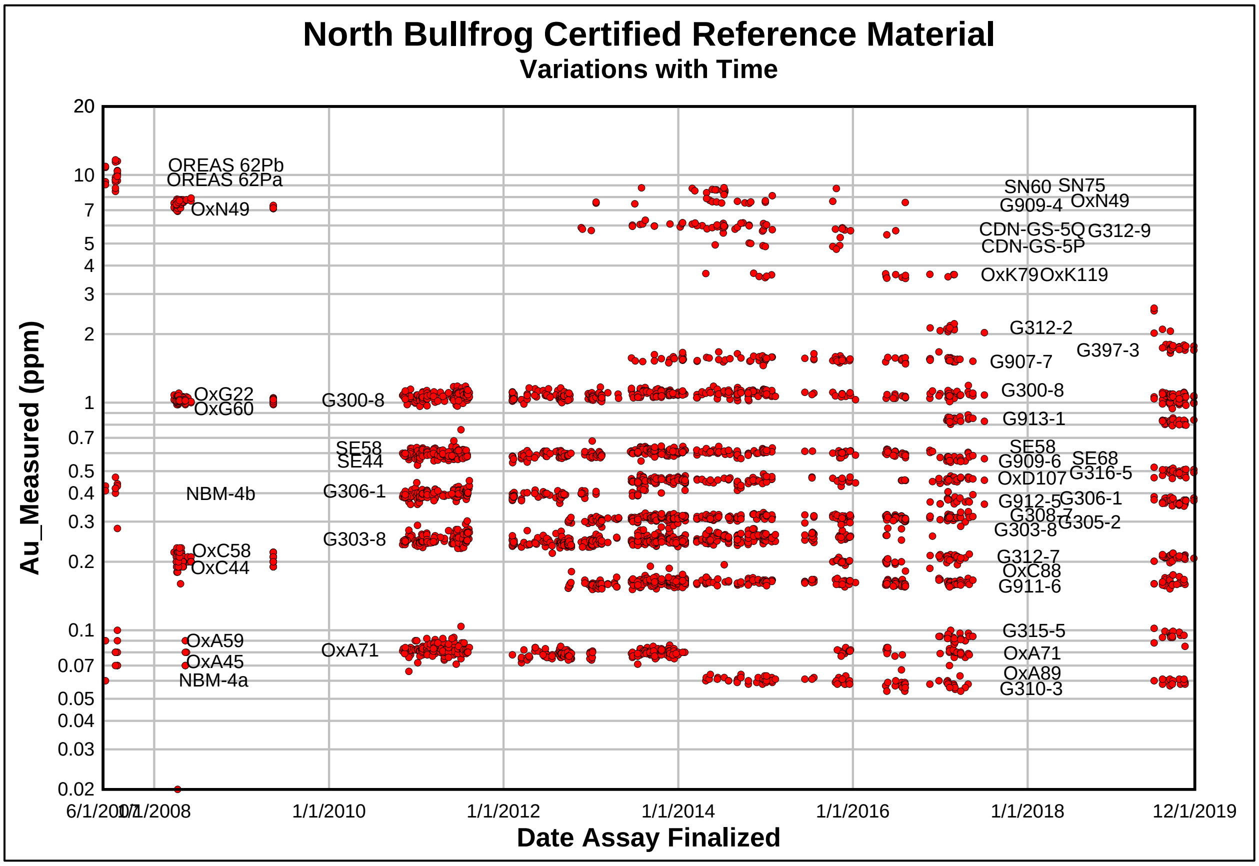


Figure 11-11 Performance of CRMs Over Time



Some of the CRMs used by Corvus have certified silver values while others that are certified for gold only have "reported" silver values. Nevertheless, these values were used to monitor the accuracy and precision of the silver assays as shown in Figure 11-12. It was clear that most of the CRM silver values reported within the analytical precision of the ME-MS61 and ICP-61-UT method. There were clearly some problems with precision in the CRMs with values less than 0.1ppm silver. Figure 11-13 illustrates that the range of concentrations that were monitored by the CRM suite used at the NBP covered the range of expected assay values.

Figure 11-12 Silver Assays for CRMs

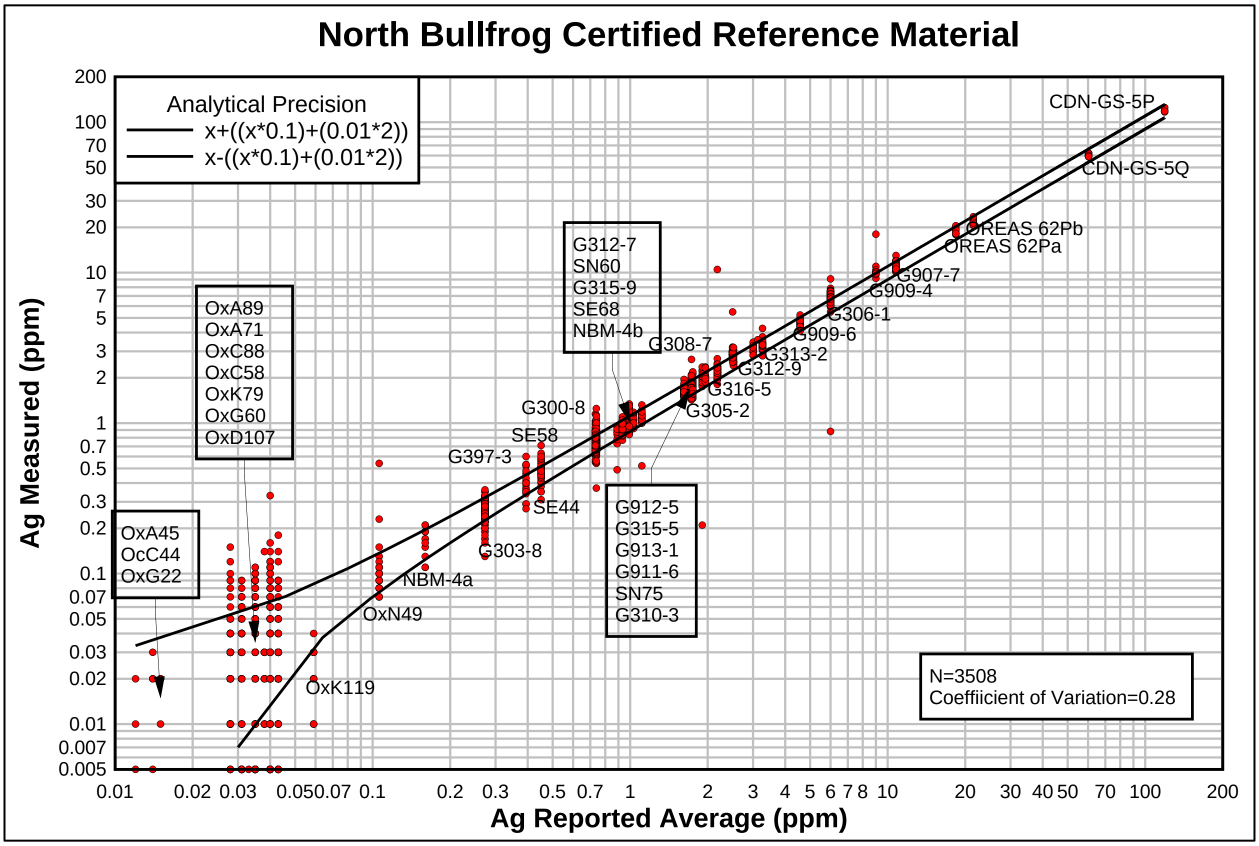
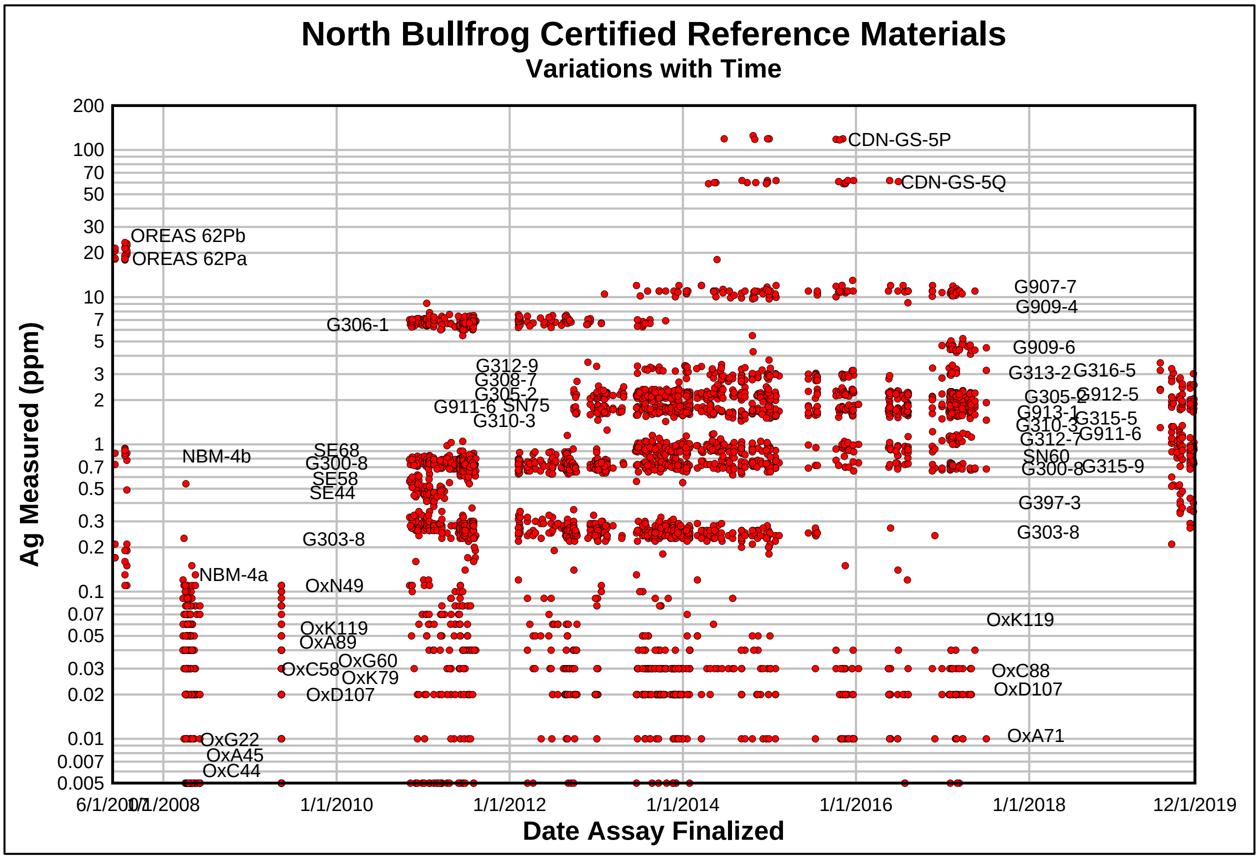


Figure 11-13 Silver Performance of CRMs Over time



11.3 Data Adequacy

In the opinion of the principal author of this Technical Report, Mr. Scott Wilson, C.P.G., SME, sample preparation, security and analytical procedures as described in this Section 11 are adequate and can be relied upon in the Mineral Resource estimate and for the PEA, each as described herein.

12. DATA VERIFICATION

The principal author of this Technical Report, Mr. Scott Wilson, C.P.G., SME, has verified the data used in this Technical Report by:

- Visiting the Project to confirm the geology and mineralization
- Visits to the core and RC storage areas and inspection of the core cutting facility
- Review of the drill core in the logging facility
- Verifying the location of drillholes in the field
- Reviewing the QA/QC protocols

The principal author of this Technical Report, Mr. Scott Wilson, C.P.G., SME, concluded that:

- Exploration drilling, drillhole surveys, sampling, sample preparation, assaying, and density measurements have been carried out in accordance with CIM Best Practice Guidelines and are suitable to support the Mineral Resource estimates and PEA contained herein
- Exploration and drilling programs are well planned and executed and supply sufficient information for Mineral Resource estimates and Mineral Resource classification and PEA contained herein
- Sampling and assaying include sufficient quality assurance procedures.
- Exploration databases are professionally constructed and are sufficiently error free to support Mineral Resource estimates and PEA contained herein

Therefore, in the opinion of the principal author of this Technical Report, Mr. Scott Wilson, C.P.G., SME, the data is adequate and can be relied upon to estimate Mineral Resources at the NBP and for the purposes of the PEA as described in this Report.

12.1 DATABASE ERROR CHECKS

The drill database was reviewed by the principal author of this Technical Report, Mr. Scott Wilson, C.P.G., SME by selecting approximately 10% of the gold sample records in the database. The certified assay certificates were crosschecked with the data entry in the database. The data entry procedures have been verified by Mr. Wilson and are accurate as compared to the certificates.

12.2 DATA VERIFICATION SAMPLES

The principal author of this Technical Report, Mr. Scott Wilson, C.P.G., SME, has independently collected field duplicates during visits to the site in 2015 and 2017 and has submitted them for laboratory analysis at the ALS Laboratory in Reno, NV and to Inspectorate America Corporation in Sparks, Nevada to independently verify the existence of the mineralization and to review the reproducibility of the original Corvus assays. No limitations were placed on Mr. Wilson's ability to review data or to independently verify the data used in the Mineral Resource estimate and the PEA. Samples were marked by Mr. Wilson with information regarding the selected sample (Date, Sample#, Hole ID, From, To, Original Assay).

The principal author of this Technical Report, Mr. Scott Wilson, C.P.G., SME, independently collected three verification core samples. The samples were photographed, and core boxes were marked by Mr. Wilson with information regarding the selected sample (Date, MMC Sample#, HoleID, From, To). The core boxes were then taken to the core cutting facility where a technician cut the quarter core. Mr. Wilson bagged the quarter core samples in the core shed and sent them to ALS for analysis. Figure 12-1 shows the cut quarter core, an example of the sample tag placed on the core box and the sample bag used to collect the sample that was shipped to the laboratory by Mr. Wilson. The analysis results are compared to the original assay results from the initial sampling in Table 12-1.

Figure 12-1 - Core Sampling Procedure used by RDA in 2015 Data Verification

Table 12-1 - Results of Data Verification Samples Collected by RDA at NBP 2015

Corvus Sample #	MMC Sample #	Hole ID	From	To	Original Value		Verification Value		MMC Weight	ALS Weight
					Au	Ag	Au	Ag	kg	kg
P491238	MMC_SB1_141102	NB-14-392	261.71	262.43	0.973	0.77	1.155	0.79	0.89	0.9

Q795239	MMC_SB2_141102	NB-14-380	89.44	90.03	2.03	23	2.19	22.2	0.93	0.93
Q795685	MMC_SB3_141103	NB-14-384	104.55	105.63	0.485	5.9	0.438	6.65	1.62	1.63

The Table 12-1 comparison indicates that the results show that check samples grades range from 7% to 11% of the original individual sample grades.

The principal author of this Technical Report, Mr. Scott Wilson, C.P.G., SME, independently collected seven field duplicates during a visit to the North Bullfrog property on June 6-8, 2017. The two verification samples were submitted to Inspectorate America Corporation in Sparks, Nevada. The purpose of these data verification samples was to independently verify the existence of the mineralization and to review the reproducibility of the original Corvus assays. No limitations were placed on the Mr. Wilson's ability to review data or to independently verify the data used in the Mineral Resource estimate. Samples were marked by Mr. Wilson with information regarding the selected sample (Date, Sample#, Hole ID, From, To, Original Assay). The results show that check samples grades range within acceptable limits when compared to the original individual sample grades. Table 12-2 compares the results of the data verification testing for the 2017 visit to North Bullfrog. The results show that check samples grades range within acceptable limits from the original individual sample grade.

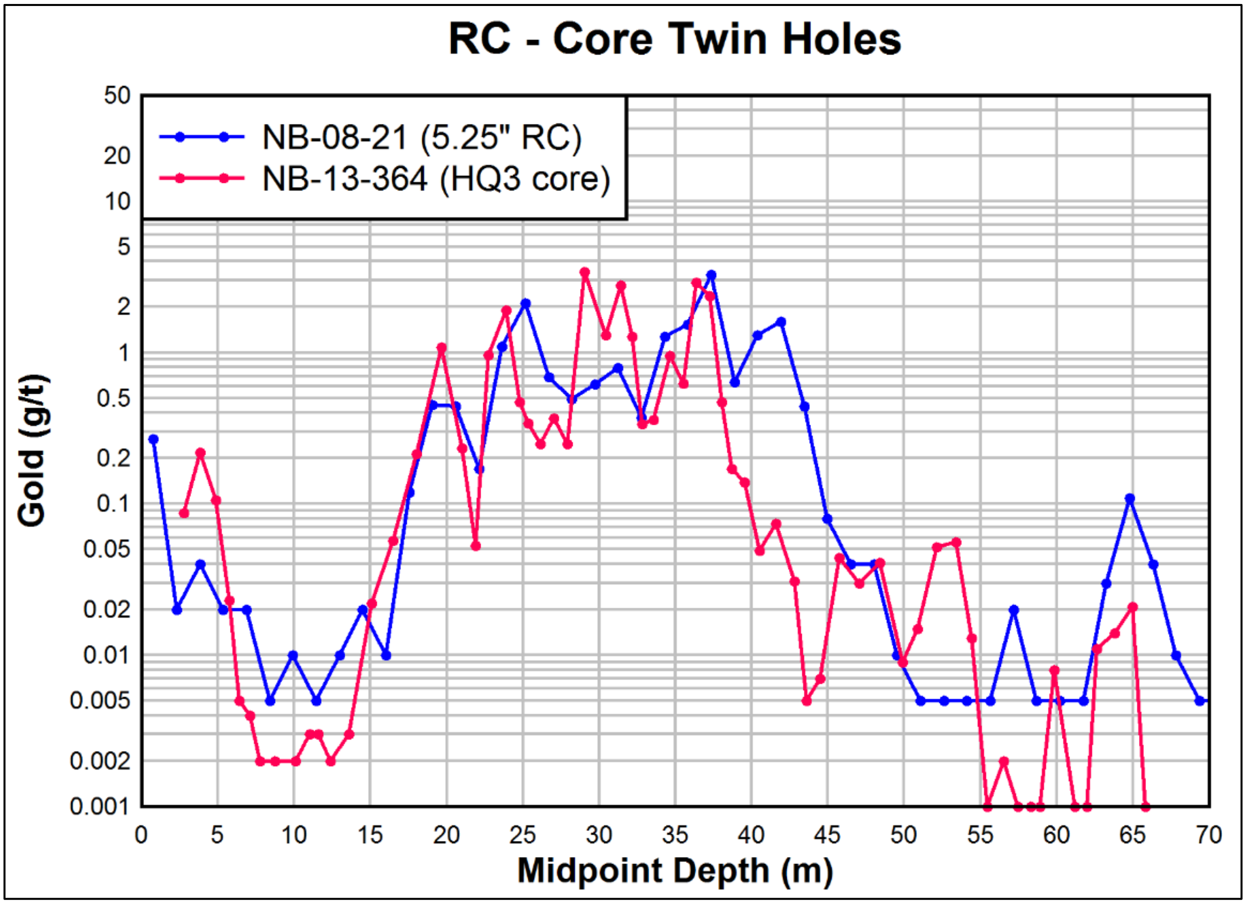
Table 12-2 - Data Verification Samples (RDA-2017)

Sample #	Hole ID	From	To	Original Value	Verification Value	Lithology
				Au	Au	
NB187273	Blank	-	-	-	-	Blank
NB187274	NB-17-329	290	295	0.722	0.767	Tsb
NB191471	NB-16-314	915	920	0.931	0.976	Tpf
NB191592	NB-16-315	860	865	0.700	0.724	Tnb
NB191667	NB-16-316	560	565	0.610	0.554	Tnb
NB191773	NB-16-318	620	625	1.070	0.718	Tpf
NB191794	NB-16-319	515	520	0.525	0.531	Tnb
NB192224	NB-16-325	595	600	0.736	0.733	Tsb
G913-1	NB-36	-	-	0.820	0.783	CRM

12.3 TWIN HOLE COMPARISONS

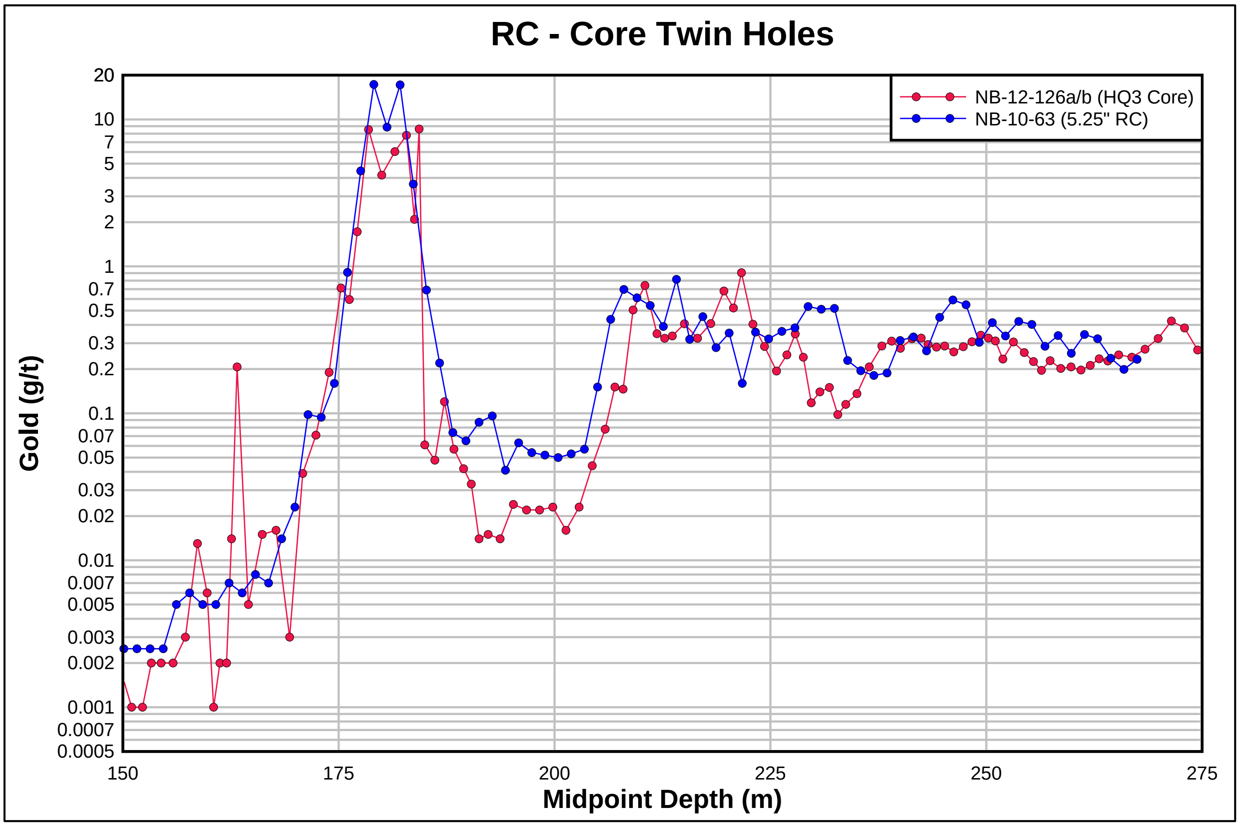
RC hole NB-08-21 was twinned with a core hole in 2013 to determine the nature of gold mineralization in the hole. The precise location of the collar for NB-08-21 could not be determined because the drill pad had been rehabilitated but the twin hole was within 4 meters of the original location. There is a good correlation between the gold values in both holes. It is noticeable that the detection limit for gold in NB-08-21 was 0.01g/t whereas in NB-13-364 it was 0.001 g/t. It is possible that the high gold peak at 30 meters in NB-13-364 could be the same structure as the high gold spike at 42 meters in NB-08-21 indicating a very oblique angle of intersection of that particular zone (Figure 12-2).

Figure 12-2 - Gold Assays for Twin Holes NB-08-21 and NB-13-364



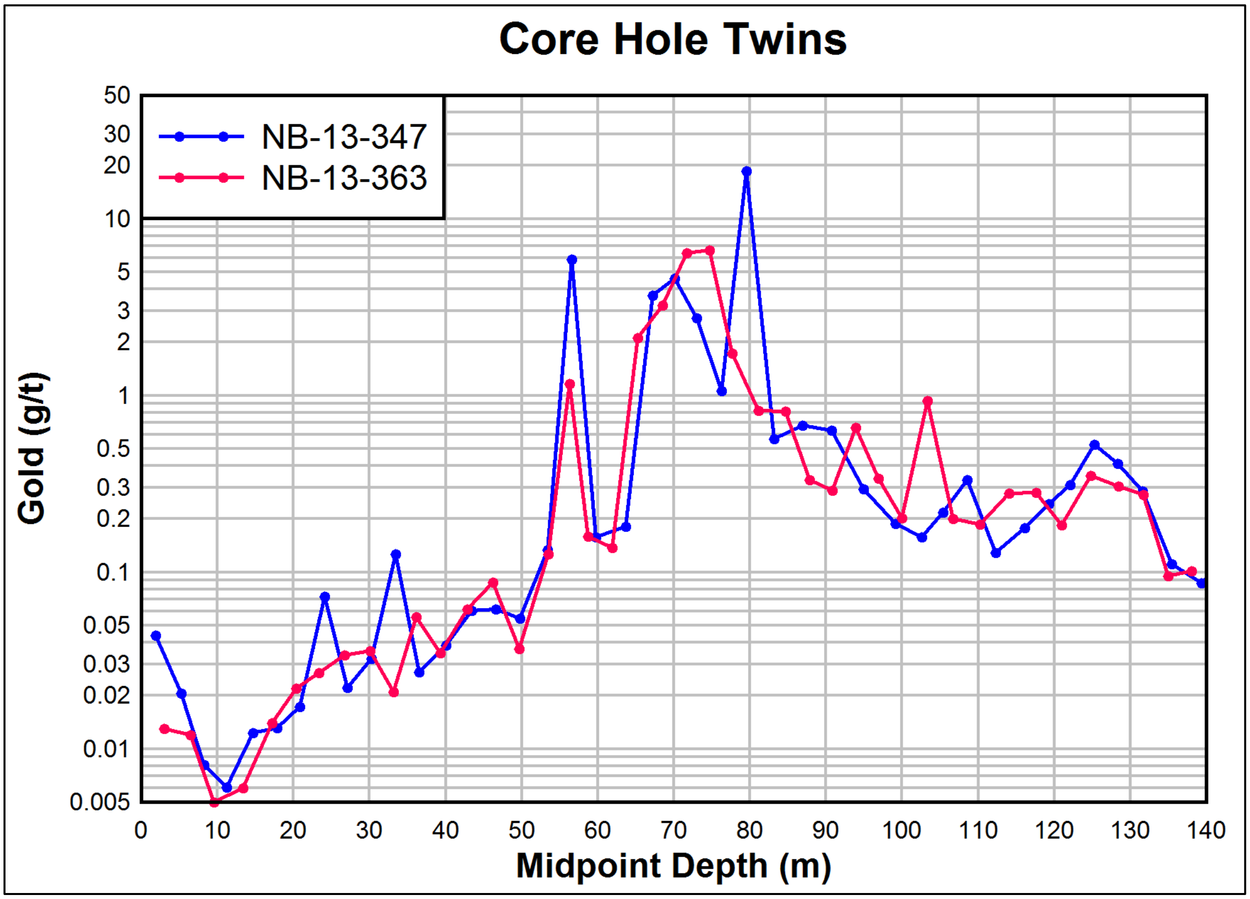
Drill hole NB-12-126 was drilled as an HQ3 core twin of RC hole NB-10-63 in order to determine the actual style of mineralization encountered in the original hole. A comparison of the two holes is shown in Figure 12-3 and indicates that the gold assays generally track each other quite well. Note that the detection limit for gold in NB-10-63 was 0.005 ppm while the detection limit for NB-12-126 was 0.001 ppm. In both holes all samples from depths less than 150 meters had values below detection. The assays from NB-12-126 do appear to have a low bias when compared to NB-10-63, particularly noticeable in the intervals 190-210 meters and 225-235 meters These intervals coincide with zones of clay alteration and faulting and it is possible that the grade bias reflects sample loss during core cutting.

Figure 12-3 - Gold Assays for Twin Core Holes NB-10-63 and NB-12-126.



Drill hole NB-13-363 is a PQ3 diameter core hole drilled as a twin of HQ3 diameter NB-13-347 in order to collect metallurgical sample material. The data are compared down-hole in Figure 12-4. The average sample length of NB-13-363 was 3.3 meters while the sample length in NB-13-347 ranges from 0.1 meter to 1.4 meters. In order to facilitate comparison between the hole's samples in hole NB-13-347 were combined into sample intervals of approximately 3 meters. It is important to understand that the original samples in NB-13-347 were taken in such a way that once the core was sawn in half the high-grade samples were never split again and the entire sample was assayed using a metallic screen fire assay. This means that the nugget effect was minimized in the sampling. In contrast, for the metallurgical samples the entire core was sampled and crushed to 19mm before a 4 kg split was taken for the head assay. The 4 kg split was then crushed to 1.7mm and 1 kg was extracted for a metallic screen fire assay. This means the metallurgical samples were certainly affected by the nugget effect during sampling. This shows that there is a good correlation overall between the two holes with higher grade spikes in the original hole almost certainly reflecting the absence of nugget effect in those samples compared to the larger metallurgical samples.

Figure 12-4 - Gold Assays for Twin Holes NB-13-347 and NB-13-363



13. MINERAL PROCESSING AND METALLURGICAL TESTING

Extensive metallurgical test work has been undertaken on various deposits at the NBP by Corvus since 2008. The results of the test work that directly relate to the processing approach described in this Technical Report are summarized in this section.

The current project envisions milling high-grade mineralization from the YellowJacket vein and vein stockwork and recovering gold with a gravity circuit. The gravity tail would be filtered and sent to a heap leach pad that would simultaneously process ROM low-grade mineralization from the YellowJacket, Sierra Blanca, Savage Valley, Jolly Jane and Mayflower deposits. The relevant milling and column leach test work was performed at Resource Development Inc. (RDi, 2020) in 2019-2020 and is also discussed in this section.

13.1 PREVIOUS METALLURGICAL TEST WORK AT NBP

Metallurgical test work undertaken includes testing for comminution, bottle roll and column leaching for the several deposits, and gravity concentration and leaching of the gravity concentrate and gravity tails during 2010-2015 which are discussed in this section.

13.1.1 COMMINUTION TEST WORK

The comminution test work consisted of Crushability Work Indices (CWi), Abrasion Indices, Bond's Ball Mill Work Indices (BMWi) and Bond's Rod Mill Work Index (RMWi). The data are presented in Tables 13-1 to 13-3. The crushability work index ranged from a low of 4.87 kwh/t to a high of 16.68 kwh/t. No CWi was obtained for Savage Valley mineralization. The average CWi for the mineralization types from the four different deposits was 11.51 kwh/t. The abrasion index ranged from a low of 0.06 to a high of 0.71. The average abrasion index was 0.411. This value indicates that the mineralized materials are generally abrasive. The average BMWi for the YellowJacket mineralization was determined to be 22.2 kWh/t and the RMWi to be 15.9 kwh/t.

Table 13-1 - CWi for Different Deposits

Mineralized Material	CW_i kWh/t		
	Measurements	Average	Design
YellowJacket	12, 3, 9.8, 7.8, 10.9, 10.5, 10.6	10.32	-
Sierra Blanca	16.68	16.68	-
Savage Valley	-	-	-
Jolly Jane	5.49	5.49	
Mayflower	5.82, 5.09, 6.51, 5.79, 4.87, 14.05, 10.42	7.51	-
Overall	-	11.51	13.34

Table 13-2 - Abrasion Index (Ai) for Different Deposits

Mineralized Material	Ai (g)		
	Measurements	Average	Design
Yellow Jacket	0.7154, 0.4259, 0.2718, 0.5729, 0.5766, 0.4778	0.5067	-
Sierra Blanca	0.4577	0.4577	-
Savage Valley	-	-	-
Jolly Jane	0.4952	0.4952	
Mayflower	0.3006, 0.2879, 0.2324, 0.2062, 0.0677, 0.3946, 0.2680, 0.3868	0.2680	-
Overall	-	0.432	0.572

Table 13-3 - Bond's Ball Mill and Rod Mill Work Indices (BMWi/RMWi, kwh/t)-YellowJacket

Mineralized Material	Measurements	Average BMWi (kWh/t)	RMWi (kWh/t)
Yellow Jacket	22.8, 22.1, 21.5, 22.1, 22.5, 22.0	22.2	15.9

Note: High-grade YellowJacket mineralization will be the only mineralization processed in the milling circuit.

13.1.2 MAYFOWER BOTTLE ROLL AND COLUMN TESTS

Mayflower deposit gold dissolution data has been developed from 74 bottle roll tests and 10 column tests on drill core, RC drill cuttings and surface grab samples from the David Adit and the main dumps. Test results are presented in Tables 13-4 and 13-5.

Bottle roll results indicate that gold dissolution by NaCN on Main Dump grab material increases from an average of 37.7% to 98.9% as the P80 decreases from 38000 to 75 µm. Sodium cyanide consumption ranged 0.1-0.2 kg/tonne. Lime consumption averages 1.0 kg/tonne.

Bottle roll result indicate gold dissolution on David Adit grab material increases from an average of 34.3% to 97.9% as the P80 decreases from 38000 to 75 µm. Sodium cyanide consumption averaged 0.2 kg/tonne. Lime consumption averages 1.5 kg/ton.

Column leach test results on Mayflower core material indicate gold dissolution at a P80 19,000 µm averaged 88.0%±1.6% and ranged from 85.7% to 90.8%. Sodium cyanide consumption averaged 1.2 kg/tonne. Lime consumption averaged 1.3 kg/tonne. Leach time ranged from 90 to 156 days and averaged 117 days.

Table 13-4 - Mayflower Bottle Roll and Column Tests 2008-2013

Deposit	No. Tests	Test Type	Material Type	Leach Time	Material Size, ~P_{80-100} or Retained	Au Calculated Head	Au Tail Assay	Au Dissolution	Ag Calculated Head	Ag Tail Assay	Estimated Ag Recovery	NaCN Consumed	CaO Consumed
				days	um	gpt	gpt	%	gpt	gpt	gpt	kg/t	kg CaO/t
Main Dump	2	Bottle Roll	Grab Sample	4	38000	2.029	1.263	37.7	--	--	--	0.1	0.6
Main Dump	2	Bottle Roll	Grab Sample	4	19000	1.035	0.492	51.9	--	--	--	0.1	0.8
Main Dump	2	Bottle Roll	Grab Sample	4	6300	2.289	1.157	49.5	--	--	--	0.1	1.3
Main Dump	2	Bottle Roll	Grab Sample	4	1700	4.512	1.571	65.1	--	--	--	0.1	0.9
Main Dump	2	Bottle Roll	Grab Sample	4	75	4.523	0.049	98.9	--	--	--	0.2	1.3
David Adit	2	Bottle Roll	Grab Sample	4	38000	2.246	1.485	34.3	--	--	--	0.1	0.8
David Adit	2	Bottle Roll	Grab Sample	4	19000	1.676	0.777	53.6	--	--	--	0.1	1.2
David Adit	2	Bottle Roll	Grab Sample	4	6300	2.140	1.009	52.9	--	--	--	0.1	1.3
David Adit	2	Bottle Roll	Grab Sample	4	1700	2.146	0.461	78.6	--	--	--	0.1	1.1
David Adit	2	Bottle Roll	Grab Sample	4	75	1.984	0.042	97.9	--	--	--	0.1	1.5
MayFlower	12	Bottle Roll	Core	3	19000	0.545	0.178	69.8	--	--	--	0.1	1.3
MayFlower	12	Bottle Roll	Core	3	6300	0.522	0.113	80.4	--	--	--	0.1	1.4
MayFlower	18	Bottle Roll	Core	3	1700	1.345	0.213	84.6	--	--	--	1.1	1.5
MayFlower	12	Bottle Roll	Core	3	75	0.515	0.027	92.8	--	--	--	0.2	1.9
Mayflower	10	Column	Core	118	19000	0.391	0.048	88.0	1.1	1.0	10.7	1.2	1.3

The test data for the core material indicates the following:

- The column leach gold extraction was higher than that obtained in the bottle roll tests. This could be due to coarse gold leaching slowly.
- The extractions in the bottle roll tests for various size fractions indicate that the gold extraction in general is independent of the particle size in the feed.

Table 13-5 - Summary Metallurgical Results, NBP Mayflower Composites from the Core Material

Composite	Test Type	Leach Time (days)	Feed Size	Au Recovery (%)[2]	Extracted Au (g/t)[2]	Tail Au (g/t)[2]	Calc'd Head (g/t)[2]	Avg. Head (g/t)[2]	NaCN Cons.[2] (kg/t)	Lime Add[2] (kg/t)
MFC 001	CLT	155	79% -19mm	88.5	0.430	0.056	0.486	0.488	1.53	1.2
MFC 001	BRT	4	79% -19mm	61.7	0.337	0.209	0.546	0.488	0.07	1.2
MFC 001	BRT	4	80% -6.3mm	73.6	0.380	0.136	0.516	0.488	0.09	1.3
MFC 001	BRT	4	80% -1.7mm	83.8	0.398	0.077	0.475	0.488	0.08	1.9
MFC 001	BRT	4	80% -75µm	97.2	0.491	0.014	0.505	0.488	0.23	2.0
MFC 002	CLT	94	91% -19mm	89.7	0.279	0.032	0.311	0.356	1.10	1.3
MFC 002	BRT	4	91% -19mm	66.8	0.237	0.118	0.355	0.356	<0.07	1.3
MFC 002	BRT	4	80% -6.3mm	83.9	0.296	0.057	0.353	0.356	0.13	1.5
MFC 002	BRT	4	80% -1.7mm	93.1	0.325	0.024	0.349	0.356	0.10	1.8
MFC 002	BRT	4	80% -75µm	95.1	0.367	0.019	0.386	0.356	0.22	2.1
MFC 003	CLT	156	85% -19mm	87.2	0.469	0.069	0.538	0.534	2.11	1.2

MFC 003	BRT	4	85% -19mm	74.5	0.438	0.150	0.588	0.534	0.09	1.3
MFC 003	BRT	4	80% -6.3mm	81.4	0.430	0.098	0.528	0.534	0.08	1.2
MFC 003	BRT	4	80% -1.7mm	88.7	0.441	0.056	0.497	0.534	0.09	1.7
MFC 003	BRT	4	80% -75µm	92.8	0.453	0.035	0.488	0.534	0.14	2.0
MFC 004	CLT	82	89% -19mm	89.0	0.138	0.017	0.155	0.160	0.48	1.4
MFC 004	BRT	4	89% -19mm	80.9	0.140	0.033	0.173	0.160	0.10	1.4
MFC 004	BRT	4	80% -6.3mm	86.1	0.143	0.023	0.166	0.160	0.09	1.6
MFC 004	BRT	4	80% -1.7mm	89.9	0.142	0.016	0.158	0.160	<0.07	1.7
MFC 004	BRT	4	80% -75µm	92.2[3]	0.154	0.013	0.167	0.160	0.27	2.0
MFC 005	CLT	94	86% -19mm	85.7	0.401	0.067	0.468	0.522	0.94	1.3
MFC 005	BRT	4	86% -19mm	73.5	0.394	0.142	0.536	0.522	0.08	1.2
MFC 005	BRT	4	80% -6.3mm	82.0	0.388	0.085	0.473	0.522	0.08	1.4
MFC 005	BRT	4	80% -1.7mm	88.1	0.473	0.064	0.537	0.522	0.10	1.7
MFC 005	BRT	4	80% -75µm	91.7	0.474	0.043	0.517	0.522	0.18	1.9
MFC 006	BRT	4	88% -19mm	60.8	0.657	0.424	1.081	1.046	0.08	1.3
MFC 006	BRT	4	80% -6.3mm	74.6	0.820	0.279	1.099	1.046	0.10	1.4
MFC 006	BRT	4	80% -1.7mm	87.2	0.905	0.133	1.038	1.046	<0.07	1.7
MFC 006	BRT	4	80% -75µm	98.7	0.990	0.013	1.003	1.046	0.11	1.8

1 – CLT Column Leach Test; BRT Bottle Roll Test.

2- Results presented in this data are the averages of duplicate tests.

3- The recovery results shown for MFC 004 at a 75µm feed size is from a single test. A calculated average is not shown due to poor head grade agreement for the duplicate test.

13.1.3 SAVAGE VALLEY BOTTLE ROLL AND COLUMN TESTS

Savage Valley deposit gold dissolution data has been developed from 63 bottle roll tests and 12 column tests on drill core, RC drill cuttings for oxide, mixed, and sulphide materials. Test results are presented in Tables 13-6 and 13-7.

Bottle roll test results indicate gold dissolution on oxide material increases from an average of 71.8% to 89.1% as the P80 decreases from 19000 to 75 µm. Sodium cyanide consumption ranges 0.1-0.2 kg/tonne. Lime consumption averages 1.2 kg/tonne.

Bottle roll test results indicate gold dissolution on mixed oxide material decreases from an average of 70.1% to 64.7% as the P80 decreases from 1700 to 75 µm. Sodium cyanide consumption was 0.1 kg/tonne. Lime consumption averaged 1.8 kg/tonne.

Bottle roll results indicate gold dissolution on sulfide material averaged 4% at 75 µm. Sodium cyanide consumption was 0.8 kg/tonne. Lime consumption averaged 0.1 kg/tonne.

Column leach test results on Savage Valley core material indicate gold dissolution at a P80 19000 µm averaged 81.7%±8.4% and ranged from 68.7% to 92.2%. Sodium cyanide consumption averaged 0.7 kg/tonne. Lime consumption averaged 0.9 kg/tonne. Leach time ranged from 63 to 136 days and averaged 87 days.

Table 13-6 - Savage Valley Bottle Roll and Column Tests 2008-2013

Deposit	No. Tests	Test Type	Material Type	Leach Time	Material Size, ~P_{80-100} or Retained	Au Calculated Head	Au Tail Assay	Au Dissolution	Ag Calculated Head	Ag Tail Assay	Estimated Ag Recovery	NaCN Consumed	CaO Consumed
				days	um	gpt	gpt	%	gpt	gpt	gpt	kg/t	kg CaO/t
Savage Valley Oxide	12	Bottle Roll	Core	3	19000	0.380	0.105	71.8	--	--	--	0.1	0.8
Savage Valley Oxide	12	Bottle Roll	Core	3	6300	0.395	0.083	74.0	--	--	--	0.1	1.1
Savage Valley Oxide	15	Bottle Roll	Core & RC	3	1700	0.376	0.064	82.4	--	--	--	0.1	1.1
Savage Valley Mixed	2	Bottle Roll	RC	3	1700	0.275	0.080	70.1	--	--	--	0.1	1.5
Savage Valley Oxide	16	Bottle Roll	Core & RC	3	75	0.402	0.040	89.1	--	--	--	0.2	1.7
Savage Valley Mixed	3	Bottle Roll	RC	3	75	0.310	0.120	64.7	--	--	--	0.1	2.1
Savage Valley Sulphide	3	Bottle Roll	RC	3	75	0.503	0.483	4.0	--	--	--	0.8	0.1
Savage Valley	12	Column	Core	87.9	19000	0.376	0.061	81.7	1.1	1.0	11.8	0.7	0.9

Table 13-7 - Summary of Column Leach Tests for NBP Savage Valley Drill Core Composites

MLI Test No.	Sample I.D.	Feed Size	Leach/Rinse Time, days	Au Recovery, %	g Au/mt Mineralized Material				Reagent Requirements Kg/mt mineralization	
					Extracted	Tail	Calc'd Head	Average Head	NaCN Cons.	Lime Added
P-11	SVC001	78%-19 mm	76	86.3	0.659	0.105	0.764	0.748	0.76	0.8
P-12	SVC001	78%-19 mm	78	87.4	0.627	0.090	0.717	0.748	0.88	0.8
P-13	SVC002	77%-19 mm	88	87.2	0.553	0.081	0.634	0.665	0.76	0.8
P-14	SVC002	77%-19 mm	88	87.7	0.582	0.082	0.664	0.665	0.70	0.8
P-15	SVC003	79%-19 mm	63	92.2	0.107	0.009	0.116	0.12	0.31	1.1
P-16	SVC003	79%-19 mm	63	91.7	0.100	0.009	0.109	0.12	0.32	1.1
P-17	SVC004	85%-19 mm	89	71.1	0.150	0.061	0.211	0.21	0.68	0.7
P-18	SVC004	85%-19 mm	89	68.7	0.147	0.067	0.214	0.21	0.56	0.7
P-19	SVC005	83%-19 mm	75	81.1	0.292	0.068	0.360	0.35	0.66	1.0

P-20	SVC005	83%-19 mm	74	82.1	0.289	0.063	0.352	0.35	0.66	1.0
P-21	SVC006	86%-19 mm	136	72.4	0.131	0.050	0.181	0.18	1.30	0.7
P-22	SVC006	86%-19 mm	136	72.3	0.133	0.051	0.18	0.18	1.20	0.7

13.1.4 SIERRA BLANCA BOTTLE ROLL AND COLUMN TESTS

Sierra Blanca deposit gold dissolution has been developed from 91 bottle roll tests and 18 column tests on drill core, RC drill cuttings for oxide, mixed, and surface bulk grab materials. Test results are presented in Tables 13-8 and 13-9.

Bottle roll test results indicate gold dissolution on bulk surface oxide material increases from an average of 83.4% to 85.9% as the P80 decreases from 1700 to 75 µm. Sodium cyanide consumption ranges 0.1-0.2 kg/ton. Lime consumption averages 1.5 kg/ton.

Bottle roll test results indicate gold dissolution on mixed oxide material ranges from 69.6% to 73.1% as the P80 decreases from 1700 to 75 µm. Sodium cyanide consumption was 0.2 kg/ton. Lime consumption averaged 1.8 kg/ton.

Bottle roll results indicated gold dissolution on Sierra Blanca oxide material decreases from 76.1% to 70.5% as the P80 decreases from 19000 to 75 µm. Sodium cyanide consumption averages 0.1 kg/ton. Lime consumption averages 1.1 kg/ton.

Column leach test results on Sierra Blanca core material indicated gold extraction of 83.3% for mineralization having a feed grade of 0.51 g/t Au. The cyanide and lime consumptions were reasonable at 0.42 kg/t and 1.0 kg/t, respectively.

Table 13-8 - Sierra Blanca Bottle Roll and Column Tests 2008-2013

Deposit	No. Tests	Test Type	Material Type	Leach Time	Material Size, ~P_{80-100} or Retained	Au Calculated Head	Au Tail Assay	Au Dissolution	Ag Calculated Head	Ag Tail Assay	Estimated Ag Recovery	NaCN Consumed	CaO Consumed
				days	um	gpt	gpt	%	gpt	gpt	gpt	kg/t	kg CaO/t
Sierra Blanca Surface Bulk Sample	3	Bottle Roll	Grab Sample	3	1700	0.326	0.054	83.4	--	--	--	0.2	1.6
Sierra Blanca Surface Bulk Sample	1	Bottle Roll	Grab Sample	3	75	0.270	0.038	85.9	--	--	--	0.1	1.3
Sierra Blanca Mixed	2	Bottle Roll	RC	3	1700	0.315	0.095	69.6	--	--	--	0.1	2.0
Sierra Blanca Mixed	2	Bottle Roll	RC	3	75	0.309	0.083	73.1	--	--	--	0.2	1.5
Sierra Blanca	14	Bottle Roll	Core	3	19000	0.255	0.056	76.1	--	--	--	0.1	0.9
Sierra Blanca	14	Bottle Roll	Core	3	6300	0.221	0.043	78.3	--	--	--	0.1	1.0
Sierra Blanca	17	Bottle Roll	Core	3	1700	0.251	0.040	82.3	--	--	--	0.1	1.2
Sierra Blanca	38	Bottle Roll	Core	3.0	75.0	0.353	0.115	70.5	--	--	--	0.1	1.2
Sierra Blanca	14	Column	Core	100	19000	0.222	0.030	83.5	1.0	--	0.1	0.6	1.0
Seirra Blanca Bulk Outcrop	2	Column	Grab Sample	117	50000	0.288	0.086	73.9	--	--	--	--	--
Seirra Blanca Bulk Outcrop	2	Column	Grab Sample	97	12500	0.405	0.074	81.8	--	--	--	--	--

Table 13-9 - Summary of Metallurgical Results for Column Leach Tests on Sierra Blanca Drill Core Composites

MLI Test No.	Sample I.D.	Feed Size	Leach/Rinse Time, Days	Extraction		Average Head Grade	Reagent Requirements, Kg/mt mineralization	
				% of High Grade	g Au/mt Mineralization		NaCN Cons.	Lime Added
P-43	SB01	86%-19mm	39	70.8	0.102	0.144	0.33	0.8
P-44	SB01	86%-19mm	39	70.1	0.101	0.144	0.32	0.8
P-45	SB02	88%-19mm	39	75.2	0.112	0.149	0.27	1.1
P-46	SB02	88%-19mm	39	59.7	0.089	0.149	0.30	1.1
P-33	SB03	90%-19mm	41	57.3	0.141	0.246	0.40	0.8
P-34	SB03	90%-19mm	41	48.4	0.119	0.246	0.36	0.8
P-35	SB04	94%-19mm	41	83.4	0.426	0.511	0.39	1.0
P-36	SB04	94%-19mm	41	83.2	0.425	0.511	0.45	1.0
P-37	SB05	93%-19mm	38	72.2	0.096	0.133	0.29	1.0
P-38	SB05	93%-19mm	38	66.9	0.089	0.133	0.27	1.0
P-39	SB06	89%-19mm	39	91.5	0.324	0.354	0.31	1.0
P-40	SB06	89%-19mm	39	102.8	0.364	0.354	0.34	1.0
P-41	SB07	91%-19mm	38	77.6	0.097	0.125	0.28	1.1
P-42	SB07	91%-19mm	38	92.0	0.115	0.125	0.34	1.1

13.1.5 JOLLY JANE BOTTLE ROLL AND COLUMN TESTS

Jolly Jane deposit gold dissolution has been developed from 54 bottle roll tests and 14 column tests on drill core, RC drill cuttings and bulk surface gab samples. Test results are presented in Table 13-10.

Bottle roll results indicate gold dissolution on Jolly Jane oxide material increases from an average of 68.5% to 87% as the P_{80} decreases from 19,000 to 75 µm. Sodium cyanide consumption ranges 0.1-0.2 kg/ton. Lime consumption ranges from 0.8-1.8 kg/ton.

Bottle roll test result indicate gold dissolution on Jolly Jane Oxide material, using bulk sampling, increases from an average of 74.4% to 80.0% as the P_{80} decreases from 1,700 to 75 µm. Sodium cyanide consumption ranges 0.1-0.2 kg/ton. Lime consumption ranges from 0.8-1.8 kg/ton.

Bottle roll test result indicate gold dissolution on Jolly Jane surface material using RC cuttings increases from an average of 66.2% to 78.8% as the P_{80} decreases from 1,700 to 75 µm. Sodium cyanide consumption ranges 0.1-0.2 kg/ton. Lime consumption averages 1.8 kg/ton.

Column leach test results on Jolly Jane core material indicate gold dissolution at a P80 19,000 µm average 74.8%±7.3% and ranged from 62.3 to 85.9. Sodium cyanide consumption averaged 0.8 kg/ton. Lime consumption averaged 0.8 kg/ton. Leach time ranged from 70 to 161 days and averaged 107 days.

Column leach test results on Jolly Jane surface outcrop indicate gold dissolution at a P80 50,000 µm was 64.0%. Sodium cyanide consumption was 0.9 kg/ton. Lime consumption was 1.6 kg/ton and leach time was 117 days.

Column leach test results on Jolly Jane surface outcrop indicate gold dissolution at a P80 12,500 µm was 63.8%. Sodium cyanide consumption averaged 1.0 kg/ton. Lime consumption averaged 1.6 kg/ton and leach time averaged 104 days.

Table 13-10 – Jolly Jane Bottle Roll and Column Tests 2008-2013

Deposit	No. Tests	Test Type	Source Material	Leach Time	Size ~P_{80-100} or Retained	Au Calculated Head	Au Tail Assay	Au Dissolution	NaCN Consumed	Lime Consumed
				days	µm	gpt	gpt	%	Kg/t	Kg CaO/t
Jolly Jane	10	Bottle Roll	Core	3	1900	0.364	0.125	68.5	0.1	0.8
Jolly Jane	10	Bottle Roll	Core	3	6300	0.355	0.095	75.1	0.1	0.9

Jolly Jane	10	Bottle Roll	Core	3	1700	0.322	0.069	79.6	0.1	1.2
Jolly Jane	14	Bottle Roll	Core & RC	3	75	0.336	0.044	87.0	0.1	1.4
Jolly Jane Oxide	3	Bottle Roll	RC	3	1700	0.343	0.117	66.2	0.1	1.3
Jolly Jane Oxide	3	Bottle Roll	RC	3	75	0.301	0.063	78.8	0.1	0.9
Jolly Jane Surface	3	Bottle Roll	Bulk Sample	3	1700	0.333	0.086	74.4	0.1	1.8
Jolly Jane Surface	1	Bottle Roll	Bulk Sample	3	75	0.315	0.063	80.0	0.2	1.8
Jolly Jane	10	Column	Core	107	19000	0.324	0.087	74.8	0.78	0.8
Jolly Jane Bulk Surface	1	Column	Bulk Sample	117	50000	0.328	0.120	64.0	0.93	1.6
Jolly Jane Bulk Surface	3	Column	Bulk Sample	104	12500	0.348	0.122	63.9	1.0	1.5

13.1.6 YELLOWJACKET METALLURGICAL TESTING 2013-2015

The YellowJacket deposit, located on the eastern edge of the Sierra Blanca Resources, is a west-dipping quartz vein and vein stockwork containing high-grade gold mineralization. A series of metallurgical tests performed on the samples from the deposit include bottle roll cyanidation tests, column leach tests, gravity concentration tests and intense cyanide leach of the gravity concentrate and cyanide leaching of the gravity tailing. The data was reported by McClelland Laboratories (2015(a) and 2015(b)) and is reviewed here.

Scoping level cyanidation bottle roll tests were performed in duplicate on twelve drilling core samples at P80 of 75 micrometers. The test results, given in Table 13-11, indicate gold extraction of over 85% in most of the tests. The silver extraction ranged from 60% to 89%. The cyanide consumption was reasonable at less than 0.25 kg/t.

The core from the drilling program (2013-2014) was used to create five composite samples above the oxidation surface (designated YJPQ) and two samples below the oxidation surface where sulfide minerals remained un-oxidized (YJJV). The composites represented the types of mineralization in the Josh Vein (quartz vein) and adjacent stockwork zones. Cyanidation bottle roll, column leach and gravity concentration tests were performed on these composites. The leaching test data are given in Tables 13-12 to 13-14 for YJPQ composites. The results indicate the following:

- The composites assayed 0.35 g/t to 9.74 g/t Au and 1.9 g/t to 47.5 g/t Ag.
- The gold in the deposit is size dependent. The finer the grind size, the higher the gold extraction. The gold extraction levels were measured at P80 of 150 micrometers (± 60% to 90%).

- The silver extraction followed the same trend as gold. It increased till P80 of 150 micrometers and then leveled off (60% to 70%).
- Column leach tests at P80 of 19 mm and 6.3 mm confirmed that the finer the crush size, the higher the gold extraction. A maximum recovery of 60% was achieved in the column leach tests.

Table 13-11 - Summary of Metallurgical Results, Bottle Roll Tests on YellowJacket Drill Core Composites, 80%-75 UM Feed Size

Composite	Au Recovery %	g Au/mt mineralization				Ag Recovery %	g Ag/mt mineralization				Reagent Requirements kg/mt mineralization	
		Extracted	Tail[1]	Calculated Head	Head[2] Assay		Extracted	Tail[1]	Calculated Head	Head[2] Assay	NaCN Cons.	Lime Added
C226950	96.2	2.27	0.09	2.36	2.16	85.7	8.4	1.4	9.8	11	0.08	1.2
C226950	96.4	2.40	0.09	2.49	2.16	85.6	8.3	1.4	9.7	11	<0.07	1.3
C226985	91.2	11.02	1.06	12.08	12.70	67.8	28.9	13.7	42.6	49	0.09	1.1
C226985	89.9	9.32	1.05	10.37	12.70	65.1	27.4	14.7	42.1	49	0.14	1.2
C226986	78.2	3.08	0.86	3.94	3.90	61.0	26.8	17.1	43.9	49	0.12	1.6
C226986	79.9	3.06	0.77	3.83	3.90	73.0	33.2	12.3	45.5	49	0.21	1.6
C226989	68.7	1.36	0.62	1.98	1.90	69.9	14.6	6.3	20.9	25	0.23	1.1
C226989	73.1	1.41	0.52	1.93	1.90	76.4	15.2	4.7	19.9	25	0.13	1.0
C226990	85.1	6.34	1.11	7.45	6.70	70.7	22.7	9.4	32.1	35	0.20	1.0
C226990	85.6	6.10	1.03	7.13	6.70	72.8	24.3	9.1	33.4	35	0.21	0.9
M612658	85.4	0.70	0.12	0.82	0.80	75.8	2.5	0.8	3.3	3	0.08	1.3
M612658	86.9	0.73	0.11	0.84	0.80	78.8	2.6	0.7	3.3	3	0.07	1.4
M612665	86.5	8.32	1.30	9.62	12.10	80.1	34.7	8.6	43.3	45	0.11	1.1
M612665	89.3	8.64	1.04	9.68	12.10	79.2	33.8	8.9	42.7	45	0.17	1.1
M612674	91.7	0.77	0.07	0.84	1.00	88.2	44.1	5.9	50.0	61	0.12	1.1
M612674	92.9	0.78	0.06	0.84	1.00	85.7	46.2	7.7	53.9	61	0.13	1.1
M612701	96.9	5.86	0.19	6.05	7.10	89.7	35.7	4.1	39.8	49	0.22	1.1
M612701	95.7	5.53	0.25	5.78	7.10	89.1	36.0	4.4	40.4	49	0.15	1.1
M612704	89.1	1.47	0.18	1.65	1.70	83.6	43.3	8.5	51.8	56	0.26	1.1
M612704	85.9	1.40	0.23	1.63	1.70	83.5	43.9	8.7	52.6	56	0.34	1.1
M612716	94.1	1.28	0.08	1.36	1.20	74.2	2.3	0.8	3.1	3	0.13	1.1
M612716	95.1	1.37	0.07	1.44	1.20	75.0	2.4	0.8	3.2	3	0.08	1.2
M612727	90.7	0.49	0.05	0.54	0.60	72.7	0.8	0.3	1.1	1	<0.07	1.2
M612727	91.7	0.66	0.06	0.72	0.60	66.7	0.8	0.4	1.2	1	0.11	1.2

(1) Average of triplicate assays

(2) Head assays were provided by Corvus

Table 13-12 - Summary of Bottle Roll Tests, YellowJacket YJ PQ Drill Core Composites

Composite	Feed Size (P_{80})	Au Recovery, %	g Au/mt mineralization				Ag Recovery %	g Ag/mt mineralization				Reagent Requirements, kg/mt mineralization	
			Extracted	Tail[1]	Calculated Head	Head[2]		Extracted	Tail[1]	Calculated Head	Head[2]	NaCN Cons.	Lime Added
YJPQ01	19mm	12.7	0.91	6.23	7.14	5.39	10.5	4.5	38.4	42.9	47.5	<0.09	0.5
YJPQ01	6.3mm	30.4	2.70	6.19	8.89	5.39	24.9	13.3	40.1	53.4	47.5	0.12	0.7
YJPQ01	1.7mm	50.0	3.88	3.88	7.76	5.39	46.9	22.5	25.5	48.0	47.5	0.17	0.7
YJPQ01	0.150mm	86.2	4.88	0.78	5.66	5.39	66.8	37.3	18.5	55.8	47.5	<0.12	1.0
YJPQ01	0.106mm	88.4	4.44	0.58	5.02	5.39	68.7	37.1	16.9	54.0	47.5	<0.12	1.0
YJPQ01	0.075mm	88.3	5.00	0.66	5.66	5.39	70.3	32.4	13.7	46.1	47.5	0.17	1.2
YJPQ02	19mm	11.3	0.60	4.69	5.29	9.74	15.0	3.4	19.2	22.6	25.0	0.11	0.7
YJPQ02	6.3mm	32.9	1.62	3.30	4.92	9.74	35.4	7.4	13.5	20.9	25.0	<0.11	0.9
YJPQ02	1.7mm	53.6	3.53	3.06	6.59	9.74	50.2	12.0	11.9	23.9	25.0	0.12	0.9
YJPQ02	0.150mm	92.0	5.29	0.46	5.75	9.74	74.3	18.2	6.3	24.5	25.0	0.10	1.3
YJPQ02	0.106mm	92.9	5.35	0.41	5.76	9.74	75.8	19.1	6.1	25.2	25.0	<0.07	1.4
YJPQ02	0.075mm	90.4	5.64	0.60	6.24	9.74	77.4	19.9	5.8	25.7	25.0	0.18	1.2
YJPQ03	19mm	13.7	0.27	1.70	1.97	1.49	11.4	0.9	7.0	7.9	9.1	0.09	0.6
YJPQ03	6.3mm	27.3	0.59	1.57	2.16	1.49	20.5	1.6	6.2	7.8	9.1	<0.07	0.8
YJPQ03	1.7mm	45.9	0.73	0.86	1.59	1.49	38.4	3.3	5.3	8.6	9.1	0.12	1.1
YJPQ03	0.150mm	88.5	1.62	0.21	1.83	1.49	68.4	5.4	2.5	7.9	9.1	<0.07	1.2
YJPQ03	0.106mm	88.3	1.58	0.21	1.79	1.49	65.9	5.6	2.9	8.5	9.1	<0.07	1.3
YJPQ03	0.075mm	89.6	1.29	0.15	1.44	1.49	76.1	7.0	2.2	9.2	9.1	0.20	1.3
YJPQ04	19mm	23.1	0.09	0.30	0.39	0.62	16.7	0.8	4.0	4.8	4.6	0.07	0.8
YJPQ04	6.3mm	33.3	0.18	0.36	0.54	0.62	41.9	1.8	2.5	4.3	4.6	<0.07	1.1
YJPQ04	1.7mm	57.8	0.37	0.27	0.64	0.62	52.3	2.3	2.1	4.4	4.6	<0.09	1.2
YJPQ04	0.150mm	61.4	0.35	0.22	0.57	0.62	62.0	3.1	1.9	5.0	4.6	0.11	1.5
YJPQ04	0.106mm	67.3	0.37	0.18	0.55	0.62	66.0	3.1	1.6	4.7	4.6	<0.07	1.6
YJPQ04	0.075mm	68.8	0.44	0.20	0.64	0.62	66.7	3.4	1.7	5.1	4.6	<0.08	2.5
YJPQ05	19mm	43.9	0.18	0.23	0.41	0.33	29.4	0.5	1.2	1.7	1.9	<0.07	1.0
YJPQ05	6.3mm	44.2	0.19	0.24	0.43	0.33	36.8	0.7	1.2	1.9	1.9	0.07	1.2
YJPQ05	1.7mm	54.0	0.27	0.23	0.50	0.33	47.4	0.9	1.0	1.9	1.9	0.16	1.3
YJPQ05	0.150mm	76.4	0.42	0.13	0.55	0.33	65.0	1.3	0.7	2.0	1.9	<0.10	1.7
YJPQ05	0.106mm	76.2	0.32	0.10	0.42	0.33	70.0	1.4	0.6	2.0	1.9	0.10	1.8
YJPQ05	0.075mm	74.4	0.32	0.11	0.43	0.33	61.9	1.3	0.8	2.1	1.9	<0.08	2.3

Table 13-13 - Summary Metallurgical Results, Gold Recovery from Column Percolation Leach Tests, YellowJacket YJ PQ Drill Core Composites (80% -6.3mm and 80% -19 mm)

Composite/Feed Size, (P_{80})	Leach/Rinse Time (days)	Au Recovery %	g Au/mt			Reagent Requirements kg/mt	
			Extracted	Tail	Calculated Head	NaCN Cons.	Lime Added

YJPQ01 -6.3mm	181	53.6	5.0	4.36	9.36	7.06	0.8
YJPQ02 -6.3mm	181	60.2	2.21	1.46	3.66	5.62	1.0
YJPQ03 -6.3mm	137	37.8	0.65	1.12	1.77	3.14	0.9
YJPQ04 -6.3mm	137	50.1	0.33	0.32	0.65	2.92	1.3
YJPQ05 -19mm	140	46.5	0.17	0.19	0.36	1.46	1.2
YJPQ05 -6.3mm	137	56.3	0.23	0.18	0.40	2.33	1.4

Table 13-14 - Summary Metallurgical Results, Silver Recovery from Column Percolation Leach Tests, YellowJacket YJ PQ Drill Core Composites (80% -6.3mm and 80% - 19mm)

Composite/Feed Size, (P_{80})	Leach/Rinse Time (days)	Ag Recovery %	g Au/mt			Reagent Requirements kg/mt	
			Extracted	Tail	Calculated Head	NaCN Cons.	Lime Added
YJPQ01 -6.3mm	181	46.2	24.2	28.2	52.4	7.06	0.8
YJPQ02 -6.3mm	181	47.7	10.5	11.6	22.1	5.62	1.0
YJPQ03 -6.3mm	137	34.0	2.4	4.7	7.0	3.14	0.9
YJPQ04 -6.3mm	137	48.4	2.4	2.6	4.9	2.92	1.3
YJPQ05 -19mm	140	38.2	0.8	1.3	2.1	1.46	1.2
YJPQ05 -6.3mm	137	48.7	0.9	1.0	1.9	2.33	1.4

The leach test data for YJJV composite samples are given in Tables13-15 and 13-16. The test results indicated the following:

- The JV Stockwork composite assayed 4.6 g/t Au and 61.2 g/t Ag and the stockwork composite assayed 1.64 g/t Au and 7.5 g/t Ag.
- The gold and silver, in general, did not appear to be size dependent. The extractions were very similar at 75 and 150 micrometers.
- These bottle roll tests indicated relatively high recovery of metal by cyanide leaching though it was lower recovery than achieved for the YJPQ tests.
- The kinetic leach data indicated that metal dissolution was still increasing at 96 hours.

Extended gravity recoverable gold (E-GRG) tests were performed on the YJPQ and YJJV composites to evaluate the potential of precious metal recovery in the gravity circuit. The test data are given in Tables 13-17 to 13-19. The test results indicated that a gravity concentration process was a viable option to recover 40% to 50% of the gold in the mineralization.

Following the gravity concentration testing, additional tests were performed on composites YJPQ where the composites were ground to P80 of 65 mesh and subjected to Knelson concentration. The gravity concentrate was cleaned, and the combined rougher and cleaner tailing cyanide leached. The test results, given in Tables 13-20 and 13-21, indicated combined gold gravity and leach extractions of 65.6% to 94.8% and silver extractions of 64.2% to 80.8%.

The YJJV composites were also evaluated for the gravity recovery followed by leaching of both the gravity concentrate and tailing. The test data, summarized in Table 13-22, indicated gold extraction of 72.6% to 87.3% and silver extraction of 63.2% to 69.7%.

These results indicated that gravity concentration followed by cyanide leaching of the gravity tailing was a potential processing option for high-grade mineralization.

Table 13-15 - Bottle Roll Tests for YJ JV Composites, Gold Recovery at Various Feed Sizes

Composite	Feed Size (P_{80})	Test	Au Rec. (%)	g Au/mt mineralization				Reagent Req., kg/mt mineralization	
				Ext'd	Tail	Calc'd Head	Head Assay	NaCN Cons.	Lime Added
JV+Stockwork	150µm	Initial	76.2	3.72	1.16	4.88	4.60	0.19	0.6
JV+Stockwork	150µm	Dup	76.4	3.72	1.15	4.87	4.60	0.21	0.6
JV+Stockwork	75µm	Initial	76.7	3.75	1.14	4.89	4.60	0.22	0.7
JV+Stockwork	75µm	Dup	77.0	3.64	1.09	4.73	4.60	0.19	0.9
Stockwork	150µm	Initial	69.2	1.17	0.52	1.69	1.64	<0.07	1.0
Stockwork	150µm	Dup	56.3	0.90	0.70	1.60	1.64	<0.07	1.1
Stockwork	75µm	Initial	75.4	1.50	0.49	1.99	1.64	0.14	1.1
Stockwork	75µm	Dup	73.5	1.39	0.50	1.89	1.64	<0.07	1.3

Table 13-16 - Bottle Roll Tests from YJ JV Composites, Silver Recovery at Various Feed Sizes

Composite		Test		g Ag/mt mineralization	Reagent Req., kg/mt mineralization

	Feed Size (P_{80})		Ag Rec. (%)	Ext'd	Tail	Calc'd Head	Head Assay	NaCN Cons.	Lime Added
JV+Stockwork	150µm	Initial	56.2	34.2	26.7	60.9	61.2	0.19	0.6
JV+Stockwork	150µm	Dup	53.5	33.0	28.7	61.7	61.2	0.21	0.6
JV+Stockwork	75µm	Initial	58.5	35.6	25.3	60.9	61.2	0.22	0.7
JV+Stockwork	75µm	Dup	58.0	34.4	24.9	59.3	61.2	0.19	0.9
Stockwork	150µm	Initial	53.8	3.5	3.0	6.5	7.5	<0.07	1.0
Stockwork	150µm	Dup	51.5	3.4	3.2	6.6	7.5	<0.07	1.1
Stockwork	75µm	Initial	57.1	3.6	2.7	6.3	7.5	0.14	1.1
Stockwork	75µm	Dup	59.1	3.9	2.7	6.6	7.5	<0.07	1.3

Table 13-17 - E-GRG Test Results for Gold Recovery from the YJ PQ Composites

Composite	Recovery, % of Total Au Nominal Grind Size				Head Grade g Au/mt mineralization	
	700µm	250µm	75µm	Total	Calculated	Average
YJPQ01	33.3	23.2	9.4	65.9	8.13	7.56
YJPQ02	34.5	33.9	11.4	79.8	5.46	5.66
YJPQ03	18.0	17.6	11.1	46.7	1.39	1.66
YJPQ04	12.3	17.8	12.8	42.9	0.74	0.60

Table 13-18 - E-GRG Test Results for Silver Recovery from the YJ PQ Composites

Composite	Recovery, % of Total Ag Nominal Grind Size				Head Grade g Ag/mt mineralization	
	700µm	250µm	75µm	Total	Calculated	Average
YJPQ01	11.5	12.2	2.8	26.5	45.9	49.3
YJPQ02	7.0	8.1	4.1	19.2	19.1	22.9
YJPQ03	2.8	3.0	2.0	7.8	7.6	8.0
YJPQ04	4.5	4.6	3.1	12.2	4.3	4.8

Table 13-19 - E-GRG Test Results for the YJ JV Composites, Gold and Silver Recovery

Composite	Recovery, % of Total Metal Nominal Grind Size				Head Grade g /mt mineralization	
	700µm	250µm	75µm	Total	Calculated	Average
Gold						
JV+Stockwork	30.4	22.3	15.6	68.3	4.56	4.80
Stockwork	31.6	25.2	11.0	67.8	1.75	1.82
Silver						

JV+Stockwork	11.4	12.1	5.5	29.0	56.1	63.8
Stockwork	8.5	7.9	5.5	21.9	6.0	6.4

Table 13-20 - Gold Recovered from Gravity Concentrate and CN Leach of Gravity Tail, YJ PQ Composites

Composite/ Feed Size (P_{80})	Au Distribution, % of total				g Au/mt mineralization					Reagent	Req.
	Au rec. from Cl. Conc.	Au rec. from gravity tail	Comb. Au rec.	Au in Tail	Extracted Cl. Conc	Extracted CN	Extracted Comb.	Tail	Calc'd Head	NaCN Cons. (kg/mt)	Lime Added (kg/mt)
YJPQ01											
150µm	50.7	43.0	93.7	6.3	4.67	3.96	8.63	0.59	9.22	0.26	2.1
75µm	58.3	37.3	95.6	4.4	4.67	3.01	7.68	0.36	8.03	0.15	1.4
YJPQ02											
150µm	56.3	38.6	94.8	5.2	4.34	3.02	7.36	0.40	7.76	0.14	1.7
75µm	64.2	31.4	65.6	4.4	3.34	1.63	4.97	0.23	5.20	0.17	1.6
YJPQ03											
150µm	25.7	62.6	88.2	11.8	0.36	0.88	1.24	0.17	1.40	0.10	1.4
75µm	25.4	67.9	93.3	6.7	0.36	0.96	1.32	0.10	1.42	0.13	1.5
YJPQ04											
150µm	23.6	49.2	72.8	27.2	0.14	0.29	0.43	0.16	0.59	0.15	1.4
75µm	24.2	49.9	74.1	25.9	01.4	0.29	0.43	0.15	0.58	0.12	2.0

Table 13-21 - Silver Recovered from Gravity Concentrate and CN Leach of Gravity Tail, YJ PQ Composites

Composite/ Feed Size (P_{80})	Ag Distribution, % of total				g Ag/mt mineralization					Reagent	Req.
	Ag rec. from Cl. Conc.	Ag rec. for gravity tail	Comb. Ag rec.	Ag in Tail	Extracted Cl. Conc	Extracted CN	Extracted Comb.	Tail	Calc'd Head	NaCN Cons. (kg/mt)	Lime Added (kg/mt)
YJPQ01											
150µm	11.7	63.4	75.0	25.0	5.5	30.1	35.6	11.9	47.4	0.26	2.1
75µm	11.7	66.5	78.2	21.8	5.5	31.6	37.1	10.4	47.5	0.15	1.4
YJPQ02											
150µm	5.8	71.2	76.9	23.1	1.2	15.0	16.2	4.9	21.0	0.14	1.7
75µm	5.9	75.0	80.8	19.2	1.2	15.5	16.7	4.0	20.6	0.17	1.6
YJPQ03											
150µm	2.3	68.9	71.1	28.9	0.2	5.6	5.8	2.4	8.2	0.10	1.4
75µm	2.2	74.9	77.1	22.9	0.2	6.2	6.4	1.9	8.3	0.13	1.5
YJPQ04											
150µm	1.9	62.3	64.2	35.8	0.1	3.1	3.2	1.8	4.9	0.15	1.4
75µm	1.8	66.5	68.2	31.8	0.1	3.4	3.5	1.6	5.1	0.12	2.0

Table 13-22 - Gold and Silver Recoveries from YJ JV Composites with Intense CN Leaching of Gravity Concentrate and CN Leach of Tail

Composite	Process	Gravity Tail Regrind (P_{80})	Leach Recoveries (%)		Reagent Requirements, kg/mt mineralization	
			Gold	Silver	NaCN	Lime
JV+Stkwrk	Conc. Int. CN w/Pretreat; Combined Tail Leach[1]	150µm	87.1	66.5	0.22	0.7
JV+Stkwrk	Conc. Int. CN w/Pretreat; Combined Tail Leach[1]	75µm	89.0	69.7	0.23	1.1
JV+Stkwrk	Conc. Int. CN no/Pretreat; Combined Tail Leach[1]	150µm	86.2	66.3	0.22	0.7
JV+Stkwrk	Conc. Int. CN no/Pretreat; Combined Tail Leach[1]	75µm	88.2	69.6	0.23	1.1
JV+Stkwrk	Conc. Int. CN w/Pretreat; Separate Tail Leach[2]	150µm	83.8	66.5	0.24	3.9
JV+Stkwrk	Conc. Int. CN w/Pretreat; Separate Tail Leach[2]	75µm	79.0	64.1	0.37	3.7
JV+Stkwrk	Conc. Int. CN w/Pretreat; Separate Tail Leach[2]	45µm	87.3	63.6	0.29	4.4
JV+Stkwrk	Conc. Int. CN no Pretreat; Separate Tail Leach[2]	150µm	79.2	66.1	0.24	3.5
JV+Stkwrk	Conc. Int. CN no Pretreat; Separate Tail Leach[2]	75µm	74.2	63.9	0.37	3.4
JV+Stkwrk	Conc. Int. CN no Pretreat; Separate Tail Leach[2]	45µm	83.1	63.2	0.29	4.0
Stockwork	Conc. Int. CN w/Pretreat; Combined Tail Leach[1]	150µm	77.6	64.1	0.15	1.0
Stockwork	Conc. Int. CN w/Pretreat; Combined Tail Leach[1]	75µm	78.1	64.6	0.11	1.2
Stockwork	Conc. Int. CN no/Pretreat; Combined Tail Leach[1]	150µm	77.0	64.2	0.14	0.9
Stockwork	Conc. Int. CN no/Pretreat; Combined Tail Leach[1]	75µm	77.6	64.7	0.10	1.1
Stockwork	Conc. Int. CN w/Pretreat; Separate Tail Leach[2]	150µm	72.6	60.8	0.11	1.2
Stockwork	Conc. Int. CN w/Pretreat; Separate Tail Leach[2]	75µm	74.2	64.4	0.10	1.3
Stockwork	Conc. Int. CN w/Pretreat; Separate Tail Leach[2]	45µm	73.7	68.0	0.11	1.8
Stockwork	Conc. Int. CN no Pretreat; Separate Tail Leach[2]	150µm	73.0	60.9	0.10	1.1
Stockwork	Conc. Int. CN no Pretreat; Separate Tail Leach[2]	75µm	74.7	64.6	0.09	1.2
Stockwork	Conc. Int. CN no Pretreat; Separate Tail Leach[2]	45µm	74.2	68.2	0.10	1.7

1)Combined recoveries and reagent consumptions for gravity concentrate at 80%-212µm, intensive cyanidation of gravity rougher concentrate at 80%-45µm regrind and leaching of gravity rougher tailings (with the intensive cyanidation residue added for re-leaching) at the indicated regrind size.

2) Combined recoveries and reagent consumptions for gravity concentrate at 80%-212µm, intensive cyanidation of gravity rougher concentrate at 80%-45µm regrind and leaching of gravity rougher tailings (with the intensive cyanidation residue added for re-leaching) at the indicated regrind size.

13.2 RESOURCE DEVELOPMENT INC. (RDI) METALLURGICAL TEST WORK 2019-2020

Resource Development Inc. (RDi) undertook metallurgical test work with the primary objective of developing a conceptual process flowsheet consisting of processing high-grade YellowJacket mineralization in the milling/gravity circuit and combining the tailing with ROM low-grade mineralization from several deposits and heap leach the tailing material.

Two tons of handpicked rock from surface cuts at the Sierra Blanca deposit and approximately 450 kgs of drill core from YellowJacket were shipped to RDi for the test work. The test work consisted of sample characterization, comminution testing, gravity concentration, leaching of gravity concentrate and tailing, thickening and filtration of gravity tailing, agglomeration of tailing and ROM mineralization and column leach testing.

13.2.1 SAMPLE CHARACTERIZATION

The ROM low-grade mineralization and the YellowJacket high-grade mineralization was crushed to P100 of 102 mm (4 inches) and P100 of 3.66 mm (6 mesh), respectively, and representative splits taken out for head analyses. The results, summarized in Tables 13-23 to 13-24, indicate the following:

- The run-of-mine (ROM) Sierra Blanca mineralization assayed 0.529 g/t Au and 1.0 g/t Ag.
- The high-grade mineralization from YellowJacket assayed 3.017 g/t Au and 25.5 g/t Ag.
- The cyanide soluble gold accounted for 90.4% and 81.6% in the low-grade and high-grade samples. The corresponding silver extractions were 28% and 92.9% respectively.
- The traces of carbon present in the mineralization is essentially organic carbon.
- Majority of the sulfur present in the low-grade mineralization is sulfate sulfur and in the high-grade mineralization is sulfide sulfur.

Table 13-23 - Head Analyses of Composite Samples

Element	Composite	
	Low Grade	High Grade
Au, g/mt	0.529	3.017
CN Soluble Au, g/mt	0.47	2.46
Ag, g/mt	1.2	25.5
CN Soluble Au, g/mt	0.28	23.70
S_{Total}, %	0.11	0.54
$S_{Sulfide}$, %	<0.01	0.51
$S_{Sulfate}$, %	0.11	0.03
C_{Total}, %	0.07	0.01
$C_{organic}$, %	0.07	0.01
$C_{inorganic}$, %	<0.01	<0.01

Table 13-23 (continued). High Grade Composite Gold Assays	
Sample	Au Assay, g/mt
Split 1 - A	**3.128**
Split 1 - B	**3.415**
Split 1 - C	**2.602**
Split 1 - Average	3.048
Split 2 - A	**2.637**
Split 2 - B	**3.067**
Split 2 - C	**2.626**
Split 2 - Average	2.777
Split 3 - A	**2.474**
Split 3 - B	**1.971**
Split 3 - C	**1.788**
Split 3 - Average	2.078
+150 mesh	**6.76**

Table 13-24 - ICP Analysis of Composite Samples

Element	Composite	
	Low Grade	High Grade
Percent		
Al	5.53	4.39
Ca	0.21	0.05
Fe	0.76	0.90
K	6.81	4.47
Mg	0.04	0.03
Na	0.08	0.06
Ti	0.06	0.05
ppm		
As	187	141
Ba	253	218
Bi	<10	<10
Cd	5	4
Co	1	2
Cr	169	204
Cu	8	22
Mn	61	429
Mo	<1	7
Ni	23	19
Pb	14	18
Sr	46	47
V	7	10
W	<10	<10
Zn	13	46

13.2.2 COMMUNITION TESTING

Bond's rod mill work index (RMWi) was generated for the YellowJacket high-grade mineralization. It was determined to be 15.8 kWh/t (Table 13-3).

13.2.3 AGGLOMERATION TESTING

Testing was conducted with various combinations of coarse low-grade material and ground high-grade material to evaluate the effectiveness of agglomeration for heap leaching. Agglomeration was completed with a combination of 1.0 g/L NaCN solution and cement. Ten-kilogram charges of sample were placed in a cement mixer without baffles for agglomeration. Cyanide solution was added by way of spray bottles while the material was mixed. Once the proper agglomeration was achieved, the amount of solution added to the sample was recorded and the agglomerates were allowed to cure for three days prior to testing.

A total of eight agglomeration tests were completed. The high-grade material was ground to P80 of 28 mesh, 48 mesh, 65 mesh, and 100 mesh. Each of the four particle sizes of high-grade material was combined with the nominal 4-inch low-grade material at ratios of 1 to 5 and 1 to 10 for testing. Each agglomeration test was completed at 2 kg/tonne cement addition.

The agglomerates produced from each test were evaluated to determine their strength and stability utilizing a wash test. The test was completed by dipping the agglomerates into a tub of water on a 10-mesh screen a total of 10 times and determining the amount of material that was washed off of the agglomerates. The results of the agglomeration evaluations are given in Table 13-25.

Table 13-25 - Agglomeration Test Results

High Grade Particle Size	Wt. % High Grade	Cement Addition (kg/mt)	Added % Moisture	% Loss in Weight
28 mesh	10	2	3.4	1.4
48 mesh	10	2	4.0	1.1
65 mesh	10	2	4.4	3.0
100 mesh	10	2	4.5	3.7
28 mesh	20	2	5.3	8.9
48 mesh	20	2	6.0	7.9
65 mesh	20	2	7.7	9.5
100 mesh	20	2	6.9	12.8

The agglomeration test results indicate that the agglomerates produced with 10% gravity tails appeared to have higher strength than the 20% gravity tail agglomerates. The strongest agglomerates were produced with the 48-mesh material. Additional cement would be necessary to improve the agglomerate strength at 20% gravity tails.

13.2.4 GRAVITY CONCENTRATION AND LEACHING OF GRAVITY TAILS

One-kilogram gravity separation testing was conducted with the high-grade composite sample at particle sizes of P80 of 28 mesh, 48 mesh, 65 mesh and 100 mesh. The samples were processed utilizing a Knelson concentrator to produce a gravity concentrate followed by a Gemini table treating the rougher gravity concentrate to produce a cleaner gravity concentrate. The gravity concentrate was submitted for assay of gold. The gravity tails were collected for bottle roll cyanide leach testing. Leach tests were completed at a pulp density of 40%, pH 11 and a maintained NaCN concentration of 1.0 g/L. Kinetic solution samples were taken at 6, 12, 24, and 36 hours. After 48 hours, the leaches were sampled. All solution and residue samples were submitted for assay of gold. The combined gravity separation and leach results are summarized in Table 13-26.

Table 13-26 - Combined Gravity and Leach Results for YellowJacket High Grade Composite

Particle Size (P_{80})	Gravity Concentrate			Leaching				Combined Au Recovery %
	Wt. %	Au Recovery %	Grade (Au g/mt)	% Au Extraction	Residue Grade (Au g/mt)	NaCN Consumption (kg/mt)	Lime Consumption (kg/mt)	
28 Mesh	0.9	12.7	44.1	50.5	1.41	0.393	1.548	56.8
48 Mesh	0.5	30.5	157.8	76.5	0.39	0.963	1.565	83.7
65 Mesh	0.5	34.5	168.3	76.7	0.38	0.964	1.509	84.7
100 Mesh	0.4	43.2	238.0	79.5	0.29	1.314	1.492	88.4

The combined gravity separation and gravity tail leaching test results indicate the following:

- The combined gold recovery was similar for the 48 mesh and 65 mesh and slightly higher for 100-mesh grind. The 28-mesh grind had significantly lower gold recovery at 56.8%
- The gravity gold recovery increased as the particle size decreased. The lowest gravity recovery was at a grind of 28 mesh (12.7%), while the highest was at a grind of 100 mesh (43.2%). The concentrate grade also increased as the particle size decreased, with a maximum grade of 238 g/mt Au at 100-mesh grind.
- Gold extraction from leaching the gravity tails were consistent with the combined recovery, ranging from 76.5% to 79.5% for the 48 mesh to 100-mesh samples while the 28-mesh sample was much lower at 50.5% Cyanide consumption was significantly higher at the 100-mesh particle size as compared to the coarse grinds.

13.2.5 THICKENING TESTS ON GRAVITY TAILING

Thickening tests were conducted with the YJ High-Grade composite at 28 mesh, 48 mesh, 65 mesh and 100 mesh. All of the samples settled very quickly. Lime addition to adjust the pH to 10.5 improved the turbidity of the overflow. Final settled densities ranging from 55% to 60% solids were achieved utilizing a high molecular weight anionic polymer. A thickener with a unit area of between 0.001 m2/mt/day and 0.041 m2/mt/day would be needed to achieve an underflow density of 55% solids, depending upon the selected particle size. The thickening data are summarized in Table 13-27.

Table 13-27 - Thickening Data for Gravity Tailings at PH=10.5

Particle Size	Final Solids (%)	Unit Area for 55% Underflow Density (m^2/mt/day)
28 mesh	60	0.001
48 mesh	57	0.008
65 mesh	55	0.077
100 mesh	58	0.041

13.2.6 VACUUM FILTRATION TESTS ON THICKENED YJ GRAVITY TAILING

Vacuum filtration testing was undertaken with the thickened solids from the thickening test work discussed in Section 13.2.4. Portions of the thickened solids from each test were placed in a vacuum filtration apparatus. The vacuum was initiated, and the slurry was allowed to filter until a cake was formed with no visible moisture on the top of the cake. Data was collected to determine the form time and cake moisture and thickness to determine the filtration rate. The filter data is summarized in Table 13-28.

Table 13-28 - Vacuum Filtration Test Data for Thickened YJ Gravity Tail

Particle Size	Form Time (min)	Cake Thickness (mm)	% Solids of Filter Cake	Filtration Rate (Dry lb./ft^2/hr.)
28 mesh	0.25	8	80.3	583.6
48 mesh	0.28	8	78.6	516.0
65 mesh	0.27	8	77.5	546.0
100 mesh	0.67	8	75.4	218.4

The leach residue filtration results indicated that a maximum percent solids of 80.3% was achieved during testing. A filtration rate of over 500 dry lb/ft^2/hr was achieved with vacuum filtration for all samples except the 100-mesh material. The filtration rate dropped by over 50% at the grind size of P80 of 100 mesh.

13.2.7 CONFIRMATION OF YJ PROCESS FLOWSHEET AT A PRIMARY GRIND SIZE OF P80 OF 48 MESH

Bench-scale 1 kg tests indicated the following:

- A primary grind of P80 -48 mesh or finer was needed to achieve acceptable gravity gold recovery.
- The cyanide leach gold extraction was similar at grind size of P80 -48 mesh or finer.
- Thickener area required increased significant at primary grind finer than P80 -48 mesh.
- The vacuum filtration rate and filter cake moisture content was reasonably good at P80 of 28 to 65 mesh and the rate dropped by over 50% at P80 of 100 mesh.

Based on these results a decision was made to generate gravity tailing for leaching with run-of-mine low-grade mineralization as well as repeat the above test work with a larger sample at P80 of 48 mesh.

Approximately 170 kg of high-grade composite was ground to P80 of 48 mesh and subjected to gravity concentration using Knelson concentrate for production of a rougher gravity concentrate and using a Model 60 Gemini table to produce gravity cleaner concentrate. The test data, given in Table 13-29, indicate a recovery 46.9% of the gold and 10.5% of the silver in a gravity concentrate assaying 930.1 g/t Au and 1,419.5 g/t Ag. The weight recovery was 0.2% of the feed.

Table 13-29 - Large-Scale Gravity Concentration Tests

Product	Assay, g/t		Distribution, %		
	Au	Ag	Wt.	Au	Ag
Gravity Cleaner Conc.	930.1	1419.6	0.2	46.9	10.5
Gravity Tailing	1.92	22.10	99.8	53.1	89.5
Cal. Feed	3.61	24.6	100.0	100.0	100.0

The gravity concentrate was leached with 5 g/L NaCN to simulate intense cyanide leach process. The test results summarized in Table 13-30 indicate the majority of the gold and silver leached in 24 hours.

Table 13-30 - Intense Cyanide Leach of YJ Gravity Concentrate

Leach Time (hrs.)	Cumulative Extraction %		NaCN Consumption, kg/t
	Au	Ag	
6	56.3	38.5	8.94

12	86.1	58.1	10.94
24	94.7	76.7	11.48
36	91.3	80.3	12.54
48	97.1	89.6	13.68
Residue, g/t	21.0	132.6	-
Cal. Feed, g/t	734.7	1280.5	-
Lime, kg/t	2.887		-

Carbon loading test was performed to determine if the pregnant solution from the leach test had any loading issues. The test data, given in Table 13-31, indicate that gold and silver are readily loaded on the carbon in two to three hours.

Table 13-31 - Carbon Loading Test Results

Sample	Solution Grade, (Mg/L)		Carbon Grade, (g/mt)		Loading % Based on Solution Assay	
	Au	Ag	Au	Ag	Au	Ag
Start	6.73	14.9	0	0	0	0
1 hr.	0.14	1.5	351.2	873.4	98.0	90.2
2 hr.	0.06	0.9	353.2	914.4	99.2	94.3
3 hr.	0.05	0.7	361.2	958.3	99.3	95.7
4 hr.	0.05	0.6	335.1	1003.7	99.3	96.4
5 hr.	0.04	0.5	325.6	889.7	99.5	97.1
6 hr.	0.05	0.5	333.4	907.2	99.5	97.1

Thickening and filtration tests were repeated on the gravity tailing sample from the large test. The results indicate the following:

- The gravity tailing at pH of 10.5 thickened to 55% solids. The unit area required was calculated to be 0.001 m2/mt/day.
- The filtration test on the thickened slurry resulted in a cake having a moisture content of 23.5% and calculated capacity of 1,369.1 dry lbs/ft^2/hr

13.2.8 COLUMN LEACH TESTING

Six column leach tests were performed to simulate the heap leach gold recovery and reagent consumptions. The four tests were run in 30.5 cm (12-inch) diameter by 1.8m (6 ft) high columns. The material tested was as follows:

- Column 1 and 2 had run-of-mine (ROM) low-grade mineralization from Sierra Blanca. The mineralization was essentially minus 4 inch. These tests were run in duplicate.

- Columns 3 and 5 were constructed of 15% mill gravity circuit tailing and 85% of ROM Sierra Blanca mineralization.
- Columns 4 and 6 were constructed of 25% mill gravity circuit tailings and 75% of ROM Sierra Blanca mineralization.

The samples for columns 1 and 2 had 1 Kg of lime blended and the mineralization was loaded into the columns. The columns were fed 1 g/L NaCN solution at a rate of 0.005 gpm/ft^2. The leach solution exiting the columns was collected and analyzed for gold and silver and free cyanide was determined. The columns were run for a total of 101 days with a four-day rest period starting day 45. The columns were shut off following draining of solution and the leach residue was dumped on a tarp and air dried. A quarter of the material, after thorough blending, was crushed to 6 mesh and a representative split taken for analyses. A similar test procedure was followed for the other four columns containing a mixture of ROM mineralization and gravity tailing.
The test results are given in Table 13-32 and Figures 13-1 to 13-6. The test results indicate the following:

- The ROM mineralization had average gold and silver extractions of 70% and 65% respectively.
- The average lime and cyanide consumptions were 5.38 kg/t and 1.16 kg/t, respectively.
- The gold extraction for the blend of 15% gravity tails and 85% ROM mineralization was 70.5-71% at 64 days of leaching (Column No. 3 and 5). The gold extraction increased to 74.9-76.4% when the blend was 25% gravity tails and 75% ROM mineralization (Column No. 4).
- The NaCN consumption for the blends of gravity tail and ROM mineralization was much lower than for ROM mineralization alone (0.585 to 0.722 kg/t versus 1.1 kg/t). Similarly, the lime consumptions were much lower for blends of gravity tail and ROM mineralization than for the ROM mineralization alone (1.45 to 1.69 kg/t versus 5.38 kg/t).

The columns with the blend of gravity tail and ROM mineralization were terminated at 64 days as compared to the ROM mineralization only columns which were terminated at 101 days. The leach kinetics indicated that the gold was still leaching in the blended columns 3 and 4. Extending the leach curves to 101 days indicated that 3% to 5% additional gold extraction could be achieved. Conservatively, an additional 3% gold extraction could be projected, which would have a positive impact on the project economic performance.

Table 13-32 - Column Leach Test Results

Parameter	Column No.					
	1	2	3	5	4	6
Feed	ROM	ROM	ROM+15% Gravity Tail	ROM+15% Gravity Tail	ROM+25% Gravity Tail	ROM+25% Gravity Tail

Leach, days	101	101	64	64	64	64
	Extraction%					
Au	65.6	74.4	71.0	70.5	76.4	74.9
Ag	62.7	67.3	79.0	79.5	86.0	84.1
	Residue, g/t					
Au	0.177	0.132	0.195	0.200	0.235	0.229
Ag	1.0	1.0	1.1	1.4	1.2	1.6
	Cal. Head, g/t					
Au	0.514	0.516	0.674	0.677	0.994	0.909
Ag	2.5	2.9	5.2	6.7	8.6	10.1
	Reagent Consumption, Kg/t					
Lime	5.365	5.397	1.46	1.68	1.448	1.686
NaCN	1.226	1.094	0.681	0.722	0.585	0.640

Figure 13-1 - Gold and Silver Extraction from Coarse Low-Grade Mineralization (Column #1)

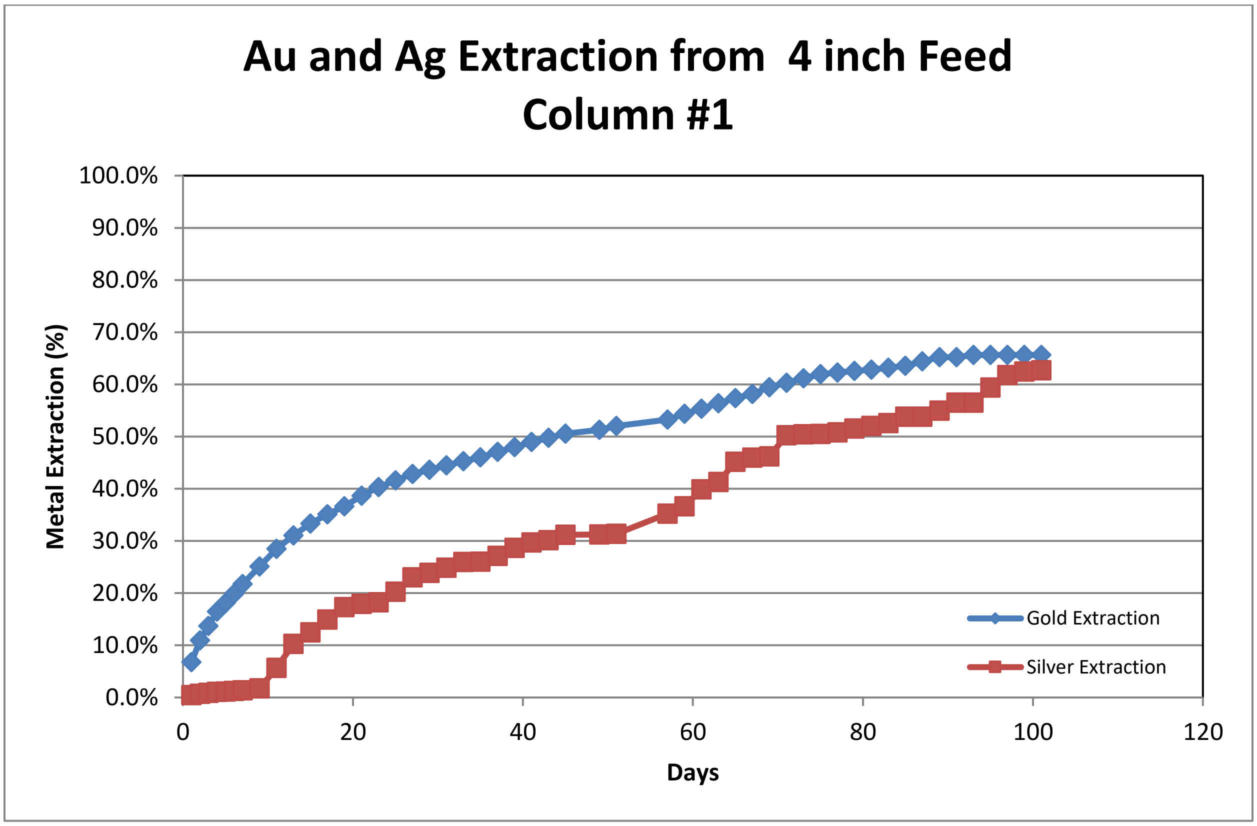


Figure 13-2 - Gold and Silver Extraction from Coarse Low-Grade Mineralization (Column #2)

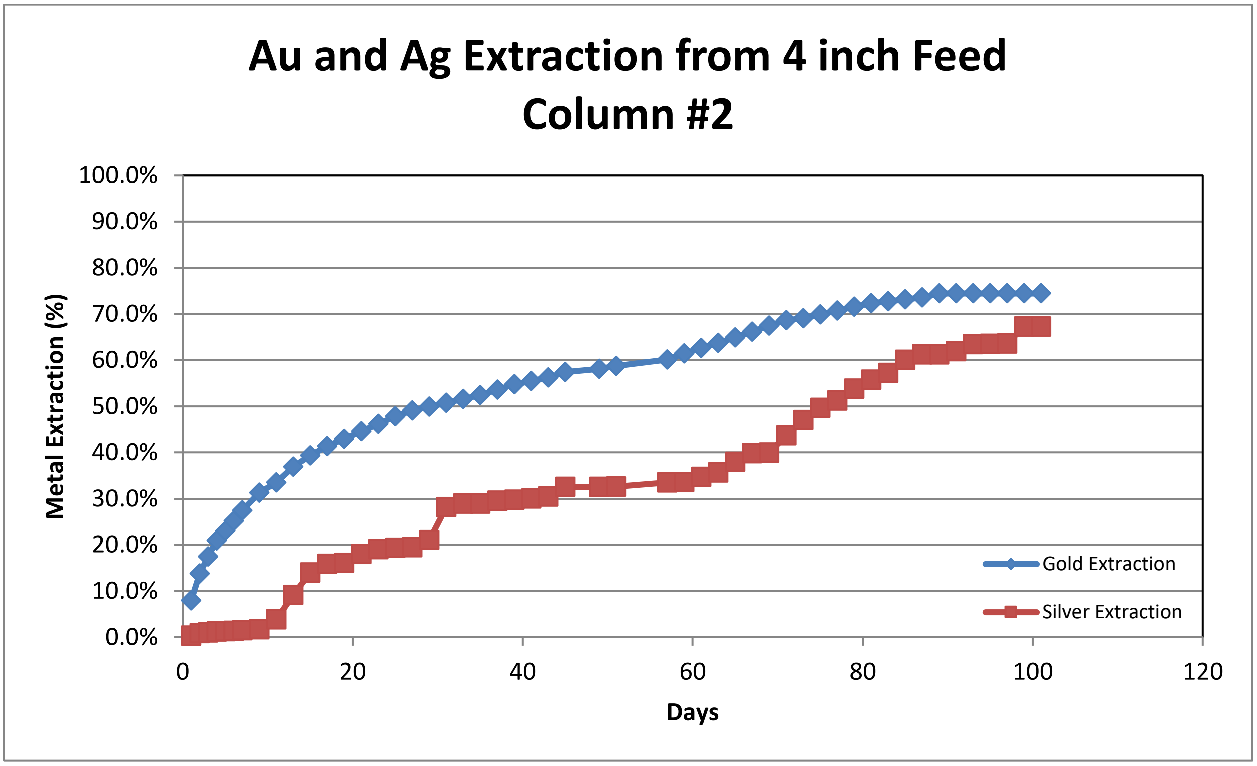


Figure 13-3 - Gold and Silver Extraction from 85% Coarse Low Grade Mineralization and 15% Gravity Tails (Column #3)

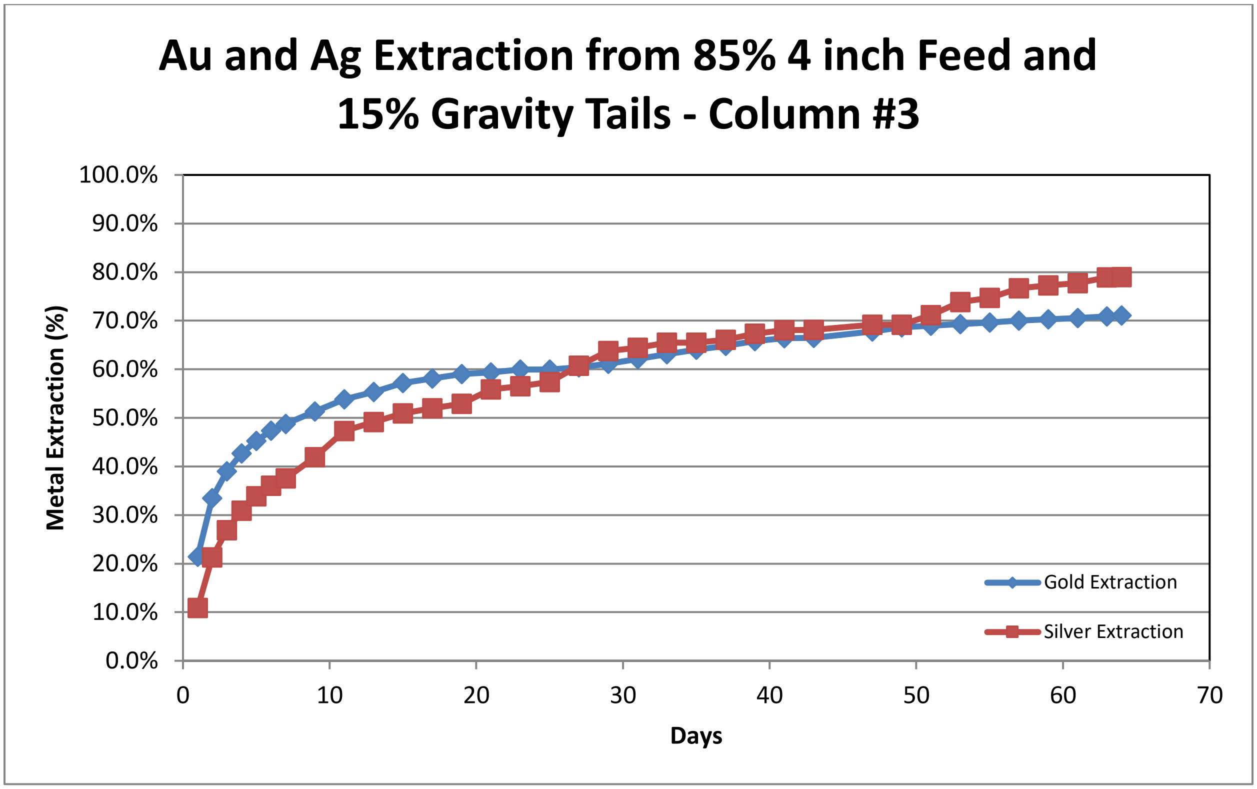


Figure 13-4 - Gold and Silver Extraction from 75% Coarse Low Grade Mineralization and 25% Gravity Tails (Column #4)

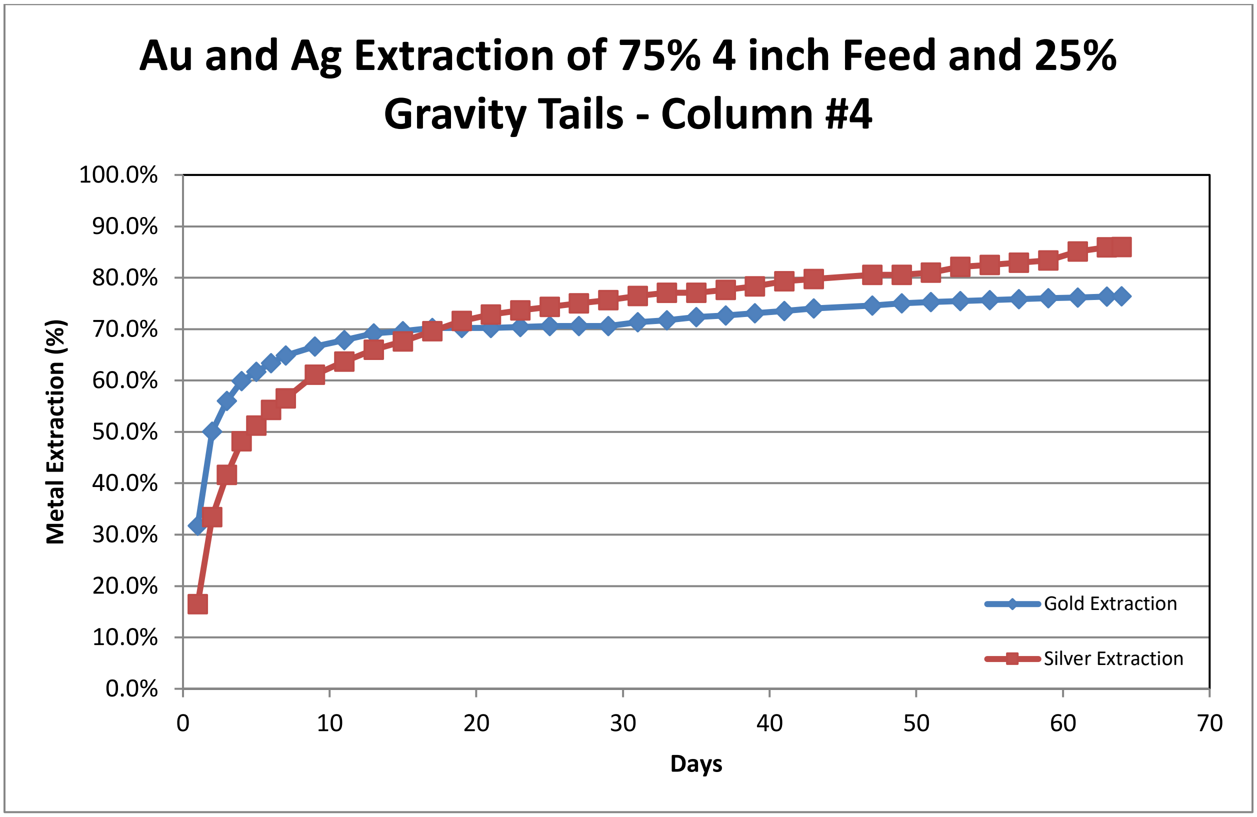


Figure 13-5 - Gold and Silver Extraction from 85% Coarse Low Grade Mineralization and 15% Gravity Tails (Column #5)

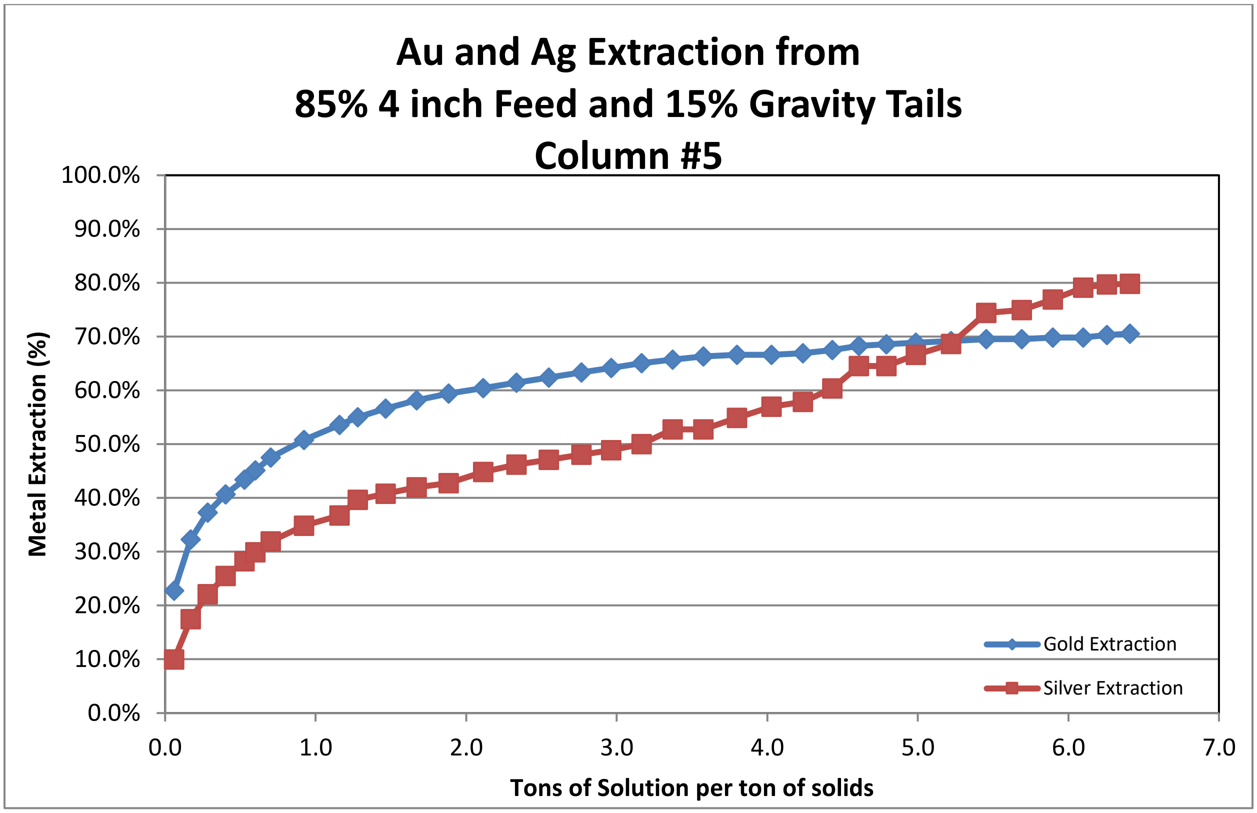
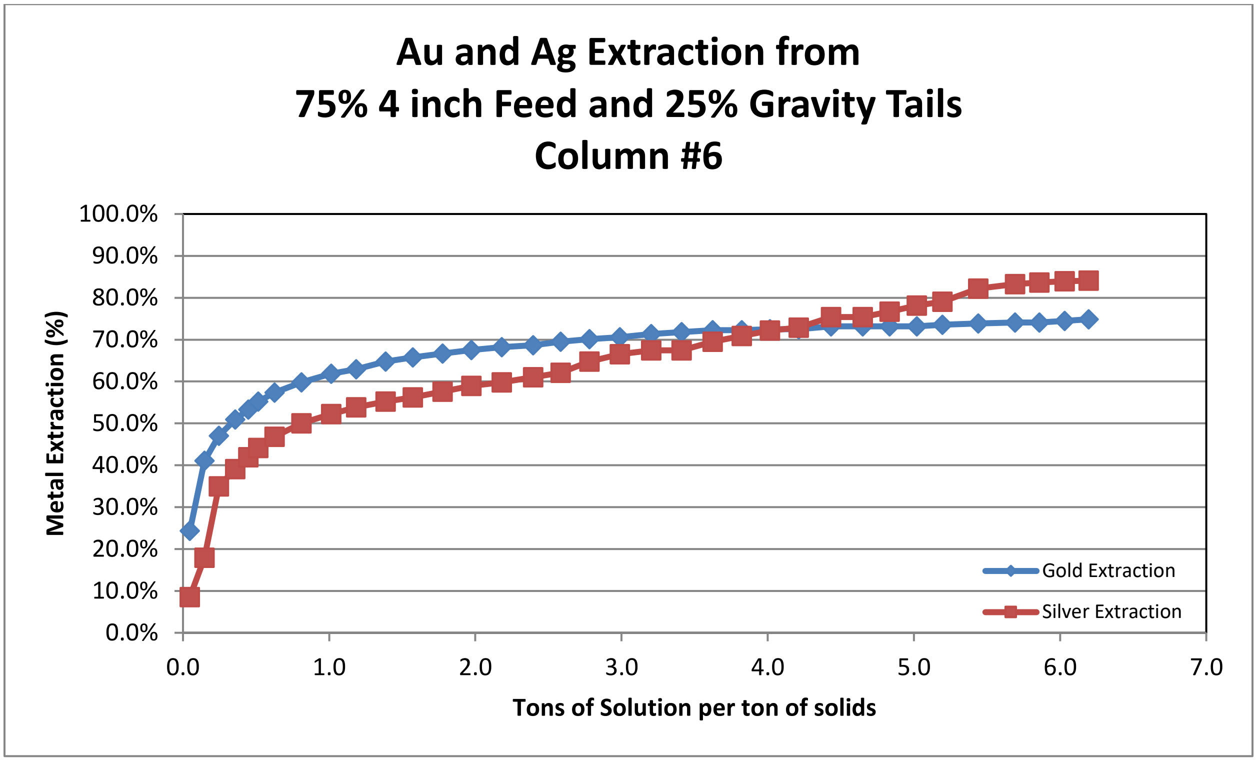


Figure 13-6 - Gold and Silver Extraction from 75% Coarse Low Grade Mineralization and 25% Gravity Tails (Column #6)



13.3 Metallurgical Conclusions

The samples tested as part of the NBP metallurgical programs and presented here are representative of the various types and styles of mineralization found at the NBP. The testing data are representative, and the qualified persons know of no known processing factors or deleterious elements that could have a significant effect on potential economic extraction.

Column leach gold recoveries of blended gravity tail material from YellowJacket vein and vein stockwork mineralization of approximately 75% were achieved for 75% ROM blended with 25% gravity tails. This is exclusive of the gold recovered in the gravity concentrate. The gravity circuit would recover 46.9% of the gold and 10.5% of the silver in the YellowJacket mineralization to the gravity concentrate. The SLR recovery would be 97% of gold and 90% of the silver contained in the gravity concentrate, therefore, the overall mill recovery would therefore be 45% of the gold and 9.5% of the silver. Projected total gold recovery of the mill feed is 85%. The blended column leach tests achieved 85% recovery of the contained silver, projected silver recovery of the mill feed is 63%.

Heap leach recoveries on ROM mineralization is projected to vary with each of the deposits as listed in Table 13-33.

Table 13-33 Project Heap Leach Recoveries by Deposit

Deposit	Projected Gold Recovery (%)	Projected Silver Recovery (%)
YellowJacket Vein and Stockwork	73.2	13.3
Sierra Blanca	73.2	13.3
Savage Valley	74.1	13.3
Jolly Jane	61.7	6.4
Mayflower	76.4	13.3

Metallurgical test results have been utilized to define design bases for the mill and heap leach facilities in Section 17.

14. MINERAL RESOURCE ESTIMATES

14.1 SUMMARY

This chapter describes the Mineral Resource estimate for Corvus' North Bullfrog Project. NBP comprises the mineral deposits of YellowJacket, Sierra Blanca, Jolly Jane and Mayflower. Each of the aforementioned deposits have been evaluated separately yet are part of the larger property of NBP. Each of these deposits has unique characteristics, yet they share common access roads, utilities, infrastructure and processing facilities. Mineral Resource estimates for the NBP contained in this Technical Report are current as of October 7, 2020.

The Sierra Blanca Mineral Resource estimate has been updated from the 2018 Technical Repot (Wilson et. al., 2018) with the addition of 44 exploration drillholes along with an updated interpretation of faults that cut off previously estimated mineralization. These faults represent hard boundaries. Definition of geologic domains have been updated based on additional information identified by drilling. The parameters for the Mineral Resource estimate were updated based upon the reinterpretation these fault locations near Air Track Hill adjacent and west of Sierra Blanca and YellowJacket. De-clustering parameters were re-evaluated and modified slightly for YellowJacket.

The Mayflower Mineral Resource estimate was updated from the 2018 Technical Report (Wilson et. al., 2018) based on a re-interpretation of geological controls and domains, which identify a high-grade corridor of mineralization through a portion of the Mayflower deposit. Mineralization transitions to lower grade disseminated mineralization in proximity to the high-grade corridor. These two separate mineralized zones share similar stratigraphy. An indicator kriging strategy was used to implicitly model and separate the high-grade corridor from low grade mineralization. Historic underground mining drifts, stopes and raises have been modeled in 3D and incorporated with the Mayflower model. These previously mined volumes have been depleted from the Mineral Resource estimate.

The Jolly Jane and Savage Valley Mineral Resource estimates are unchanged from the 2018 Technical Report (Wilson et. al., 2018) and remain current as of the effective date of this report.

The following Mineral Resource estimates have been developed according to the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines. Project mineral inventories are reported at various cut-off-grades. Mineral Resources are classified according to CIM recommendations and reported in accordance with the disclosure obligations under NI 43-101 - *Standards of Disclosure for Mineral Projects*.

Vulcan® Software was used to estimate and quantify the Project Mineral Resource estimates. Vulcan® software utilizes a block modeling approach to represent the deposit as a series of 3-D blocks to which grade attributes, and other attributes can be assigned. Mineral inventories have been pit-constrained using Whittle® in order to demonstrate the reasonable prospects of eventual economic extraction. General statistics and geostatistics were evaluated using GSLIB®, Sage2000®, Rockworks® Utilities, Excel and a variety of internally developed programs. Maps, cross sections, project layout and other visual aides were evaluated with Vulcan® and ArcGIS®.

The evaluation of Mineral Resources for the Project involved the following procedures:

- Validation of the database and wireframe models developed by Corvus;
- Data processing;
- Exploratory data analysis;
- Statistical analysis;
- Variography;
- Determination of estimation strategies writing custom estimation parameters;
- Block modelling and grade interpolation and validation of the results;
- Classification and tabulation of Mineral Resources;
- Quantifying the reasonable prospects for eventual economic extraction of Mineral Resources.

Table 14-1 lists the Mineral Resources, as classified according to CIM definitions, for the Project. Reasonable prospects for eventual economic extraction, defined in this section of the report, assume open pit mining, run-of-mine heap leach processing of oxide mineralization, mill processing of YellowJacket vein and vein stockwork mineralization with gravity recovery of a portion of the contained gold and silver, and heap leach processing of the gravity tail material. Higher grade sulphide mineralization is assumed to be stockpiled for processing in the future.

Table 14-1 North Bullfrog Pit Constrained Measured, Indicated and Inferred Mineral Resource Estimate

Mineral Resource Classification	Milling Sulphide & Oxide COG 0.204 and 0.400 g/t			Heap Leach Oxide COG 0.060 g/t			Total k-ounces		Mill k-oz		Heap Leach k-oz	
Units	K-tonnes	Au g/t	Ag g/t	k-tonnes	Au g/t	Ag g/t	Au koz	Ag koz	Au koz	Ag koz	Au koz	Ag koz
Measured	9,539	1.46	10.18	27,601	0.25	0.78	669	3,816	447	3,121	222	695
Indicated	15,130	1.21	7.61	139,867	0.19	0.62	1,438	6,490	590	3,702	848	2,788
M & I Total	24,669	1.31	8.60	167,469	0.20	0.65	2,107	10,306	1,037	6,823	1,070	3,483
Inferred	418	0.97	7.96	67,254	0.19	0.55	414	1,292	13	107	401	1,185

(1) Sulphide mineralization processing assumes bio-oxidation and would be stockpiled for future operations.

(2) The qualified person for the above estimate is Scott Wilson, C.P.G., SME.

(3) The Mineral Resources are classified as Measured, Indicated and Inferred Mineral Resources, and are based on the 2014 CIM Definition Standards.

(4) Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

(5) Mineral Resources are estimated using a gold price of $1,500/oz.

(6) Numbers may not add up due to rounding.

(7) The effective date of this Mineral Resource estimate is October 7, 2020.

(8) The quantity and grade of reported inferred Mineral Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these inferred Mineral Resources as indicated or measured Mineral Resources.

(9) Mill sulphide mineralization cut-off grade was 0.40 g/t

(10) Mill oxide mineralization cut-off grade was 0.20 g/t

(11) Heap Leach oxide mineralization cut-off grade was 0.06 g/t

(12) The qualified person knows of no environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that may materially affect the Mineral Resource estimates in this Technical Report.

14.2 SIERRA BLANCA AND YELLOWJACKET

14.2.1 SIERRA BLANCA GEOLOGY MODEL

Geologic models of the Sierra Blanca and YellowJacket litho-stratigraphy were provided by Corvus to use for the 2020 Mineral Resource estimate. Items such as material density, structure, stratigraphic order, volumes and barren material can be identified and used for proper determination of Mineral Resource. Final modelling results mirror the descriptions found in chapters 7 and 8 of the Technical Report. The Sierra Blanca and YellowJacket geology solids were checked for topologic consistency.

14.2.1.1 OXIDATION MODEL

With the exception of a few historical holes, the degree of oxidation has been determined for every sample in the drill database. Oxidation is evaluated on a scale of 1 to 5 with 1 being un-oxidized and 5 being completely oxidized. Cyanide shake leach and bottle roll tests have shown that oxidation levels 4 and 5 behave similar to level 3, which has mixed oxide and sulphide. Level 2 and 1 are dominantly un-oxidized. Oxidation states need to be modelled correctly in order to apply appropriate assumptions in the determination of the reasonable prospects of eventual economic extraction.

Oxidation was estimated based using indicators. Three variables were added to the composite database based upon oxidation picks:

- Oxide_1 (sulphide) set to 1 if logged sulphidation is 1 or 2, set to 0 if not
- Oxide_3 (mixed) set to 1 if logged sulphidation is 3, set to 0 if not
- Oxide_5 (oxide) set to 1 if logged sulphidation is 4 or 5, set to 0 if not

Probabilistic oxidation states were estimated in three runs based on the three above described indicator sets. Results stored the probability of a block being oxide, mixed or sulphide. Blocks are flagged based upon the highest estimated probability. Table 14-2 lists the oxidation variogram models used to establish estimation parameters.

Table 14-2 - Oxide Indicator Variogram Parameters

Oxidation Type	Nugget	Sill Differential	Major Radius	Semi-Major Radius	Minor Radius	Rotation about Z axis	Rotation about Y axis	Rotation about X axis
Oxide	0.05	0.75	750	390	75	138	0	-3
Mixed	0.76	0.26	430	250	115	152	-9	-6
Sulphide	0.07	0.83	920	475	85	143	-2	-1

14.2.2 MINERALIZATION MODEL

Mineralization at Sierra Blanca and YellowJacket occur in two distinct settings: low-grade disseminated mineralization and higher grade structurally controlled mineralization. Geological models have been constructed to reflect these two styles of mineralization. Two structurally controlled mineralization structures were modeled; the YellowJacket vein (Figure 14-1) and the Liberator vein (Figure 14-2). Two separate grade shells were modeled to represent the higher grades associated with the vein zones. A 0.5g/ton Gold cutoff was used to estimate these domain volumes. These two high-grade mineralized zones allow for estimation parameters to be applied to only these vein zones and to limit the effects of high-grades on grade estimation in the surrounding disseminated mineral zones.

A tabular horizon, the Swale Zone, which is interpreted as a high-grade lens within disseminated mineralization, has been modelled just to the north and east of YellowJacket (Figure 14-3). As with YellowJacket and Liberator, a 0.5 g/t Gold grade shell was created to characterize this lens and allow for separate estimation parameters to be applied for evaluation.

Disseminated mineralization has been modeled and limited according to stratigraphy models. Paleozoic basement stratigraphy comprising micritic limestone, quartzite and sandstone was considered to be barren of any mineralization. No mineralization was interpolated into the basement rocks. Likewise, the Donovan Mountain has been identified as a barren unit for the purposes of mineral resource estimates. Contact profiles suggest that gold and silver mineralization may be present through the majority of the volume in the remaining stratigraphic units at Sierra Blanca.

Mineral domains used to control mineral estimates are:

- Sierra Blanca: Pervasive disseminated gold and silver mineralization and represented in all units.
- YellowJacket: High-grade faulting and stockwork veining (Figure 14-1).
- Liberator: High-grade mineralization, faulting and stockwork veining (Figure 14-2).
- Swale: High-grade mineralization (Figure 14-3).

Figure 14-1 - Isometric View of Modeled YellowJacket Mineralization Grade Shell (Blue)

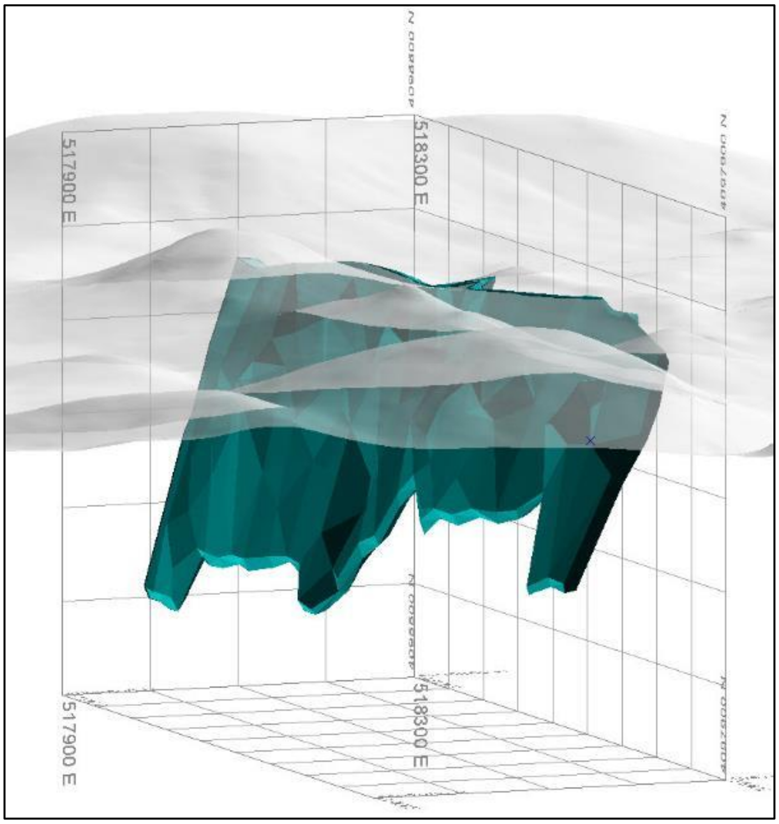


Figure 14-2 - Isometric View of YellowJacket (Blue) and Liberator Veins (Green)

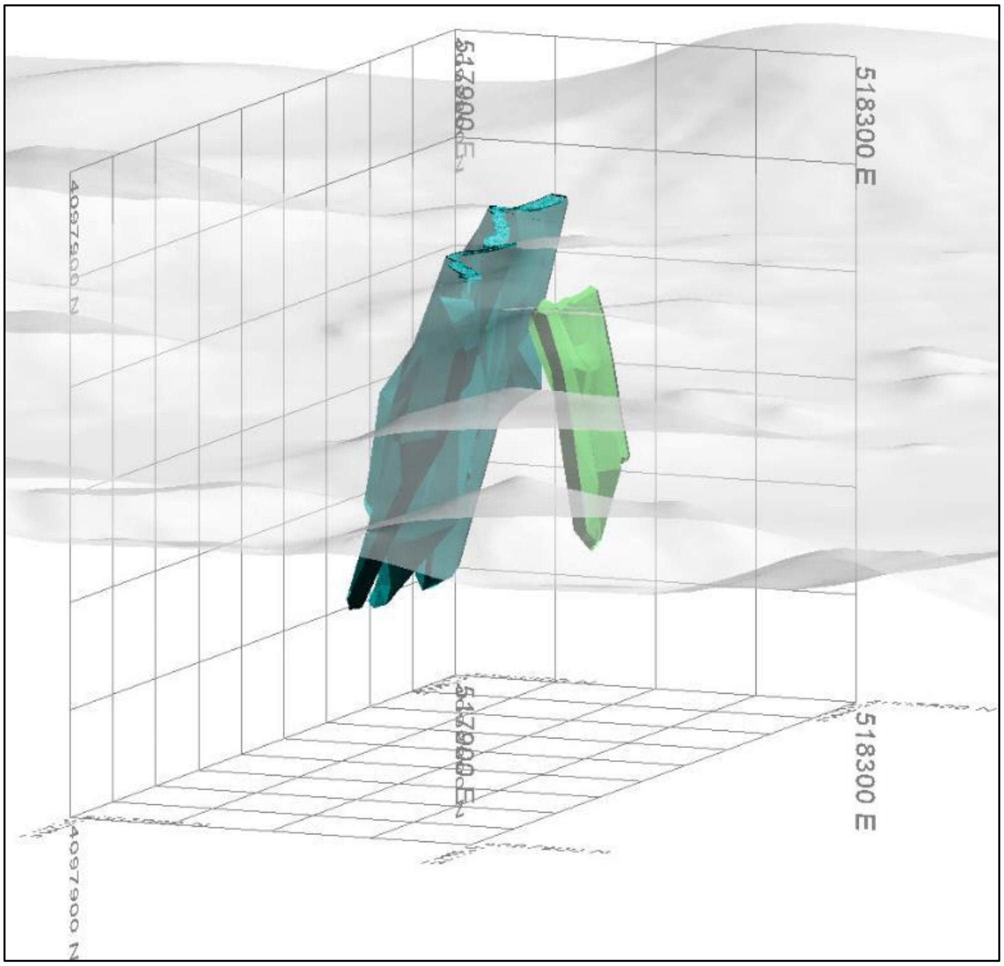
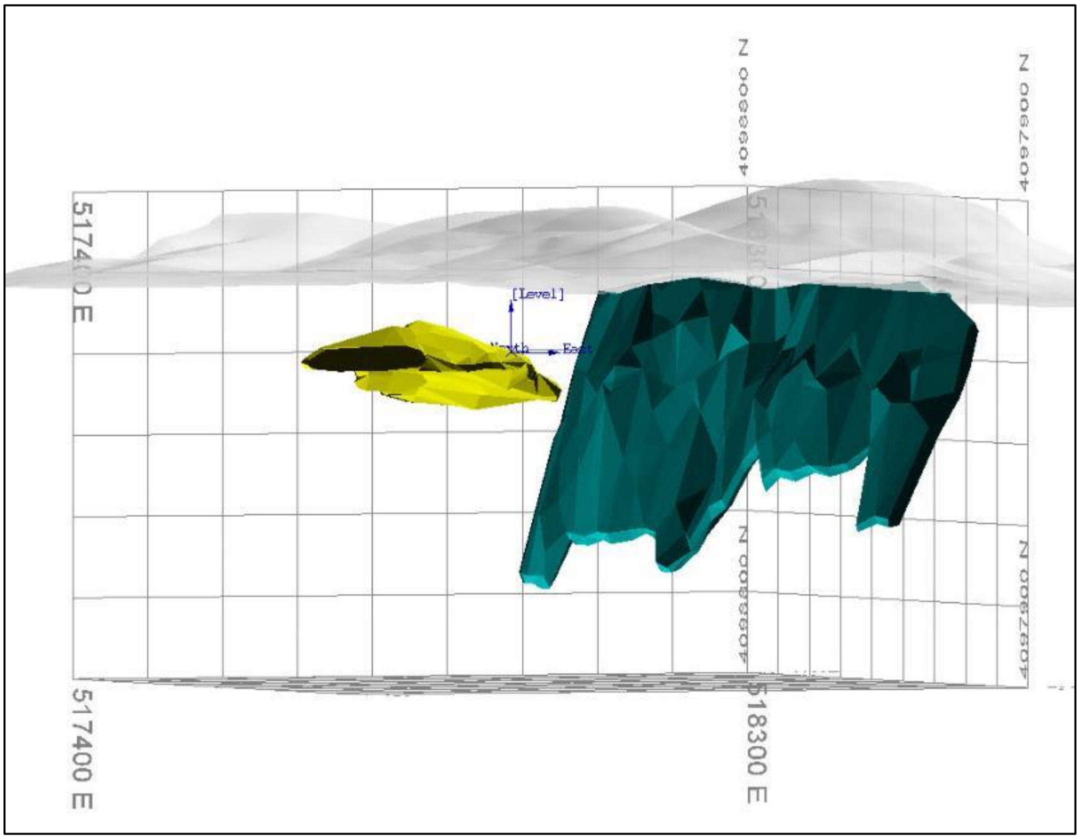


Figure 14-3 - Isometric View of YellowJacket (Blue) and Disseminated Swale (Yellow)



14.2.3 EXPLORATORY ANALYSIS

14.2.3.1 ASSAY STATISTICS

A subset of the North Bullfrog drilling database, comprising 387 holes, was selected for mineral resource estimates at Sierra Blanca-YellowJacket. Sampled intervals of 60,533 assays were provided to be used for the mineral estimate. Table 14-3 lists the gold assay statistics for each of the modeled geologic domains. Table 14-4 lists the silver assay statistics.

Table 14-3 - Gold Assay Statistics Sorted by Zone

Zone	Number	Mean Au (g/t)	Stand. Dev.	Min Assay	Max Assay	Coefficient of Variation
YellowJacket Vein	3,097	2.862	14.402	0.0005	431.0	5.032
Liberator Vein	218	0.876	2.103	0.001	17.299	2.402
Swale	657	0.660	0.500	0.0005	6.390	0.758
Disseminated	56,561	0.143	1.480	0.0005	209.0	10.372
All	60,533	0.290	3.610	0.0005	431.0	12.446

Table 14-4 - Silver Assay Statistics Sorted by Zone

Zone	Number	Mean Ag (g/t)	Stand. Dev.	Min Assay	Max Assay	Coefficient of Variation
YellowJacket Vein	3,097	23.910	162.617	0.0005	7590.0	6.803
Liberator Vein	218	2.489	5.543	0.030	65.0	2.227
Swale	657	1.805	3.236	0.0005	61.0	1.793
Disseminated	56,561	0.609	2.541	0.0005	308.0	5.818
All	60,533	1.821	37.294	0.0005	7590.0	20.485

14.2.3.2 CAPPING

Grade distributions for gold and silver within each of the different zones were examined to determine if capping was required and, if so, at what value. Assays for each zone were graphically displayed as histograms and as lognormal probability plots. YellowJacket samples were capped at 100 g/t Au and 550 g/t Ag. Both of these capping levels were determined to be where a natural break occurred in the lognormal distribution curve around the 99.5th percentile. Figures 14-4 and 14-5 show the lognormal distributions of uncapped gold and silver assays within the YellowJacket Vein. Disseminated samples were capped at 15 g/t au and 200 g/t ag. The lognormal distribution curves both have smooth distributions, so capping levels were determined to be at the 99.9th percentile. Figures 14-6 and 14-7 show the lognormal distributions of the uncapped gold and silver assays within disseminated mineralization. Assays in the Liberator and Swale zones were unaffected by assay capping. Table 14-5 describes the number of gold and silver assays capped for Mineral Resource estimates.

Figure 14-4 - Au Lognormal Graph (within YellowJacket)

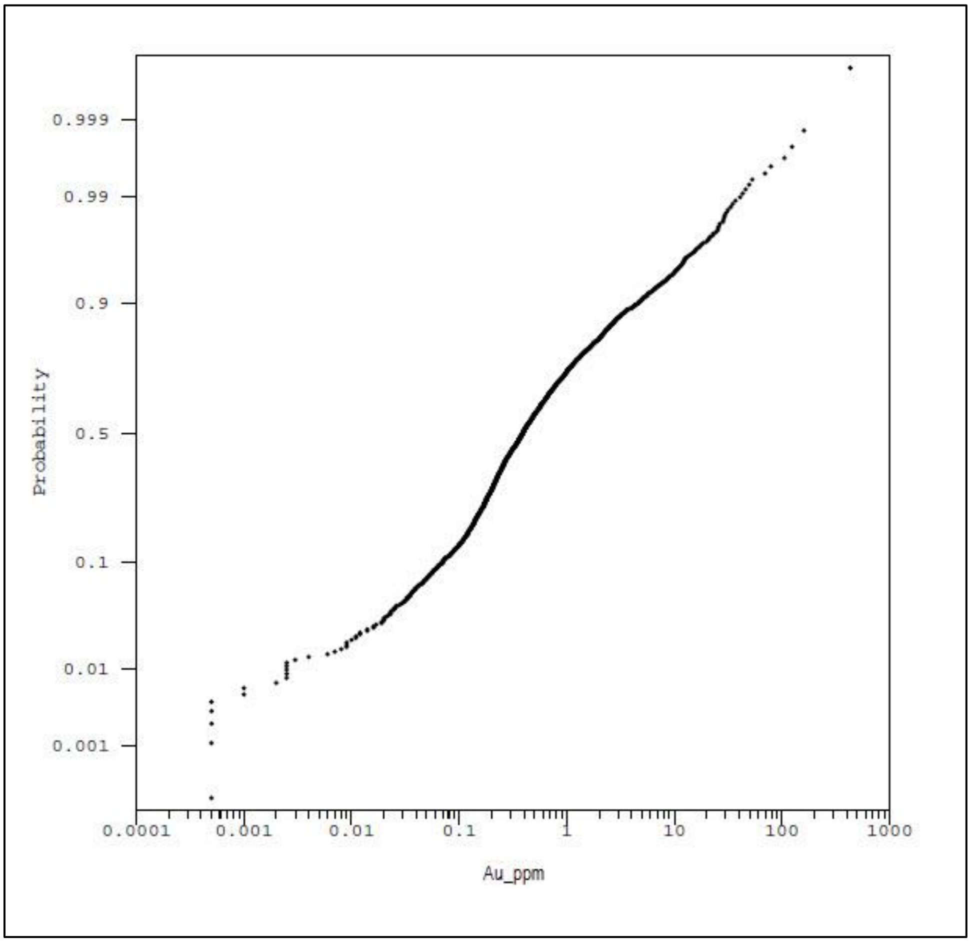


Figure 14-5 - Ag Lognormal Graph (within YellowJacket)

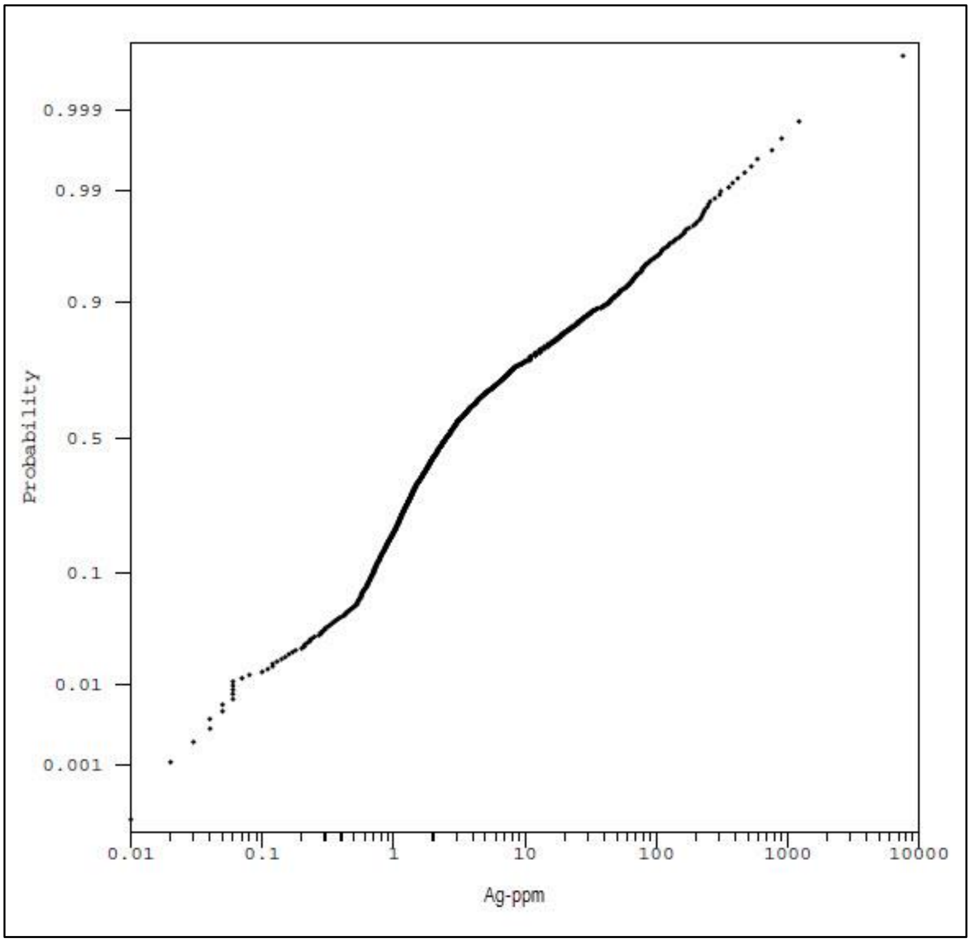


Figure 14-6 - Gold Lognormal Graph (Disseminated)

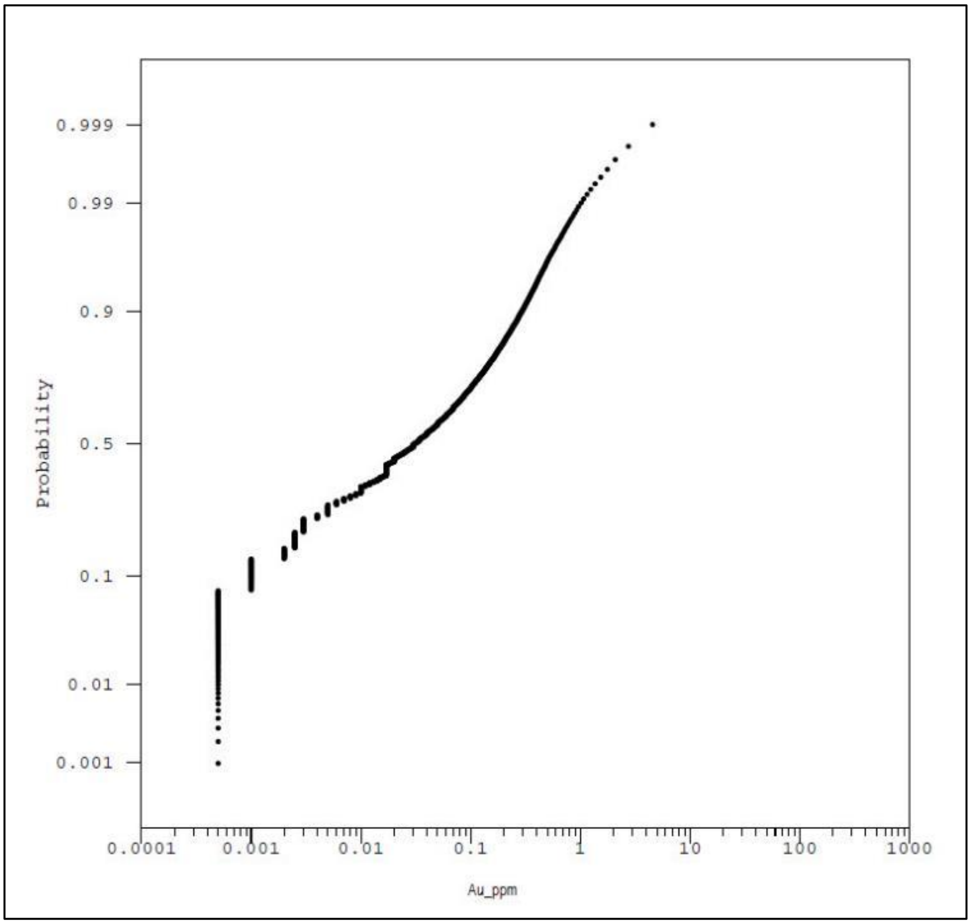


Figure 14-7 - Ag Lognormal Graph (Disseminated)

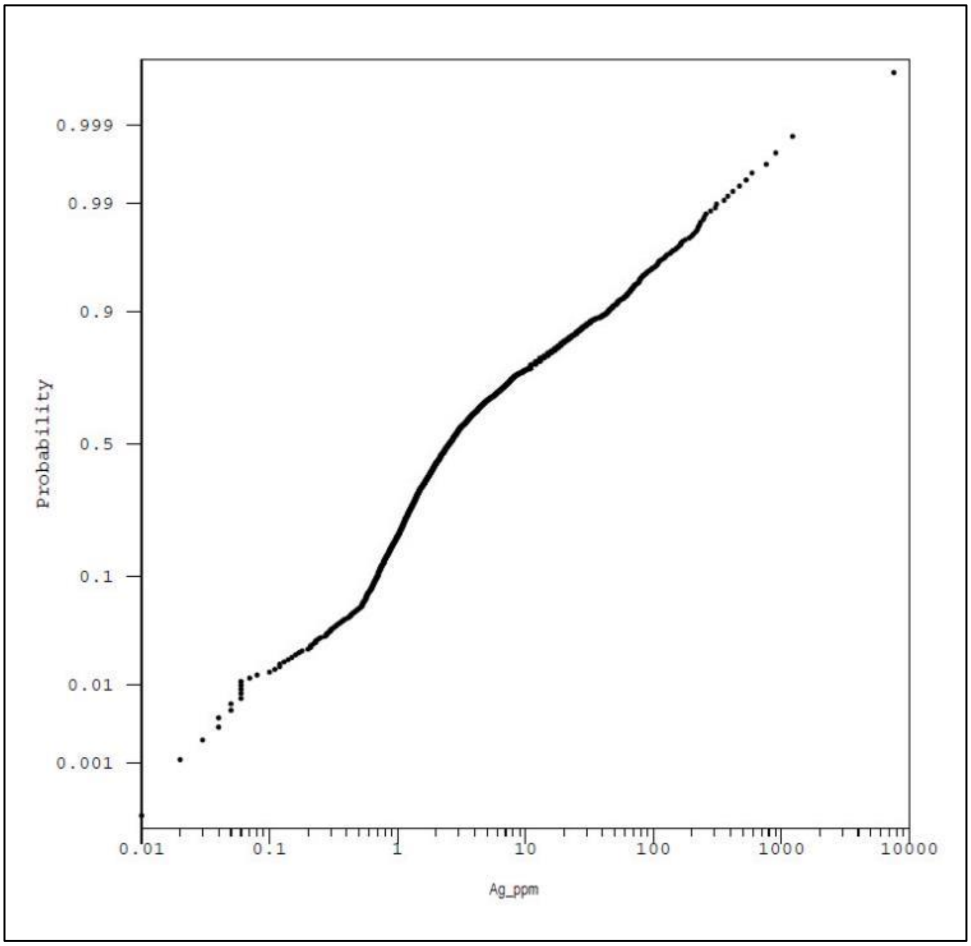


Table 14-5 - Capped Assays by Zone

Zone	Au Cap (g/t)	Number Capped	Ag Cap (g/t)	Number Capped
YellowJacket	100	15	550	15
Liberator	100	0	100	0
Swale	10	0	200	0
Disseminated	10	27	200	6

The results from capping (Tables 14-6 and 14-7) show reduced standard deviation and coefficients of variation in all groups when compared to the uncapped assays (Tables 14-3 and 14-4).

Table 14-6 - Capped Assay Statistics for Gold Sorted by Zone

Zone	Number	Mean Au (g/t)	Stand. Dev.	Min Assay	Max Assay	Coefficient of Variation
YellowJacket	3,097	2.525	8.699	0.0005	100.0	3.444
Liberator	218	0.875	2.103	0.001	17.299	2.402
Swale	657	0.659	0.500	0.0005	6.390	0.758
Disseminated	56,561	0.129	0.308	0.0005	14.22	2.390

Table 14-7 - Capped Assay Statistics for Silver Sorted by Zone

Zone	Number	Mean Ag (g/t)	Stand. Dev.	Min Assay	Max Assay	Coefficient of Variation
YellowJacket	3,097	19.321	59.378	0.010	550.0	3.073
Liberator	218	2.489	5.543	0.030	65.0	2.227
Swale	657	1.805	3.236	0.0005	61.0	1.793
Disseminated	56,561	0.607	2.984	0.0005	200.0	4.936

14.2.4 COMPOSITE FILES USED FOR GRADE ESTIMATION, VARIOGRAPHY AND STATISTICS

Capped drill hole assays for Sierra Blanca were composited using five (5) meter down–the–hole composite lengths. Composite lengths were chosen based on the anticipated mine selectivity of five meters which corresponds to the block size length in the z-direction. A total of 17,256 five (5) meter gold and silver composites were constructed. Intervals with missing assays were ignored and a new composite centroid was generated at that point. A merge tolerance of 2.5 meters was used to limit the number of “short” composites lengths in the database. Only 12 samples composites measured less than 5 meters. Composite statistics are compiled and listed in Table 14-8 and 14-9.

Table 14-8 - Composite Statistics for Gold Sorted by Zone

Zone	Number	Mean Au (g/t)	Stand. Dev.	Min Assay	Max Assay	Coefficient of Variation
YellowJacket Vein	639	1.487	3.120	0.001	37.096	2.099
Liberator Vein	56	0.869	1.635	0.005	8.825	1.882
Disseminated Swale	190	0.669	0.430	0.006	3.537	0.652
Disseminated	16,371	0.123	0.195	0.0005	4.498	1.589
All	17,256	0.186	0.691	0.0005	37.096	3.804

Table 14-9 - Composite Statistics for Silver Sorted by Zone

Zone	Number	Mean Ag (g/t)	Stand. Dev.	Min Assay	Max Assay	Coefficient of Variation
YellowJacket Vein	639	11.537	28.466	0.018	349.659	2.467
Liberator Vein	56	2.209	2.879	0.043	12.945	1.303
Disseminated Swale	190	1.811	2.651	0.001	32.104	1.464
Disseminated	16,371	0.541	1.549	0.0005	88.967	2.865
All	17,256	0.967	6.055	0.0005	349.659	6.259

14.2.5 BLOCK MODEL

The Mineral Resource model contains information about the deposit and is stored variably in each block. The information stored includes:

- Estimated characteristics of Au, Ag, S and Oxide
- Percentage of block below the surface topography
- Specific gravity defined by geologic triangulations
- Stratigraphic Unit
- Percentage of a block found within a vein and percentage of a block found within the disseminated material.

Table 14-10 outlines the framework for the Sierra Blanca block model.

Table 14-10 - Sierra Blanca Block Model Framework

Item	Easting	Northing	Elevation
Block Model Reference Point	516760	4096460	720
Number of Blocks	197	259	148
Parent Block Size	10	10	5

14.2.6 BULK DENSITY

A total of 1,365 specific gravity measurements were used to define the density value of each block based on modeled lithology types. Basic statistics were compiled and listed in Table 14-11. The final density values assigned to the model were derived by eliminating 10% of the lowest and highest density values for each lithology type and using the mean value. Table 14-12 lists specific gravity by Stratigraphy (as opposed to lithology).

Table 14-11 - Lithology Types and Corresponding Specific Gravity Measurements

Lithology	All Samples				Minus 10% of Lowest and Highest Values			
	Count	Min	max	mean	Count	min	max	mean
Post SB	462	1.88	2.64	2.33	370	2.15	2.49	2.34
Mélange	40	1.74	2.53	2.34	32	2.2	2.49	2.36
SB Middle	484	2.04	2.63	2.42	387	2.29	2.54	2.43
SB Lower	165	2.19	2.58	2.46	132	2.33	2.55	2.47
Pre SB	1	1.86	1.86	1.86	1	1.86	1.86	1.86
Camb	1	2.56	2.56	2.56	1	2.56	2.56	2.56
Gravel	1	1.85	1.85	1.85	1	1.85	1.85	1.85
Unknown Default	1	1.85	1.85	1.85	1	1.85	1.85	1.85
Rhyolite_9	68	1.85	2.50	2.18	54	2.00	2.41	2.18
Rhyolite	142	2.06	2.60	2.43	114	2.25	2.55	2.44

Table 14-12 - Specific Gravity by Stratigraphic Unit

Stratigraphy	Specific Gravity
PZ_Basement	2.56
Tsf	1.86
Tnb1	2.18
Tpf	2.47
Tsb	2.43
Tdi	2.43
Td	2.34
Tlr	2.34
Trm	2.34
YellowJacket	2.36

14.2.7 CONTRACT PROFILES

A contact profile analysis investigates the relationships between assay values in relation to the contact of geological units. This analysis was used to identify separate mineral estimation domains based on assay grades in relation to the stratigraphic units. This method takes samples from one stratigraphic unit and pairs it with samples from another stratigraphic unit based on a separation distance. The pairs are constructed over an increasing separation distance. The average grade of the first unit is plotted against the average grade calculated with the second unit. Figure 14-8 is the contact profile comparing gold assays within the YellowJacket vein vs all other stratigraphic unit within 30 meters. Figure 14-9 is the contact profile again, but with silver assays. These analyses confirm that the YellowJacket contains much higher gold and silver grades than the surrounding stratigraphy and should be used as its own estimation domain.

Figure 14-8 - YellowJacket Contact Profile (Au)

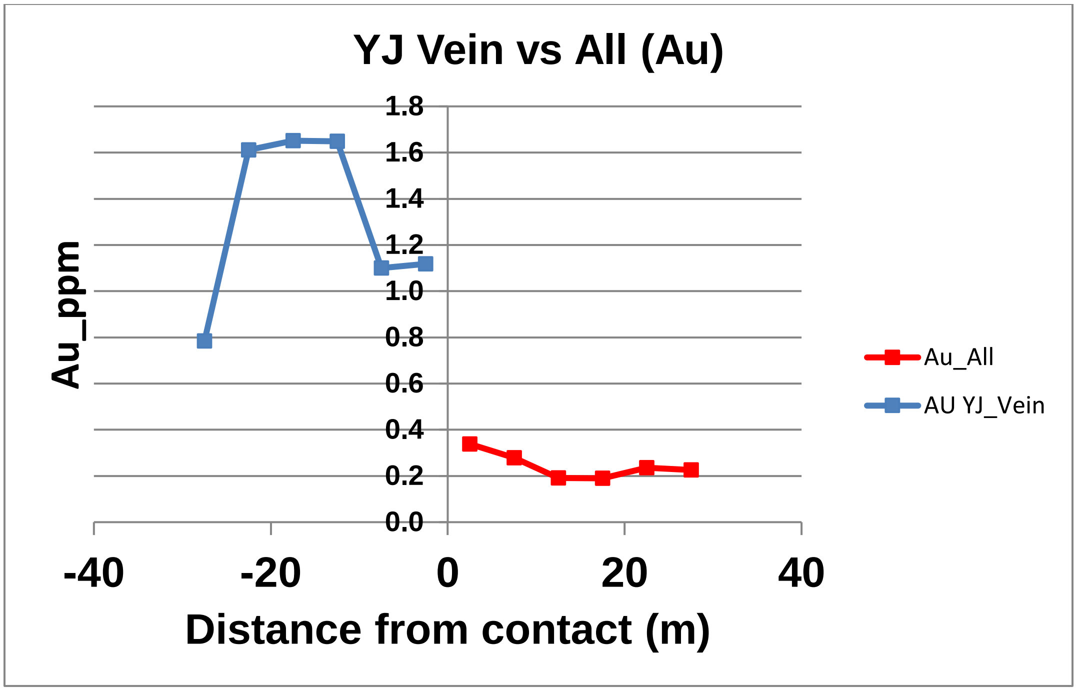
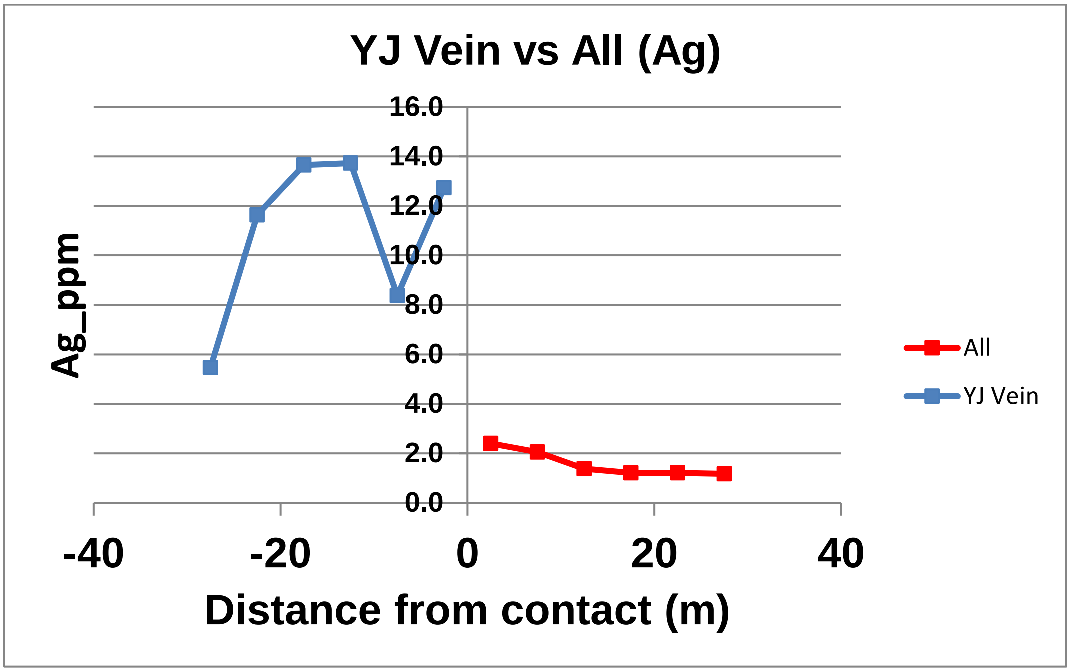


Figure 14-9 - YellowJacket Contact Profile (Ag)



Sierra Blanca stratigraphic units, external to YJ, were evaluated to determine if separate individual estimation domains were required outside of YellowJacket, Liberator and Swale. No additional domains were identified. Figures 14-10 and 14-11 are examples.

Figure 14-10 - Savage Formation vs Pioneer Formation Contact Profile (Au)

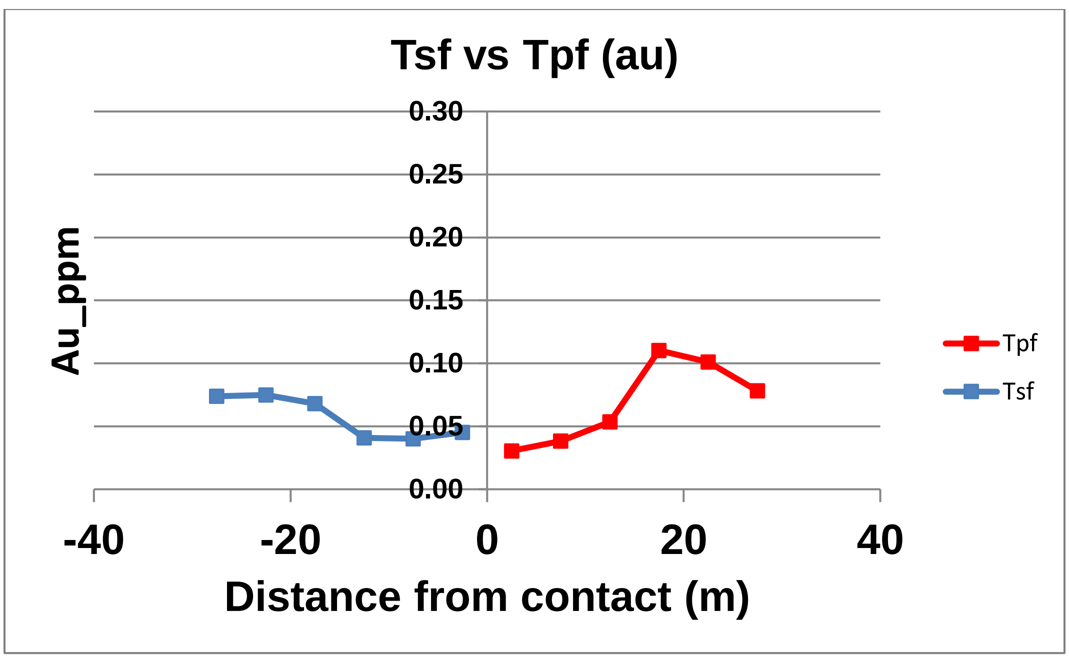
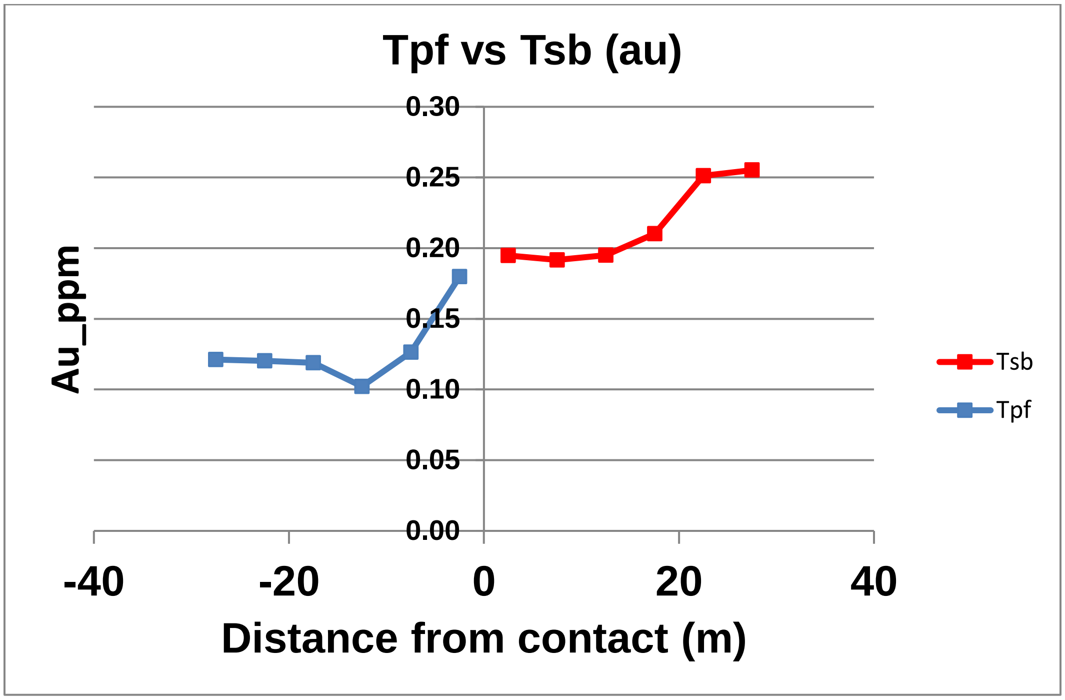


Figure 14-11 - Pioneer Formation vs Sierra Blanca Formation Contact Profile (Au)



Contact profile analyses for disseminated mineralization demonstrate consistent grades across all stratigraphic units.

14.2.8 DECLUSTERING OF YELLOWJACKET COMPOSITES

Due to the significant differences in metal content in YellowJacket mineralization as compared to the rest of the mineral deposit, Corvus drilled many holes with abundant preferential sampling of the vein. Therefore, Yellowjacket drilling is regularly spaced and dense. Declustering was applied in order to de-bias the assay data. Multiple kriging and inverse distance estimates ("ID") were evaluated in order to determine the most realistic Gold and Silver estimates for Sierra Blanca-YellowJacket. Kriging declusters data points used for block grade estimation, ID does not, therefore declustering statistics needed to be determined. Cell declustering was performed using the nearest neighbor declustering technique. Each point receives a weight inversely proportional to the number of points that fall within the same cell. Decluster weights are scaled to a mean of 1.

The weights depend on the cell size. When a cell size is very small, each datum is in its own cell and receives an equal weight. When the cell size is very large, all data fall into one cell and are equally weighted (Rossi and Deutsch 2014). To choose the appropriate cell, the declustered mean versus a range of cell sizes was plotted. Figure 14-12 is a graph showing the cell size versus the mean grade for YellowJacket. The lowest mean value along the curve was the cell size to be used. Figure 14-13 is a graph used for declustering the silver grades. Cell sizes used for gold and silver are 135 and 110 meters, respectively.

Figure 14-12 - YellowJacket Au Declustering

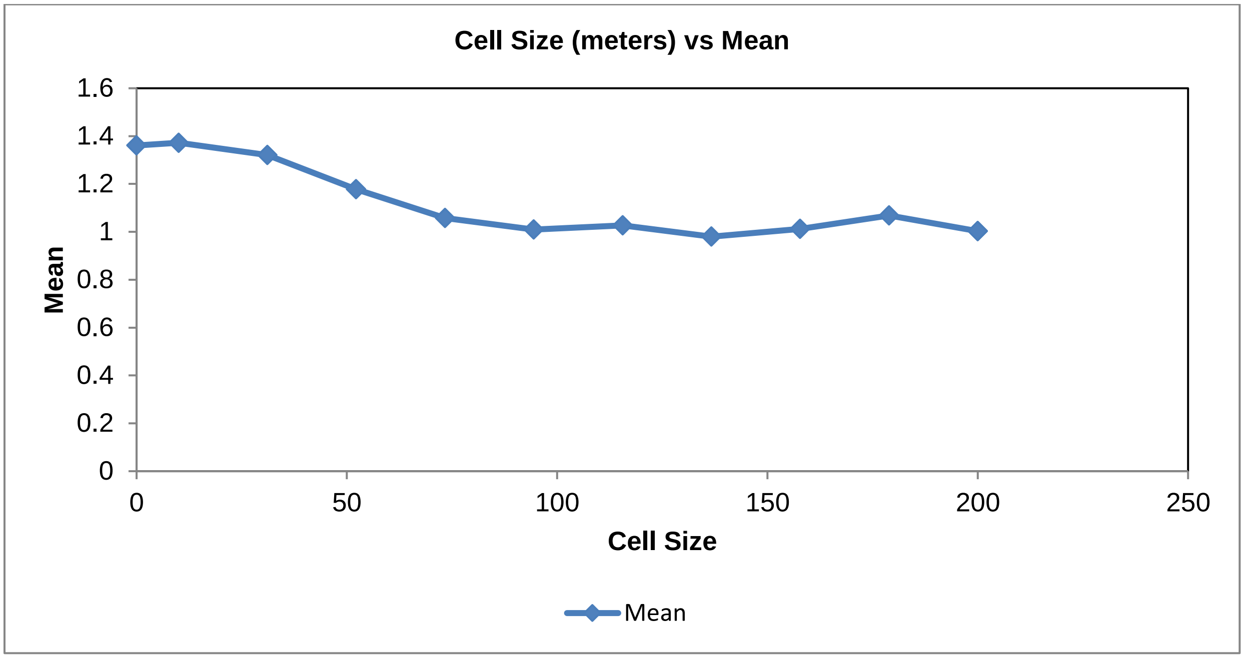
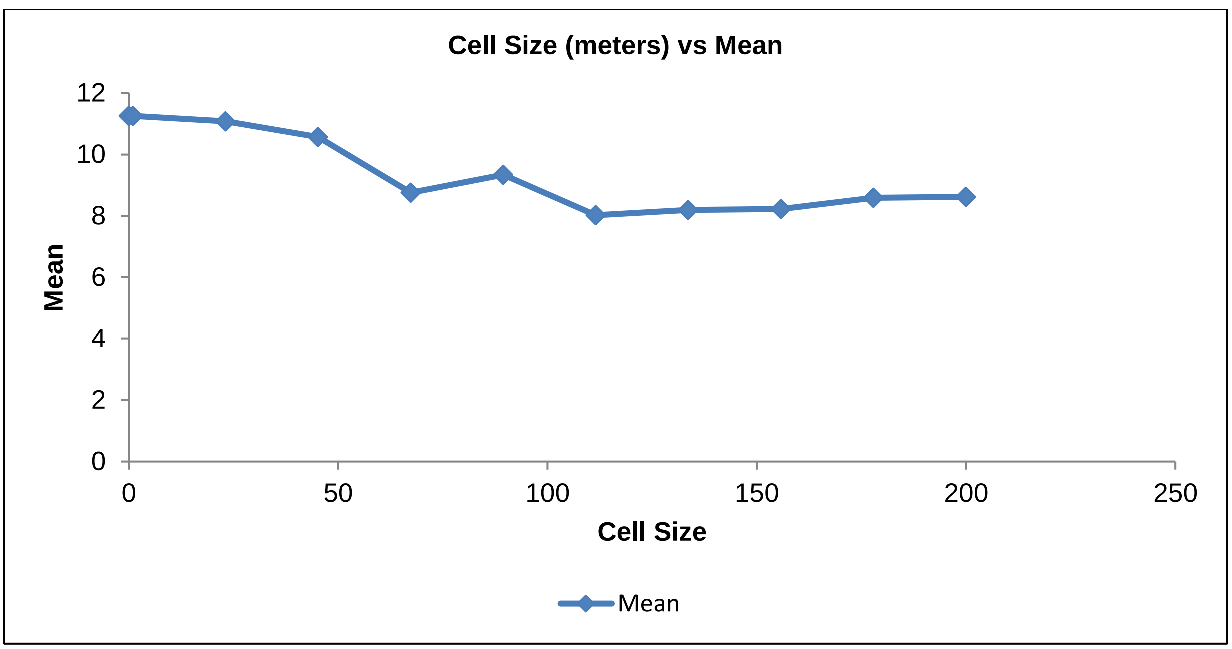


Figure 14-13 - YellowJacket Ag Declustering



14.2.8.1 YELLOWJACKET MINERALIZATION

YellowJacket mineralization metal grades were interpolated using the inverse distance squared ("ID2") estimation technique. Search ellipsoids were oriented to mimic the shape of the YellowJacket vein and vein stockworks. Search ellipsoid orientations were assigned in the block model based on the geographic bearing and plunge changes within the vein. Table 14-13 lists the breakdown of the unique orientations used for the estimate. Figure 14-14 is a plan view illustrating the change in the strike of the YellowJacket structure. Figure 14-15 is a cross section looking north showing orientation changes in the plunge of the vein with depth. A two-pass estimation was performed to ensure mineralization was estimated into the entire shape.

Table 14-13 - YellowJacket Vein Estimation Parameters for Au and Ag – Vulcan Software

YellowJacket Vein Estimation Parameters			
Estimation Type	Inverse Distance Squared (ID2)		
Search Ellipsoid	**Bearing**	**Plunge**	**Dip**
Northing > 4098500	100		0
Northing 4098275 – 4098500	65		0
Northing 4098225 – 4098275	120		0
Northing 4098175 – 4098225	90		0
Northing < 4098175	95		0
Elevation > 1140		65	0
Elevation < 1140		80	0
Search Ellipse	**Major Axis**	**Semi-Major Axis**	**Minor Axis**
Pass 1	100	60	20
Pass 2	180	130	20
Samples	**Min**	**Max**	
	4	20	
Maximum Samples per Drill hole	**Max**		
	2		

Figure 14-14 - YellowJacket Vein Estimation Bearing Changes (Elevation 1,150 m)

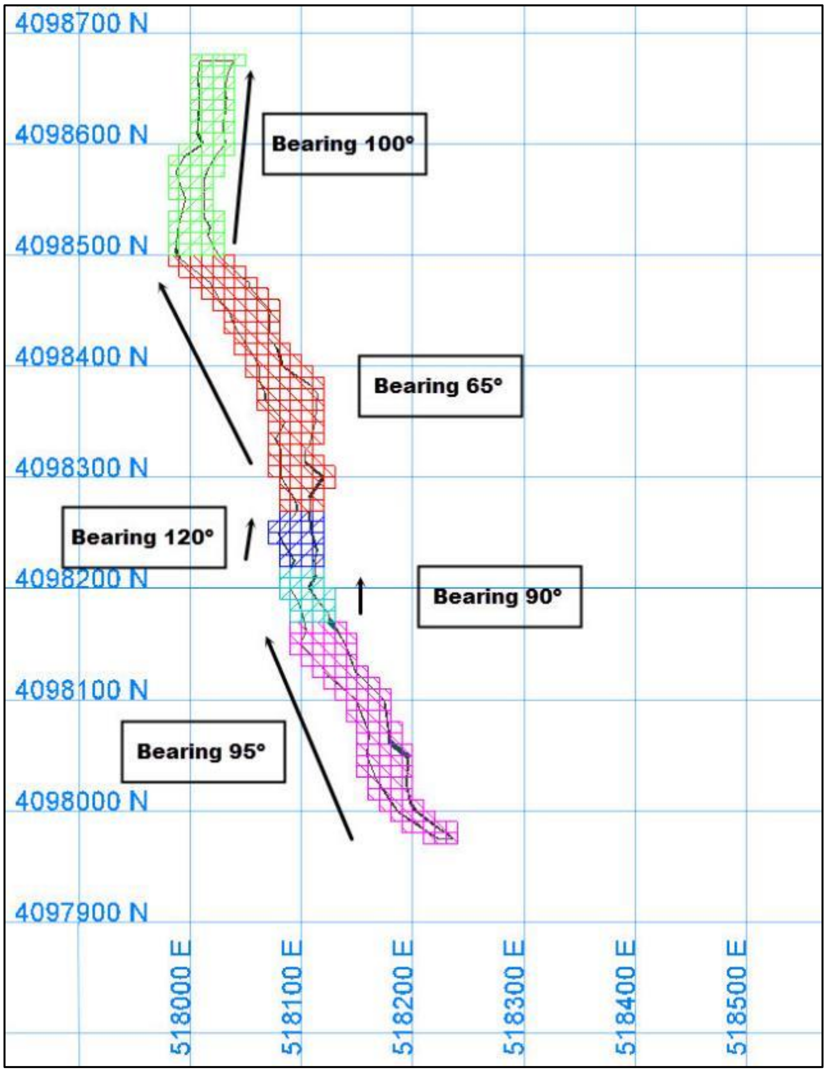
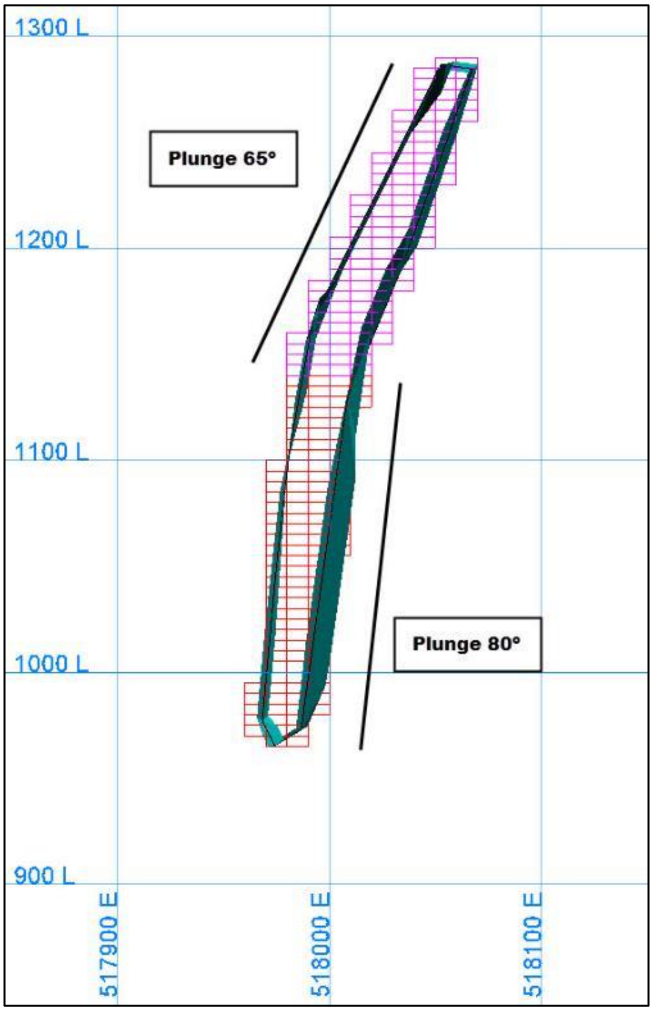


Figure 14-15 - YellowJacket Vein Estimation Plunge Changes (Northing 4098525)



14.2.8.1.1 LIBERATOR AND SAWLE ESTIMATION PARAMETERS

Gold and silver values were estimated using inverse distance squared (ID2). Search ellipsoid orientations and dimensions were determined by evaluating the structural characteristics, and drill density along the vein. Liberator estimation parameters are listed in Table 14-14 and Swale estimation parameters are listed in Table 14-15.

Table 14-14 - Liberator Au and Ag Estimation Parameters – Vulcan Software

Liberator Vein Estimation Parameters			
Estimation Type	Inverse Distance Squared (ID2)		
Search Ellipsoid	**Bearing**	**Plunge**	**Dip**
	254	80	0
Search Ellipse	**Major Axis**	**Semi-Major Axis**	**Minor Axis**
Pass 1	135	60	20

Liberator Vein Estimation Parameters			
Samples	**Min**	**Max**	
	2	20	
Maximum Samples per Drill hole	**Max**		
	2		

Table 14-15 - Swale Au and Ag Estimation Parameters

Swale Estimation Parameters			
Estimation Type	Inverse Distance Squared (ID2)		
Search Ellipsoid	**Bearing**	**Plunge**	**Dip**
	20	-8	0
Search Ellipse	**Major Axis**	**Semi-Major Axis**	**Minor Axis**
Pass 1	100	85	25
Samples	**Min**	**Max**	
	2	20	
Maximum Samples per Drill hole	**Max**		
	2		

14.2.9 DISSEMINATED MINERALIZATION

Gold, silver and sulphur values were estimated for disseminated blocks using Ordinary Kriging. This was done to best capture the nature of grade variances in the deposit. Gold and silver analysis required slightly different variograms. Eight domains were established to ensure that mineralization was estimated properly throughout the bulk of the mineral deposit without impacting the YellowJacket, Liberator and Swale estimates. Composites and blocks were required to be within the same domain in order to be used during interpolation of metal grades. Metal grades were estimated in two passes based on drilling density. Table 14-16 and 14-18 list gold and silver variogram model parameters according to the variograms in Figure 14-16 and 14-17, respectively. Variograms were auto fit and evaluated for accuracy. Table 14-17 and 14-19 list the estimation parameters used for the gold and silver kriging estimates, respectively.

Table 14-16 - Gold Variogram Model Parameters

Variogram Model Parameters							
Nugget	0.153	**Number of Structures**		1	**Distance (m)**		
Variogram Type	**Sill Differential**	**Bearing**	**Plunge**	**Dip**	**Major Axis**	**Semi-Major Axis**	**Minor Axis**
Exponential	0.819	330	15	-25	105	54.5	43.4

Figure 14-16 - Gold Variogram Model-meters

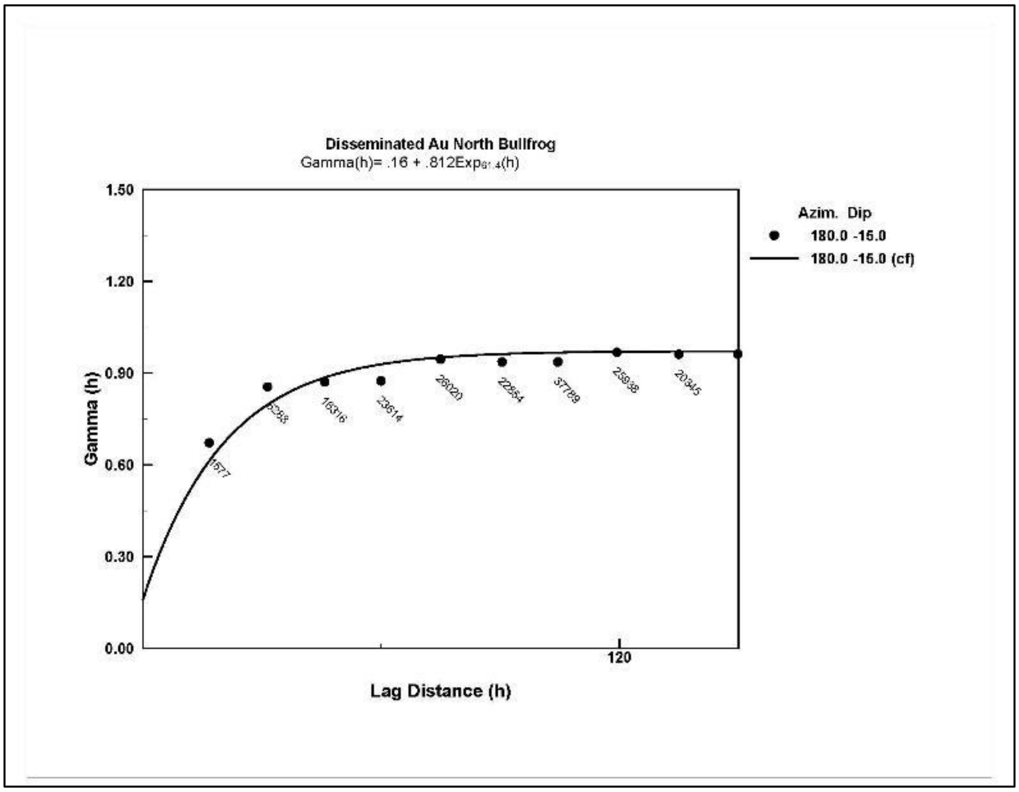


Table 14-17 - Au Estimation Parameters in Vulcan® Format

Disseminated Mineralization			
Estimation Type	**Ordinary Kriging**		
Search Ellipsoid	**Bearing**	**Plunge**	**Dip**
	330	15	-25
Search Distance	**Major Axis**	**Semi-Major Axis**	**Minor Axis**
Pass 1	105	54.5	43.4
Pass 2	225	150	50
Samples	**Min**	**Max**	
	4	20	
Maximum Samples per Drill hole	**Max**		
Pass 1 and 2	2		

Table 14-18 - Silver Variogram Model Parameters

Variogram Model Parameters							
Nugget	0.331	**Number of Structures**		1	**Distance (m)**		
Variogram Type	**Sill Differential**	**Bearing**	**Plunge**	**Dip**	**Major Axis**	**Semi-Major Axis**	**Minor Axis**
Exponential	0.581	160	0	7	140	111.8	91.3

Figure 14-17 - Silver Variogram Model-meters

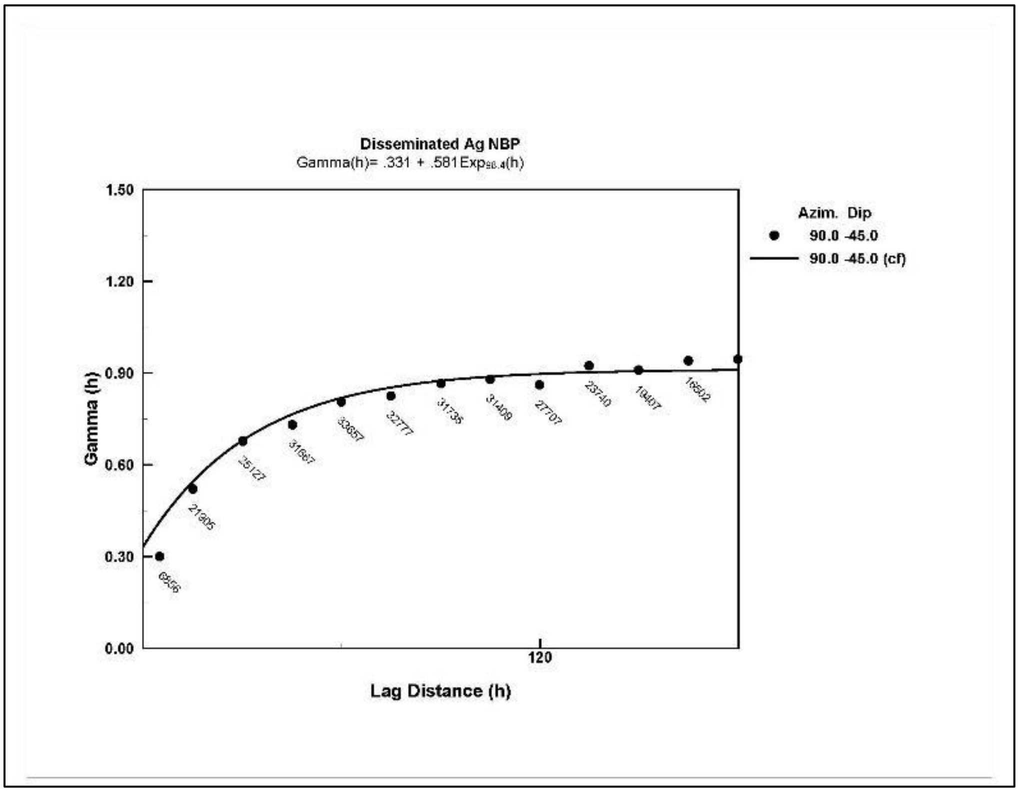


Table 14-19 - Ag Estimation Parameters in Vulcan® Format

Disseminated Mineralization			
Estimation Type	**Ordinary Kriging**		
Search Ellipsoid	**Bearing**	**Plunge**	**Dip**
	160	0	7
Search Distance	**Major Axis**	**Semi-Major Axis**	**Minor Axis**
	140	112	92
Samples	**Min**	**Max**	
	4	20	
Maximum Samples per Drill hole	**Max**		
	2		

Sulphur was kriged into all rock types and domains. Sulphur content will be useful for potential processing evaluations and metallurgical testing. Sulphur content was important for determination of oxidation potential of various mineralization types. Variograms and kriging estimation parameters for Sulphur are listed in Tables 14-20 and 14-21. The Sulphur variogram model is shown graphically in Figure 14-18.

Table 14-20 - Sulphur Variogram Model Parameters

Variogram Model Parameters							
Nugget	0.132	**Number of Structures**		2	**Distance (m)**		
Variogram Type	**Sill Differential**	**Bearing**	**Plunge**	**Dip**	**Major Axis**	**Semi-Major Axis**	**Minor Axis**
Exponential	0.299	349	-64	23	104	56	27.5
Exponential	0.368	122	-3	5	408	267	123

Figure 14-18 - Sulphur Variogram Model-meters

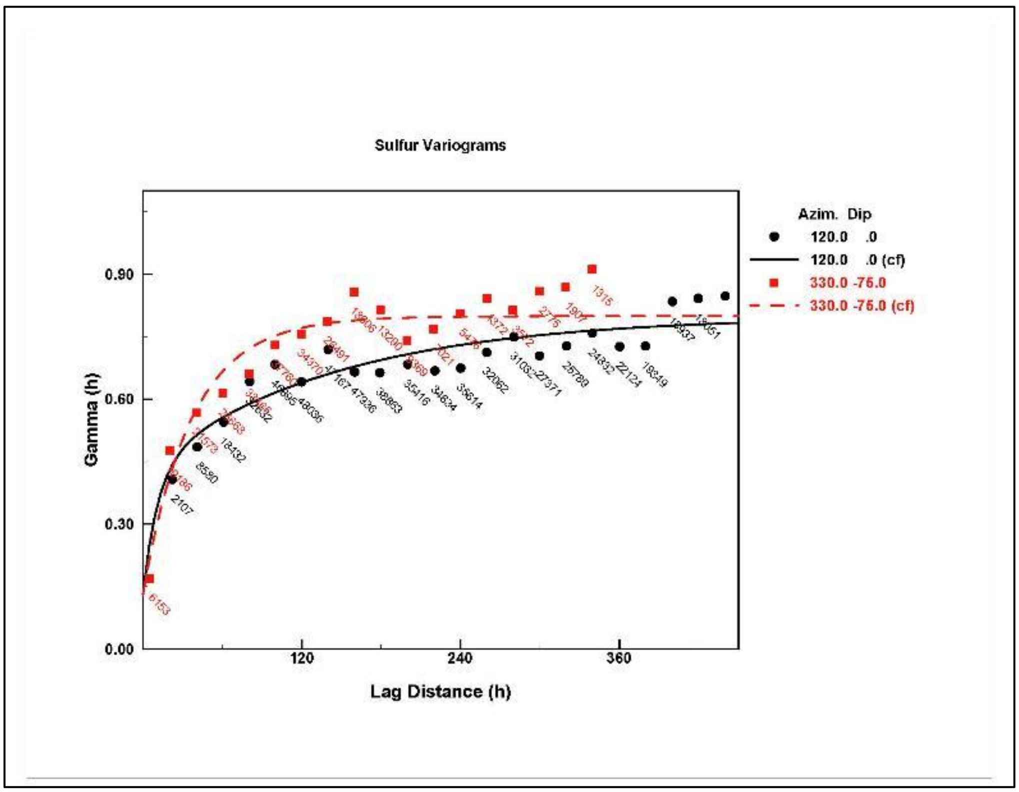


Table 14-21 - Sulphur Estimation Parameters in Vulcan® Format

Disseminated Mineralization			
Estimation Type	**Ordinary Kriging**		
Search Ellipsoid	**Bearing**	**Plunge**	**Dip**
	122	-3	5
Search Distance	**Major Axis**	**Semi-Major Axis**	**Minor Axis**
	408	267	123
Samples	**Min**	**Max**	
	4	20	
Maximum Samples per Drill hole	**Max**		
	2		

14.2.10 SWATH PLOTS

A swath plot is an analysis which compares estimated block grades to composite grades for a slice taken from the block model. This is a useful tool to help determine whether grade estimation parameters correlate well with expected values based on composite grades. Figure 14-19 is an overview map that shows the spatial relationship of the section line (Northing 4098300) with the YellowJacket, Liberator and Swale grade shells. Figure 14-20 shows the results of the swath plot analysis. Grade shells used for estimation domains are identified. The swath plot analysis indicates that the grade estimates for the Swale, YellowJacket and Liberator grade shells correlate well with sample composite grades.

Figure 14-19 - North Bullfrog Project Swath Plot Section Line-meters

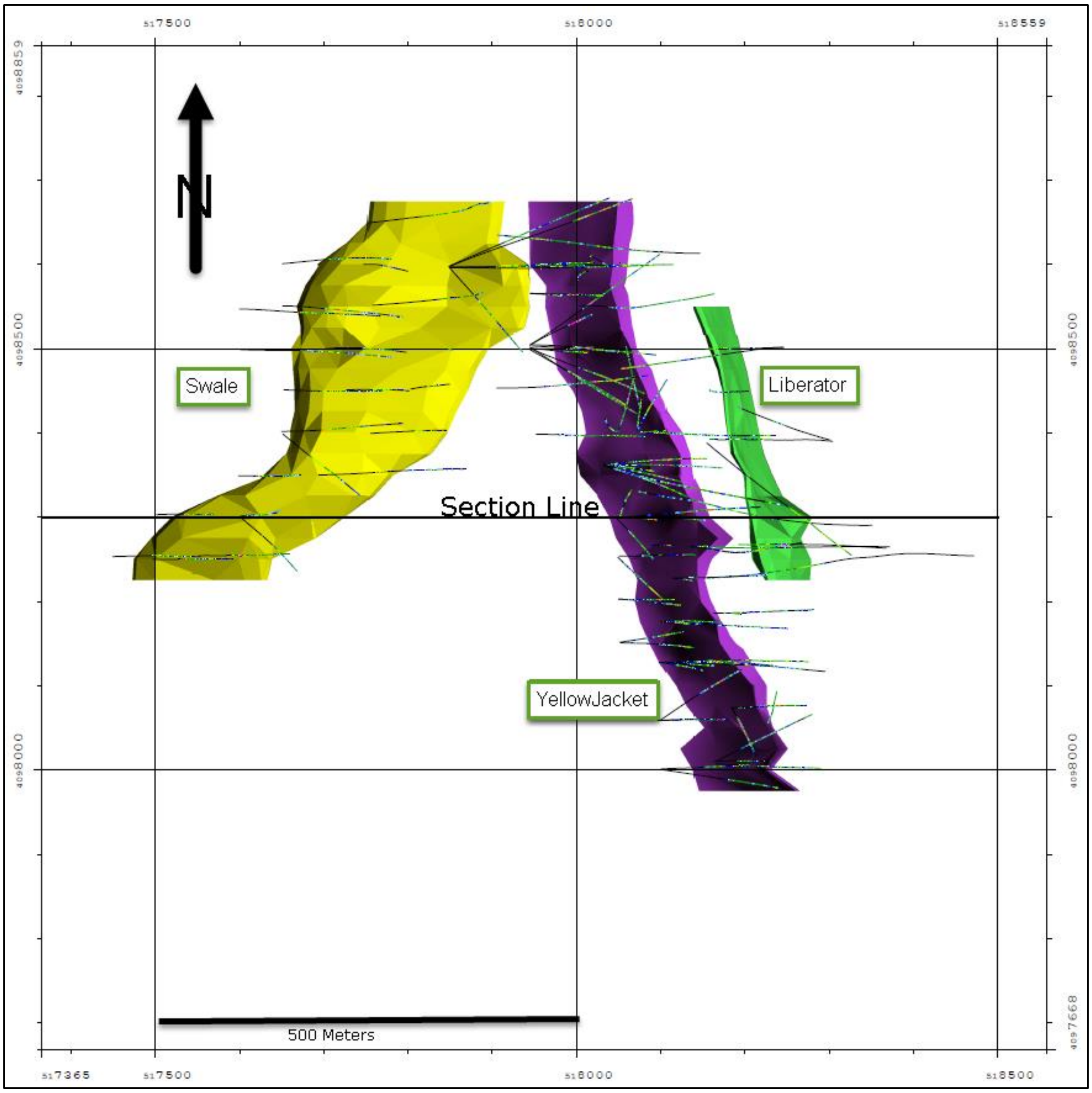
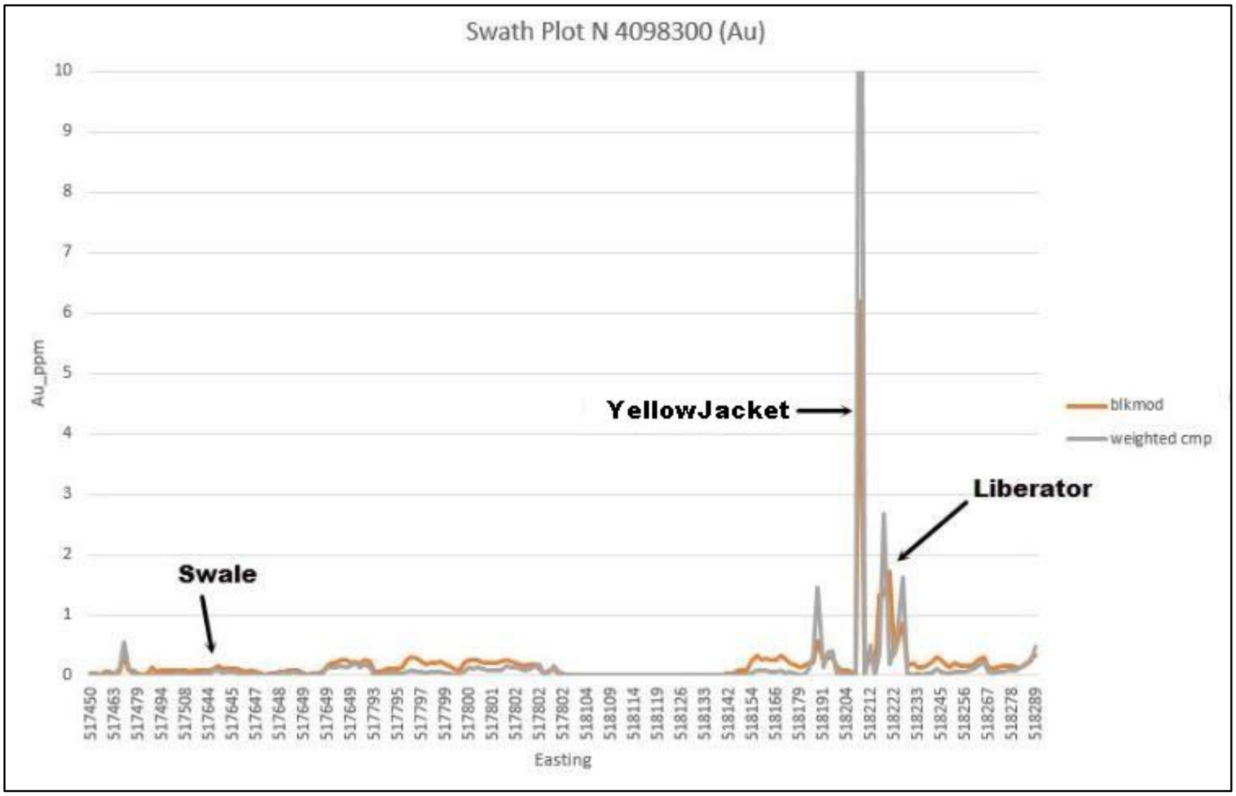


Figure 14-20 - Swath Plot Graphical Analysis-meters



14.3 MAYFLOWER

14.3.1 MAYFLOWER GEOLOGIC MODEL

Mayflower drilling data is stored as a subset of the North Bullfrog drilling database. Of the 811 total drillholes at NBP, 91 holes were selected to define the mineralized grades and volume for the Mayflower deposit. Assayed intervals, comprising 8,993 Au and Ag assays, were used for statistical analyses, grade estimation and mineral resource estimates. Figure 14-21 is a plan view showing drill hole collar locations and drill hole traces at the Mayflower deposit.

Two stratigraphic units, the Rainbow Mountain Debris Flow Sequence (Tdf) and the Donovan Mountain Latite (Trl), host the known volume of mineralization at Mayflower. Mineralization appears to be controlled by a complex fracture network associated with the David, 1_fault, and South 200 faults which bisect the deposit at N35W dipping 70° southwest. There is a northwest trending zone of higher-grade, greater than 0.50g/t Au, mineralization associated with proximity to the aforementioned faults; generally, in the hanging wall of the 1 Fault. Lower grade mineralization occurs distally in both directions from the faults.

An indicator variogram model was evaluated to identify the high-grade corridor within the deposit. This is necessary for two reasons, 1) to limit the influence of high grades on the surrounding lower grade country rock and 2) to capture the higher-grade mineralization as a separate domain within the deposit.

Indicators of 1 were set for assay intervals with grades >= 0.50 g/t Au. The indicators were then kriged throughout the block model to determine the potential probability of a model block to host mineralization of at least 0.5 g/t Au from 0% to 100%. Blocks with a 40% or higher probability to host high grade mineralization were flagged. The resulting domain closely matches the distribution of mineralization within Mayflower.

Figure 14-21 - Mayflower Resource Model Drill Holes.

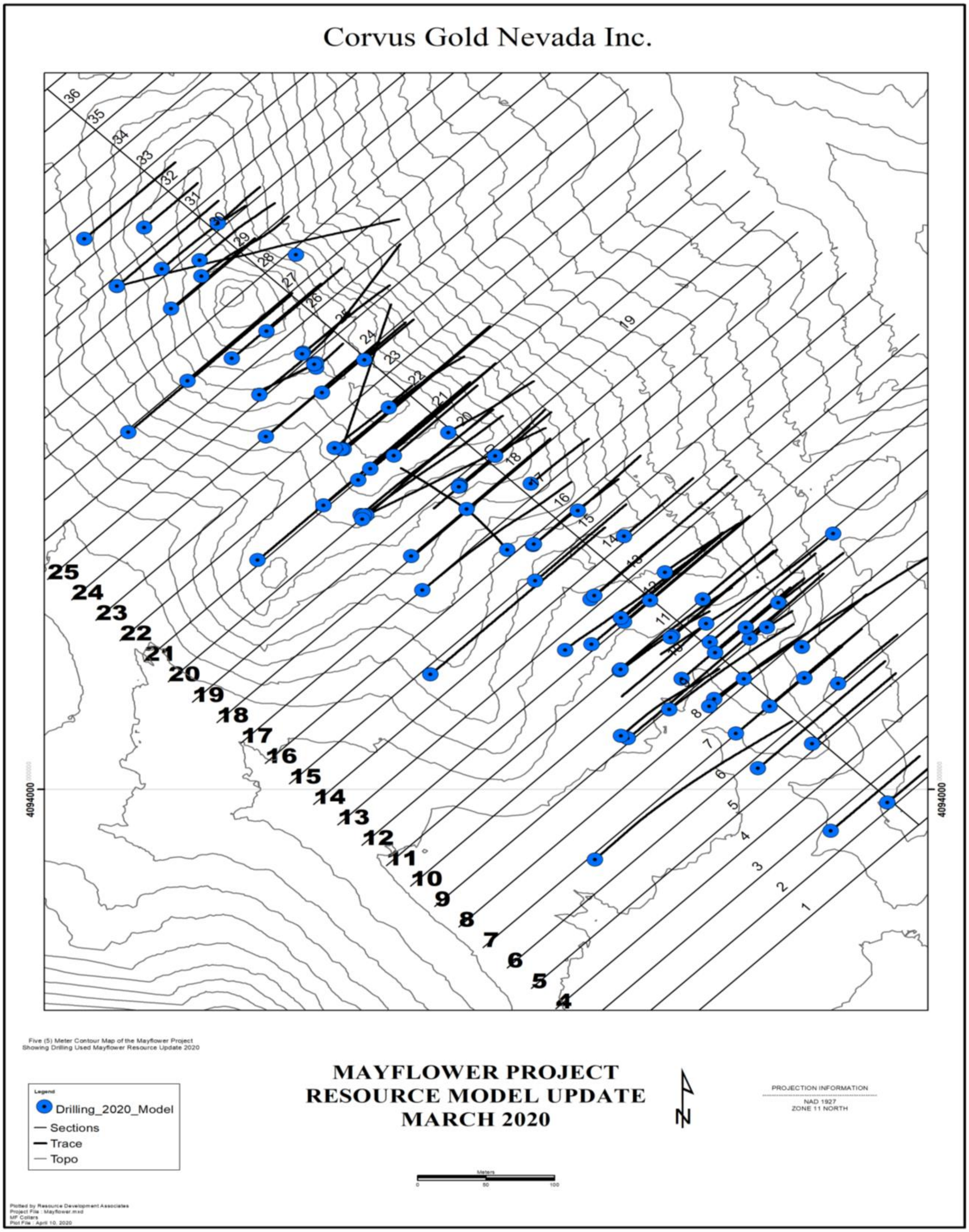
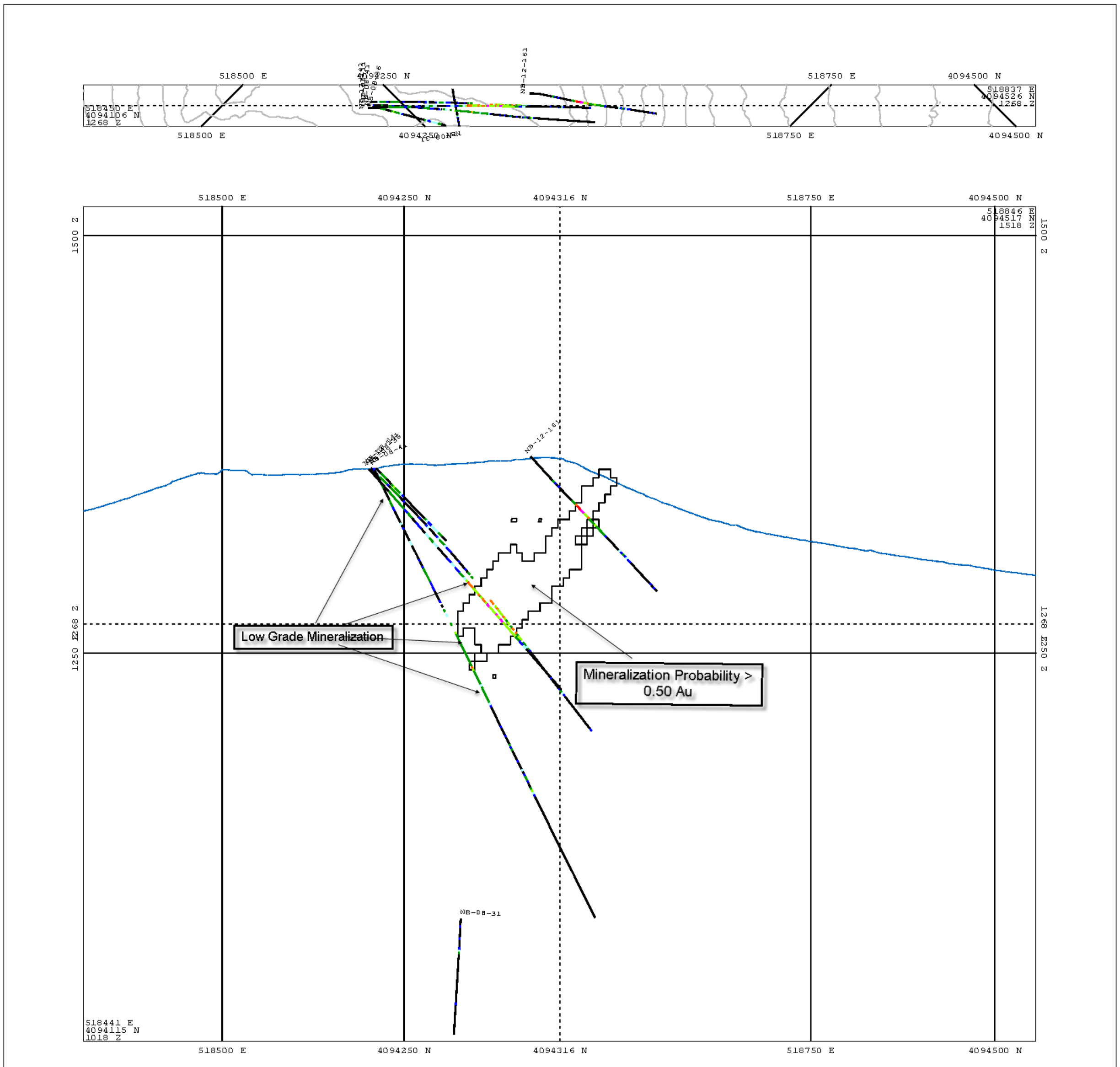


Figure 14-22 - Typical Section Through Mayflower showing drill indicated mineralization and the probability indicator model.



Cross Section 20 Showing Drilled Mineralization and the Indicator Model. Lower Grade Mineralization Distal to the High Grade Indicator Model

14.3.2 EXPLORATORY DATA ANALYSIS

Assays were flagged both inside and outside of the indicator model for comparative statistics. Exploratory Data Analysis is used to identify the characteristics of mineralization, the relationship to geology and assist with the identification of outliers or assay values which may potentially bias the grade estimate. Gold and Silver assay statistics for the drill hole data are listed in Table 14-22. Three views of the mineralization population have been evaluated; 1) all assayed material, 2) mineralization at or above 0.06 Au g/t which is representative of mineralization which may meet the reasonable prospects of eventual economic extraction and 3) mineralization greater than or equal to 0.13 Au g/t which represents an operational cutoff grade for the North Bullfrog Project.

Table 14-22 - Summary Assay Statistics at Mayflower

	Samples Within 0.50 Gram Indicator Au Shell 0.00 Cutoff		Samples External to 0.50 Gram Indicator Au Shell 0.00 Cutoff	
91 Drillholes 8,993 Assays	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)
Number of Samples	390	390	8,603	8,603
Mean Grade	1.51	0.77	0.09	0.19
Standard Deviation	3.43	1.13	0.49	0.91
Minimum Value	0.012	0.050	0.001	0.001
Maximum Value	41.50	15.60	38.3	75.9
Coefficient of Variation	3.43	1.46	5.47	4.78
	Samples Within 0.50 Gram Indicator Au Shell 0.06 Cutoff		Samples External to 0.50 Gram Indicator Au Shell 0.06 Cutoff	
91 Drillholes 2,713 Assays	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)
Number of Samples	380	380	2,333	2,333
Mean Grade	1.55	0.79	0.28	0.37
Standard Deviation	3.47	1.14	0.92	1.68
Minimum Value	0.060	0.050	0.060	0.010
Maximum Value	41.50	15.60	38.30	75.90
Coefficient of Variation	2.23	1.44	3.24	4.55
	Samples Within 0.50 Gram Indicator Au Shell 0.13 Cutoff		Samples External to 0.50 Gram Indicator Au Shell 0.13 Cutoff	
91 Drillholes 2,713 Assays	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)
Number of Samples	373	373	1,370	1,370
Mean Grade	1.58	0.80	0.42	0.47
Standard Deviation	3.49	1.14	1.18	2.17
Minimum Value	0.13	0.050	0.13	0.04
Maximum Value	41.50	15.60	38.30	75.90
Coefficient of Variation	2.22	1.43	2.80	4.61

Global gold statistics were used to determine a global cap for mineralization in the assay database. Capping is used to control the impact of higher-grade outliers on the global estimate of mineralization at Mayflower. A review of the assays at the cutoff grade of 0.13 g/t gold, the graph in Figure 14-23, suggests a cap 4.66. A rounded cap of 5 g/t gold is appropriate for Mayflower as this is an estimate. Assays were capped at these values prior to compositing. Figure 14-24 shows the capped histogram. Table 14-23 lists the capped and uncapped statistics. Twenty-three samples were set to 5 g/t Au prior to composting. Silver grades were not capped.

Figure 14-23 - Au Assay Histogram

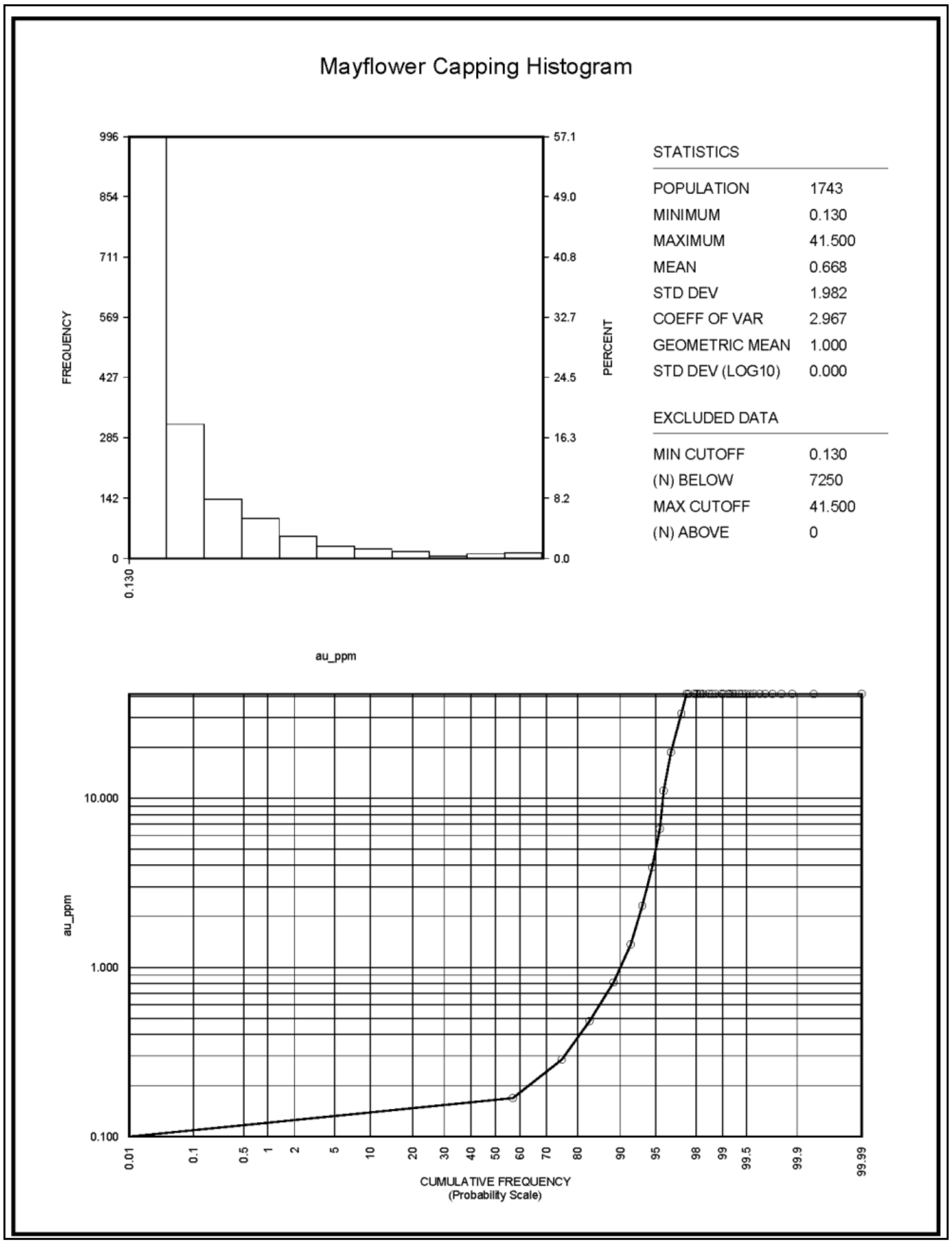


Figure 14-24 - Assay Statistics Capped at 5g

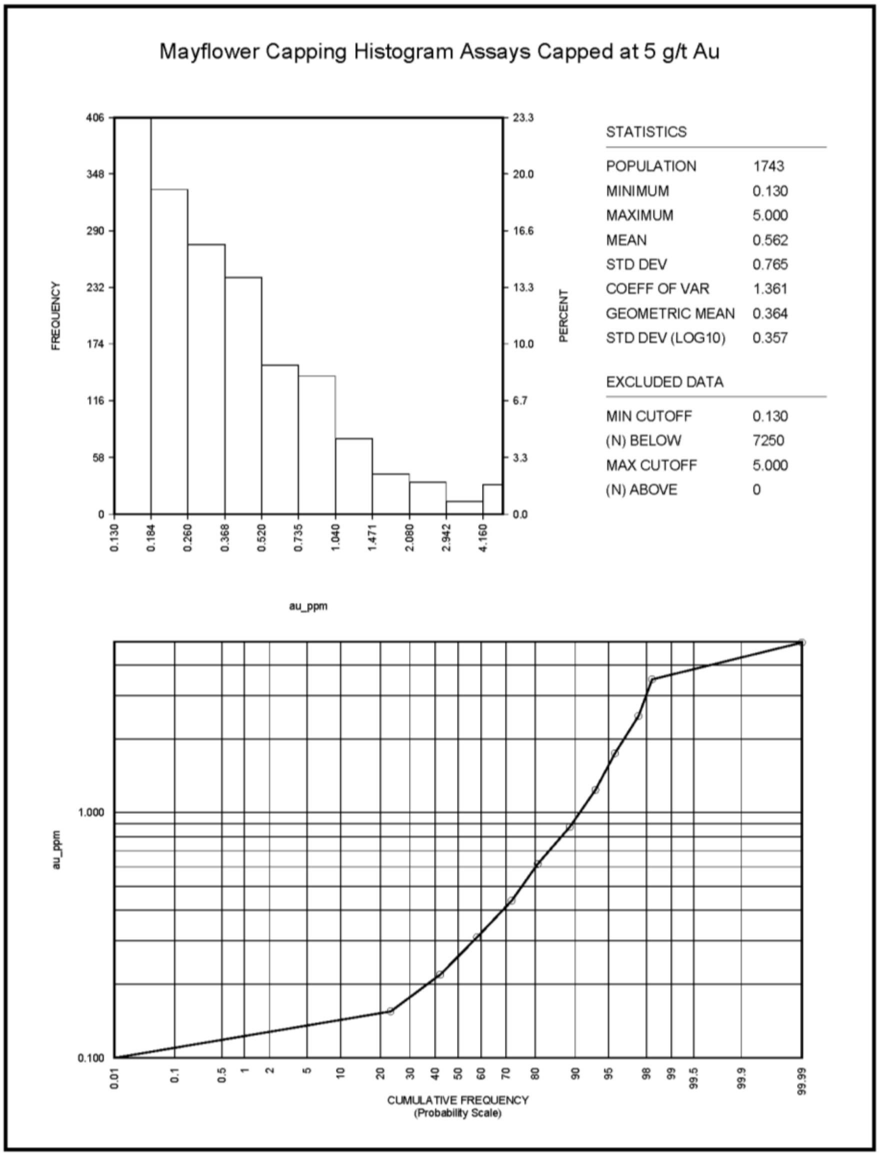


Table 14-23 - Summary of Capping of Significant Mineralization at Mayflower

	Capping	
	Uncapped	Capped
Number of Samples	1743	1743
Mean Grade	0.67	0.56
Standard Deviation	1.98	0.77
Minimum Value	0.13	0.13
Maximum Value	41.5	5.00
Coefficient of Variation	2.97	1.36

14.3.3 COMPOSITES

Assay intercepts were composited to 5-meter lengths across the entire database of 8,993 assay intervals. Composite statistics are listed in Table 14-24.

Table 14-24 - Composite Statistics

	Au (g/t)	Ag (g/t)
Number of Samples	3,029	3,029
Mean Grade	0.13	3,029
Standard Deviation	0.316	0.20
Minimum Value	0.001	0.23
Maximum Value	3.76	0.005
Coefficient of Variation	2.46	3.14
-	-	-

14.3.4 VARIOGRAPHY

A single semi-variogram was produced from the Au composites. The variograms model the visual strike and dip of the mineralization at Mayflower. A spherical model was fit to the anisotropy of the mineralization. Figure 14-25 displays the graph; variogram parameters are listed in Table 14-25.

Figure 14-25 - Mayflower Variogram

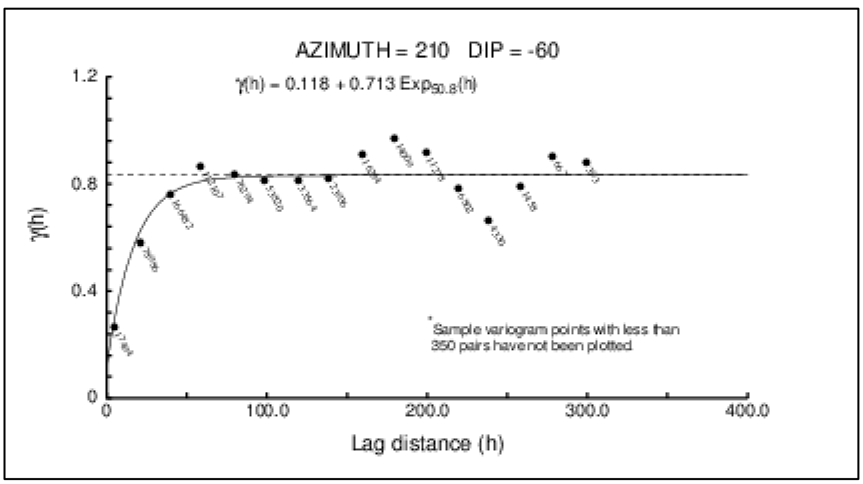


Table 14-25 - Mayflower Au Variogram

Axis	Bearing, Dip	Range	Nugget	Sill	C1
Major	Azimuth 330, 0	60	0.118	0.831	0.713
Semi-Major	Plunge 118, 0	46			
Minor	Dip 214, -60	31			

14.3.5 BULK DENSITY

During the 2012 drill program, a total of 271 specific gravity measurements were made from drill core using the weight in air/weight in water method. These determinations came from holes NB-12-132, -133, -140, -141, -142 and -143. The results can be sorted by lithology and by gold grade. While there is a range of specific gravities for the various lithologies sampled, lithology has not been modeled so it is not of any use in assigning density to estimated blocks. The average specific gravity of .27, based on the 2012 drilling, was applied to the entire mineral deposit.

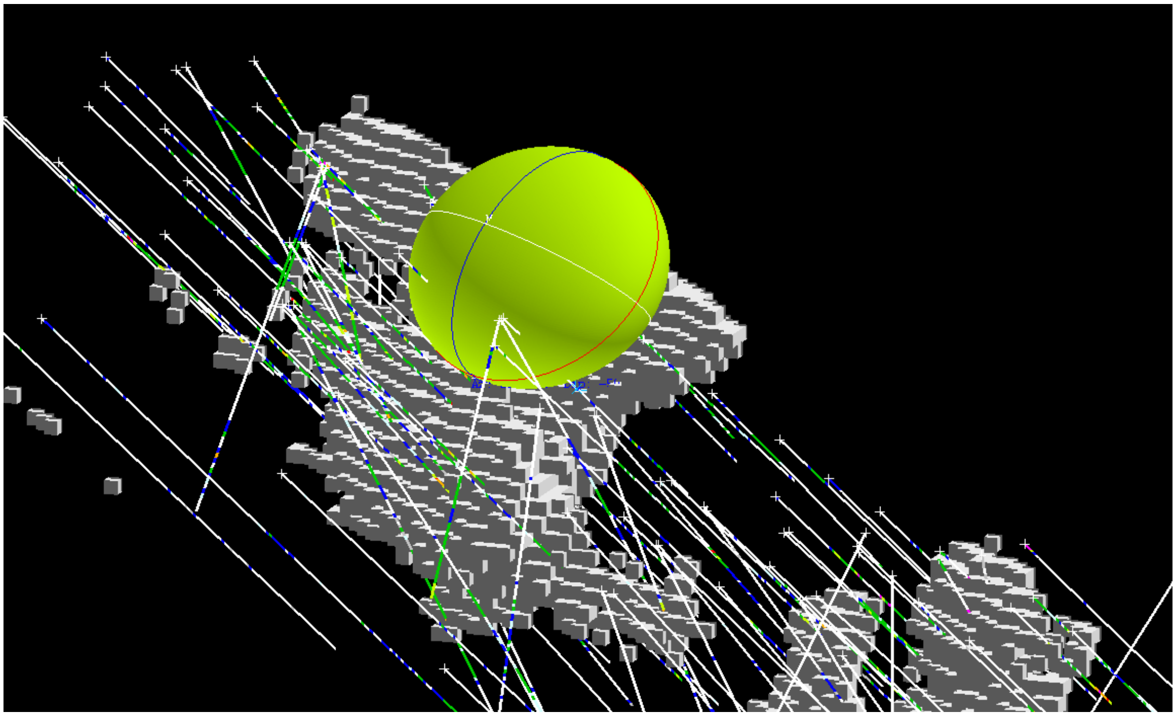
14.3.6 GRADE INTERPOLATION

Grades for gold and silver were determined using four separate estimation techniques; kriging, inverse distance squared, inverse distance cubed and nearest neighbor. The results for each type of estimate were compared to the composite statistics. The kriged estimate was chosen as the most representative for Mayflower because this method most closely matches the gold histogram for the deposit. In all cases a two-pass estimate was used to 1) properly assign higher grades to the model and 2) to limit the impact of higher grades on the global estimate of contained metal. Pass 1 used blocks and samples contained within the indicator model. The indicator model captures blocks that may host mineralization greater than 0.50 g/t Au. Pass 2 estimated gold and silver into blocks external to the indicator model. All composites were used for the pass 2 estimation, though there were more samples used for each estimate ensuring that lower grade material had a higher weight on the stored gold and silver grades. Pass 1 used a minimum of 3 and maximum of 10 samples for estimation while pass 2 used a minimum of 4 and maximum of 20 samples for an estimate. Estimation parameters are listed below in Table 14-26. An Oblique View looking North in Figure 14-26 shows the search ellipsoid and anisotropy of the mineralization.

Table 14-26 - Summary of Kriging Search Parameters - Mayflower

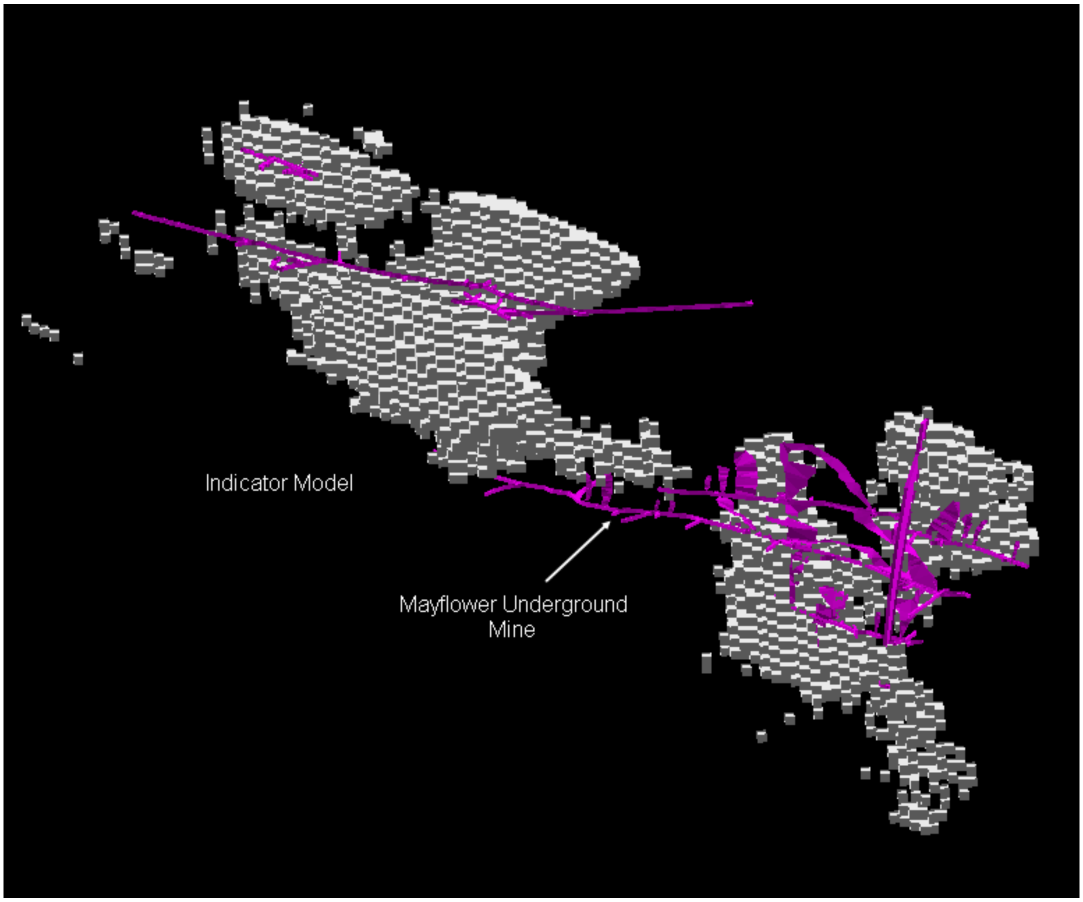
Variable	Pass	Bearing/Plunge/Dip	Distance Major/Semi/Minor	Samples Min/Max	Nugget	Sill	C1
Au	1	305/0/50	60/45/30	3/10	0.118	0.831	0.713
	2	305/0/50	60/45/30	4/20	0.118	0.831	0.713
Ag	1	305/0/50	60/45/30	3/10	0.118	0.831	0.713
	2	305/0/50	60/45/30	4/20	0.118	0.831	0.713

Figure 14-26 - Oblique View of Search Ellipsoid



Historic mining occurred at Mayflower. The volumes of mineralization extracted by underground mining, derived from historic survey data, have been removed from the model. Figure 14-27 shows the underground mining in relation to the high-grade indictor model at Mayflower.

Figure 14-27 - Mayflower Underground Mine

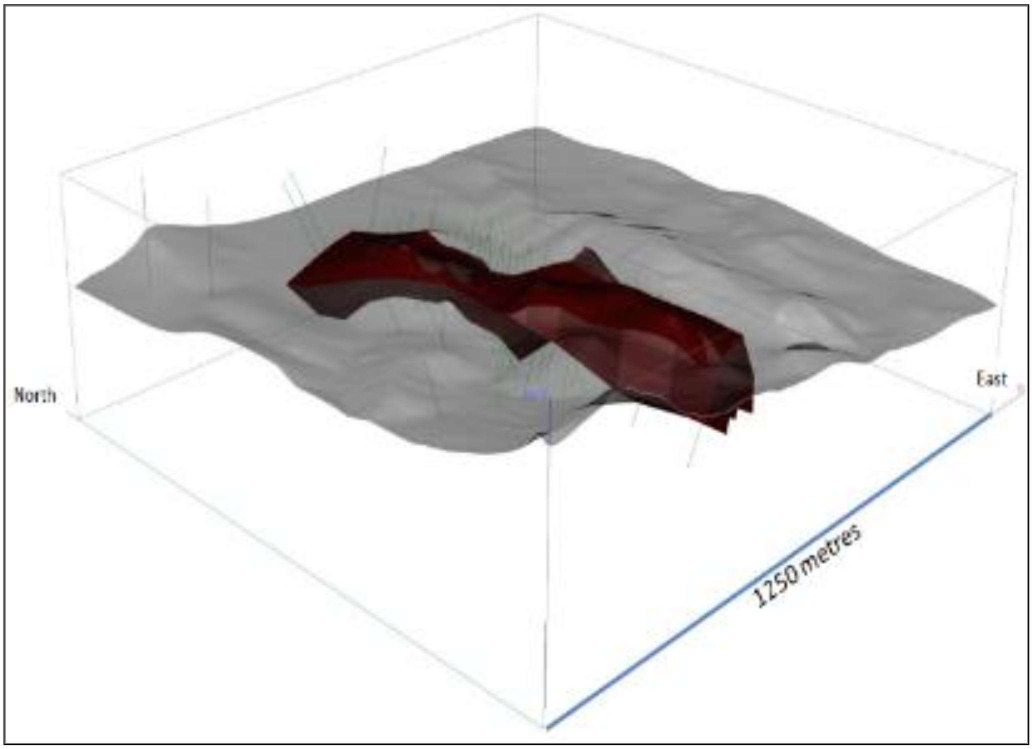


14.4 JOLLY JANE

The geologic model for Jolly Jane was built by Corvus geologists. Ninety-four (94) reverse circulation drill holes and 5 surface outcrop channel sample lines define the Jolly Jane zone. Within the 9,450 gold assays supplied for Jolly Jane a total of 61 gaps in the assay record were identified. These gaps were filled with values of 0.001 g/t Au which is an acceptable practice and has no impact on the results.

The 3D geology for Jolly Jane was modeled as two surfaces, one describing the lower contact of the mineralized Crater Flat Tuff and the other describing the upper contact. These 3D surfaces were constructed by Corvus geologists and are shown in Figure 14-28. The lower contact is sometimes the original depositional contact on Tertiary sediments or the basement Paleozoic sediments. However, in other places the lower contact is with post-mineral dacite intrusions. The lower contact has been offset by a series of west dipping faults. The upper contact is generally defined by post-mineral dacite intrusions or locally the next stratigraphic unit. Because the dacites are post-mineral they are not offset by the same faults as the lower contact. There are some minor internal dacite intervals. These dacites are a different composition to the post mineral intrusions and they are generally mineralized so they have been included in the volume between the upper and lower contacts. The upper and lower contacts have been extended north and south to the limits that should be modeled. Consequently, the volume to model should be defined by the upper and lower contacts together with the topography and then the ends should just be clipped with vertical planes which coincide with the edge of the triangulated surfaces.

Figure 14-28 - Isometric View of Jolly Jane Looking Northeasterly Showing Mineralization Solid in Red, Drill holes in Green and Surface Topography in Grey



14.4.1 DATA ANALYSIS JOLLY JANE

Drill holes were compared to the geologic solid and the assays were back-tagged with a mineralized code if inside the solid. The sample statistics are listed in Table 14-27.

Table 14-27 - Summary of Assay Statistics for Jolly Jane Mineralized Solid

	Inside Solid		Outside Solid	
	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)
Number of Samples	4,585	4,417	4,549	4,390
Mean Grade	0.143	0.361	0.033	0.288
Standard Deviation	0.164	0.378	0.075	0.639
Minimum Value	0.001	0.005	0.001	0.005
Maximum Value	1.45	4.46	0.93	17.75
Coefficient of Variation	1.14	1.04	2.29	2.22

The grade distribution for gold was evaluated using a lognormal cumulative frequency plot for samples within the mineralization solid. Five overlapping lognormal populations made up the gold distribution for the mineralized zone. The highest-grade population, with a mean value of 0.85 g/t Au, represented 0.73% of the data or 33 samples and was not considered erratic high grade. A cap level was chosen at two standard deviations above the mean of this highest-grade population. A cap value of 1.78 g/t Au was used, and no assays required capping.

A similar exercise was completed for silver within the mineralization solid. No silver assays within the mineralization zone required capping.

For assays outside the mineralization solid a total of 55 assays were capped at 0.35 g/t Au and 13 assays were capped at 4.4 g/t Ag.

14.4.2 COMPOSITES JOLLY JANE

Drill holes at Jolly Jane were compared to the mineralization solid and the points at which each hole entered and left the solid were recorded. Uniform down hole composites, 5 meters in length, were formed and made to honor the solid boundaries. Intervals less than ½ the composite length at the solid boundaries were joined with adjoining samples to produce a composites file of uniform support, 5± 2.5 meters in length. The statistics for 5 meters composites are summarized below (Table 14-28). A similar exercise was completed for samples outside the solid.

Table 14-28 - Summary of 5m Composite Statistics for Mineralization Solid Jolly Jane

	Mineralized Solid		Outside Solid	
	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)
Number of Samples	1,441	1,408	1,414	1,396
Mean Grade	0.144	0.37	0.031	0.31
Standard Deviation	0.152	0.41	0.057	0.55
Minimum Value	0.001	0.005	0.001	0.005
Maximum Value	1.28	4.46	0.35	4.40
Coefficient of Variation	1.06	1.10	1.85	1.78

14.4.3 VARIOGRAPHY JOLLY JANE

Pairwise relative semi variograms were used to model the gold continuity at Jolly Jane. The direction of longest continuity for gold in the horizontal plane was along azimuth 0° dipping -30°. In the plane perpendicular to this the longest continuity was along azimuth 90° dipping -40°. Nested spherical models were fit to all directions. The nugget-to-sill ratio of 16% for Au and 11% for Ag were very good. For Au and Ag in waste, isotropic spherical models were produced. The parameters are tabulated below.

The parameters are tabulated in Table 14-29.

Table 14-29 - Summary of Jolly Jane Gold and Silver Semi-variogram Parameters

Domain	Azimuth	Dip	C_o	C_1	C_2	Short Range (m)	Long Range (m)
Mineralized Solid Au	0º	-30º	0.10	0.10	0.43	20	120
	90º	-20º	0.10	0.10	0.43	30	110
	270º	-70º	0.10	0.10	0.43	20	50
Mineralized Solid Ag	0º	-30º	0.05	0.10	0.30	30	120
	90º	-40º	0.05	0.10	0.30	40	80
	270º	-50º	0.05	0.10	0.30	15	60
Waste Au	Omni Directional		0.15	0.15	0.40	25	90
Waste Ag	Omni Directional		0.10	0.10	0.35	30	100

14.4.4 BULK DENSITY JOLLY JANE

During the 2010 drill campaign on the NBP, a total of 102 samples of RC chips were sent to ALS Minerals for specific gravity measurements by pycnometer (method OA-GRA08b). The average specific gravity from 46 samples within the oxidized Tuff units from mineralization zones drilled in 2010 was 2.60.

During the 2012 drill campaign 74 specific gravity determinations were made from drill core which is far more representative than RC Chips, as porosity is included. Of these samples 59 were within the Crater Flat Tuff unit which hosts the mineralization at Jolly Jane. The average specific gravity from these samples, listed in Table 14-30, was 2.34.

Table 14-30 – Specific Gravity Determinations for Tuff Units – Jolly Jane

HoleID	SampleID	From_m	To_m	SG	StratUnit1
NB-12-130	M610829	17.68	20.73	2.48	fault zone
NB-12-130	M610836	35.97	39.01	2.43	fault zone
NB-12-130	M610838	42.06	45.11	2.44	fault zone
NB-12-130	M610851	79.56	82.76	1.93	fault zone
NB-12-130	M610855	89.31	92.50	1.96	fault zone
NB-12-131	M612269	11.44	13.50	2.38	fault zone
NB-12-131	M612279	36.10	39.01	2.21	fault zone
NB-12-131	M612283	44.40	46.33	2.23	fault zone
Average				2.26	Fault zones
NB-12-131	M612265	0.00	3.05	2.36	Crater Flat Tuff
NB-12-131	M612266	3.05	6.27	2.37	Crater Flat Tuff
NB-12-131	M612267	6.27	9.40	2.41	Crater Flat Tuff
NB-12-130	M610859	101.80	105.55	2.44	lower Crater Flat Tuff

HoleID	SampleID	From_m	To_m	SG	StratUnit1
NB-12-130	M610860	105.55	109.42	2.43	lower Crater Flat Tuff
NB-12-130	M610824	2.20	5.38	2.34	middle Crater Flat Tuff
NB-12-130	M610825	5.38	8.45	2.43	middle Crater Flat Tuff
NB-12-130	M610826	8.45	11.58	2.40	middle Crater Flat Tuff
NB-12-130	M610827	11.58	14.63	2.39	middle Crater Flat Tuff
NB-12-130	M610828	14.63	17.68	2.41	middle Crater Flat Tuff
NB-12-130	M610830	20.73	23.77	2.38	middle Crater Flat Tuff
NB-12-130	M610831	23.77	26.82	2.47	middle Crater Flat Tuff
NB-12-130	M610832	26.82	29.87	2.41	middle Crater Flat Tuff
NB-12-130	M610834	29.87	32.92	2.40	middle Crater Flat Tuff
NB-12-130	M610835	32.92	35.97	2.47	middle Crater Flat Tuff
NB-12-130	M610837	39.01	42.06	2.39	middle Crater Flat Tuff
NB-12-130	M610839	45.11	48.11	2.43	middle Crater Flat Tuff
NB-12-130	M610840	48.11	51.21	2.50	middle Crater Flat Tuff
NB-12-130	M610841	51.21	54.25	2.52	middle Crater Flat Tuff
NB-12-130	M610842	54.25	57.30	2.47	middle Crater Flat Tuff
NB-12-130	M610844	57.30	60.35	2.45	middle Crater Flat Tuff
NB-12-130	M610845	60.35	63.74	2.43	middle Crater Flat Tuff
NB-12-130	M610846	63.74	67.18	2.33	middle Crater Flat Tuff
NB-12-130	M610847	67.18	70.30	2.32	middle Crater Flat Tuff
NB-12-130	M610848	70.30	73.34	2.34	middle Crater Flat Tuff
NB-12-130	M610849	73.34	76.48	2.01	middle Crater Flat Tuff
NB-12-130	M610850	76.48	79.56	2.06	middle Crater Flat Tuff
NB-12-130	M610852	82.76	86.26	1.98	middle Crater Flat Tuff
NB-12-130	M610854	86.26	89.31	1.95	middle Crater Flat Tuff
NB-12-130	M610856	92.50	95.52	2.17	middle Crater Flat Tuff
NB-12-130	M610857	95.52	98.63	2.30	middle Crater Flat Tuff
NB-12-130	M610858	98.63	101.80	2.27	middle Crater Flat Tuff
NB-12-131	M612276	27.53	30.29	2.34	middle Crater Flat Tuff
NB-12-131	M612277	30.29	33.22	2.39	middle Crater Flat Tuff
NB-12-131	M612278	33.22	36.10	2.32	middle Crater Flat Tuff
NB-12-131	M612280	39.01	42.06	2.35	middle Crater Flat Tuff
NB-12-131	M612282	42.06	44.40	2.31	middle Crater Flat Tuff
NB-12-131	M612284	46.33	49.68	2.36	middle Crater Flat Tuff
NB-12-131	M610862	49.68	52.73	2.28	middle Crater Flat Tuff
NB-12-131	M610863	52.73	56.66	2.21	middle Crater Flat Tuff
NB-12-131	M610864	56.66	60.35	2.34	middle Crater Flat Tuff
NB-12-131	M610865	60.35	64.10	2.38	middle Crater Flat Tuff
NB-12-131	M610866	64.10	67.97	2.40	middle Crater Flat Tuff
NB-12-131	M610867	67.97	71.60	2.49	middle Crater Flat Tuff
NB-12-131	M610868	71.60	75.24	2.45	middle Crater Flat Tuff

HoleID	SampleID	From_m	To_m	SG	StratUnit1
NB-12-131	M610869	75.24	78.64	2.37	middle Crater Flat Tuff
NB-12-131	M610870	78.64	81.69	2.42	middle Crater Flat Tuff
NB-12-131	M610872	81.69	84.73	2.22	middle Crater Flat Tuff
NB-12-131	M610873	84.73	87.78	2.30	middle Crater Flat Tuff
NB-12-131	M610874	87.78	90.83	2.29	middle Crater Flat Tuff
NB-12-131	M610875	90.83	93.88	2.31	middle Crater Flat Tuff
NB-12-131	M610876	93.88	97.88	2.22	middle Crater Flat Tuff
NB-12-131	M610877	97.88	101.72	2.50	middle Crater Flat Tuff
NB-12-131	M610878	101.72	105.58	2.47	middle Crater Flat Tuff
NB-12-131	M610879	105.58	109.32	2.43	middle Crater Flat Tuff
NB-12-131	M610880	109.32	112.34	2.22	middle Crater Flat Tuff
NB-12-131	M610882	112.34	116.30	2.13	middle Crater Flat Tuff
NB-12-131	M610883	116.30	119.40	2.34	middle Crater Flat Tuff
NB-12-131	M610884	119.40	120.94	2.10	middle Crater Flat Tuff
Average				**2.34**	**Crater Flat Tuff**
NB-12-131	M612268	9.40	11.44	2.16	dacite breccia
NB-12-131	M612270	13.50	16.68	2.33	dacite breccia
NB-12-131	M612272	16.68	19.80	2.51	dacite breccia
NB-12-131	M612273	19.80	23.49	2.48	dacite breccia
NB-12-131	M612274	23.49	26.92	2.45	dacite breccia
NB-12-131	M612275	26.92	27.53	2.25	dacite breccia
NB-12-131	M610885	120.94	122.40	2.32	dacite breccia
Average				**2.36**	**Dacite Breccia**

For Jolly Jane, a specific gravity of 2.34 was used to determine tonnage.

14.4.5 GRADE ESTIMATION

Grades for gold were interpolated by ordinary kriging into all blocks, with some percentage within the Jolly Jane mineralization solid. Kriging was completed in a series of passes with the dimensions and orientation of the search ellipse for each pass tied to the semi-variogram for gold. The first pass used dimensions equal to ¼ of the semi-variogram range in the three principal directions. If a minimum of 4 composites were found within this ellipse centered on a block, the block was estimated. For blocks not estimated, the search ellipse was expanded to ½ the semi-variogram range. Again, a minimum of 4 composites within the search ellipse were required to estimate any given block. A third pass using the full semi-variogram range was completed for blocks not estimated during the first two passes. Finally, a fourth pass using roughly twice the range was completed. In all cases if more than 12 composites were located in any search, the closest 12 were used. A maximum of three composites from any individual hole were allowed in all passes. The search parameters for the Kriging procedure are tabulated below (Table 14-31).

A similar procedure was used to estimate silver with the pass four ellipse expanded to the pass four gold search to ensure all blocks estimated for gold had a silver value.

Volumes for each block estimated were determined by multiplying the block volume by the percentage of block below topography and within the solid. The tonnage was determined by multiplying the block volume by the S.G. (2.34).

Table 14-31 – Summary of Kriging Search Parameters for Jolly Jane

Domain	Pass	Number Estimated	Az/Dip	Dist. (m)	Az/Dip	Dist. (m)	Az/Dip	Dist. (m)
Mineralized Solid Au	1	3,145	0 / -30	30.0	270 / -70	12.5	90 / -20	27.5
	2	35,918	0 / -30	60.0	270 / -70	25.0	90 / -20	55.0
	3	34,632	0 / -30	120.0	270 / -70	50.0	90 / -20	110.0
	4	7,968	0 / -30	240.0	270 / -70	100.0	90 / -20	220.0
Mineralized Solid Ag	1	1,473	0 / -30	30.0	270 / -50	15.0	90 / -40	20.0
	2	28,358	0 / -30	60.0	270 / -50	30.0	90 / -40	40.0
	3	38,531	0 / -30	120.0	270 / -50	60.0	90 / -40	80.0
	4	13,301	0 / -30	240.0	270 / -50	120.0	90 / -40	160.0

14.5 MINERAL RESOURCE CLASSIFICATION

Mineral Resources are characterized according to CIM Definitions Standards which are incorporated by reference in NI 43-101. Mineralization at NBP has been categorized as Inferred Mineral Resources, Indicated Resources and Measured Resources, based upon increasing levels of confidence in various physical characteristics of each Project. Drill hole spacing, search neighborhoods, metallurgical characterization, geological confidence, kriging variance and many other factors were used to give the author confidence in the Mineral Resource estimate. Appropriate classification criteria should aim to integrate all these concepts. The author is satisfied that the geological modelling for the Phase I mining study honors the geological information and knowledge of each of the deposits. The location of the samples and the assay data are sufficiently reliable to support resource evaluation.

The author has chosen to use kriging variance versus distance as the approach to determining the Mineral Resource classification at Sierra Blanca, YellowJacket and Mayflower. Kriging variance was used for the classification of mineral resources which were derived using ordinary kriging.

14.5.1 SIERRA BLANCA

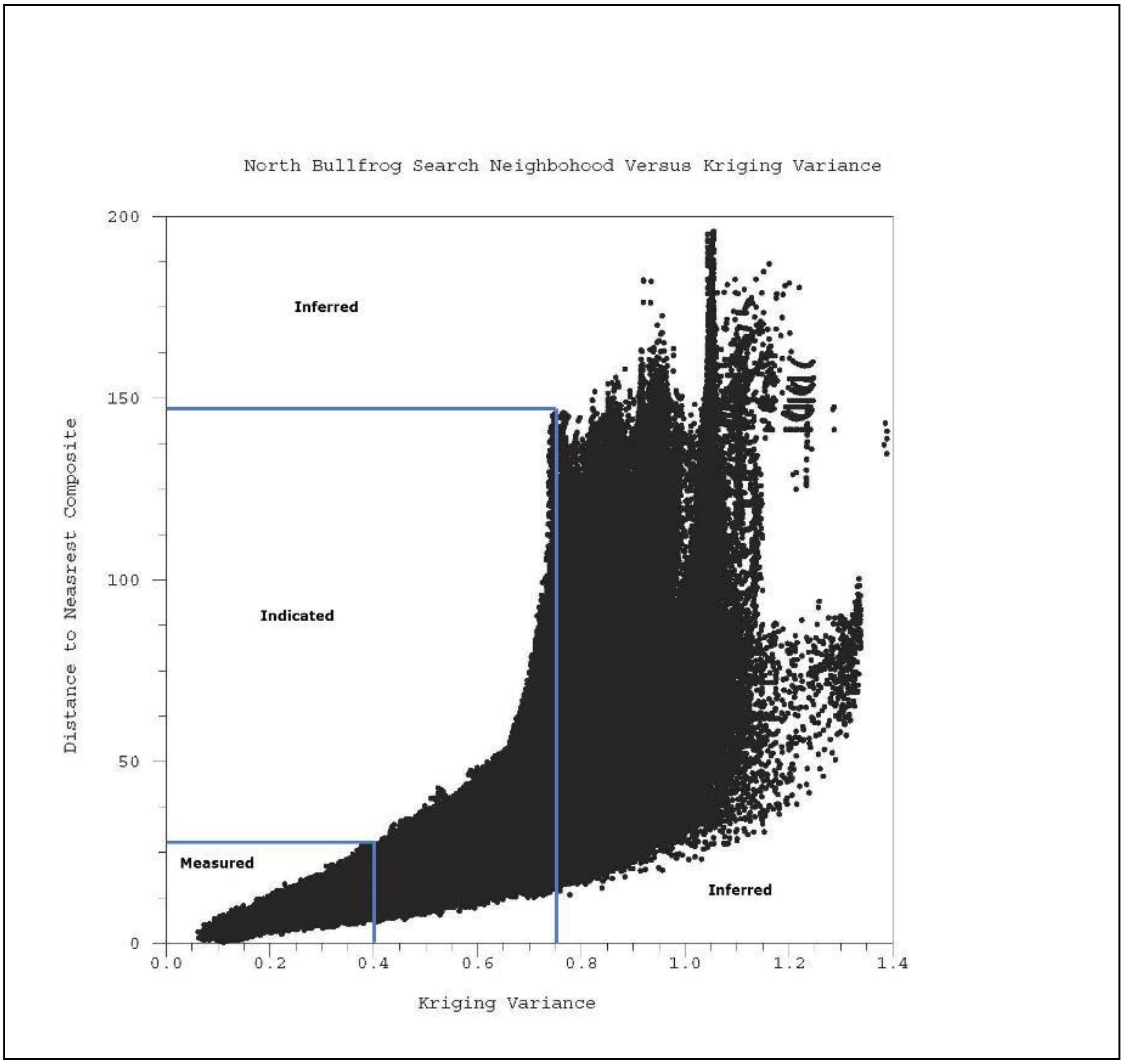
Disseminated mineralization at Sierra Blanca has been categorized according to kriging variance. Kriging is an interpolation technique that minimizes the squared error between the estimated value and the unknown true value.

The resulting error variance, stored as kriging variance in the block model, is dependent only on the estimation location, the location of samples used in the estimate and the variogram. Kriging variance thresholds were used to differentiate between measured, indicated and inferred categories. The advantage of using kriging variance is the consideration of the spatial structure of the gold and silver grades and redundancy between the samples.

Drill hole NB-16-309 was flagged and determined to have a variance of 0.406, the highest kriging variance in the deposit. A graph of kriging variance versus distance as a point cloud was evaluated and displayed in Figure 14-29 as the basis for classification. A variance of 0.406 correlated well with a distance of approximately 25 meters. No extraneous holes or islands of mineralization have been interpreted at this threshold. The kriging variance of 0.406 was chosen to be the boundary between measured and indicated classification as shown in point cloud in Figure 14-29.

Indicated Mineral Resource has been interpreted for the kriging variance versus distance graph at a threshold of 0.768. This correlates with the near vertical departure of the lower end of the point cloud with the majority of the mineralization being within 50 meters of a drill hole. There are a few model blocks between 50 and 105 meters that fall into the indicated category, but the geological confidence is very high for these few blocks. Globally, any model block greater than 0.768 is neither supported by minimal samples nor is located greater than 50 meters from a drill hole as shown in Figure 14-29.

Figure 14-29 - Kriging Variance vs. Distance, Sierra Blanca Classification-meters



14.5.2 MAYFLOWER

Both Pass 1 and Pass 2 of the Mayflower estimation were evaluated separately and categorized according to a combination of kriging variance and distances to the nearest sample. Kriging is an interpolation technique that minimizes the squared error between the estimated value and the unknown true value. The resulting error variance, stored as kriging variance in the block model, is dependent only on the estimation location, the location of samples used in the estimate and the variogram. Kriging variance thresholds were used to differentiate between measured, indicated and inferred categories. The advantage of using kriging variance is the consideration of the spatial structure of the gold and silver grades and redundancy between the samples.

Drill hole NB-12-133 was flagged and determined to have the highest kriging variance in pass 1 at 0.013. Likewise, hole NB-12-180 showed the highest kriging variance of 0.31 for blocks estimated with Pass 2. The highest kriging variance estimated in the deposit is 1.14. A graph of kriging variance versus distance was evaluated for both passes. A variance of 0.013 correlated well with a distance of approximately 25 meters for Pass 1 and those blocks have been classified as measured. Mineralization from 25 to 55 meters are considered to be indicated and greater than 55 meters are inferred blocks. Pass 2 mineralization has been classified as measured within 25 meters, 25 – 40 meters are considered indicated and blocks greater than 40 meters are inferred. No extraneous holes or islands of mineralization have been interpreted at this threshold. There are cases where mineralization has been interpolated within the distances with minimal sample support. For these cases, the slope and intercept were used to reduce the confidence of those blocks. Figures 14-30 and 14-31 show the point cloud relationships of kriging variance to distance for the Mayflower estimation passes. A typical cross section through the Mayflower block mode in Figure 14-32 shows the classification of model blocks and their relationship to drillholes.

Figure 14-30 - Pass 1 Kriging Variance versus Distance

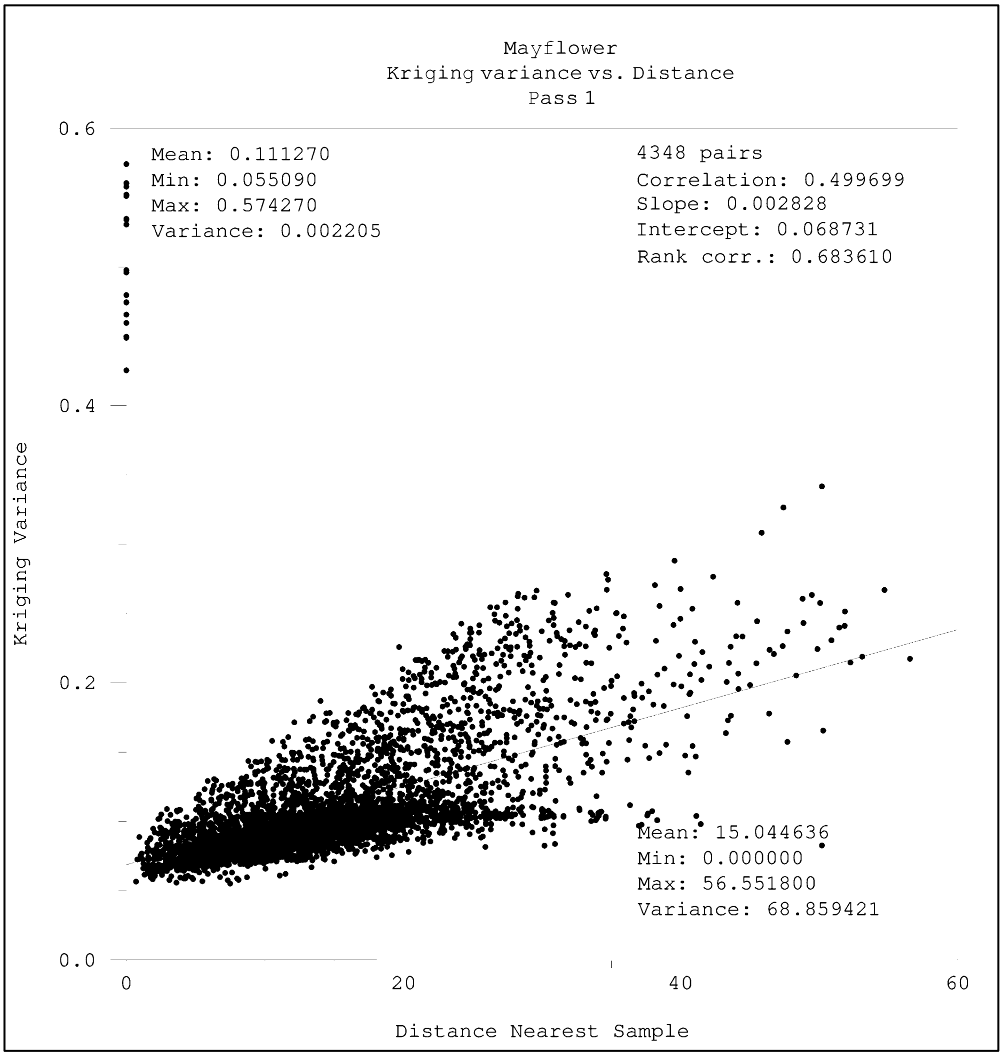


Figure 14-31 - Pass 1 Kriging Variance versus Distance

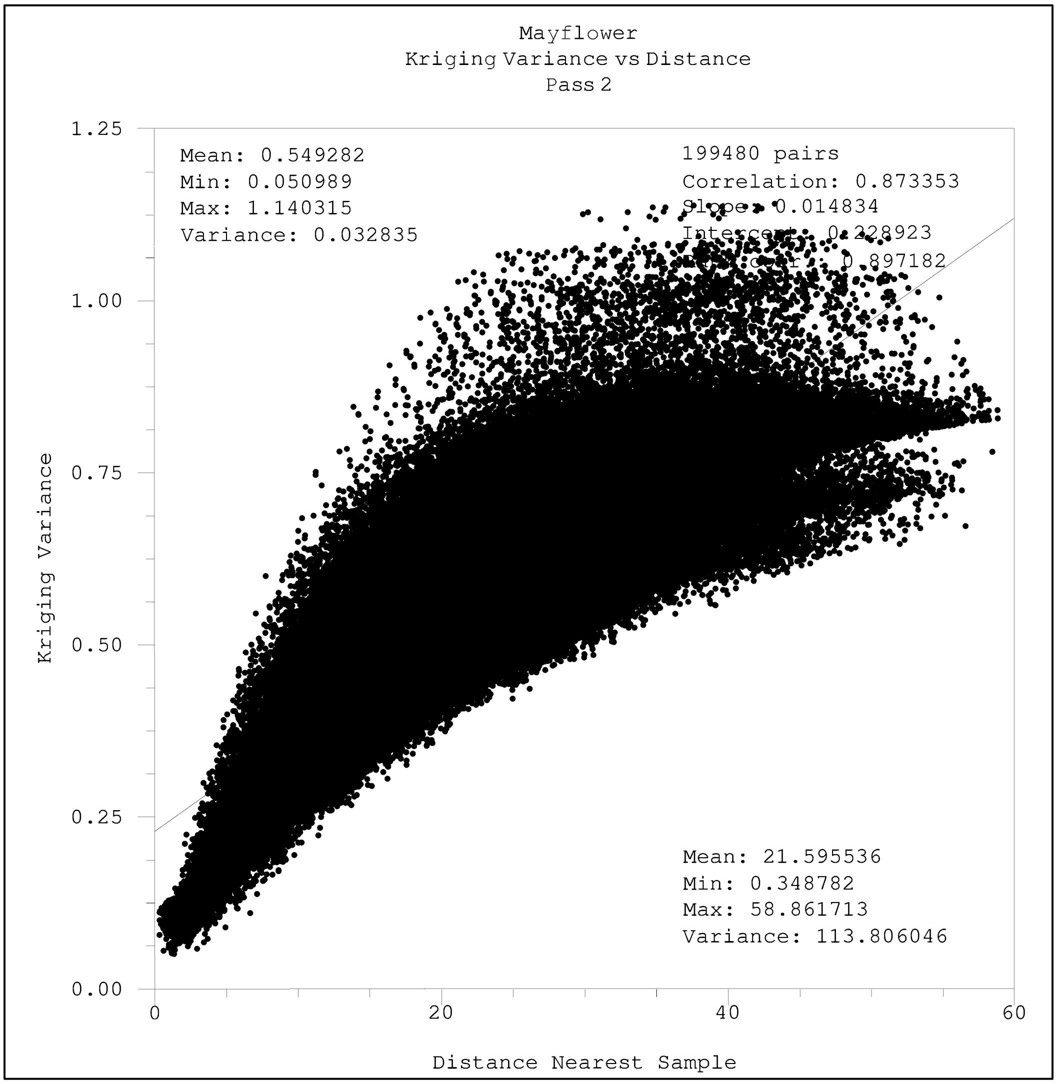
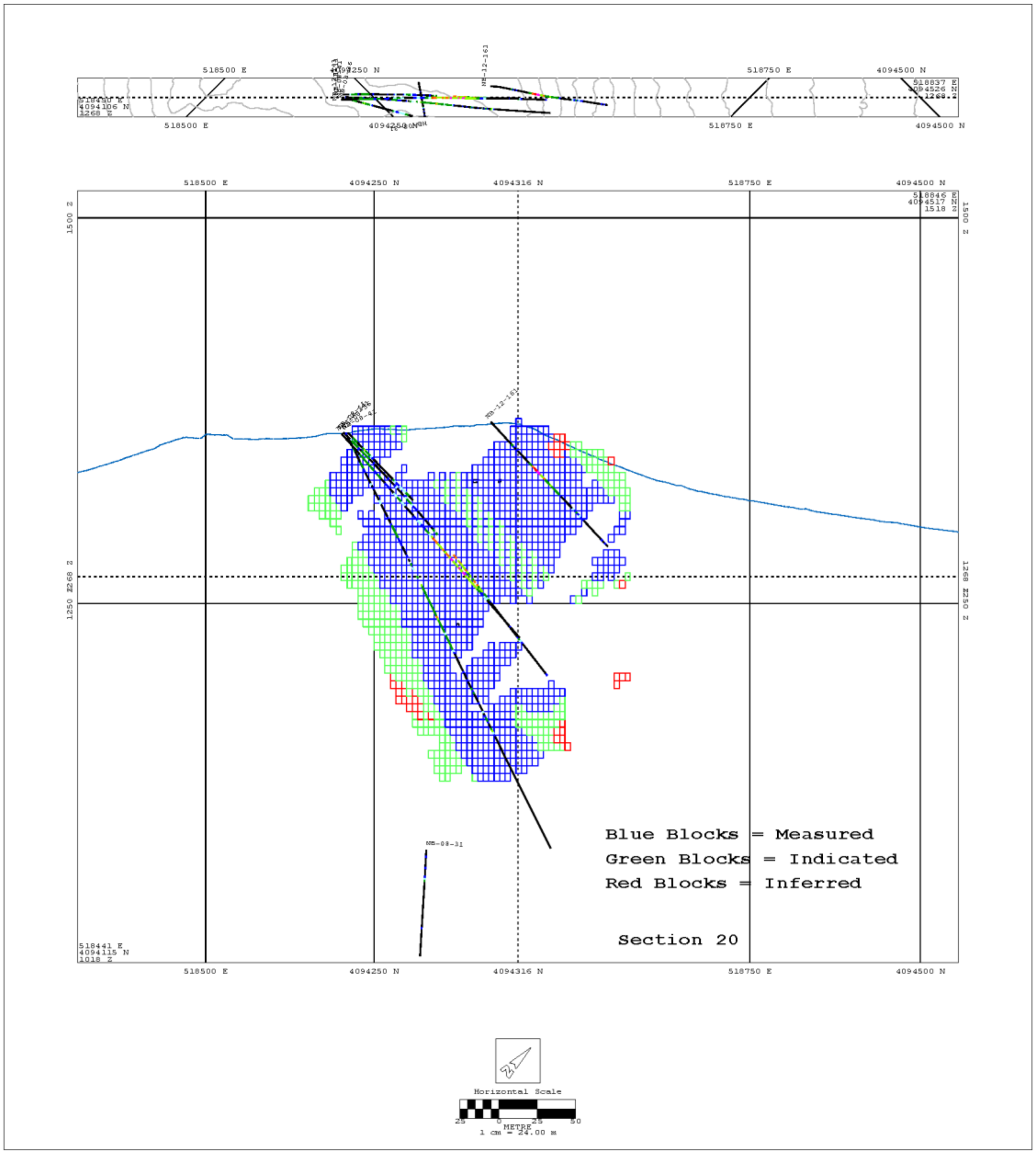


Figure 14-32 - Cross Section Showing Classification of Mineralization at Mayflower



14.6 MINERAL RESOURCES

14.6.1 PIT CONSTRAINING PARAMETERS

Mineral Resources must demonstrate reasonable prospects for eventual economic extraction in accordance with the CIM Definition Standards. Pit constraining limits were determined using the Lerchs-Grossman© economic algorithm which constructs lists of related blocks that should or should not be mined. The final list defines a surface pit shell that has the highest possible total value, while honoring the required surface mine slope and economic parameters.

Mineral Resources are not Mineral Reserves and do not demonstrate economic viability. There is no certainty that all or any part of the Mineral Resource will be converted to mineral reserves. The author knows of no environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that may materially affect the Mineral Resource estimate in this report. Quantity and grade are estimates and are rounded to reflect the fact that the resource estimate is an approximation.

Economic parameters used in the analysis are listed in Table 14-32 and are based on the following processes:

- Gravity mill processing of YellowJacket mineralization
- Heap leaching of the gravity tail component of YellowJacket mineralization
- Heap Leach processing of disseminated mineralization
- Stockpiling of sulphide mineralization for future processing

Table 14-32 - Pit Constraining Parameters Used for the Mineral Resource Update, October 7, 2020

Parameter	Unit	Mayflower	Jolly Jane	Sierra Blanca	YellowJacket Vein/Vein Stockwork	YellowJacket Sierra Blanca Sulphides
Mining Cost	$/tonne	2.23	1.70	1.72	1.72	1.72
Au Cut-Off	g/tonne	0.06	0.065	0.065	0.17	0.17
Processing Cost	S$/tonne	1.72	1.70	1.72	3.75	25.60
Au Recovery	%	70	62	74	85	91
Ag Recovery	%	8	8	6.3	65	57
Administrative Cost	$/tonne	0.50	0.50	0.35	0.50	0.50
Dewatering Below Water Table	$/tonne	0	0	0.15	0.15	0.15
Re-handle	$/tonne	0	0	0	0.75	0.75
Refining & Sales	$/oz	5.00	5.00	5.00	5.00	5.00
Reclamation	$/tonne	0.12	0.12	0.12	0.12	0.12
Au Selling Price	$/oz	1,500	1,500	1,500	1,500	1,500
Ag Selling Price	$/oz	19.20	19.20	19.20	19.20	19.20
Slope Angle	Degrees	50	50	58 – East Highwall 38 - West Highwall	58 – East Highwall 38 - West Highwall	58 – East Highwall 38 - West Highwall

The parameters listed in Table 14-32 define a realistic basis to estimate the Mineral Resources for the Project and are representative of similar mining operations throughout Nevada. The Mineral Resource has been limited to mineralized material that occurs within the pit shells and which could be scheduled to be processed based on a defined cut-off grade. All other material within the defined pit shells was characterized as non-mineralized material.

14.7 MINERAL RESOURCES

14.7.1 SIERRA BLANCA AND YELLOWJACKET ESTIMATED MINERAL RESOURCES

Resource tables for the individual NBP mineralization areas are presented in the following sections and are accumulated according by separate classification.

The estimate considers three processing methods: 1) mill processing of gravity-separable gold and silver from the YellowJacket vein and vein stockwork mineralization; 2) heap leach processing of the gravity tail from YellowJacket vein and vein stockwork mineralization; and 3) heap leach processing of disseminated oxide gold and silver mineralization. Mineralization within the YellowJacket vein, including the surrounding stockwork veining, associated with the YellowJacket structural corridor, is situated within a well-defined zone that holds together at higher cutoff grades within resource constraining pit. Since this part of the mineral deposit contains the highest grades, it is reasonable to expect this part of the Sierra Blanca deposit would be economically extracted prior to economically extracting lower grade, heap-leachable mineralization. The limits of the constraining pit were determined by assuming only mill mineralization and higher grade disseminated mineralization would be processed. Mineralization is reported at higher cutoff grades for this portion of the deposit. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

NBP Mineral Resources are estimated according to the following break-even cutoff grades and are reported in Table 14-33:

- Mill Resources (Oxide, Vein and Stockwork Mineralization) - >= 0.204 g/t Au
- Heap Leach Resources (Oxide Vein Mineralization) – (>= 0.06 and <0.204) g/t Au
- Heap Leach Resources (Oxide Disseminated Mineralization) - >= 0.06 g/t Au
- Sulphide Resources (Sulphide Mineralization) - >= 0.40 g/t Au

Silver is assumed to be recovered as a byproduct of processing and is reported from model blocks that meet the gold cutoff grade criteria.

Mineral Resources which meet the reasonable prospects of eventual economic extraction, based upon the costs and parameters in Table 14-33, are reported at the breakeven cutoff grades for the mineralization and processing methods described. There is no assurance that any or all of the Mineral Resources will be converted to mineral reserves.

Table 14-33 - North Bullfrog Project Pit Constrained Mineral Resource Estimate

Classification	Tonnes (k)	Au (g/t)	Ag (g/t)	Contained Au (000's)	Contained Ag (000's)
Measured	37,140	0.56	3.20	669	3,816
Indicated	154,997	0.29	1.30	1,438	6,490
Inferred	67,672	0.19	0.59	414	1,292

(1) The qualified person of the above estimate is Scott Wilson, C.P.G., SME.

(2) The Mineral Resources are classified as Measured, Indicated and Inferred Mineral Resources, and are based on the 2014 CIM Definition Standards.

(3) Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

(4) Mineral Resources are estimated using a gold price of $1,500/oz.

(5) Numbers may not add up due to rounding.

(6) The effective date of this Mineral Resource estimate is October 7, 2020.

(7) The quantity and grade of reported inferred Mineral Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these inferred Mineral Resources as indicated or measured Mineral Resources.

(8) The qualified person knows of no environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that may materially affect the Mineral Resource estimates in this Technical Report.

14.8 NBP MINERAL RESOURCES BY DEPOSIT

14.8.1 SIERRA BLANCA AND YELLOWJACKET MINERAL RESOURCES

Table 14-34 shows the resources separated by process method for Measured, Indicated and Inferred resources for Sierra Blanca and YellowJacket. Resources are pit constrained and reported at cutoff grades based on processing methods.

Table 14-34 - Sierra Blanca and YellowJacket Mineral Resources Estimate

Milling Sulphide & Oxide COG 0.204 and 0.400 Au gpt	Heap Leach Oxide COG 0.060 Au gpt

Classification	Tonnes (Kt)	Au g/t	Ag g/t	Tonnes (Kt)	Au g/t	Ag g/t	Au Ounces (x1,000)	Ag Ounces (x1,000)
Measured	9,539	1.46	10.18	18,538	0.21	1.01	573	3,720
Indicated	15,130	1.21	7.61	119,829	0.18	0.66	1,280	6,231
Inferred	418	0.97	7.96	55,803	0.19	0.58	362	1,151

(1) Sulphide mineralization processing assumes bio-oxidation and would be stockpiled for future operations.

(2) The qualified person of the above estimate is Scott Wilson, C.P.G., SME.

(3) The Mineral Resources are classified as Measured, Indicated and Inferred Mineral Resources, and are based on the 2014 CIM Definition Standards.

(4) Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

(5) Mineral Resources are estimated using a gold price of $1,500/oz.

(6) Numbers may not add up due to rounding.

(7) The effective date of this Mineral Resource estimate is October 7, 2020.

(8) The quantity and grade of reported inferred Mineral Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these inferred Mineral Resources as indicated or measured Mineral Resources.

(9) The qualified person knows of no environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that may materially affect the Mineral Resource estimates in this Technical Report.

14.8.2 MAYFLOWER MINERAL RESOURCES

Table 14-35 lists the Mineral Resources for Mayflower. All material is assumed to be processed by heap leaching methods. Resources are pit constrained and reported at a gold cut-off grade of 0.08 g/t Au.

Table 14-35 - Mayflower Pit Constrained Mineral Resource Estimate

Classification	Tonnes (kt)	Au g/t	Ag g/t	Contained Au (x1,000)	Contained Ag (x1,000)
Measured	9,063	0.33	0.33	97	97
Indicated	1,461	0.31	0.30	14	14
Inferred	147	0.41	0.37	2	2

(1) The qualified person of the above estimate is Scott Wilson, C.P.G., SME.

(2) The Mineral Resources are classified as Measured, Indicated and Inferred Mineral Resources, and are based on the 2014 CIM Definition Standards.

(3) Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

(4) Mineral Resources are estimated using a gold price of $1,500/oz.

(5) Numbers may not add up due to rounding.

(6) The effective date of this Mineral Resource estimate is October 7, 2020.

(7) The quantity and grade of reported inferred Mineral Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these inferred Mineral Resources as indicated or measured Mineral Resources.

(8) The qualified person knows of no environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that may materially affect the Mineral Resource estimates in this Technical Report.

14.8.3 JOLLY JANE MINERAL RESOURCES

Table 14-36 lists the Mineral Resources for Jolly Jane. All material is assumed to be processed by heap leaching methods. Resources are pit constrained and reported at a gold cut-off grade of 0.08 g/t Au.

Table 14-36 – Jolly Jane Pit Constrained Mineral Resource Estimate

Classification	Tonnes (kt)	Au g/t	Ag g/t	Contained Au (x1,000)	Contained Ag (x1,000)
Indicated	18,577	0.24	0.41	143	245
Inferred	7,744	0.20	0.56	50	139

(1) The qualified person of the above estimate is Scott Wilson, C.P.G., SME.

(2) The Mineral Resources are classified as Measured, Indicated and Inferred Mineral Resources, and are based on the 2014 CIM Definition Standards.

(3) Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

(4) Mineral Resources are estimated using a gold price of $1,500/oz.

(5) Numbers may not add up due to rounding.

(6) The effective date of this Mineral Resource estimate is October 7, 2020.

(7) The quantity and grade of reported inferred Mineral Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these inferred Mineral Resources as indicated or measured Mineral Resources.

(8) The qualified person knows of no environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that may materially affect the Mineral Resource estimates in this Technical Report.

14.9 VISUAL VALIDATION

The portions of the mineralization block models bounded by the Whittle^{TM} defined open pit mining shells are illustrated in Figures 14-33 through 14-37. Figures 14-33 and 14-34 contain a long section through the YellowJacket corridor and cross-section through Sierra Blanca and YellowJacket, respectively. Figures 14-35 and 14-36 are cross-sections and long sections, respectively, through Mayflower. A representative cross-section through Jolly Jane is shown in Figure 14-37.

Figure 14-33 - Long Section through Sierra Blanca/YellowJacket Mineral Resource Model

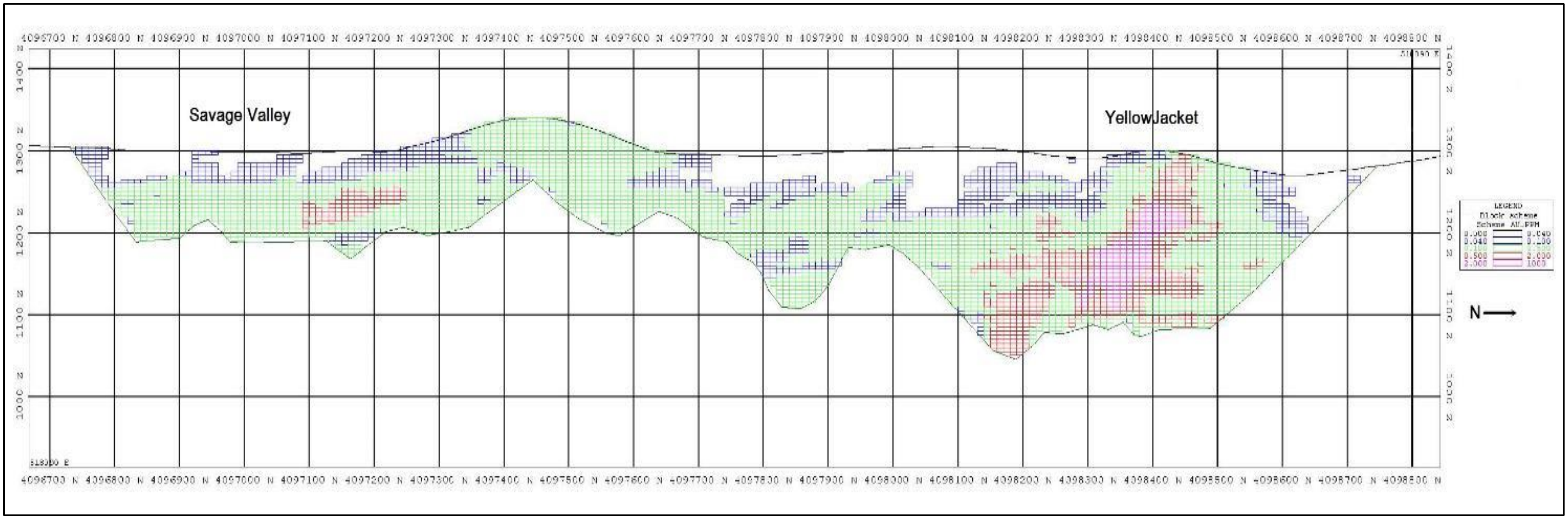


Figure 14-34 - Cross Section across Sierra Blanca/YellowJacket Looking North

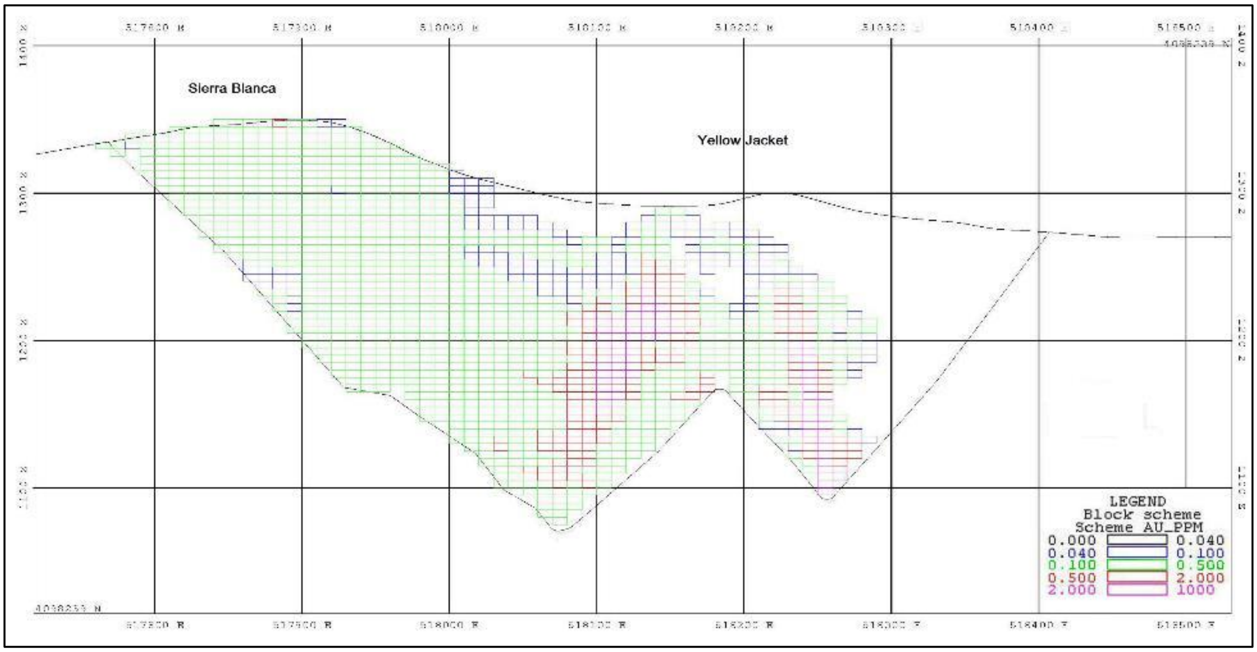


Figure 14-35 - Mayflower Section 20 Gold Cross Section

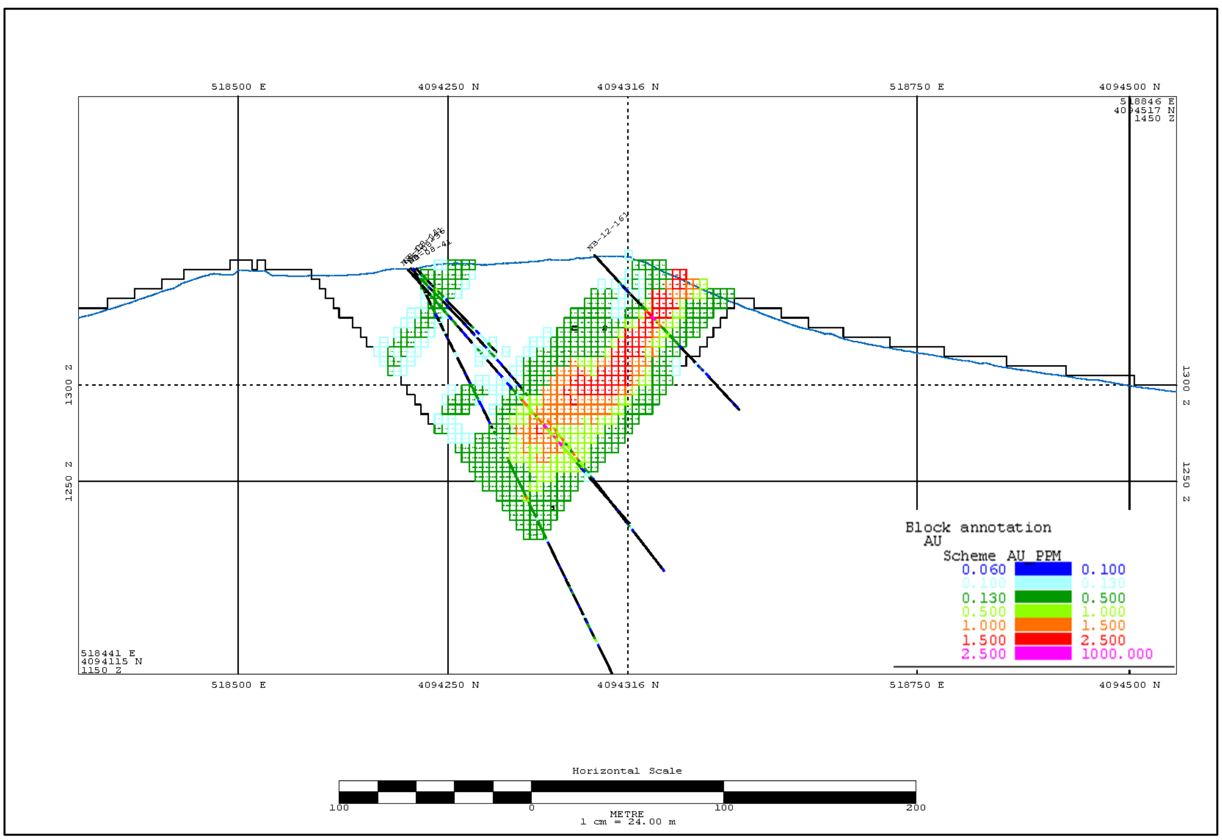


Figure 14-36 - Long Section through Mayflower

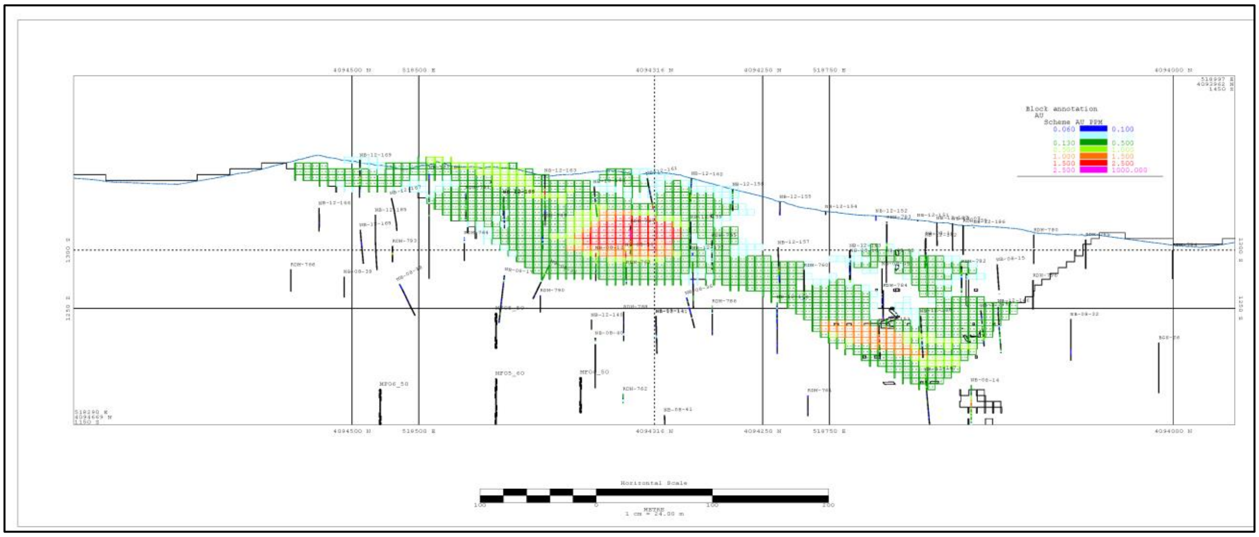
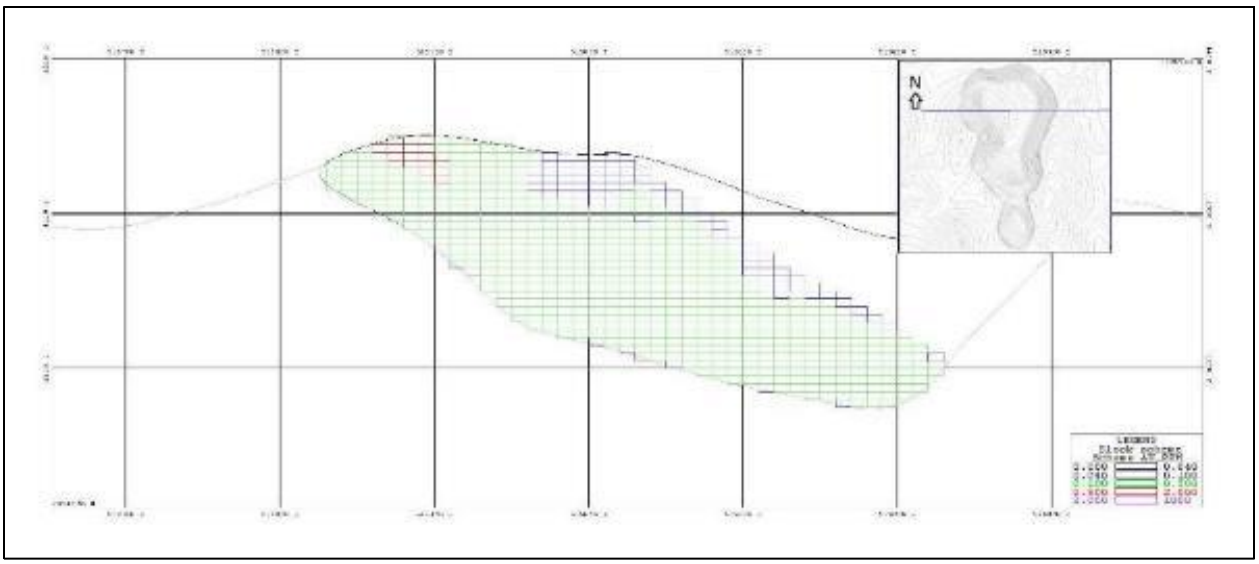


Figure 14-37 - Cross Section through the Jolly Jane Mineral Resource Model. Section Location Identified on Plan Inset Map



15. MINERAL RESERVE ESTIMATE

There are no Mineral Reserves estimated for the NBP.

16. MINING METHODS

This PEA is preliminary in nature and is based on technical and economic assumptions which will be evaluated in more advanced studies. The PEA is based on the North Bullfrog Mineral Resource models which includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. The basis for the PEA is to demonstrate the economic viability of the Project. The PEA results are only intended as an initial, first-pass review of the Project economics based on preliminary information. There are no advanced studies on the Project that would be impacted by the PEA.

The NBP contains mineralization at or near the surface that is suitable for open pit mining methods. Gold grade distribution and the results of preliminary mineral processing test work indicate that North Bullfrog resources can be processed by conventional heap leaching methods for the bulk of the material and a gravity milling circuit followed by conventional heap leach for the YellowJacket vein and stockwork material. The method of material transport evaluated for this PEA is open pit mining using 20 cubic meter loaders as the main loading units with 133-t rigid frame haul trucks.

16.1 PROPOSED MINING METHODS

The proposed mining method is conventional open pit mining. Mineralized material and waste are drilled and blasted, then loaded into 133-t payload haul trucks with ~ 20 m^3 bucket capacity wheel loaders. The loading and haulage fleet would be supported by track dozers, motor graders, and water trucks. Waste would be hauled to Waste Rock Management Facilities (WRMF) near each pit. Mill resources would be hauled to the mill stockpile while run-of-mine (ROM) resources would be hauled and placed directly on the heap leach pad. Mill tailings would be hauled and placed directly on the heap leach pad and mixed into the ROM material by dozing.

The proposed mining operation would employ owner operated equipment. The general site layout, including pits, waste dumps, mill site, ponds, and heap leach pad, is shown in Figure 16-8.

Mill resource production was planned at a nominal rate of 4,700 t/d, equivalent to 1.7 Mt/y with the majority being mined in the first 7 years. Run of Mine (ROM) heap leach resource would be mined at a nominal rate of 32,300 t/d, equivalent to 11.8 Mt/y for the first 5 years followed by a nominal rate of 51,200 t/d, equivalent to 18.7 Mt/y for years 6 through 13. Mining was planned on a 7 day per week 24-hour per day schedule, 360 days per annum. The average mineralized material and waste production would be approximately 91,000 t/d. The average LOM mining stripping ratio would be 0.96:1 waste-to-mineralized material, using a 0.13 Au gpt cut-off for ROM resources and a 0.23 Au gpt cut-off for mill resources in production years 1-5. The selection cutoff grade for years 6-13 is 0.06 Au gpt cut-off for ROM mineralized material and a 0.23 Au gpt cut-off for mill mineralized material.

16.2 PARAMETERS RELEVANT TO MINE OR PIT DESIGNS AND PLANS

The Yellowjacket vein material would be processed by gravity separation and high intensity leaching of the concentrate. The gravity tail material would be added to the conventional heap leach. Metallurgical test work indicates approximately 45% of the gold would be recovered in the gravity mill circuit and 72% of the mill tail gold would be recovered in the heap leach process, for a total recovery of approximately 84%. The ROM mineralization will be direct shipped to the leach pad and metallurgical data indicate approximately 74% recovery of gold. The metallurgical testing assumed that the mineralized material would be blasted with the Ultra-High-Intensity-Blasting (Brent, et. al., 2014) method [UHIB] to create a highly fractured material.

16.2.1 GEOTECHNICAL DESIGN – PITS

The Preliminary pit designs were based on an overall 50° inter-ramp angle. A pit slope analysis was completed by Knight Piésold and Company in May 2020. Table 16-1 summarizes the criteria from the Knight Piésold analysis of the Yellowjacket Pit. This inter-ramp angle was consistent with analyses conducted for the Jolly Jane open pit (Engineering Analytics, 2013a) and the Mayflower open pit (Engineering Analytics, 2013b).

Table 16-1 - Geotechnical Criteria for Mine Design

Pit Design Criteria	All Rock Units
Inter-Ramp Angles	58°
Face Angles	80°
Catch Bench Berm	8 m
Catch Bench Vertical Spacing	20 m
Road Widths (Including Berm)	32 m
Road Grade	10%

16.2.2 WASTE ROCK MANAGEMENT FACILITIES

The Waste Rock Management Facilities (WRMF) were based on the current mine plan, which predicts that approximately 206.8 Mt of waste rock will require placement and management. The waste rock would be placed in four WRMFs at an overall reclaimed slope of 3H:1V (18.4°). Approximately 91.5 Mt (YellowJacket, Sierra Blanca and Savage pits) would be placed in the East WRMF, approximately 80.9 Mt (Sierra Blanca pit) would be placed in the West WRMF, approximately 20.1 Mt would be placed in the Mayflower WRMF and approximately 14.3 Mt would be placed in the Jolly Jane WRMF. The WRMFs are located adjacent to the pits, as shown in Figure 16-8. A summary of basic design assumptions and dimensions for the proposed WRMFs is shown in Table 16-2.

Vegetation would be cleared from the WRMF footprints; plant growth medium would be salvaged and placed in separate stockpiles. The final surfaces of the WRMF would be constructed by end dumping to create typical mining waste rock facilities. On sloped terrain, where safe and practicable, some weathered geologic materials below the plant growth medium may be pushed downhill to construct toe berms, to prevent rocks from scattering on the hillside below the toes of the WRMF.

Table 16-2 - Design Criteria Assumptions for Waste Rock Management Facilities

WRMF	As-built slope (degrees)	Height (m)	Footprint (acres)
East	18.4	120	365
West	18.4	90	263
Mayflower	18.4	60	145
Jolly Jane	18.4	50	111

16.2.3 PIT HYDROGEOLOGY AND PREDICATED WATER INFLOW

The extent of the YellowJacket vein and vein stockwork mineralization reaches below the local water table, so the mining operation is projected to require local dewatering. Evaluation is ongoing for the Sierra Blanca and Jolly Jane pit areas to determine if there will be a pit lake based on the current mine plan. The Mayflower pit is projected to be a dry pit. Existing, but limited, hydrogeologic data are available and have been used to assess the potential magnitude of pit water inflow during mining (HydroGeoLogica, Inc., 2020).

Water level data and bottom-hole water production rates during drilling in the YellowJacket area have been used, in conjunction with regional ground water information for projective estimates of pit water inflow during mining. A range of K values (hydraulic conductivity) have been estimated and used to calculate pit water inflow during mining. Table 16-3 shows the range of K assumptions. The water level in the YellowJacket and Sierra Blanca area was approximately 1,200 m amsl.

Groundwater elevations from the Dewatering Model indicated that the model was simulating the groundwater flow field reasonably. The Intermediate-K case indicated the 1,200 m amsl contour did not reach the no-flow model boundary of the Dewatering Model indicating this approach is reasonable for the predictions.

Table 16-3 - Simulated Dewatering Rates (gpm) Over the YellowJacket Mining Life (HydroGeoLogica, Inc., 2020)

Projection Case	Year 1	Year 2	Year 3	Year 4	Year 5
High Conductivity - K	0	0	299	598	600
Intermediate Conductivity -K	0	0	47	105	97

Intermediate Conductivity-K with Carbonate Aquifer	0	0	49	328	268
Low Conductivity-K	0	0	12	51	29

The Intermediate-K base case results without the presence of a carbonate aquifer have been used to project pit water production for the purpose of this study and a project peak dewatering rate of 105 gpm. Based on limited site data and regional hydrogeologic analogs, both the Intermediate- and High-K cases are reasonable and possible, with peak dewatering rate ranging from of 105 to 600 gpm. If a high-conductivity carbonate unit is present at depth, the effect was that this range may be higher yet. Hydrogeologic characterization would be conducted in the Pit area to help constrain hydraulic properties of the HGUs (and potential presence of carbonate aquifer). This information will decrease model input uncertainty and improve the accuracy of predictions for mining design and permitting.

16.3 PIT OPTIMIZATION

Pit constraining limits were determined using the Lerchs-Grossman© (LG) economic algorithm which constructs lists of related blocks that should or should not be mined. The pit optimization was discussed in Section 14.

16.3.1 MINERAL RESOURCE MODEL

Measured, Indicated and Inferred resources were considered in the evaluation; Inferred resources represent approximately 1% of the mill resource and approximately 30% of the ROM resource based on the mine plan. A full description of the resource model and modeling techniques is provided in Section 14.

16.3.2 TOPOGRAPHIC DATA

Base topographic data was from publicly available USGS data (www.USGS.gov). No site-specific topographic data has been collected.

16.3.3 OPTIMIZATION PARAMETERS AND CONSTRAINTS

Overall pit slope angles of 50^{O} were used in the LG runs. Dilution was not accounted for in the optimization for this preliminary design exercise. The material surrounding the mineralized material is known to be a gradational low-grade halo therefore the dilution near the cut-off grade is expected to be minimal. The pit optimization parameters and constraints were discussed in Section 14.

16.4 DESIGN CRITERIA

The mine designs were considered preliminary. Pit designs were developed for the first 7 years and LG shell limits ($1,500 Au) were used for the remaining mine life. The LG shells selected for pit designs for the early mining (years 1-5) varied for each deposit; YellowJacket - $975 pit, Sierra Blanca - $875 pit, and Savage - $875 pit. These LG shells were used as a guide for the preliminary pit designs. The design goal was to capture the majority of the mineralized material in the LG pits which resulted in laybacks for haul road access. The Mayflower pit was an existing pit design provided by Corvus from a previous study. The Mayflower design included multiple slope angles, based at on site-specific geotechnical study (Engineering Analytics, 20143b). Haul roads were designed at a width of 32 m and a maximum gradient of 10%. Corvus's pit design criteria are listed in Table 16-4.

Table 16-4 - Pit Design Criteria

Sierra Blanca, YellowJacket, Savage Pits	
Pit Design Criteria	**All Rock Units**
Inter-Ramp Angles	50°
Road Widths	32 m
Road Grade (Max)	10%

Figures 16-1 and 16-2 show the starter pit and year 5 designs for the YellowJacket pit. The Year 5 pit designs for Sierra Blanca and Savage pits are shown on Figures 16-3 and 16-4 and Mayflower is shown on Figure 16-5.

Figure 16-1 - YellowJacket Starter Pit Design

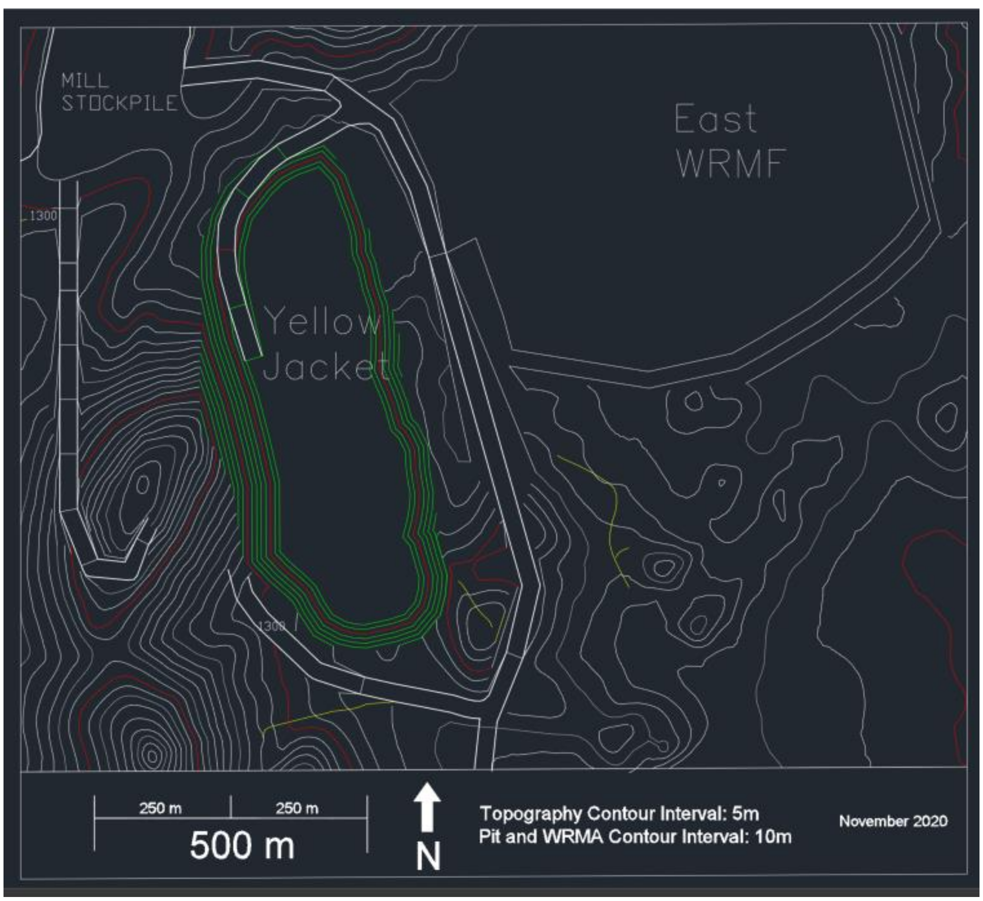


Figure 16-2 - YellowJacket Pit Year 5 Design

Figure 16-3 - Sierra Blanca Pit Year 5 Design

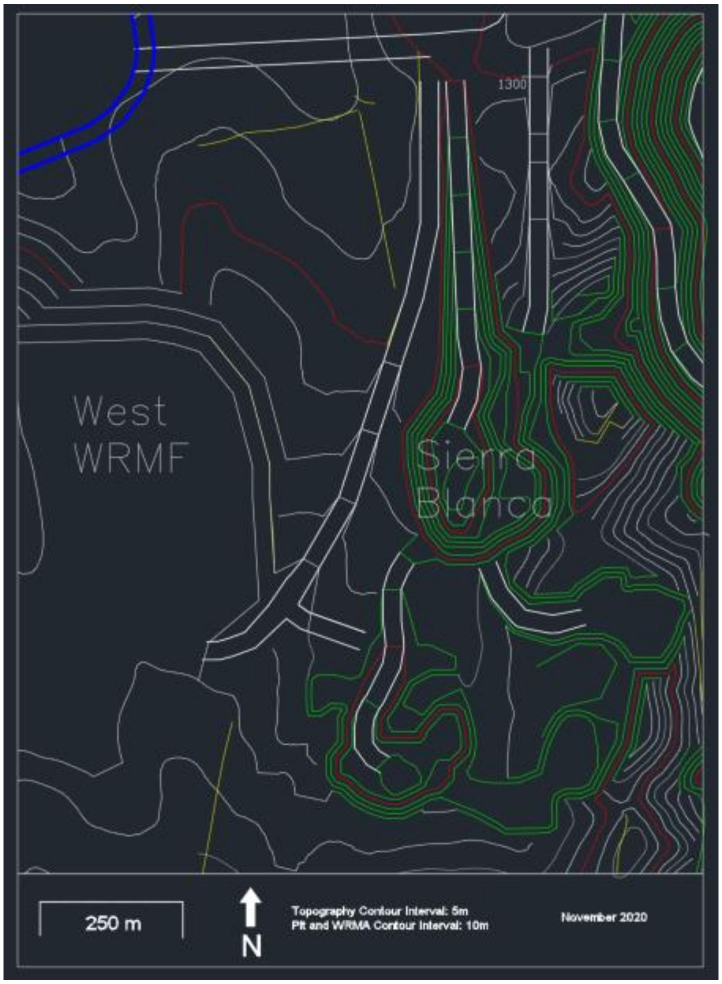
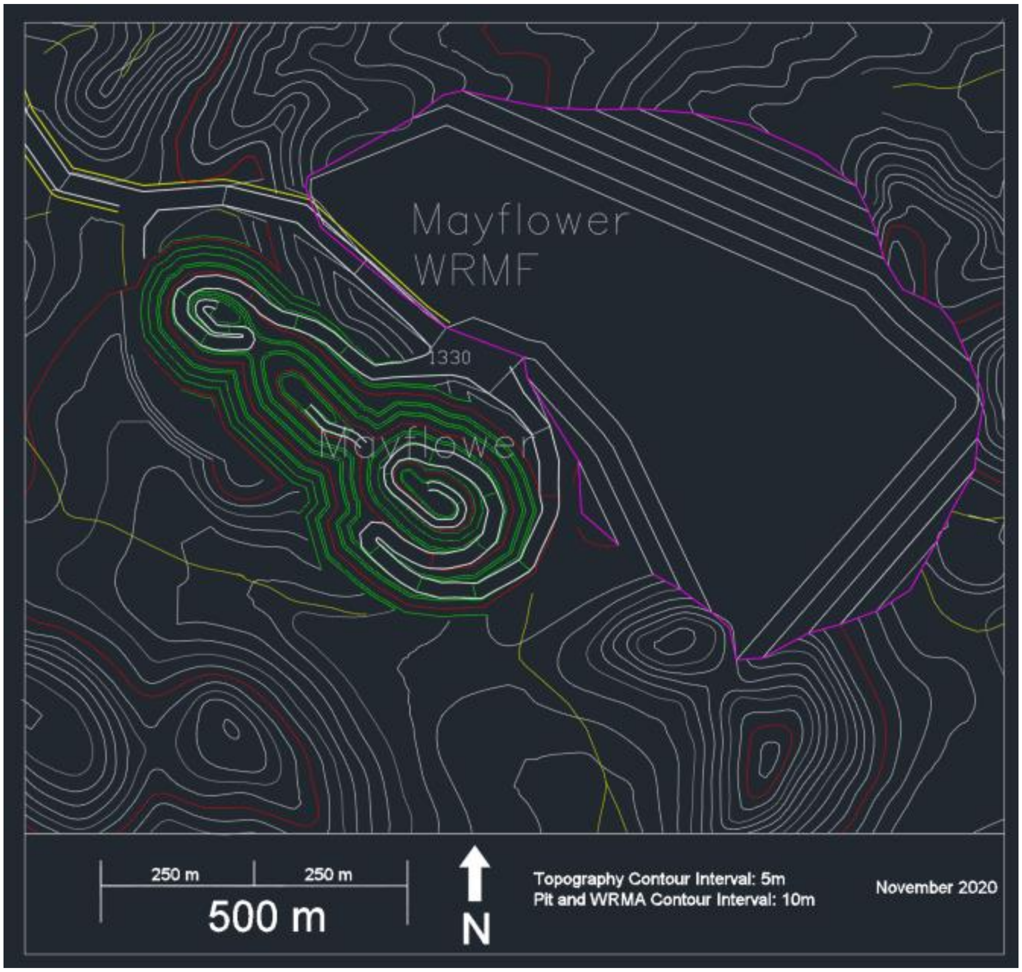


Figure 16-4 - Savage Pit Year 5 Design

Figure 16-5 - Mayflower Pit and Dump Final Design



16.5 MINE PRODUCTION SCHEDULE

The mine plan is depicted in annual periods (12-month periods). The mine production schedule focused on opening the YellowJacket pit to provide the annual gravity mill feed of approximately 1.7 Mt. The other pits were sequenced to provide approximately 11.5 Mt of ROM heap leach resource annually for the first 5 years increasing to 18.7 Mt for years 6 through 13.

16.5.1 MINE PRODUCTION

The yearly mine production schedule was presented in Table 16-5, beginning in period -1 through period 13, for a total of 14 years of mining. The schedule was completed monthly for 24 months, from period -1 through period 1, and annually thereafter. The production schedule is driven by the nominal rate of 4,700 t/d (1.7 Mt/y) of gravity mill resource. Peak mineralized material and waste production is approximately 102,000 t/d and occurs in period 3.

The schedule was developed using the pit bench data from the design pits for years -1 (construction and per-stripping) through year 5, and LG output data was used for years 6 through 13. Initial mining was focused on the YellowJacket pit in order to meet the gravity mill feed demand of 1.7 Mt per year. Sierra Blanca and Savage pits would begin in period 2 and continue through period 13. The Mayflower pit would begin in period 3 and continue through period 5. The Jolly Jane pit would begin in period 5 and continues through period 9.

Table 16-5 - Annual Mining Production Schedule

Production Schedule		-1	1	2	3	4	5	6	7	8	9	10	11	12	13	Total
Total Production	Units															
Mill Mineralized Material[1]	Tonnes (000's) Au g/t Au Oz (000's)	635 0.93 19.0	1,106 1.37 48.7	1,698 2.06 112.3	1,755 2.33 131.7	1,718 1.92 106.0	1,698 1.45 78.9	1,034 0.83 27.6	1,034 0.83 27.6	0 0.00 0.0	0 0.00 0.0	872 0.58 16.3	800 0.58 14.9	0 0.00 0.0	0 0.00 0.0	12,350 1.47 583
ROM Mineralized Material1	Tonnes (000's) Au g/t Au Oz (000's)	2,303 0.21 15.5	10,092 0.23 74.6	9,573 0.25 78.2	12,802 0.30 121.7	13,559 0.32 138.8	16,772 0.26 137.8	20,990 0.19 127.9	21,400 0.19 130.8	21,400 0.19 130.8	21,402 0.19 130.8	16,000 0.14 72.0	16,000 0.14 72.0	18,000 0.13 76.5	8,817 0.16 44.2	209,110 0.20 1,352
Total Mineralized Material[1]	Tonnes (000's) Au g/t Au Oz (000's)	2,938 0.37 34.5	11,198 0.34 123.3	11,272 0.53 190.5	14,556 0.54 253.4	15,277 0.50 244.9	18,470 0.36 216.7	22,024 0.22 155.5	22,434 0.22 158.4	21,400 0.19 130.8	21,402 0.19 130.8	16,872 0.16 88.3	16,800 0.16 87.0	18,000 0.13 76.5	8,817 0.16 44.2	221,460 0.27 1,935
Waste	Tons (000's)	18,406	23,764	17,163	22,794	16,819	14,589	12,497	10,967	10,000	12,000	16,214	15,214	13,000	7,545	211,972
Total	**Tons (000's)**	21,344	34,962	28,434	37,351	32,096	33,059	34,521	33,401	31,400	33,402	33,086	33,014	31,000	16,362	433,432
Strip Ratio		6.26	2.12	1.52	1.57	1.10	0.79	0.57	0.49	0.47	0.56	0.96	0.97	0.72	0.86	0.96
Contained Oz.	Oz (000's)	34.5	123.3	190.5	253.4	244.9	216.7	155.5	158.4	130.8	130.8	88.3	87.0	76.5	44.2	1,934.7

1- Includes Measured, Indicated and Inferred Resource

16.5.2 PIT SCHEDULE SEQUENCE

The mining schedule is described in annual periods would begin in Period -1, with mining starting from the YellowJacket open pit. ROM resource mined in period -1 would be used for overliner crushing feed for the leach pad. Excess ROM during Period -1 would be stockpiled for the initial phase of the leach pad overliner, then placed on the leach pad, once constructed. Gravity mill resource would be placed in a stockpile adjacent to the primary crusher location. The mining sequence would continue in Period 1 through 5 meeting the gravity mill feed target of 1.7M t per year and providing approximately 11.5 M t per year of ROM resource. In years 6 through 13 the gravity mill stockpile would receive additional tonnes as available and the ROM resource will increase to approximately 18.7 Mt per year.

The production schedule by pit is presented in Table 16-6. Charts showing contained gold ounces mined and mineralization and waste production by mining period, are shown in Figure 16-6 and Figure 16-7, respectively. Figure 16-8 shows the original topography and locations of the facilities boundaries. The end of year mine configurations for year -1 through 5 are shown in Figure 16-9 through Figure 16-14. The final mine configuration is shown in Figure 16-15.

Table 16-6 - Production Schedule by Pit

Production Schedule		-1	1	2	3	4	5	6-9[1]	10-13[1]	Total
YellowJacket										
	Units									
Mill Resource	Tonnes (000's)	635	1,106	1,698	1,755	1,718	1,698	2,068	1,672	12,350
	Contained Au g/t	0.93	1.37	2.06	2.33	1.92	1.45	0.83	0.58	1.47
	Au Oz (000's)	19.0	48.7	112.3	131.7	106.0	78.9	55.2	31.2	583.0
ROM Resource	Tonnes (000's)	2,303	10,092	4,411	2,211	361	330	Note 1	Note 1	19,708
	Contained Au g/t	0.21	0.23	0.32	0.27	0.25	0.31			0.25
	Au Oz (000's)	15.5	74.6	45.6	18.9	2.9	3.3			160.8
Waste[2]	Tonnes (000's)	18,406	23,764	10,656	6,269	4,544	2,955	3,722	2,428	72,744
Total	**Tonnes (000's)**	**21,344**	**34,962**	**16,765**	**10,235**	**6,623**	**4,983**	**5,790**	**4,100**	**104,802**
Strip Ratio		6.26	2.12	1.74	1.58	2.14	1.46	1.80	1.45	2.27
Sierra Blanca										
ROM Resource	Tonnes (000's)	0	0	2,298	4,705	4,916	8,472	60,000	58,817	139,208
	Contained Au g/t	0	0	0..22	0.28	0.27	0.23	0.18	0.14	0.17
	Au Oz (000's)	0	0	16.0	43.0	42.0	63.9	345.3	264.7	774.9
Waste	Tonnes (000's)	0	0	633	1,246	2,095	5,365	27,000	54,545	90,884
Total	**Tonnes (000's)**	**0**	**0**	**2,931**	**5,951**	**7,011**	**13,837**	**87,000**	**113,362**	**230,092**
Strip Ratio		0	0	0.28	0.26	0.43	0.63	0.45	0.93	0.65
Savage[1]										
ROM Resource	Tonnes (000's)	0	0	2,864	3,082	4,736	4,283	Note 1	Note 1	14,965
	Contained Au g/t	0	0	0.18	0.24	0.28	0.27			0.25
	Au Oz (000's)	0	0	16.6	23.8	42.6	37.2			120.2
Waste	Tonnes (000's)	0	0	5,874	5,807	1,683	596	Note 1	Note 1	13,960
Total	**Tonnes (000's)**	**0**	**0**	**8,738**	**8,889**	**6,419**	**4,879**	**0**	**0**	**28,925**
Strip Ratio		0	0	2.05	1.88	0.36	0.14	0	0	0.93
Mayflower										
ROM Resource	Tonnes (000's)	0	0	0	2,804	3,546	1,687	0	0	8,037
	Contained Au g/t	0	0	0	0.40	0.45	0.36	0	0	0.41
	Au Oz (000's)	0	0	0	36.1	51.3	19.5	0	0	106.9
Waste	Tonnes (000's)	0	0	0	9,472	8,497	2,138	0	0	20,107
Total	**Tonnes (000's)**	**0**	**0**	**0**	**12,276**	**12,043**	**3,825**	**0**	**0**	**28,144**
Strip Ratio		0	0	0	3.38	2.40	1.27	0	0	2.50
Jolly Jane										
ROM Resource	Tonnes (000's)	0	0	0	0	0	2,000	25,192	0	27,192
	Contained Au g/t	0	0	0	0	0	0.22	0.22	0	0.22
	Au Oz (000's)	0	0	0	0	0	13.9	174.9	0	188.8
Waste	Tonnes (000's)	0	0	0	0	0	3,535	10,742	0	14,277
Total	**Tonnes (000's)**	**0**	**0**	**0**	**0**	**0**	**5,535**	**35,934**	**0**	**41,469**
Strip Ratio		0	0	0	0	0	1.77	0.43	0	0.53
Project Total										

Mill Resource	Tonnes (000's) Contained Au g/t Au Oz (000's)	635 0.93 19.0	1,106 1.37 48.7	1,698 2.06 112.3	1,755 2.33 131.7	1,718 1.92 106.0	1,698 1.45 78.9	2,068 0.83 55.2	1,672 0.58 31.2	12,350 1.47 583.0
ROM Resource	Tonnes (000's) Contained Au g/t Au Oz (000's)	2,303 0.21 15.5	10,092 0.23 74.6	9,573 0.25 78.2	12,802 0.30 121.7	13,559 0.32 138.8	16,772 0.26 137.8	85,192 0.19 520.2	58,817 0.14 264.7	209,110 0.20 1,351.6
Waste	Tonnes (000's)	18,406	23,764	17,163	22,794	16,819	14,589	45,464	52,973	211,972
Total	**Tonnes (000's)**	**21,344**	**34,962**	**28,434**	**37,351**	**32,096**	**33,059**	**132,724**	**113,462**	**433,432**
Strip Ratio		6.26	2.12	1.52	1.57	1.10	0.79	0.52	0.988	0.96

[1] ROM material reported in years 6-13 as Sierra Blanca includes Yellowjacket and Savage pit areas

[2] Sulfide mineralization mined and stockpiled is recorded as waste in Yellowjacket

Figure 16-6 – Contained Gold Ounces Mined by Mining Year

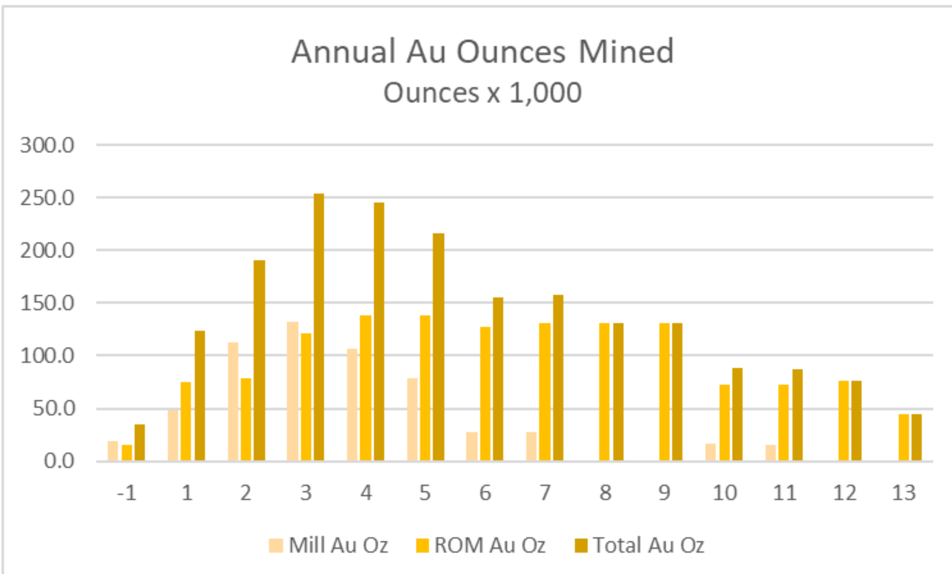


Figure 16-7 - Mineralization and Waste Production by Mining Year

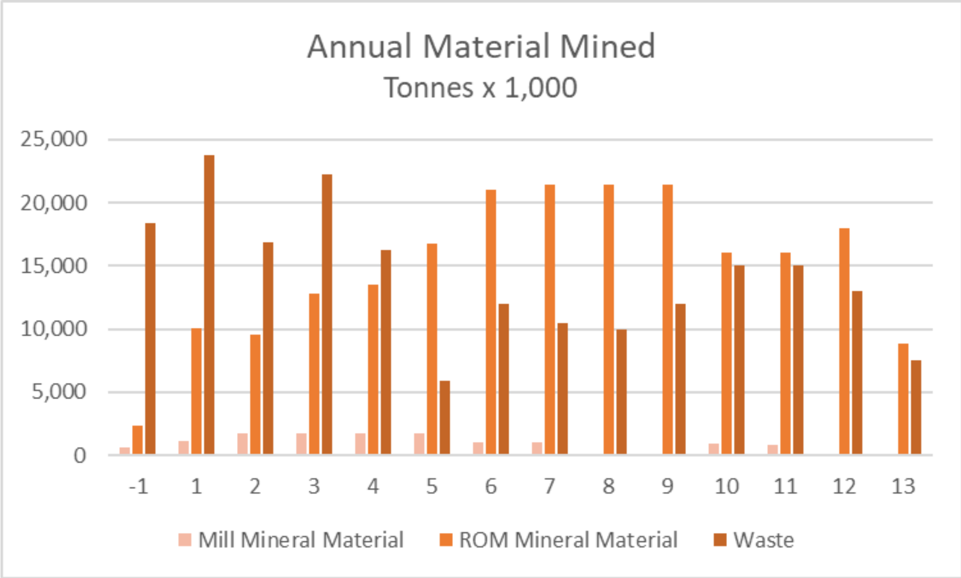


Figure 16-8 - Original Topography with Facilities Boundaries

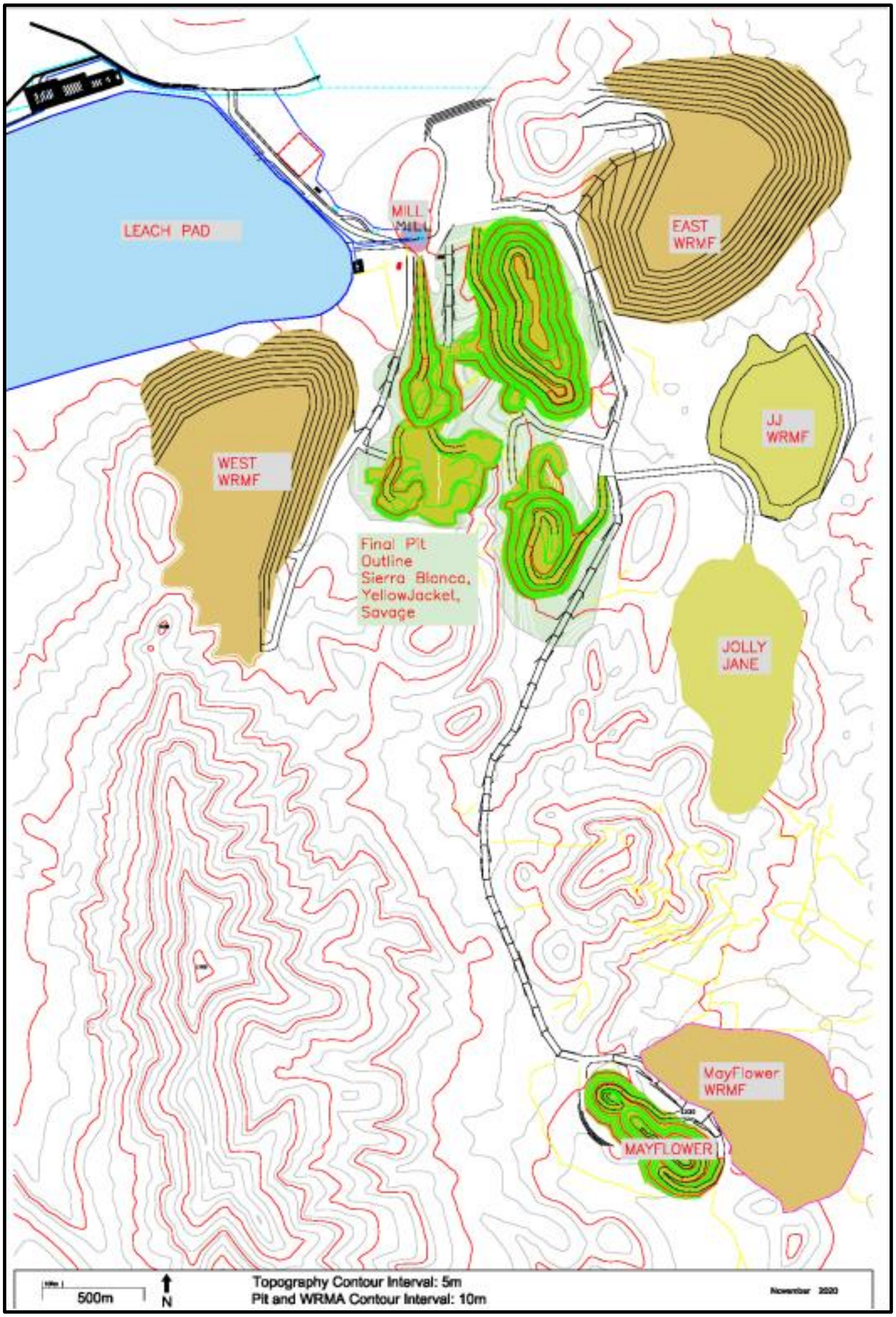


Figure 16-9 - Mine Configuration - End of Period -1

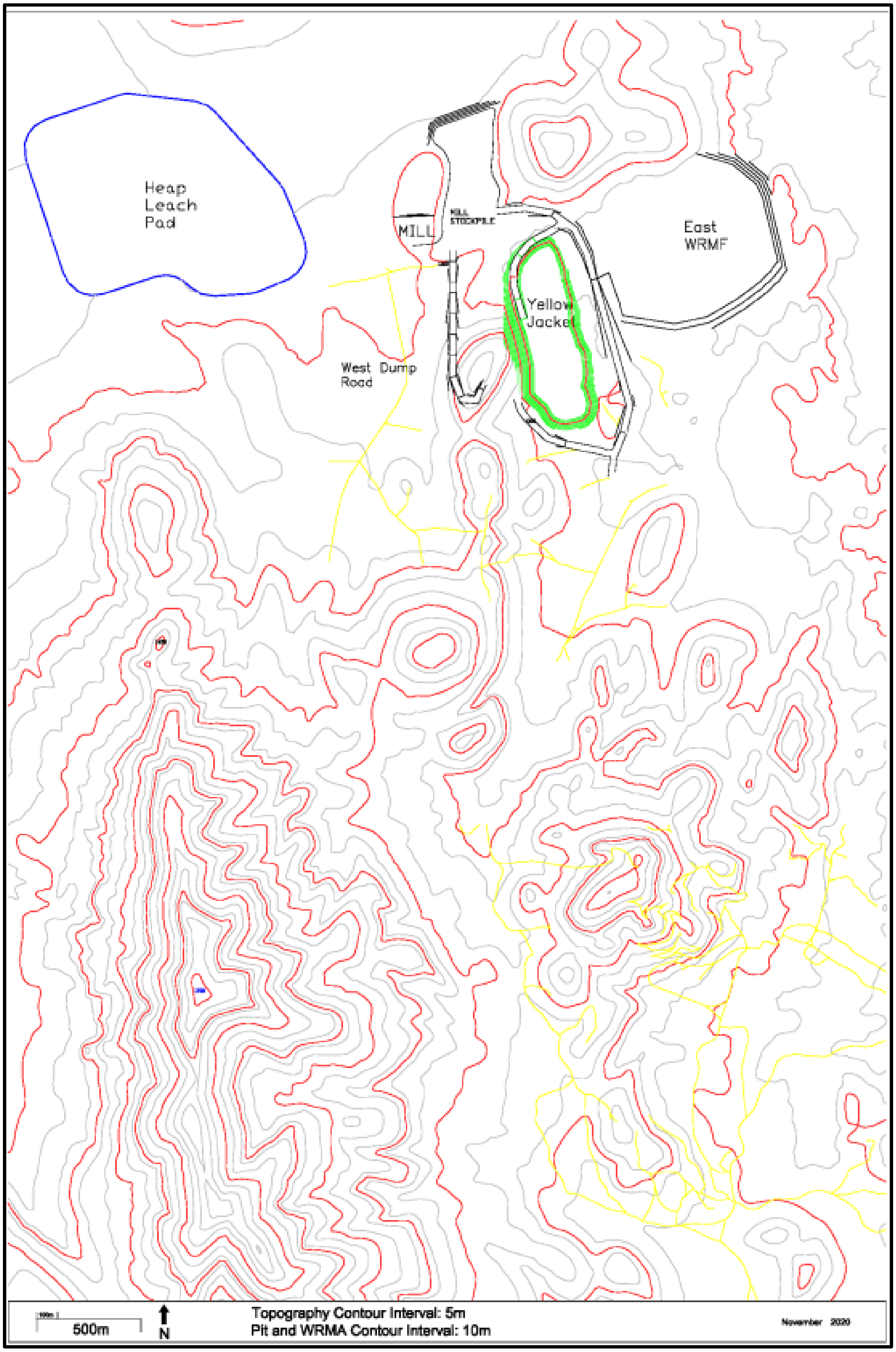


Figure 16-10 - Mine Configuration - End of Period 1

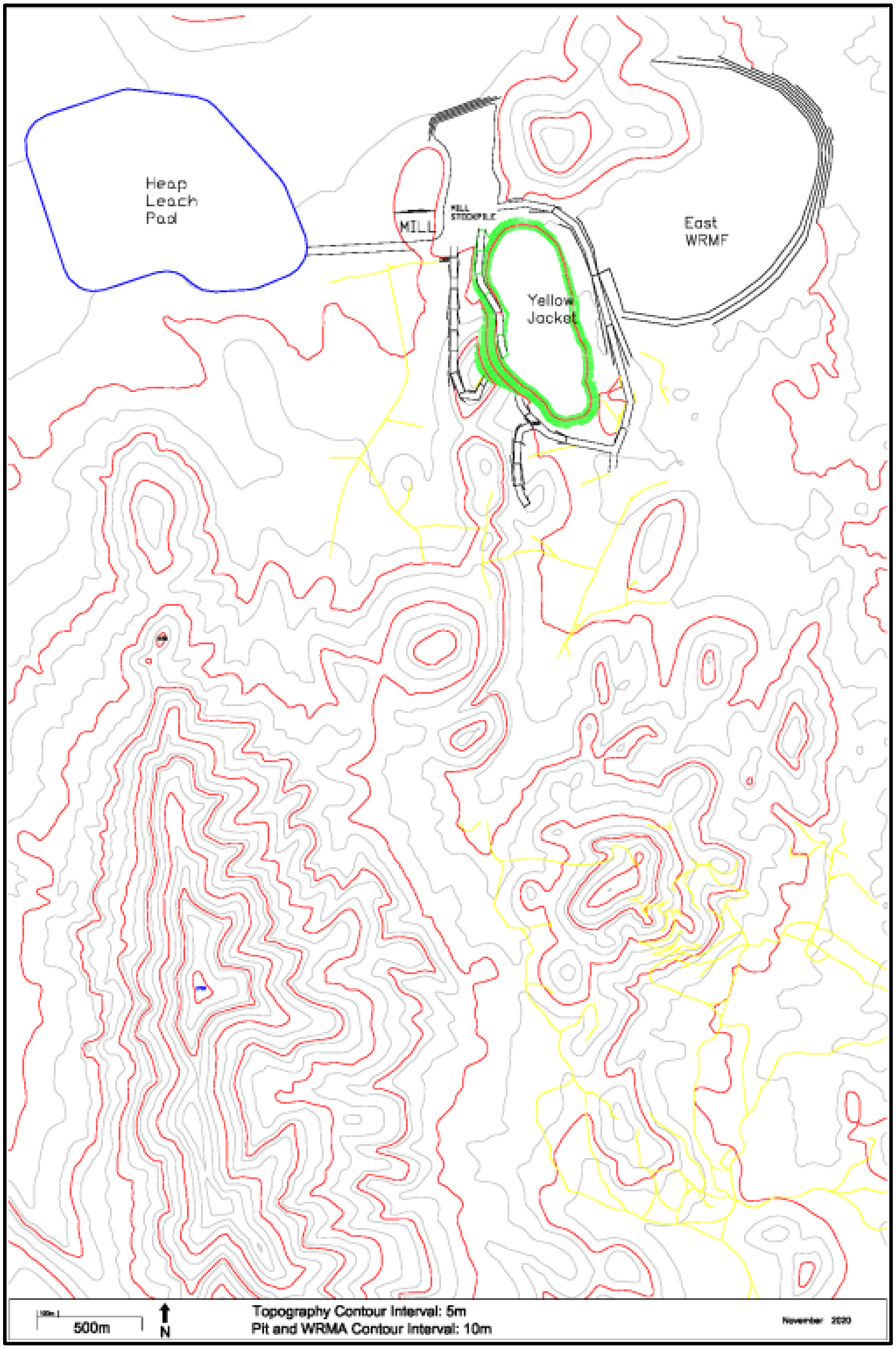


Figure 16-11 - Mine Configuration - End of Period 2

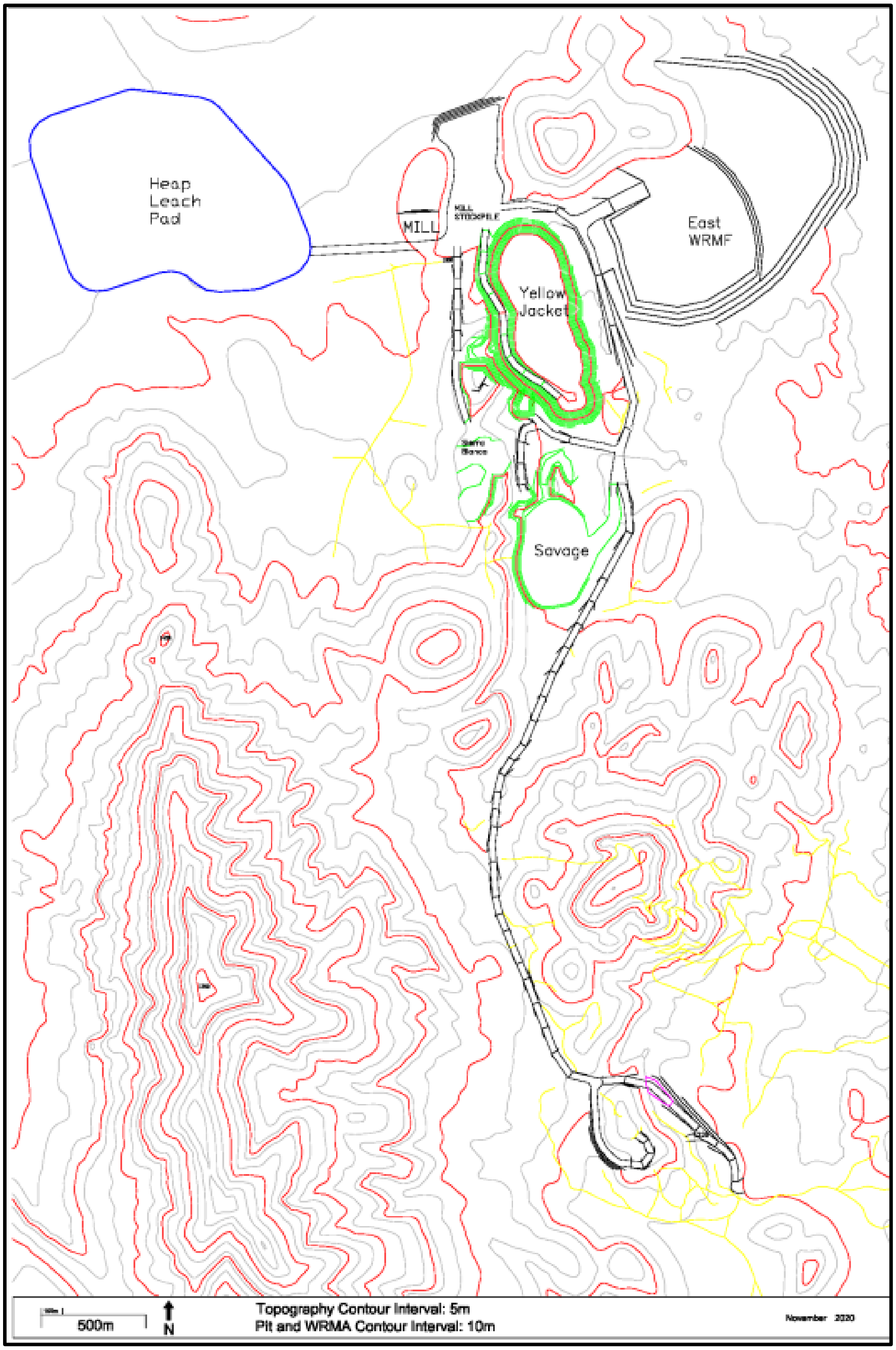


Figure 16-12 - Mine Configuration - End of Period 3

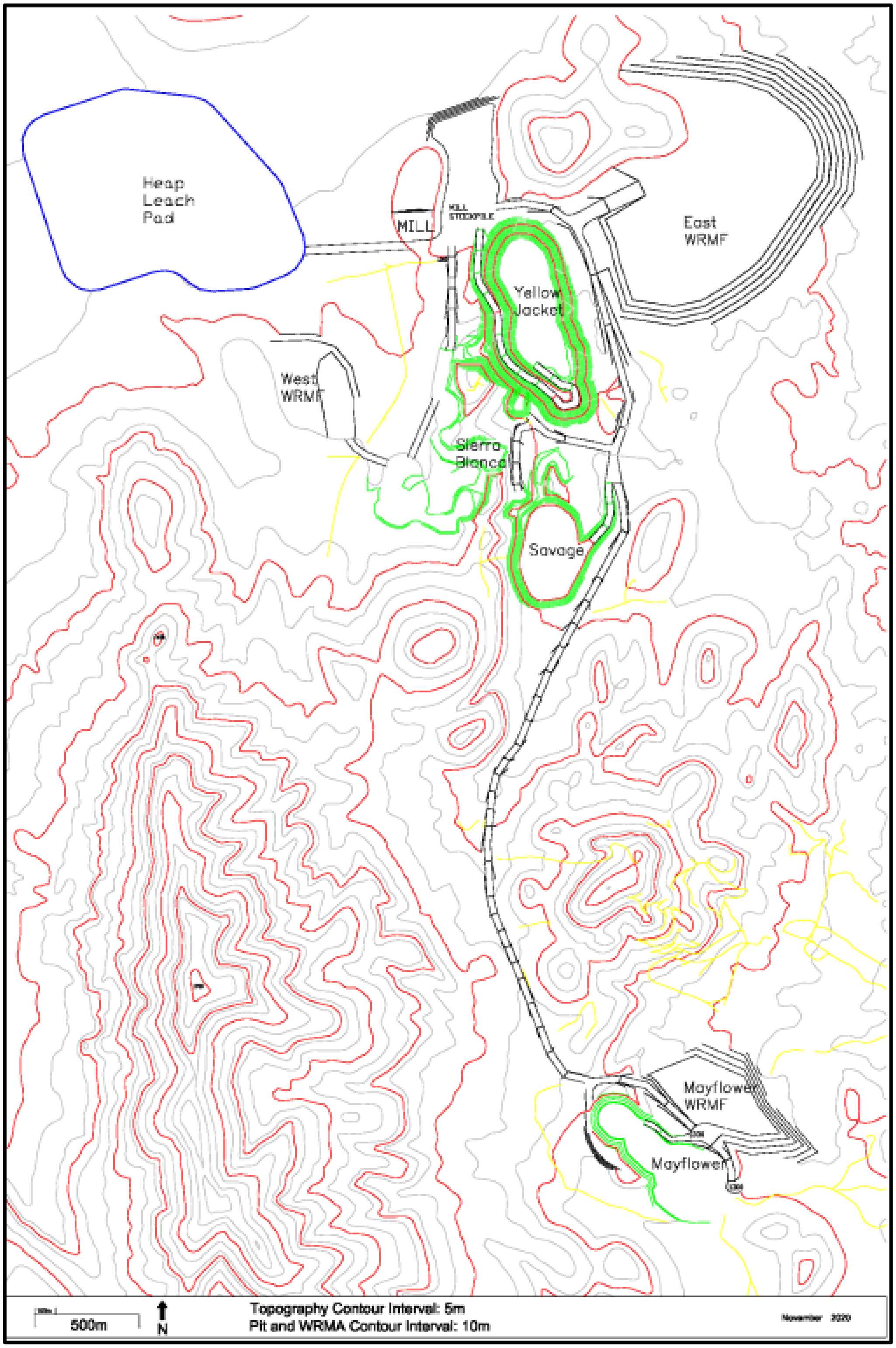


Figure 16-13 - Mine Configuration - End of Period 4

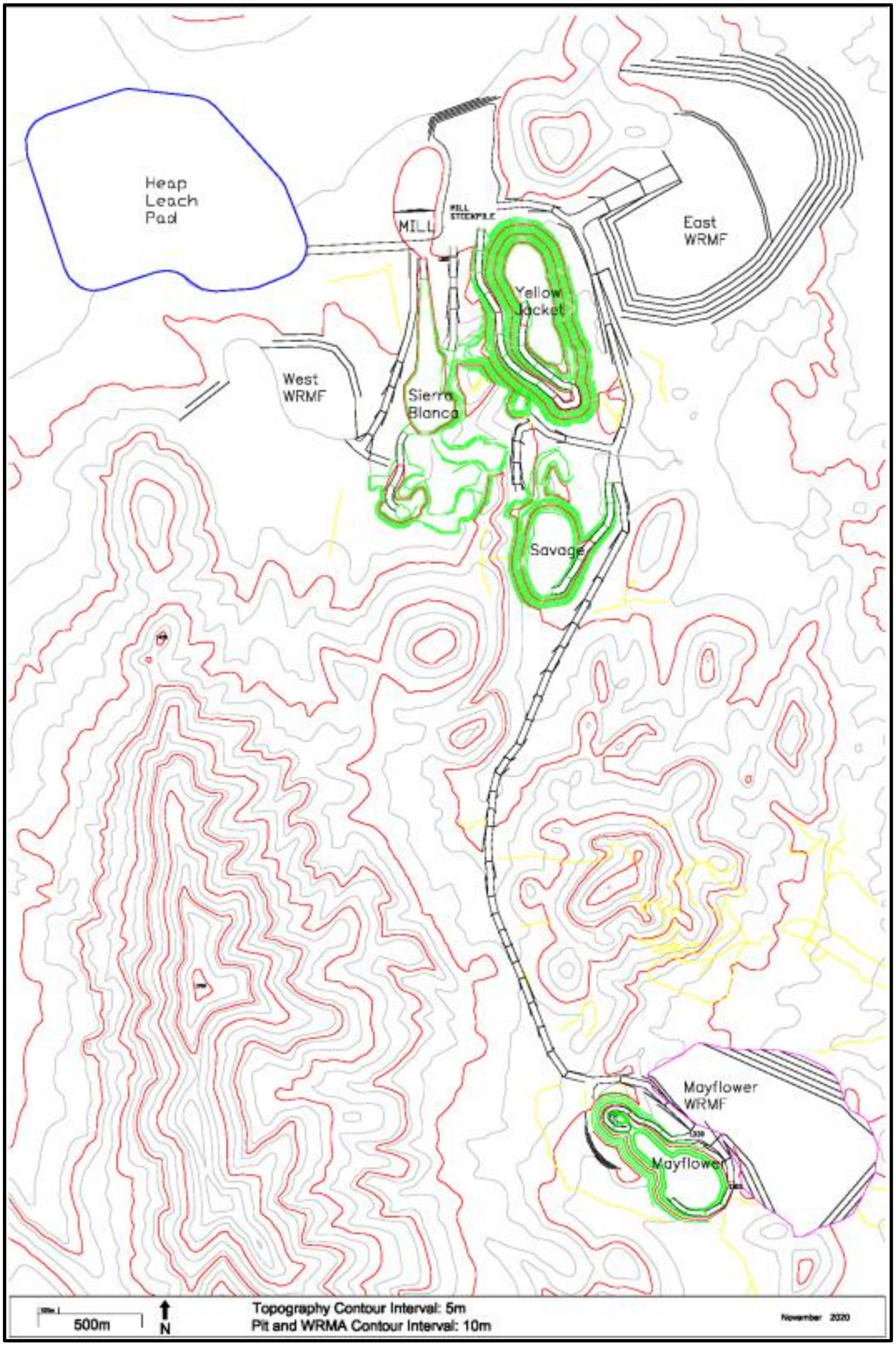


Figure 16-14 - Final Mine Configuration End of Period 5

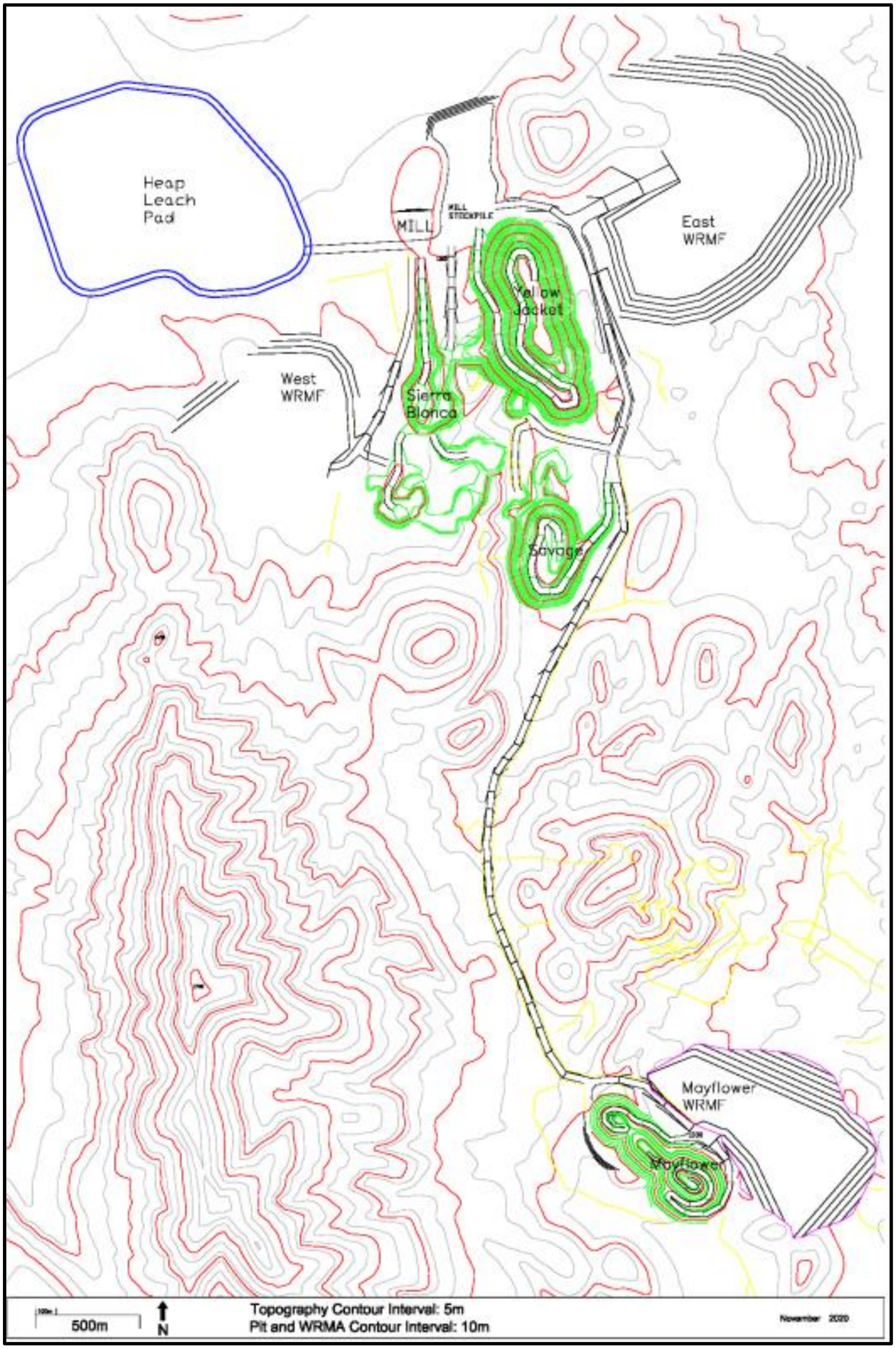
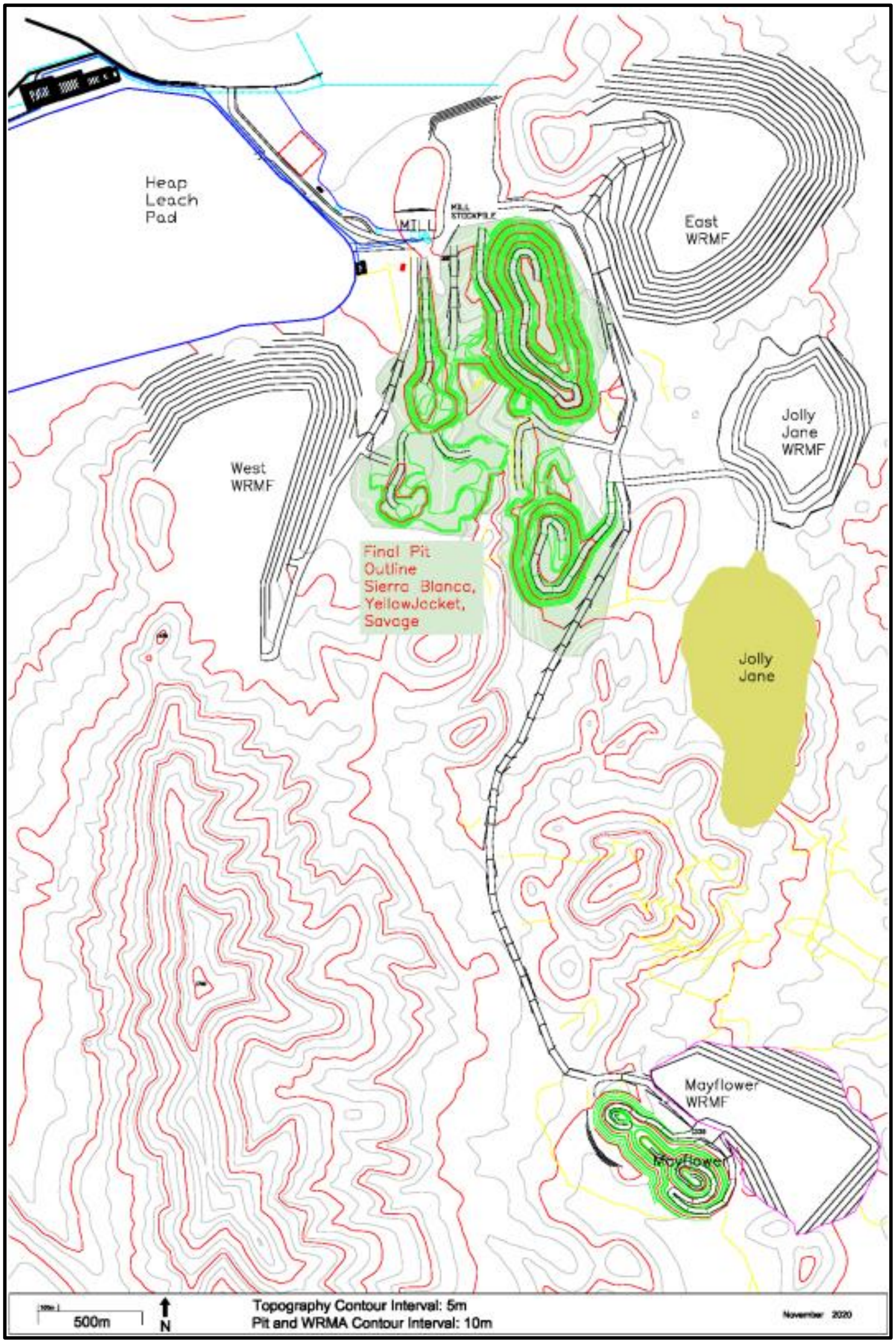


Figure 16-15 - Final Mine Configuration End of Period 13



16.6 WASTE ROCK MANAGEMENT FACILITY AND STOCKPILE DESIGN

The Waste Rock Management Facilities (WRMF) were designed to accommodate typical haul and end dump placement methods. The maximum slope angle of the reclaimed waste dumps would be limited to 3H:1V. The West WRMF was designed with a capacity of approximately 91 Mt, the East WRMF was designed with a capacity of approximately 82 Mt, the Mayflower WRMF had a design capacity of approximately 21 Mt and the Jolly Jane WRMF had a design capacity of approximately 15 Mt.

The West WRMF, Mayflower WRMF, and the Jolly Jane WRMF were planned to be reclaimed concurrently during the mining period. A 12-inch thick growth media cover would be placed over the dumps.

Mineralized Material Stockpiles

Mineralized material stockpiles would be relatively small and would be constructed at angle of repose. The mill stockpile would be a “live” pile throughout the mine life. A small ROM stockpile would be developed in period -1 and would be used to crush overliner for the subsequent Leach Pad phases. The mill resource stockpile would be placed at the crusher to feed the crushing system.

16.7 MINING FLEET AND REQUIREMENTS

16.7.1 GENERAL REQUIREMENTS AND FLEET SELECTION

The mine plan and associated mining cost were based on an owner-operated fleet. The planned equipment would include wheel loaders and rigid frame haul trucks and supporting equipment. Table 16-7 lists the peak mining fleet requirement.

Table 16-7 - Mine Production Equipment

Category	Class	Number of Units
Truck	133t	12
Water Truck	80t	2
Grader	216kW	2
Front End Loader	13m³	1
Front End Loader	20m³	2
Dozer	447 kW	3
Blasthole Drill	12.2 m	5

16.7.2 DRILLING AND BLASTING

Production drilling and blasting was planned to be owner operated. The design parameter used to define drill and blast requirements were based on a 6.75” blasthole and a 3.1m by 3.1m pattern in the mineralized material zones and a 4.3m by 4.3m pattern in the waste zones. Benches would be blasted and mined on 10m levels with 1m of sub-drill. Corvus plans to implement the Ultra-High-Intensity-Blasting (UHIB) techniques (Brent et. al., 2014) in the mineralized material zones to generate an optimum particle size. Perimeter/ Buffer rows were planned to allow for controlled blasting and minimize damage to the highwalls. The powder factor for the blasting was estimated to be 0.31 kg/t for waste and 1.24 kg/t for mineralized material.

16.7.3 LOADING AND HAULING

The primary production loading units for the NBP were based on 20 m^3 front end loaders. One smaller loader (13m^3) would be used for the mill tailings rehandle and for smaller pit benches. The hauling units were based on 133t capacity rigid frame haul trucks; the loaders would require 4 to 5 passes to load the trucks. A haulage study for years -1 through 5 was conducted to ensure that the mining fleet was sufficient to meet production targets in the mine schedule. Table 16-8 lists the equipment demand by mining period.

Table 16-8 - Equipment Demand by Mining Period

	Mining Period/ Units Required						
Equipment	**-1**	**1**	**2**	**3**	**4**	**5**	**6-13**
13m^3 Loader	1	1	1	1	1	1	1
20m^3 Loader	1	2	2	2	2	2	2
133t Haul Truck	5	9	10	12	12	12	12
Blasthole Drill	3	5	5	5	5	5	5

16.7.4 SUPPORT AND AUXILIARY EQUIPMENT

Support equipment would include three track dozers for dump and stockpile management. One road grader would service the access road and, haul roads, in Period -1 and an additional unit would be added for the remainder of the mine life. Two water trucks would provide dust control. Mobile light plants would be utilized for lighting the working areas during production in low light conditions.

16.7.5 MANPOWER

Mining personnel would be sourced locally with some skilled positions recruited from outside the local area. The personnel requirements are presented by mining period in Table 16-8.

Table 16-9 - Personnel Requirements

Area	-1	1	2	3	4	5	6-13
Drilling	12	20	20	20	20	20	20
Blasting	4	8	8	8	8	8	8
Loading	8	12	12	12	12	12	12
Hauling	20	36	40	48	48	48	48
Mine Support	29	33	33	33	33	33	33
Maintenance	19	19	19	19	19	19	19
Technical Support	13	13	13	13	13	13	13
Total	105	141	145	153	153	153	153

16.7.6 ORE CONTROL

Corvus would utilize a blasthole sampling system for ore control. Blasthole cuttings would be collected in a pie-shaped sample pan, which sits on the ground adjacent to the blasthole. No subdrill would be collected in the sample. The sample would then be bagged and identified with a unique sample number. The drillhole would then be staked and tagged with the same number. Samples would then be delivered to the on-site laboratory for cyanide solution and fire assay analysis.

Once a drill pattern is completed, and prior to blasting, the drillhole locations would be surveyed and plotted with the corresponding assay results. This data would then be used to generate ore / waste boundaries (bench maps) for layout in the field. Mill ore, ROM ore, and waste blocks would be staked after blasting with numbered lath and colored pin flags to guide mining. Movement due to blasting would be accounted for in the field by staking depending on the observation of the blast. Bench maps would be provided to the mine foreman and loader operators for each blast area. Corvus geologists would monitor mining to maintain ore and waste control for proper material routing. Samples from the waste dumps and ore stockpiles would be used to determine if adjustments to the ore control plan would be necessary.

16.8 MINE DEWATERING

16.8.1 SURFACE WATER

Surface water from precipitation would be diverted within the open pits by using berms and ditches routed to sediment basins for evaporation, infiltration or overflow.

Best management practices (BMP) would be used to limit erosion and reduce sediment in precipitation runoff from mining facilities and disturbed areas during construction, operations, and initial stages of reclamation.

16.8.2 PIT DEWATERING SYSTEM

A dewatering system may be necessary for the lower benches of the YellowJacket Pit and possibly the Sierra Blanca and Jolly Jane pits. The Mayflower pit area is not expected to encounter groundwater inflows. The modeling performed by HydroGeoLogica indicates that inflows will be approximately 105 gpm. Provisions would be made for in-pit sumps and pumps to remove any water accumulation. The water would be pumped into water trucks for dust control use.

17. RECOVERY METHODS

17.1 PROCESS DESIGN BASIS

Test work has been performed on the NBP mineralized material by a number of metallurgical laboratories, as described in Section 13. The test work has indicated that the Yellowjacket, Sierra Blanca, Mayflower, Jolly Jane and Savage Valley mineralized materials are amenable to cyanide leaching for the recovery of gold and silver. The processing plan has been divided into two facilities:

- Heap Leach/ADR: Mineralization from the Yellowjacket, Sierra Blanca, Mayflower, Jolly Jane and Savage Valley open pits would be mined and placed onto a heap leach pad. Run of Mine (ROM) mineralized material would be mined at an average rate of approximately 44,700 tonnes per day and placed on a conventional heap leach pad in nominal 9.1 m (30 ft) lifts. Through the use of ultra-high-intensity blasting, as described in Section 16, the mineralization would be expected to have a P80 of approximately 84 mm (3.3 in). Quicklime (CaO) would be added to the mineralized material prior to placement on the leach pad for pH control. Barren solution containing dilute sodium cyanide would be applied via drip emitters at an application rate of 6.1 L/h/m2 (0.0025 gpm/ft2). Pregnant solution would be collected in a divided pond where sediment would settle out of the solution before overflowing an internal berm. The pregnant solution pumps would feed a pair of vertical carbon-in-columns (VCIC). A second pair of VCIC's would be added to manage the increased barren flow rate in Year 5. Gold and silver extracted from the pregnant solution onto activated carbon would be further processed via the desorption and recovery circuit to produce a final doré product.

- Gravity Mill: High grade vein and vein stockwork mineralization from the Yellowjacket deposit would be processed through a mill circuit at a rate of approximately 4,700 tonnes per day. The mineralized material would be run through a two-stage portable crushing circuit and conveyed into a fine crushed product bin at a P80 of 19 mm (¾ in). The fine crushed product bin would feed a rod mill with a target product size of 48 mesh. Slurry classification would be via multi-stage screens. The screen undersize would feed a gravity concentrator circuit. Gravity concentrate would be processed via an intense leach reactor with the pregnant solution processed via a dedicated electrowinning cell for doré production. The gravity tails would be thickened and filtered prior to stockpiling. The filtered tails would be loaded into haul trucks and dumped on the leach pad for residual gold and silver recovery. A staggered dump pattern would be created on leach pad, then pushed with a dozer to assure mixing with the ROM mineralization.

A simplified Process Flow Diagram (PFD) for the heap leach/ADR circuit and a block flow diagram for the mill circuit are shown below in Figures 17-1 and 17-2.

Figure 17-1 - Heap Leach & ADR Simplified Process Flow Diagram

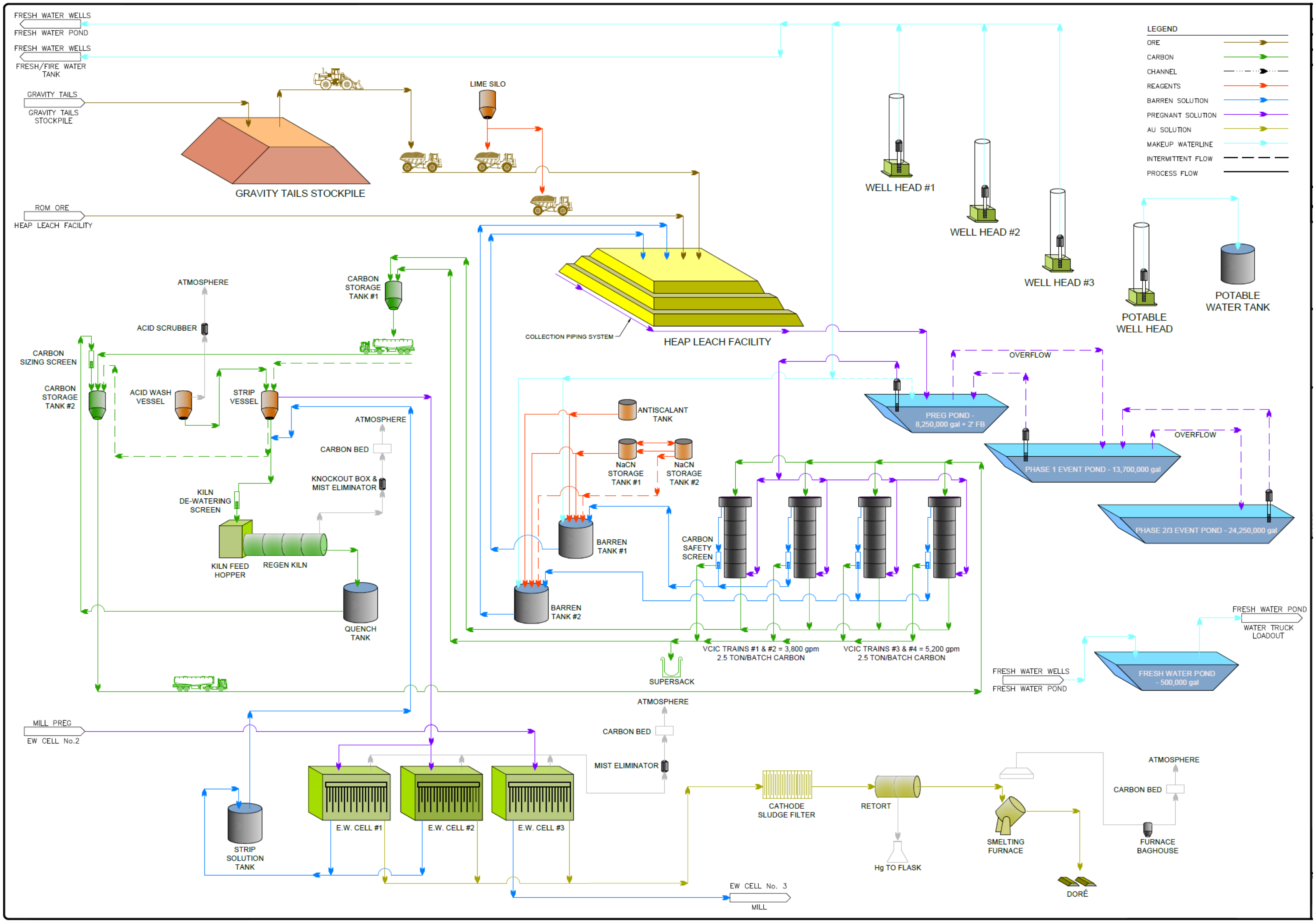
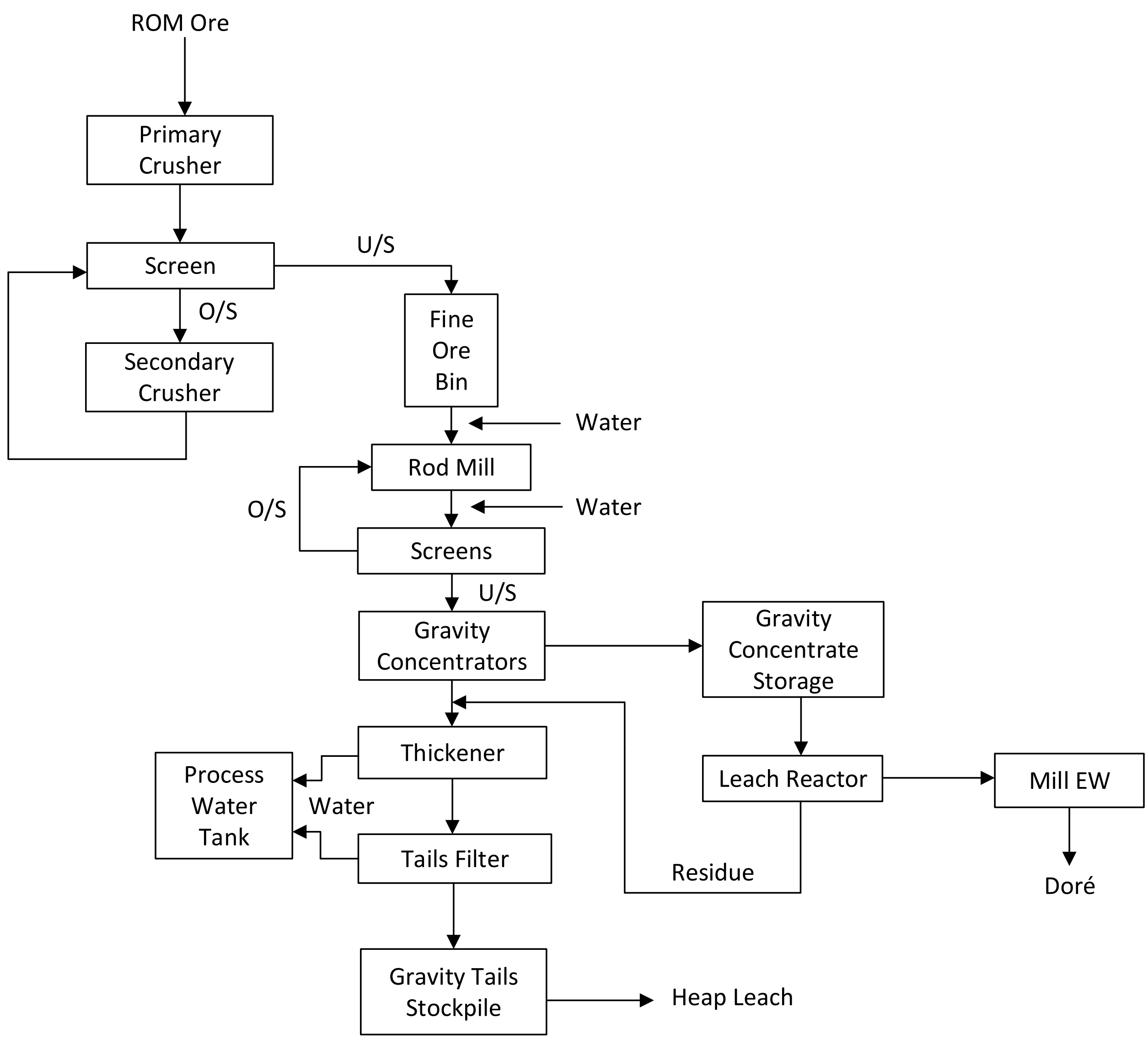


Figure 17-2 - Mill Simplified Block Flow Diagram



17.2 HEAP LEACH AND ADR

The summary process design criteria (PDC) is shown below in Table 17-1.

Table 17-1 - Process Design Criteria

Item	Design Criteria
Mineralized Material Characteristics	
ROM Mineralized Material to Leach Pad	44.7k tonnes per day
Mill Tails to Leach Pad	4.7k tonnes per day
Total Mineralized Material to Leach Pad	**47.3k tonnes per day**
LOM Mineralized Material to Leach Pad	**221.4M tonnes**
P80, mm	84 mm
Crusher Work Index (CWi), kWh/t	11.5 kWh/t
Design Bulk Density (t/m^3)	1.65 t/m^3
Head Grade	
AVG ROM Mineralized Material Grade, Au	0.20 gpt
AVG ROM Mineralized Material Grade, Ag	0.55 gpt
AVG Mill Tails Grade, Au	0.80 gpt
AVG Mill Tails Grade, Au	9.71 gpt
LOM AVG Leach Pad Grade, Au	**0.23 gpt**
LOM AVG Leach Pad Grade, Ag	**1.12 gpt**
Heap Leach Gold Recovery	
Yellow Jacket, %	73.2%
Sierra Blanca, %	73.2%
Savage Valley, %	74.1%
Jolly Jane, %	61.7%
Mayflower, %	76.4%
Mill Tails, %	71.8%
Heap Leach Silver Recovery	
Yellow Jacket, %	13.3%
Sierra Blanca, %	13.3%
Savage Valley, %	13.3%
Jolly Jane, %	6.4%
Mayflower, %	13.3%
Mill Tails, %	59.1%

Leach Pad Properties	
Phase 1A Area, m^2	630k m^2
Phase 1A Tonnes	38.5M tonnes
Phase 1B Area, m^2	342k m^2
Phase 1B Tonnes	29.6M tonnes
Phase 2 Area, m^2	953k m^2
Phase 2 Tonnes	92.9M tonnes
Phase 3 Area, m^2	605k m^2
Phase 3 Tonnes	60.5M tonnes
Barren Application Method	Drip Emitter
Solution Application Rate, L/h/m^2	6.1 L/h/m^2
Design Barren Flow Rate, Phase 1, m^3/h	949 m^3/h
Design Barren Flow Rate, Phases 2-3, m^3/h	2,249 m^3/h
Average Leach Cycle, Phase 1, Days	60 days
Average Leach Cycle, Phases 2-3, Days	75 days
ADR Plant Design	
Adsorption Method	Vertical Carbon-In-Column (VCIC)
Nominal Flow Rate, Phase 1, m^3/h	863 m^3/h
Nominal Flow Rate, Phases 2-3, m^3/h	2,044 m^3/h
Number of Trains, Phase 1	2
Number of Trains, Phases 2-3	4
Adsorption Stages per Train	6
Superficial Velocity, (m^3/h)/m^2	73 (m^3/h)/m^2
Acid Wash Method	Hydrochloric Acid
Number of Acid Wash Vessels	1
Carbon Elution Method	Modified Pressure Zadra
Number of Strip Vessels	1
Carbon Regeneration Method	Electric Kiln
Kiln Throughput, t/d	9 t/d
Precious Metal Recovery Method	Electrowinning
Number of Heap Leach EW Cells	2
Number of Mill EW Cells	1
Heap Leach/ADR Reagents	

Sodium Cyanide Solution Concentration, %	30%
Heap/ADR Cyanide Consumption, kg/t Mineralized Material	0.25 kg/t
Sodium Hydroxide Solution Concentration, %	50%
Heap/ADR Caustic Consumption, kg/t carbon	160 kg/t
Hydrochloric Acid Solution Concentration, %	35%
Heap/ADR Hydrochloric Acid Consumption, kg/t carbon	124 kg/t
Quicklime, %	> 90%
Heap/ADR Lime Consumption, kg/t Mineralized Material Consumption	1.25 kg/t
Antiscalant Consumption, kg/t Mineralized Material	0.01 kg/t
Carbon Consumption, kg/t Carbon	30 kg/t

17.2.1 HEAP LEACH PAD AND SOLUTION DISTRIBUTION

The leach pad would be designed as a double-lined facility and would consists of a layer of geosynthetic clay liner (GCL) having a hydraulic conductivity less than or equal to $1x10^{-6}$ centimeters per second (cm/s) which acts as the secondary liner system. A layer of 80-mil geosynthetic liner made from high density polyethylene (HDPE) would be placed over the GCL to act as the primary liner. A leak detection system would be installed between the GCL and the HDPE liner to alert operations to any potential leaks in the primary liner. A series of pregnant solution collection pipes would be installed in a "herring bone" arrangement to collect the solution and direct it into the pregnant solution pond. The main collection trunk lines generally would run from southeast to northwest. Overliner, consisting of crushed mineralized material to provide both liner protection and a hydraulic conductivity of at least $1x10^{-1}$ cm/s, would be screened to 100% passing 51 mm (2") and limited to a maximum of 10% passing 200 Mesh. The overliner would be placed in a 0.6 m (2 ft) thick layer over the HDPE liner and solution collection piping.

The mixed ROM mineralization and gravity mill tails would be stacked in lifts averaging 9.1 m (30 ft) high to a maximum of 91.4 m (300 ft) (an elevation of 1,355 m (4,445 ft) AMSL) at a nominal rate of 47,315 tonnes per day. The leach pad would be constructed in four phases. The first phase of the pad would have an area of approximately 630k m^2 (6.8M ft^2) with a capacity to hold 38.5M tonnes (42.4M tons) of mineralized material. Phase 1B of the pad would have an area of approximately 342k m^2 (3.5M ft^2) with a capacity of 29.6M tonnes (32.6M tons) of mineralized material. Phase 2 of the leach pad would have a lined area of approximately 953k m^2 (10.3M ft^2) and a capacity of 92.9M tonnes (102.4M tons) of mineralized material. The Phase 3 expansion would include an additional 605k m^2 (6.5M ft^2) of lined area with a capacity of 60.5M tonnes (66.7M tons)

Dilute cyanide solution would be distributed to the leach pad by three vertical turbine pumps installed in a common barren solution sump. One pump would serve as an installed spare, with two pumps operating to deliver the barren solution at a flow rate of 949 m^3/h (4,180 gpm). The barren sump would have a volume of approximately 255 m3 (67,324 gal) giving an operating residence time of approximately 15 minutes. During the Phase 2 leach pad construction, an additional barren sump and pumps would be added to deliver an additional 1,300 m^3/h (5,200 gpm) for a combined barren flow up to 2,249 m^3/h (9,900 gpm). Concentrated cyanide solution (30%) would be added directly to the barren sump, as would antiscalant and freshwater makeup.

The solution would be pumped through an 18-inch distribution pipeline around the toe of the leach pad. The solution would be conveyed to each consecutive lift via a series of 12-inch riser pipes. Seven risers would be installed for Phase 1A, with an additional four risers installed to service the Phase 1B of the leach pad. Approximately seven risers would be installed for the Phase 2 leach pad expansion, with an additional seven risers installed for the Phase 3 leach pad expansion. The 12-inch header lines would carry solution across the pad for distribution. Reusable, flexible 4-inch sub-header lines would be connected to the main headers. The barren solution would be applied to the leach pad with a cyanide concentration of approximately 250 mg/L via drip emitters connected to the sub-header lines at 24-in intervals with using an application rate of 6.1 L/h/m2 (0.0025 gpm/ft^2) to extract the recoverable gold and silver from the mineralized material. The average primary leach cycle was designed at 60-days, followed by a 7-day draindown period before placing new mineralized material. The leach solution would flow by gravity into the preg pond via the collection piping system. In Phase 2, the primary leach cycle is increased to 75-days to account for the slower leach kinetics of the mineralized material.

17.2.2 ROM TRUCK STACKING

Blasting, loading, hauling, and placement of the mineralized material on the leach pad would be by an owner mining fleet. The haul trucks would place mineralized material on the lift being constructed. A ramp would be constructed between each consecutive lift. The mineralized material would be truck stacked (end dumped) and then pushed out and cross-ripped by a dozer to eliminate any compaction that occurred during placement.

Quicklime (CaO) would be added to each truck for pH control with an estimated consumption of 1.25 kg/t, based on the metallurgical test work. Pebble quicklime with at least 90% active product would be delivered to the site in bulk trucks and stored in a 227 tonne silo which would be equipped with a variable speed feeder. This would allow for variable addition rates while maintaining cycle times for the mining fleet.

Prior to stacking mineralized material on a new lift, the ROM material would be allowed to drain down so that the irrigation piping can be reclaimed for use on future cells. Depending on the degree of compaction or solidification observed, the operation may elect to rip the surface again prior to placement of new mineralized material. Drip emitters would generally be abandoned in place and dumped over.

17.2.3 PONDS

The heap leach facility would initially include two ponds – the pregnant solution pond and the event pond. A second event pond would be added with the Phase 2 leach pad expansion and would be adequate for Phases 2 and 3. The pregnant solution pond would receive the preg solution from the leach pad as described above. Solution would enter the pond on the west side. Any entrained solids would settle out of the solution before overflowing an internal divider berm. Pumps would then transfer the clarified preg solution into the VCIC's for adsorption. The flow could also be diverted directly into the pumping side of the pond so that the settling side could periodically be bypassed, and the solids could be removed. The operating volume of the pond would be approximately 31.2k m^3 (8.25M gallons) with an additional 15.3 cm (two feet) of freeboard. This pond would consist of a 30.5 cm (12 in) prepared subbase layer with a hydraulic conductivity less than or equal to $1x10^{-5}$ cm/s, followed by two layers of 80-mil geosynthetic liner, with geonet and a leak detection system installed between the two geosynthetic layers.

The pregnant solution will be pumped from the pond into the carbon adsorption circuit using a permanent pumping system. The pumping system would consist of three submersible pumps with two operating and one standby. The pumps would feed into a common solution header before splitting between the two VCIC trains. Three additional pregnant solution pumps would be added during Phase 2 to accommodate the increased flow to the third and fourth VCIC trains

The event pond was designed as an emergency pond. This pond was sized to capture inflow from a 100-year, 24-hour storm event plus an 8-hour draindown from the leach pad. The pregnant solution pond would be connected to the event pond so that during upset conditions, any overflow would be directed into the event pond avoiding any release to the environment. A second event pond would be added during the Phase 2 leach pad construction. The Phase 1 pond would have a volume of approximately 51.7k m^3 (13.65M gallons). The event pond would consist of a layer of a 30.5 cm (12 in) prepared subbase layer with a hydraulic conductivity less than or equal to $1x10^{-5}$ cm/s and a single layer of 80-mil geosynthetic liner. The subbase for the ponds would be obtained from onsite or nearby sources. Any solution accumulated in the pond would be evacuated via a temporary submersible pump and reclaimed into the pregnant solution pond. Design and construction of the Phase 2/3 event pond would be the same as the Phase 1 pond, but the Phase 2/3 pond would have a volume of 91.8k m^3 (24.25M gallons).

The freshwater piping would be configured such that water can be added to the event pond and the process pond. This was intended to accommodate water testing of the ponds prior to startup, as well as offering additional storage volume for the initial wetting of the mineralized material on the leach pad. During operations, fresh water would generally be added to the barren solution sump.

17.2.4 CARBON ADSORPTION

Pregnant solution would flow from the heap leach pad into the pregnant solution pond. The solution would then be pumped into two parallel vertical carbon-in-column (VCIC) trains at a nominal flowrate of 863 m3/h (3,800 gpm). As noted previously, there would be two operating preg pumps and one installed spare as a standby pump. All three pumps would be piped to operate in parallel with a common discharge manifold, with flow splitting prior to entry into the VCIC's. Each VCIC feed line would have independent sampling and flow measurement to perform the mass balance around each train of carbon columns. During the Phase 2 leach pad construction, a second set of VCIC's and pregnant solution pumps would be installed to accommodate the necessary increase in flow. The second VCIC's would operate at a nominal 1,181 m3/hr (5,200 gpm) for a combined pregnant solution flow of 2,044 m3/hr (9,000 gpm).

Pregnant solution would enter the bottom of each VCIC at a nominal flowrate of 432 m^3/hr (1,900 gpm). Solution would pass through distribution plates to distribute the flow between each of the chambers. Six separate chambers would contain 2.7 tonnes (3 tons) of carbon each for gold and silver adsorption. The solution would continue to flow up through the column until it reaches the top where it would overflow the column as barren solution. Each VCIC would be approximately 2.7 m (9 ft) in diameter and 11.3 m (37 ft) tall. The VCICs added during Phase 2 would be approximately 3.2 m (10.5 ft) in diameter and 8.9 m (29.3 ft) tall. These columns would be designed to accommodate the higher pregnant solution flow rate required, but each chamber would still contain 2.7 tonnes (3 tons) of carbon.

The barren solution would be sampled prior to passing over a carbon safety screen before it entered the barren solution sump to prevent carbon from leaving the system. Cyanide, antiscalant, and make-up water would be added to the barren sump before the solution was pumped back to the leach pad via two multi-stage vertical turbine pumps. A third pump would also be installed as a spare. The barren pumps would be designed for the head requirements at the maximum leach pad elevation. Only the necessary pump impellers, installed at startup, would prevent overpressure of the system at the lower pad heights. Impellers with higher head capacity, would be installed successively as the leach pad increased in height. In Phase 2, a second barren solution sump and carbon safety screen would be installed, as would three additional barren solution pumps.

Precious metal loading onto the activated carbon in the VCIC's would be a continuous process. Solution would flow from chamber to chamber with the gold load decreasing as it passes each successive chamber. Carbon would move countercurrent to the solution flow. The design efficiency would be 98.4% for gold and 87.7% for silver. Precious metal values would increase until the design metal loading was achieved, at which time, the carbon in the first column would be removed from the circuit for further processing in the desorption circuit. Carbon would be advanced sequentially from two to one, three to two and so on, with the carbon in chamber six replaced with virgin (attritted) regenerated carbon.

17.2.5 CARBON HANDLING

Carbon movement would be countercurrent to the pregnant solution flow. Loaded carbon would be removed daily from chamber one by pumping to the VCIC area carbon storage tank. Carbon would then be advanced from chamber two to one, three to two, and so on. This process would be repeated for each VCIC, resulting in 2.7 tonnes (3 tons) of carbon from each vessel being transferred into the storage tank. A total of 5.4 tonnes (6 tons) of carbon would be transferred for further processing. Since the DR plant would be located approximately 1.8 km (1.1 miles) from the VCIC's, carbon would be transported using a specially designed truck between the facilities.

The loaded carbon would then be transferred from the truck into an identical carbon storage tank at the DR plant. The plant would be designed so that carbon could also be transferred directly from the truck into the acid wash or strip vessels, if desired.

Each new supersack of virgin activated carbon would first be attritted prior to being introduced into the adsorption circuit. The carbon would be placed into the Carbon Attrition Tank with process solution, and mechanically agitated for 20-30 minutes. This process would break-off any platelets or sharp corners of the carbon particles, which may have easily broken-off while in the adsorption column.

Regenerated (regen) and the attritted virgin carbon would be screened to remove any fines prior to filling the DR carbon storage tank. The carbon would then be delivered by truck to the VCIC carbon storage tank for distribution to each train. Carbon fines from screening would be captured in a carbon fines clarifier tank where they would be processed through a plate and frame filter press for recovery.

A single carbon advance pump is provided for each VCIC circuit and would be used for all inter-tank transfers, as well as transfer to the VCIC area carbon storage tank. A dedicated carbon transfer pump would be used to transfer loaded carbon into the truck from the storage tank, as well as to transfer regen or virgin carbon back into the VCIC's. Likewise, a dedicated carbon transfer pump would be installed at the DR carbon storage tank to transfer carbon into the tank or to the acid wash or strip vessels. The acid wash and strip vessels would each have a dedicated carbon transfer pump.

Overflow and drainage from the carbon storage tanks would be captured by area sumps and returned to the process to minimize solution or carbon losses. Solution from the VCIC area sump would be pumped to the carbon safety screens. Solution from the DR carbon handling area sump pump would be pumped to the carbon fines clarifier tank.

Carbon transfer water would be used to assist in pushing carbon through the transfer lines to prevent plugging. At the VCIC area, barren solution would be used for carbon transfers. At the DR area, a dedicated carbon transfer water tank would be used, which would receive recycled process water from the carbon fines clarifier overflow, as well as freshwater make-up as needed.

17.2.6 CARBON ACID WASHING

The loaded carbon would be introduced into the acid wash vessel. The carbon would then be rinsed in a dilute (3%) hydrochloric acid solution to remove carbonate scale by passing solution through the carbon for approximately three hours or six bed volumes. The dilute acid solution would be pumped through the carbon in an up-flow manner. After the scale was removed, the acid solution would be drained from the acid wash tank and neutralized with sodium hydroxide in a neutralization tank. The neutralized solution would be transferred to the mill water tank for re-use in the process. The solution could also periodically be transferred to the barren solution tank via the carbon truck to bleed the solution and reduce the buildup of contaminants in the mill circuit. The carbon would be rinsed with sodium hydroxide and fresh water and would then be transferred to carbon stripping. Acid washing would be performed on every batch. The total cycle duration would take approximately seven hours to complete, including the transfer to the strip vessel.

17.2.7 CARBON STRIPPING (ELUTION)

After acid washing, the loaded carbon would be stripped of the adsorbed metal species using a Pressurized ZADRA strip scheme. The ZADRA strip would utilize 10-12 bed volumes of a circulated 149°C (300°F) solution of a minimum 1.25% caustic (NaOH) and 0.2% sodium cyanide (NaCN) concentration at a minimum pressure of 380 kPa (55 psig) to strip gold and silver from the carbon. In the process, the strip solution would be chemically prepared in the strip solution tank and heated via a heater. This strip solution would be pumped in an up-flow manner through the carbon bed in the strip vessel, where it would strip the carbon of precious metals. The pregnant strip solution would exit the top of the strip vessel and be routed to the electrowinning cells, where the metal in solution is electrochemically precipitated into a sludge on the cathodes. The strip solution would then be routed back to the strip solution tank. The strip solution tank would hold about 4 bed volumes, or the volume occupied by the carbon.

The elution circuit would also include the water boiler, heat exchangers, and pumps to heat the solution prior to the strip process and cool the eluate prior to electrowinning.

The strip cycle time was estimated at 10 hours per day. The metal production profile would require approximately one strip per day to process, LOM. The facility was designed to accommodate up to two strips per day for peak production. The increased leach rate in the later phases of the project would not require an associated increase in carbon handling or strip capacity, as the grade and overall metal production do not increase during this period. At the completion of each strip, the carbon would be cooled, rinsed, and transferred to the carbon dewatering screen ahead of the kiln feed hopper.

17.2.8 CARBON REGENERATION

As carbon is utilized in the adsorption and recovery circuits, the surface and internal pore structure would become contaminated with organic species. The organics would foul the carbon, slow the gold and silver adsorption rate, and decrease the metal loading capacity of the carbon. The fouling organics would be removed by heating the carbon to 650 to 750 °C in a slightly oxidizing atmosphere and burning them from the carbon.

After passing the dewatering screen, the carbon would enter the kiln feed hopper. Excess moisture would drain down in the hopper and would be discharged. The dewatered carbon would then be fed by screw feeder into the electric rotary kiln where it would be in a steam environment. The regenerated carbon at the discharge would be quenched and passed over the carbon sizing screen prior to being placed into the DR area storage tank for return to the VCIC's. The kiln was sized to allow for 100% carbon regeneration.

Any mercury vapor contained in the kiln exhaust gases would be captured in a mercury abatement system. The majority of the mercury would be condensed with the cooled vapor passing through a sulfur-impregnated carbon bed to capture any residual mercury prior to the off-gas discharge to atmosphere.

17.2.9 ELECTROWINNING

The heap leach electrowinning (EW) cells would be designed to treat 30 to 45 gpm per cell of pregnant strip solution in two cells operated in parallel with each cell containing 34 anodes and 33 cathodes. The cells would be designed to recover gold and silver from hot sodium cyanide solutions with a pH of 10.5 or greater. Solution flow would be continuous during the strip process. A dedicated 20 anode/19 cathode EW cell would receive the pregnant solution from the gravity mill intense cyanidation reactor.

The caustic in the eluate solution would act as an electrolyte to encourage free flow of electrons and promote the precious metal transfer from solution. To keep the electrical resistance of the solution low during desorption and the electrowinning cycle, make-up caustic soda may be added to the strip solution tank. Barren solution leaving the EW cells would be pumped back to the strip solution storage tank for recycle through the elution column.

Periodically, all or part of the EW barren solution would be transferred to the barren solution tank. Approximately one-third of the barren eluate would be bled after each elution or strip cycle. Caustic and cyanide would be added as required from the reagent handling systems to the strip solution tank during strip solution make-up.

The cathodes would load with the precious metals and would be removed and processed periodically to produce the final doré product. The loaded cathodes would be pressure washed in place, removing precipitated precious metals in the form of a sludge. The resulting sludge would be pumped to a plate-and-frame filter press to remove water and the filter cake would be loaded into pans for retorting. After mercury has been removed in the retort, the dried sludge would be mixed with fluxes and smelted in an induction furnace to produce doré bullion.

Each cell's electrical power would be supplied by a dedicated local rectifier. Fumes generated during electrowinning would be removed via the electrowinning cell exhaust system which would be common for all cells. The EW cells would also utilize a common filter press to capture the sludge and remove the majority of the water prior to the retort.

17.2.10 REFINING

Gold and silver would be recovered as a soft sludge from the stainless-steel mesh cathodes in electrowinning. The sludge would be washed from the cell cathodes into the electrowinning sludge filter to remove the majority of the water. It would then be dried in a mercury retort to remove the remainder of the moisture and capture any entrained mercury. The sludge would be placed into pans with the contents heated in the retort for a minimum of six hours at 482°C (900°F) to volatilize the mercury.

The mercury vapor would be removed from the retort via a vacuum system. The vapor would pass through a water-cooled condenser, allowing the condensed mercury vapor to be collected in a trap and stored in flasks. The cooled vapor leaving the trap would pass through a sulfur-impregnated carbon bed to remove any residual mercury.

The dried sludge would be mixed with fluxes and fed into a tilting induction furnace. Once at temperature, the charge is held at temperature until the entire contents are molten. The slag would then be poured off into cast iron slag pots. The remaining contents would be poured into the bar molds, where the gold and silver doré bars would be cooled, cleaned, and stored in the vault for shipment to a third-party refiner. Slag can be periodically re-melted to recover residual precious metals, or it can be reprocessed through the mill circuit.

17.2.11 REAGENTS

The estimated average annual reagent consumption and onsite storage availability for the North Bullfrog Project are listed in Table 17-2.

Table 17-2 - Estimated Reagent Consumption and Storage

Heap Leach/ADR Reagents			
Reagent	**Form**	**Average Annual Usage**	**Onsite Storage**
Sodium Cyanide	Bulk Liquid Delivery, 30% NaCN by Weight	4,100 tonnes	113.5 m^3
Sodium Hydroxide	Bulk Liquid Delivery, 50% NaOH by Weight	520 tonnes	75.7 m^3
Hydrochloric Acid	Bulk Delivery, 35% HCl by Weight	370 tonnes	26.5 m^3
Quicklime (Pebble Lime)	Bulk Delivery, > 90% CaO	21,100 tonnes	226.8 tonnes
Antiscalant	Bulk Delivery, Liquid	160 tonnes	38.6 m^3
Activated Carbon	500 kg Supersacks	90 tonnes	16.3 tonnes
Propane	Bulk Delivery, LPG	45,200 L	7,533 L

Lime for pH control would be delivered as quicklime with at least 90% calcium oxide. The lime would be stored in a 227 tonne lime silo and delivered to the haul trucks by an automatic addition system prior to placing the mineralized material on the leach pad.

Liquid sodium cyanide (cyanide) would be received and used as a 30% solution for make-up to the barren solution tank. Sodium cyanide solution would be automatically added to the barren solution tank as the primary location for cyanide addition. It may also be added to the incoming pregnant solution ahead of the VCIC circuits depending on operational needs. Cyanide would also be used in the DR plant for the strip solution to remove the gold and silver from the carbon. Two 47.3 m^3 storage tanks would be installed at the leach pad and adsorption area with a single 19 m^3 tank installed at the Mill/DR area.

Antiscalant would be added to the barren solution tank and the strip solution tanks to minimize the effects of scale buildup. A 19 m^3 tank for bulk antiscalant storage will be available at both the adsorption area and the Mill/DR area. If specialty products are required, such as for the strip circuit, they would be added via 1 m^3 totes.

Liquid sodium hydroxide (caustic) would be delivered as a 50% solution and used in the acid neutralization and carbon strip systems. The caustic would be used to neutralize the solution after the acid wash cycle is complete. Caustic will also be added to the water in the strip solution tank to increase the pH ahead of cyanide addition. Onsite storage is via a 75.7 m^3 bulk tank.

Hydrochloric acid would be delivered as a 35% solution to a 26.5 m^3 bulk storage tank. The hydrochloric acid would be diluted to a 3% concentration prior to acid washing the carbon.

17.3 GRAVITY MILL

The high-grade mineralization from the YellowJacket vein and vein stockwork deposit would be processed through a mill circuit at a rate of approximately 4,700 tonnes per day. The process design criteria for the milling circuit is listed in Table 17-3.

The run-of-mine (ROM) YellowJacket vein and vein stockwork mineralization, having a particle size of P80 of 84 mm (3.3 in), would be trucked to a stockpile located in the vicinity of the mill. The mineralized material would be re-handled with a front-end loader into a heavy-duty horizontal vibratory grizzly feeder which would feed the primary 30 x 48 jaw crusher. The crushed product would be conveyed to a double deck (6 ft. x 20 ft.) inclined screen. The screen oversize, +25 mm (+ 1 inch) material, would be conveyed to a 300 HP standard Cone Crusher. The cone crusher product would be recycled to the double deck screen. The screen undersize (P80 of 19 mm [3/4 inch]) would be conveyed to two fine material bins. The bins would have a total of capacity of 1,600 tonnes.

The mineralization from the bins would be conveyed on a conveyor belt to the rod mill. The conveyor belt would have a magnet to remove tramp iron and a scale to measure the tonnage to the mill. The 4.6 m diameter x 6.5 m long rod mill with 2,680 HP motor will process 220 mtph of fresh mineralized material at 70% solids. The ground product would be classified on four 1.0 x 1.4 m stack sizer with 5 decks on each screen. The screen aperture would be 500 microns (28 mesh) to produce a product of P80 48 mesh. The screen oversize would be returned to the mill for further grinding.

The finished product (screen undersize) would be sent to a sump. The slurry would be pumped from the sump to two SB 5200B-SFC Falcon gravity concentrators. The tailings from these concentrators would be sent to another set of Falcon gravity concentrators. The concentrate from the four concentrators would be sent to the Sepro Leach Reactor (SLR) for gold extraction. The SLR consists of two tanks linked by a peristaltic pump. The concentrate holding tank would be equipped with load cells to track batch sizes. The SLR would use peroxide to achieve the elevated levels of dissolved oxygen required to accelerate the leach process. After the leach cycle, the pregnant solution would be drained using Sepro’s unique drainage system. The solids would be re-pulped and pumped to a thickener. The pregnant solution would be transferred to the electrowinning circuit in the ADR plant for gold recovery. The reagent consumptions are listed in Table 17-4.

The gravity circuit would recover 46.9% of the gold and 10.5% of the silver in the YellowJacket mineralization to the concentrate. The SLR recovery would be 97% of gold and 90% of the silver contained in the gravity concentrate. The overall mill recovery would therefore be 45% of the gold and 9.5% of the silver.

The tailings from the gravity circuit would also be pumped to the thickener. The thickened tailings at 55% solids would be filtered using a disk filter. The filtrate would be sent to the process water tank along with the thickener overflow. The process water would be reused for the milling plant. Only a small portion of the freshwater circulating will be required for the plant once it is operating at a steady state.

The filter cake would be dumped a pad, then be picked up with the front-end loader and put in trucks which will transport the gravity tailing to the heap leach pad. The tails would be plug dumped on the leach pad, then pushed to blend it with ROM mineralization being placed on the leach pad.

Table 17-3 - Design Criteria for Gravity Milling Circuit for Processing YellowJacket Vein and Vein Stockwork Mineralization

	Units	Nominal	Design	Source
Plant Throughput	Dry tpy	1,7000,000		Client
	Dry tpd	4722		Calculated
	Dry tph	360		Pro Solv
	Hrs./d	24		Pro Solv
CRUSHING CIRCUIT				
Feed Moisture	%	3		Pro Solv
Feed Rate	Dt/d	4722		Calculated
	t/d	4868		Calculated
Availability	%	75		Pro Solv
Feed rate	Dt/hr	263		Calculated
	t/hr	270		Calculated
F_{80}	mm/ins	75/3		Pro Solv
P_{80}	mm/ins	19/0.75		Pro Solv
MINERALIZED MATERIAL CHARACTERISTICS				
Mineralized Material Specific Gravity (in-situ)		2.42		Calculated
Dry bulk density @ 48 mesh size: Loose	g/cc	1.33		RDi

Topped Specific Gravity		1.54 2.48		RDi RDi
WORK INDICES AND ABRASION				
Crusher work index (CW_i) Bond Work Index (BW_i) Rod Mill Work Index (RMW_i) Abrasion Index (A_i)	Kwh/t Kwh/t Kwh/t g	16.68 22.2 15.9 0.4577		RDi RDi RDi RDi
ROD MILL				
Plant Availability	%	90	91	Sepro/Cemtec
F_{80}	mm/ins	19/0.75	19/0.75	Sepro/Cemtec
	dry tph	220	220	Sepro/Cemtec
Circulating Load	%		100	Sepro/Cemtec
	tph		220	Sepro/Cemtec
SCREEN				
P_{100}	Mesh	28	6 (5 duty 7 1 standby)	Sepro
P_{80}	Mesh	48	28	Sepro
Feed, Solids	%		48	Sepro
Rate	Tph		55	Sepro
Product, solids	%		440	Sepro
Rate	tph	220	38	Sepro
			220	Sepro
FALCON CONCENTRATES				
No. of units		4	4	Sepro
Size		SB5200 (48”)	SB5200 (48”)	Sepro
Feed rate	tph	220	400	Sepro
Solids	%	38	38	Sepro
Concentrate	tph	0.5	0.5	Sepro
Solids	%	10	10	Sepro
Tailing	tph	219.5	219.5	Sepro

Solids	%	30	30	Sepro
LEACH CIRCUIT				
Feed	tph	0.5	0.5	Sepro
Solids	%	10	10	Sepro
Residue time	hrs.	16	8-16	Sepro
TAILING THICKENER				
Feed rate	tph	220		
Solids	%			
Unit Area	M^2/t/d	0.001	0.008	RDi
Safety Factor	%	25	25	
Thickener	M		9.1	Calculated
	HP		3	
Thickener U/F Solids	%		55	RDi
TAILING FILTER				
Feed solids	%		55	RDi
	tph		220	RDi
Filter rate	t/m^2/hr.		5.58	RDi
Safety factor	%		25	Pro Solv
Disk Filter	M^2		46.5	Pro Solv
No. of disks			8	Pro Solv
Diameter	M		1.82	Pro Solv
Cake Moisture	%		23.5	Pro Solv

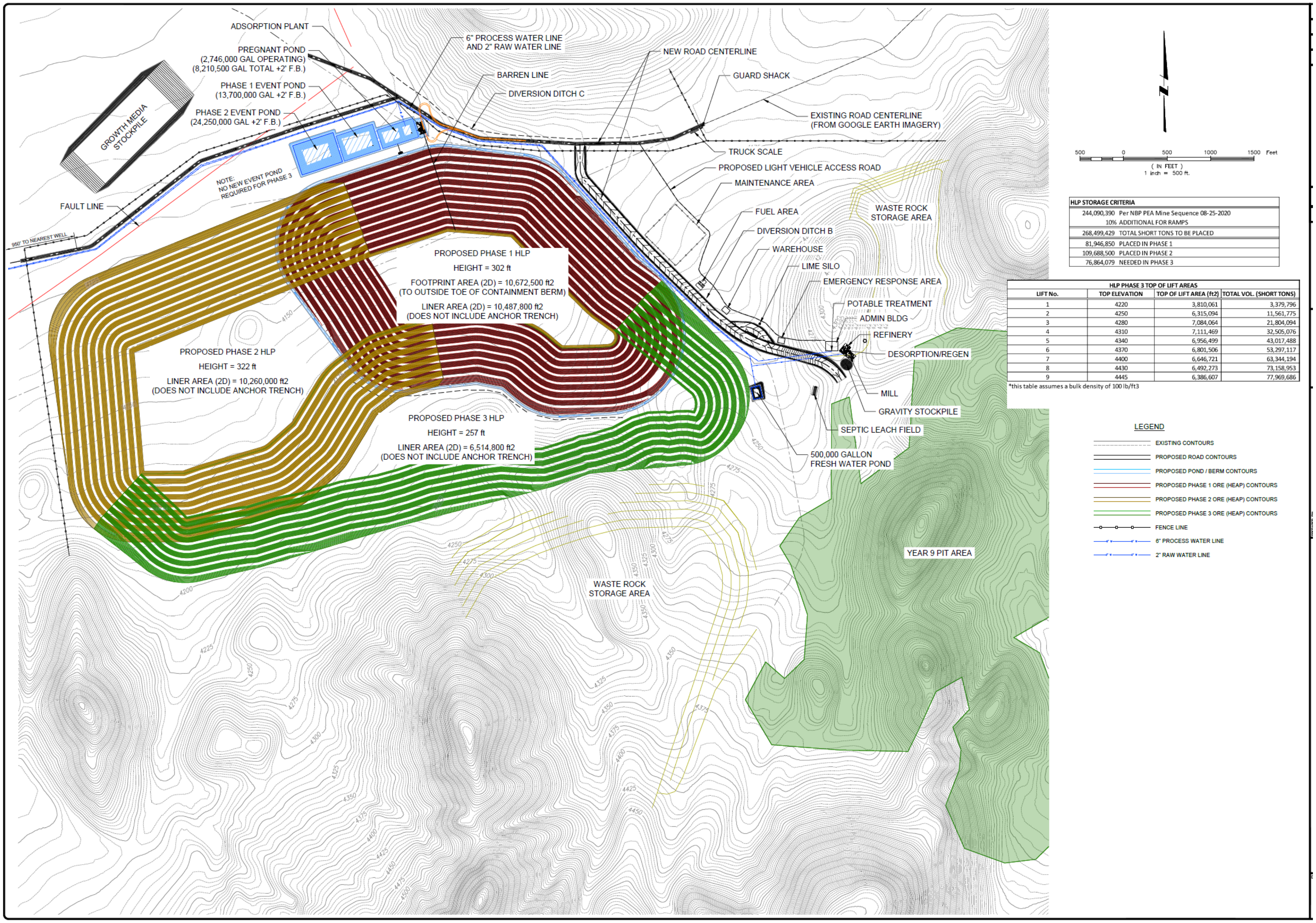
Table 17-4 - Estimated Gravity Mill Reagent Consumptions

	Reagent Consumption	
Reagent	Based on Concentrate (kg/t)	Based on Plant Feed (kg/t)
Sodium Cyanide	13.6	0.031
Flocculant	-	0.015
Lime/Sodium Hydroxide	2.9	0.007

17.4 SITE PROCESS FACILITIES LAYOUT

Figure 17-3 below shows the proposed layout of the leach pad, process facilities, and site infrastructure for the project.

Figure 17-3 - Process and Facilities Layout



HLP STORAGE CRITERIA	
244,090,390	Per NBP PEA Mine Sequence 08-25-2020
10%	ADDITIONAL FOR RAMPS
268,499,429	TOTAL SHORT TONS TO BE PLACED
81,946,850	PLACED IN PHASE 1
109,688,500	PLACED IN PHASE 2
76,864,079	NEEDED IN PHASE 3

HLP PHASE 3 TOP OF LIFT AREAS

LIFT No.	TOP ELEVATION	TOP OF LIFT AREA (ft2)	TOTAL VOL. (SHORT TONS)
1	4220	3,810,061	3,379,796
2	4250	6,315,094	11,561,775
3	4280	7,084,064	21,804,094
4	4310	7,111,469	32,505,076
5	4340	6,956,499	43,017,488
6	4370	6,801,506	53,297,117
7	4400	6,646,721	63,344,194
8	4430	6,492,273	73,158,953
9	4445	6,386,607	77,969,686

*this table assumes a bulk density of 100 lb/ft3

17.5 PROCESS LABOR

Table 17-5 below provides the breakdown of the process labor for the heap leach/ADR and mill facilities.

Table 17-5 - Process Personnel Requirements

Salaried Personnel	**Headcount**	**Area**
Process Manager	1	Mill/Heap/ADR
Chief Metallurgist	1	Mill/Heap/ADR
Plant Metallurgist	1	Mill/Heap/ADR
Process General Foreman	1	Mill/Heap/ADR
Maintenance Foreman	1	Mill/Heap/ADR
Process Foreman	4	Mill/Heap/ADR
Maintenance Planner	1	Mill/Heap/ADR
Chief Chemist / Assayer	1	Mill/Heap/ADR
Refiners	2	Mill/ADR
Salaried Personnel Subtotal	**13**	**Mill/Heap/ADR**
Shared Personnel	**Headcount**	**Area**
Control Room Operator	4	Mill/Heap/ADR
Process Mechanics	6	Mill/Heap/ADR
Process Electricians	1	Mill/Heap/ADR
Process Instrumentation Techs	1	Mill/Heap/ADR
Assay / Lab Techs	4	Mill/Heap/ADR
Shared Personnel Subtotal	**16**	**Mill/Heap/ADR**
Heap Leach / ADR Personnel	**Headcount**	**Area**
Adsorption Area Operator	4	Heap/ADR
DR Area Operator	4	Heap/ADR
Leach Operators	6	Heap/ADR
Heap/ADR Subtotal	**14**	**Heap/ADR**
Mill Personnel	**Headcount**	**Area**
Crusher/Loader Operator	4	Mill
Grinding/Gravity Operator	4	Mill
Leach Operator	4	Mill
Thickner/Filter Operator	4	Mill
Helper	4	Mill
Mill Subtotal	**20**	**Mill**
Total Salaried Personnel	**13**	
Total Hourly Personnel	**50**	
Total Headcount	**63**	

17.6 PROCESS MAKE-UP WATER

A site water balance was generated using GoldSim™ modeling software. Modeling evaluated dry, wet, and average year scenarios, as well as stochastic model runs. Normal operations are anticipated to experience a regular water deficit in the water balance due to the climate characteristics, including high evaporative losses and low precipitation. The water balance varies from year to year, but the makeup water requirements range from an average 550 gpm during a wet year to 1,000 gpm in a dry year, with an average year requirement of approximately 775 gpm. The inputs to the model, included mine usage, mill consumption, site potable consumption, and freshwater addition to the heap leach pad. The input conditions for each scenario are shown below in Table 17-6.

Table 17-6 - Water Balance Scenario Inputs

	Dry	Wet	Average	Stochastic
Climate Input	All Dry years (0.46 in/yr)	All Wet years (12.65 in/yr)	All Average years (5.10 in/yr)	Random Historical Period
Initial Mineralized Material Moisture Content, %	2	6	4	4, ± 2
Initial Mill Tailings Moisture Content, %	20	20	20	20
Residual Moisture Content, %	12	7	9.5	8, -1/+4
Drip Emitter Evap., %	5	1	3	3, ± 2
Process Pond Initial Fill Volume, gal	8,000,000	1,000,000	8,000,000	8,000,000
Event Pond Initial Fill Volume, gal	13,000,000	0	8,000,000	0

17.7 PROCESS POWER REQUIREMENT

Power usage for the process equipment and infrastructure was estimated based on the connected loads assigned to the powered equipment in the electrical load analysis. The equipment power demands for normal operations are calculated using equipment availability factors to determine the average energy usage.

The Heap Leach, ADR and Infrastructure power requirements are as follows:

- Connected Load: 4.9 MW
- Peak Demand Load: 3.5 MW
- Average Demand Load: 3.2 MW

The Crusher and Mill power requirements are as follows:

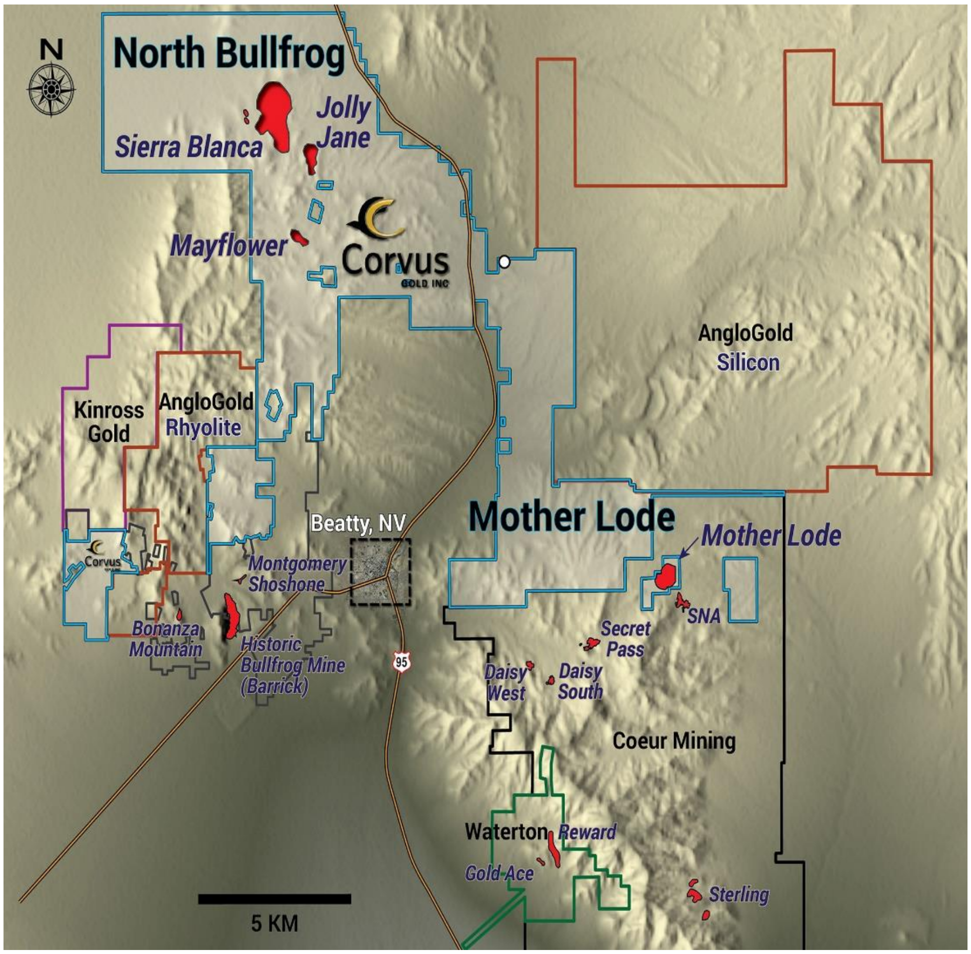
- Connected Load: 3.5 MW
- Peak Demand Load: 3.2 MW
- Average Demand Load: 2.6 MW

18. INFRASTRUCTURE

18.1 GENERAL AREA RESOURCES

Much of the primary infrastructure required to develop a surface mine is available in close proximity to the North Bullfrog Project (NBP). The availability of the key infrastructure elements is described in the following sections, addressing the project location and access, human resources, electrical power, water resources, and project infrastructure elements. The location of the project relative to Beatty, NV is shown in Figure 18-1.

Figure 18-1 - Map with Property Boundaries and Mining Resources



18.1.1 LOCATION AND ACCESS

The North Bullfrog Project is located approximately 16 kilometers (9.3 miles) north of the community of Beatty, in Nye County Nevada. The property is immediately west of U.S. Highway 95 which connects the major cities of Las Vegas and Reno. Access to the property from the highway is currently by dirt roads that are maintained and provide access for an existing commercial aggregate producer, as well as cattle grazing operations to the west. Strozzi Ranch Road is a dirt road at the north end of the property and will provide access to the major facilities, including the heap leach pad, process facilities, and administration facilities. The incoming powerline for the project will be an extension of the line currently running along Highway 95 and will run parallel to Strozzi Ranch Road.

Major mining and construction equipment sales and service are readily available throughout Nevada; however, most major mining operations are located in the northern part of the state and are serviced from the cities of Reno and Elko. Las Vegas is located approximately 193 km (125 miles) south of the project offering a major construction industry as well as heavy equipment sales and service. The town of Beatty is a small residential community with motels, restaurants and stores.

18.1.2 HUMAN RESOURCES

Human resources are available within the community of Beatty, which has a population of approximately 1,100 people, and has historically provided a substantial workforce for the Bullfrog Mine, which operated in the area between 1989 and 1998 as both an open pit and underground gold mining operation. The community has a long association with the mining industry and could contribute some experienced personnel to a mining project. The community has schools, a medical clinic, motels, fuel service, and food stores.

Pahrump, NV is located approximately 110 km (68 miles) to the southeast of Beatty. It is a larger community with a population of 36,000 people. Pahrump is a local regional center, with a hospital and emergency medical services, a college campus with technical training for industrial support and expanded service sectors. Pahrump has traditionally provided human resources for the Nevada Test Site, which had numerous high technology and underground construction projects. The Test Site is approximately 64 km (40 miles) from Pahrump, so locals are used to relatively long commutes on a daily basis. Similarly, Tonopah, NV is a regional center to the north of Beatty, with the capability to contribute to the available labor supply. Tonopah also has an extensive mining history, including providing labor for currently operating mines in the area.

18.1.3 ELECTRICAL POWER

Electrical power is available in the immediate area of the Project by Valley Electric Association (VEA), Inc., which is headquartered in Pahrump, Nevada. Current substation capacity at Beatty is projected to be sufficient to meet the needs of the project. To provide power at the NBP, a 24.9 kV line runs north from Beatty along US Highway 95. VEA has recently upgraded the main powerline, which now exceeds the projected requirements for NBP with ~15 MW capacity. Corvus's NBP requirements were considered in the upgrade of the powerline. As noted above, the Project connected load is estimated at 8.4 MW with an average demand of 5.8 MW and a peak demand at 6.7 MW.

At NBP, two existing electrical feeder lines run west from the main line, one to the perimeter of the NBP property to power an aggregate crushing plant operating in the southern portion of NBP and a second line traverses the property to power a centrally located communication station and the Company's weather station which has been installed on Corvus controlled patented mining claims near Mayflower.

The incoming powerline for the NBP main project supply will be a new 5 km (3.2 mile) extension of the line currently running along Highway 95 and will run parallel to Strozzi Ranch Road. A new primary substation will be constructed near the access road and enclosed by a security fence. A series of 3-phase, 24.9 kV, 60 Hz overhead lines will be installed for site power distribution. Two primary power distribution centers (PDC) will be installed with pad mounted transformers utilized to supply power at both medium voltage (4.16 kV) and low voltage (480/208/120 V). One PDC will be constructed at the heap leach and adsorption area with the second located at the mill, desorption, and refinery area. Pole mounted transformers will be utilized for remote applications, such as the lime silo and freshwater wells. Additional electrical upgrades would be included in the Phase 2 leach pad construction to accommodate the additional power requirements associated with the VCIC expansion and added preg and barren solution pumping.

18.1.4 WATER RESOURCES

Water resources for mining and processing at NBP must be obtained from the ground water in the Sarcobatus (Basin 146) hydrographic basin.

In 2014, Corvus purchased a 430-acre property located 48 kilometers (30 miles), along Highway 95, to the north of NBP in the Sarcobatus hydrographic basin which included a 1600-acre-foot water right. In 2018, Corvus applied for and received conversion of the water right to "mining application" by the Nevada Division of Water Resources. Historic production testing of a single well at the property indicated that a 270-foot-deep well could produce water at the permitted flowrate. In 2018, Corvus installed a well at Basin 146 in Sarcobatus Flat at a depth of 128 meters (420 feet). The well is currently servicing the exploration activities for the Project and could be utilized to supply a portion of the freshwater demand during operations. This well is also intended as the water source for construction water, though a larger pump will be required.

The northeast corner of NBP lies within the Sarcobatus hydrographic basin. The base case plan for the Project is to evaluate a water production well field in the basin near the Project and make application to the Nevada State Engineer for a temporary relocation of the production location. The Project currently assumes three production wells with an average production of approximately 68 m^3/h (300 gpm) each. A dedicated potable water well will also be installed in the same area with an average production rate of approximately 4.5 m^3/h (20 gpm).

Water wells, power, and conveyance pipelines would be developed to transport water to the mines, process facilities and ancillary structures. Electrical power would be distributed by new overhead lines fed from the project substation. Pole-mounted transformers would be installed at each well location.

For the leach area, water can be added to the ponds if required, but the primary location for makeup water is the barren solution sump. A combined fresh and fire water tank will be installed for the mill and desorption area. The mill and desorption areas will also retain as much water as possible through clarification and filtration of the carbon fines, and by thickening and filtration of the mill tails slurry to minimize the fresh water make up to the extents possible.

18.2 PROJECT SPECIFIC INFRASTRUCTURE

18.2.1 SITE AND ACCESS ROADS

Access to the NBP would be from US 95 just north of the Town of Beatty and west along the Strozzi Ranch Road. The site access would follow this existing roadway corridor for approximately 7.5 kilometers (4.6 miles) and would require improvement to allow two-lane traffic. The access to the operations area from Strozzi Ranch Road will be designed to include a road relocation of approximately 3.7 km (2.3 miles) long. This will allow a public diversion around the property so that access along Strozzi Ranch Road by the public would not be impeded. Along the main mine access portion of the road, a security building, including access gates and perimeter fencing will be installed. All persons accessing the mine property will be required to pass through the security gate. The general site fencing will be standard four-strand barbed wire range fencing.

Site access roads will also be constructed to allow light vehicle traffic and bulk deliveries access to the administration and process facilities without traveling on active haul roads. A leach pad perimeter road would also be constructed for access to the leach pad and process ponds.

18.2.2 HAUL ROADS

Leach pad loading will be by haul trucks. The trucks will travel along the east side of the leach pad, entering the pad via the access ramp. Mineralized material will be end dumped and pushed with dozers. The Mayflower deposit requires the longest haul road with an approximate haul distance of 4.5 kilometers (2.8 miles) along the western edge of the property to the leach pad location. The mine plan calls for early mining of the Sierra Blanca and YellowJacket deposits prior to mining of Savage Valley, Jolly Jane, or Mayflower.

A haul road from YellowJacket for mill mineralized material is planned to the east side of the mill facility. Trucks will initially dump directly onto a dedicated mill stockpile area. As the stockpile increases, dozers will push the material up, creating a ramp. This will allow the trucks to dump on the top of the pile while the dozer pushes the material out to create a larger stockpile. This stockpile will be immediately adjacent to the crusher to provide access for the loader feeding the mill.

18.2.3 HEAP LEACH PAD AND PONDS

The heap leach pad would be located northwest of Sierra Blanca and is shown in Figure 17-3 above with an approximate total storage capacity of 244 M tonnes of mineralized material, with approximately 221 M tonnes of material that will be placed. The design of the leach pad was described in detail in Section 17 above but will include a prepared GCL secondary liner system with an HDPE primary liner, as well as the solution collection system and overliner layer. The leach pad would be constructed in four phases with the first expansion (Phase 1B) added to the initial leach pad constructed in Year 3. The Phase 2 leach pad construction would occur in Year 5, with Phase 3 construction occurring in Year 9.

A solution management system connects the solution collection system within the leach pad, to the process pond and VCIC array. The design includes the compartmentalization of the leach pad into solution collection cells with the individual cells having flow measuring points and sampling locations. From these points, the solution will be directed to the settling side of the pregnant solution pond or to the pumping side, as necessary. Adjacent to the process pond would be an event pond and the two ponds would be connected by a spillway. An event pond for Phases 2 and 3 would be constructed during the Phase 2 leach pad construction, but the volume would be adequate to accommodate both phases of the leach pad. The pond sizing is based on a working volume for the preg pond, as well as the 100-yr, 24-hr storm event and a 24-hr drain down of the leach pad, as well as, 0.6 meters (2 ft) of freeboard. The process pond would consist of a 30.5 cm (12 in) prepared subbase layer with a hydraulic conductivity less than or equal to $1x10^{-5}$ cm/s, followed by two layers of 80-mil geosynthetic liner, with geonet and a leak detection system installed between the two geosynthetic layers. The event ponds would consist of a layer of a 30.5 cm (12 in) prepared subbase layer with a hydraulic conductivity less than or equal to $1x10^{-5}$ cm/s and a single layer of 80-mil geosynthetic liner.

18.2.4 ANCILLARY FACILITIES

The NBP would require the design and the construction of several ancillary structures to support the day to day operations of the mine. These facilities include a dedicated mine maintenance shop, wash bay, bulk fuel storage and distribution (off-road diesel), combined mine and process warehouse, administration building, emergency response vehicle bay, light vehicle fuel island (gasoline and on highway diesel), security building and potable water treatment facility. The mine maintenance shop would be located near the leach pad, immediately adjacent to the haul road.

To the extents possible, the facilities would be robust, yet cost effective, including the use of sprung structures for the warehouse and emergency response bay. The administration building and security building are intended as modular buildings. The security building (guard shack) is a 2.4 m x 6m structure located at the main security gate. A truck scale for weighing bulk deliveries will also be installed near the guard shack. The administration building is approximately 32.7 m x 42.6 m and will provide office space, conference rooms, the main IT infrastructure, and the operations change rooms for the site.

An assay and metallurgical laboratory would also be constructed onsite. The lab will provide operational support for blasthole analysis, process assays, as well as metallurgical testing for the site. The lab would be located near the administration building and the mill/desorption area.

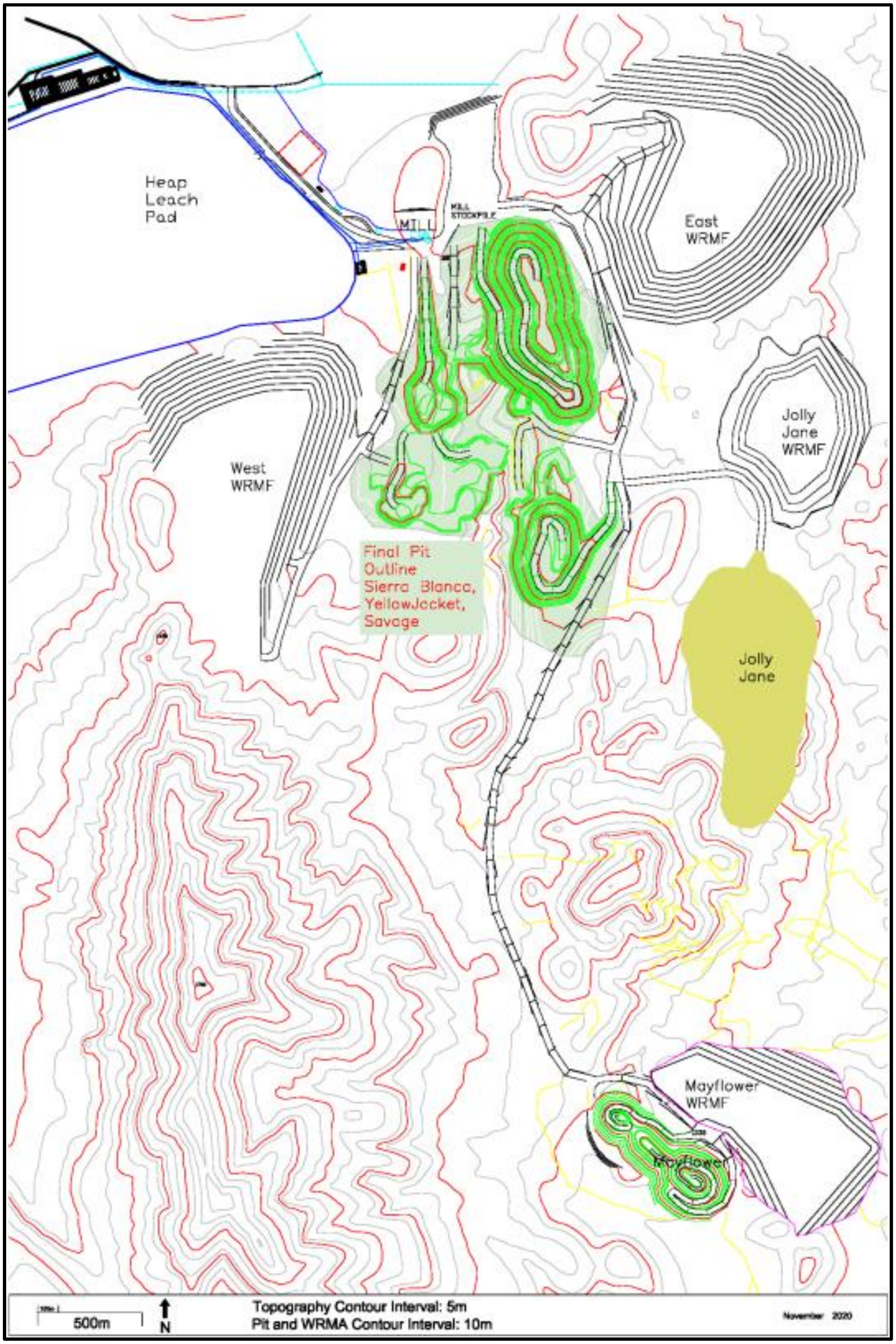
18.2.5 SURFACE WATER MANAGEMENT FACILITIES

The location of the heap leach pad, ponds, and carbon columns would require diversion structures to redirect surface runoff around these facilities and discharge into natural drainage channels. Distinct channels would be constructed for the NBP surface water management. The dimensions for each channel would vary based on the peak flow estimates, but in general, the channels would have an average depth of 0.8 m and an average width of 7 m. Each channel would also include geotextile and rip rap for erosion control. The rip rap would be placed at an average depth of 0.4 m.

18.2.6 WASTE ROCK MANAGEMENT FACILITIES

Based on the most recent mine production schedule, 207 M tonnes of waste rock would need to be stockpiled. The WRMF facilities are described in Section 16.2.2. Figure 18- 2 shows the proposed locations of the WRMFs.

Figure 18-2 - Facility Topography with Dump Locations



18.2.7 PROCESS FACILITIES

The process facilities are generally open air. Process tanks and piping are positioned over secondary containment. Containment areas for all process tanks are designed at 110% of the largest tank. The adsorption area containment is also designed to overflow back into the process pond if necessary. The refinery is the only process building and would be constructed of reinforced masonry walls. Security fencing would also be installed around the refinery entrance to allow secure access for the doré shipments.

The process ponds would be enclosed with wildlife exclusion fencing to prevent inadvertent access by area wildlife. HDPE bird balls would be installed on the surface of the process pond to discourage access by migratory waterfowl, as well as to reduce surface evaporation from the pond.

18.2.8 COMMUNICATION SYSTEMS

The site would be connected to the local phone and internet data network via an extension of existing fiber optic lines by the local utility. Phone and internet are available through Valley Communications Association, which is a division of the electric utility. A business network would be established for Corvus business services. A separate process control network would also be established to serve as the operating system for the process plant. The process control system would utilize programmable logic controllers (PLC) with a human-machine interface (HMI) system in the control room. A data historian would also be utilized for data collection and reporting. A sitewide VHF radio network would be installed for the project with multiple channel capacity. This radio network would be maintained, and additional repeaters put in place if needed.

18.2.9 FIRE WATER AND PROTECTION

A combination fresh and fire water tank would be installed with a total capacity of 1,438 m^3 (380,000 gal). The fire water reserve would account for a volume of 984 m^3 (260,000 gal) with 454 m^3 (120,000 gal) for process water distribution. A fire pump system would be installed, including a primary electric pump with a diesel-powered backup pump. A jockey pump, piping, valves and controls would also be included. The fire ring main would provide coverage through a buried piping system and hydrants to service the process facilities and ancillary buildings. The site's main IT infrastructure, electrical rooms, and process control network would be protected with a clean agent, such as FM-200 or equivalent.

18.2.10 SEWAGE AND WASTE MANAGEMENT

The site would be a zero-discharge facility, so no effluents other than the discharge from the septic system could be released to the environment. Lavatory and wash facilities would be located in the administration, laboratory, and refinery buildings. The sanitary waste from these facilities would flow by gravity to a central septic system for treatment and disposal. The septic tank and leach field would be sized for the facility occupancy. Additional portable restroom facilities would be utilized for remote operations, such as at the leach pad and adsorption areas, as well as in the mining areas. These facilities would be serviced by local provider.

Site solid waste would be managed in dumpsters or other appropriate containers. All solid waste at site would be covered to reduce the potential for blowing trash or access by wildlife. Office trash would be bagged for disposal. The project would permit and construct an NDEP approved Class III landfill on site for disposal of non-hazardous materials.

Any hazardous waste generated at site, such as that from the fire assay process, would be placed in drums, on pallets, labelled, and stored in a secure location. The pallets would be placed in an area offering secondary containment where the material would be stored until such time as it could be hauled offsite by a licensed contractor for disposal in a safe and environmentally sound manner.

19. MARKET STUDIES AND CONTRACTS

No market studies have been undertaken by the Project at this time, and no contracts have been discussed for the sale of the gold which may be produced at the Project. It is assumed that the process facilities at the North Bullfrog ADR Plant would produce a gold doré with high purity, which will be shipped to a commercial refiner such as Johnson Matthey in Salt Lake City. All-in charges from such refiners are currently in the range of US $1.50-2.50/Oz, based on a minimum one-year contract at quantity levels consistent with this Project. Sales price would be based on the spot price of gold. A gold price of $1,500 per ounce has been assumed for the NBP PEA.

Gold is readily sold on the spot market, and historically has not been a demand limited commodity. This PEA assumes that gold with be sold at spot price and this assumption is considered to be reasonable by the authors of this Technical Report.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Corvus currently has permits to conduct exploration activities at the North Bullfrog Project (NBP) with both the Nevada Division of Environmental Protection (NDEP)-Bureau of Mining Regulation and Reclamation (BMRR) and the Bureau of Land Management (BLM). Those permits allow 20 acres and 120 acres of surface disturbance on the private and public land, respectively. The permits for activities on the public lands are based on an Environmental Assessment that contains environmental baseline data on wildlife, climate and local physical characteristics. Class III cultural surveys have been conducted on the permit area.

In addition, Corvus has 2 Notice of Intent permits for exploration and baseline characterization activities in the northwest corner of NBP (Baseline Characterization) and along the eastern boundary of NBP (East Bullfrog). The two permits allow up to 5 acres of surface disturbance without the development of an Environmental Assessment document.

The permitting process is described in the following sections. Mine operating permits will be required from both NDEP-BMRR and from the BLM. An Environmental Impact Statement (EIS) will require development and will include baseline characterization data to document the existing conditions, community impacts and reclamation and closure plans and bonding requirements. This information will require more detailed project designs than have been currently developed.

20.1 EXISTING ENVIRONMENTAL BASELINE CHARACTERIZATION ACTIVITIES

Corvus has developed characterization plans which describe the on-going and future collection of baseline environmental data that will be required to support a future mine permitting process at NBP. Current baseline characterization activities include:

- Geochemical characterization of waste rock geochemistry – At NBP, Acid-Base Accounting (ABA) characterization of waste rock as defined in the current mining plans using static tests have been completed and a first phase of Humidity Cell Tests (HCT) have been completed for the waste rock associated with mining the heap leach disseminated mineralization. A preliminary report on the waste rock characterization has been completed, and the second phase of HCTs has been defined to characterize waste associated with the mining of the YellowJacket mineralization.

- Hydrologic characterization testing – At NBP, hydrologic characterization testing has been performed during installation of 12 ground water monitoring wells. Collection of water quality data began in Q4 2012 on a quarterly basis for the ground water monitoring wells and a group of 12 surface springs surrounding NBP.

NDEP Profile I parameters are reported for each ground water and spring sample. A preliminary baseline groundwater characterization report has been completed, and the plan for testing and characterization for the NBP project and for characterization of a pit lake associated with mining of the YellowJacket vein deposit has been developed.

- Surveys of plants and wildlife have been conducted on a large portion of the NBP and the Mother Lode area of the MLP, including special surveys for bats and desert tortoise. Desert tortoise range is currently limited to the eastern portion of NBP, outside of the areas containing currently defined Mineral Resources.

- A meteorological station has been in operation on the NBP site since August 2012.

These studies address some of the baseline data whose collection is time critical to production of a mining plan of operations which would serve as the basis to initiate the BLM's National Environmental Policy Act (NEPA) process (likely through the preparation of an environmental impact statement (EIS) that is required for the processing of a Plan of Operations). There will be additional baseline data collection requirements of which timing could affect the schedule for completion of the EIS. In addition, other baseline characterization activities would be required but would not control the schedule for completion of the EIS.

No known environmental issues have been identified at the NBP that would materially affect the current mine design or scope of the needed environmental permits or Corvus' ability to extract Mineral Resources. Geochemical characteristics of the waste rock suggest that no acid generation and only minor metals leaching would be expected from the waste materials associated with the heap leach mineralization. Ground water quality is typical of the regional data and drilling activities suggest minimal pit water inflow because only a small portion of mining would be below the water table.

20.2 PERMITS REQUIRED FOR FUTURE MINING ACTIVITES

This section of the Technical Report summarizes the permits that will likely be required to conduct mining activities at the NBP. The details of the mine area and activities are described elsewhere in this Technical Report. NBP would be an open pit mining operation and would have associated waste rock dumps and a heap leach processing facility for mineralized material. A gravity mill would treat the YellowJacket vein and vein stockwork mineralization, but the mill tail material would be placed on the heap leach pad for final gold recovery, and to create a zero-discharge facility.

In order to conduct mining and processing activities, NBP would need specific permits from the NDEP BMRR and the BLM. The following is a list of the major permits that will be required followed by a brief discussion of each. Except for the Water Rights, which have been obtained, none of the permits are currently in application stage.

- Plan of Operations/Nevada Reclamation Permit;
- Water Pollution Control Permit;
- Air Quality Operating Permits;
- Water Rights; and
- Industrial Artificial Pond Permit.

20.3 PLAN OF OPERATIONS/NEVADA RECLAMATION PERMIT

A Plan of Operations/Nevada Reclamation Permit (Plan) is a joint application that is submitted to the BLM and NDEP-BMRR that utilizes a format accepted by the BLM and BMRR. The application would describe the operational procedures for the construction, operation and closure of the Project. As required by the BLM and BMRR, the Plan will include a waste rock management plan, quality assurance plan, a storm water spill contingency plan, reclamation plan, a monitoring plan and an interim management plan. In addition, the Plan includes a Reclamation Cost Estimate for the closure of the Project. The mine design must be completed prior to submittal of the Plan.

20.4 WATER POLLUTION CONTROL PERMIT APPLICATION

The Water Pollution Control Permit (WPCP) application must address the open pit, waste rock dump, heap leach pad, mining activities and the water management system, as well as the potential for these facilities to degrade waters of the state. The application includes an engineering design for the waste rock dump, a waste characterization report and a modeling report for the closure of the waste rock dump, as well as an engineering design for the water management system.

A Tentative Permanent Closure Plan must also be completed and submitted to the NDEP-BMRR in conjunction with the WPCP. A Final Permanent Closure Plan will need to be developed two years prior to Project closure.

20.5 AIR QUALITY OPERATING PERMITS

An application for a Class II Air Quality Permit for those portions of the stationary source that have the potential to emit pollutants must be prepared using Bureau of Air Pollution Control (BAPC) forms. The application includes a description of the facility and a detailed emission inventory. The application also includes locations, plot plans and process flow diagrams. The application must also include a fugitive dust control plan to be used during construction and operation of the plan. If the facility will process loaded carbon or electrowinning precipitate, then a Mercury Operating Permit application and a Title V Operating Permit application will also be necessary, which will have to address the necessary state and federal mercury controls, respectively.

20.6 WATER RIGHTS

Corvus is currently permitted to withdraw 1,277 acre-feet per year of water resources through permit 65756 (SoN Land and Water LLC, a wholly owned subsidiary of Corvus) with a point of diversion at the Corvus private land parcel north of NBP and in the Basin 146 (Sarcobartus Flat). The water is permitted for Mining and Milling applications by Nevada Division of Water Resources (NDWR).

20.7 INDUSTRIAL ARTIFICAL POND PERMIT

The development of the water storage pond, which is part of the water management system, will require an Industrial Artificial Pond Permit (IAPP) from the Nevada Department of Wildlife.

20.8 MINOR PERMITS AND APPLICATIONS

In addition to the above noted permits, Table 20-1 lists other notifications or ministerial permits that will likely be necessary to conduct the mining operations.

Table 20-1 - Required Minor Permits and Applications

Notification/Permit	Agency	Comments
Mine Registry	Nevada Division of Minerals	-
Mine Opening Notification	State Inspector of Mines	-
Solid Waste Landfill	Nevada Bureau of Waste Management	-
Hazardous Waste Management Permit	Nevada Bureau of Waste Management	-
General Storm Water Permit	Nevada Bureau of Water Pollution Control	-
Hazardous Materials Permit	State Fire Marshall	-
Fire and Life Safety	State Fire Marshall	-
Explosives Permit	Bureau of Alcohol, Tobacco, Firearms and Explosives	-
Notification of Commencement of Operation	Mine Safety and Health Administration	-
Radio License	Federal Communications Commission	-
Public Water Supply Permit	NV Division of Environmental Protection	-
MSHA Identification Number and MSHA Coordination	U.S. Department of Labor Mine Safety and Health Administration (MSHA)	-
Septic Tank	NDEP – Bureau of Water Pollution Control	-

21. CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATES

The NBP overall capital cost estimate was developed by Forte Dynamics, MinerMike LLC, and Pro Solv, with input from Corvus. The estimated capital costs are considered to have an accuracy of +/- 35% overall and are discussed in greater detail in the following sections. The currency for the cost estimate is expressed in second quarter 2020 US dollars. No provision is included for potential future cost escalation.

The scope of facilities addressed in the cost estimate included the following major elements:

- Heap Leach & ADR Area
    - Heap Leach Pads & Ponds
    - Adsorption
    - Carbon Handling
    - Acid Wash
    - Elution
    - Carbon Regeneration
    - Electrowinning & Refining
    - Utilities & Reagents
- Infrastructure
    - Site Power
    - Fresh & Potable Water
    - Septic
    - Buildings & Facilities
    - Fuel Service
    - Communications System
- Gravity Mill
    - Crushing & Screening
    - Fine Crushed Product Storage
    - Grinding
    - Classification
    - Gravity Concentration
    - Leaching
    - Thickening & Filtration
    - Electrowinning & Refining

    - Utilities & Reagents
- Mining
    - Drilling
    - Blasting
    - Loading
    - Hauling
    - Mine Support

The above facilities were designed to support a nominal mining rate of 31M tonnes per year. The process production design and cost estimate were based on a nominal processing rate of 46.4k tonnes per day of mineralized material. The process plants were designed for a combined gold production of approximately 118k gold ounces per year and 264k silver ounces per year.

Table 21-1 shows the initial capital costs for the Project:

Table 21-1 - Initial Capital Costs

Area	Initial Capital Cost (USD $M)
Initial Direct Capital Cost	$137.9
EPCM	$8.0
Contingency	$18.5
Owner's Cost	$3.0
Total Initial Capital	**$167.4**

The direct initial capital costs are further subdivided in Table 21-2. The mobile equipment was assumed to financed with an initial capital cost of 20%. The remaining payments were captured as sustaining capital and interest costs, calculated as 6% per annum were transferred to operating cost.

Table 21-2 - Initial Direct Capital Costs

Area	Initial Capital Cost (USD $M)
Mill	$19.3
Heap Leach	$49.4
Mobile Equipment	$8.0
Infrastructure & Facilities	$22.2
Capitalized Mining	$38.9
Total Initial Direct Capital	**$137.9**

Indirect capital costs accounted for items such as fuel, taxes, contractor overhead, freight, EPCM services, construction CQA, third-party support for survey, vendors and commissioning, as well as initial fills and spares. The Owner's costs are also included in the indirect estimate. Table 21-3 shows the project indirect costs:

Table 21-3 - Initial Indirect Capital Cost Estimates

Area	Indirect Capital Cost (USD $M)
EPCM	$8.0
Contingency	$18.5
Owner's Cost	$3.0
Total Initial Indirect Capital	**$29.5**

Sustaining capital costs have been estimated for the Phase 1, 2 and 3 leach pad expansions, as well as the incremental mining equipment required for the project, which accounts for the majority of the sustaining capital costs. The additional EPCM costs and contingency are also included as sustaining capital. Table 21-4 below provides the breakdown of the LOM sustaining capital costs. Contingency was not applied to the mobile equipment.

Table 21-4 - Sustaining Capital Cost Estimates

Area	Sustaining Capital Cost (USD $M)
Sustaining Capital	$101.4
EPCM	$8.1
Contingency	$22.7
Total Sustaining Capital	**$132.3**

Working capital was estimated to be equivalent to the operating costs for the first three (3) months of production, or $31.3M. These costs are not included in Table 21-4. Working capital is credited out of the project capital costs at the end of production Year 2. The initial fill costs are estimated at $0.6M and are recovered by transfer to the operating costs in the last year of active operations.

21.1.1 HEAP LEACH, ADR, AND INFRASTRUCTURE COSTS

The capital costs were developed from the following sources of information:

- Design Criteria
- Process Flow Diagrams
- Mechanical Equipment List

- Electrical Equipment List
- Electrical Single Line Diagrams
- Plot Plans and General Arrangement Drawings
- Quantities and Material Takeoffs (MTO) from Preliminary Design
- Budgetary Quotes from Vendors for All Major Equipment
- In-house Historical Data from Recent Projects
- External Data Sources such as CostMine
- Mining Fleet Study.

The heap leach, ADR, and infrastructure costs were developed for each area of the facilities noted above by each relevant prime commodity account, including:

- Civil (Earthworks)
- Structural (Concrete & Steel)
- Mechanical
- Piping
- Electrical
- Instrumentation
- Buildings & Facilities

Forte generated MTOs to provide the basis for most civil and structural elements of the Phase 1 facilities. A contractor quote was obtained for crushing the overliner material which verified the Engineer's estimate. Piping MTOs were created for large-bore and long run piping, including the pregnant solution collection, barren solution distribution, and the freshwater piping from the well field. Process piping for the desorption plant was included in the vendor quote. The balance of process piping was factored. Budgetary quotes were obtained for all major mechanical equipment and most minor equipment. Where quotes were not obtained, pricing from a database of recent projects or from external data sources, such as CostMine, were utilized. Budgetary pricing was obtained for the various ancillary buildings such as the administration building and warehouse. The electrical costs were a combination of recent budgetary quotes from other projects, in-house database costs, factors and allowances. The instrumentation costs were a combination of factors and allowances, though a preliminary instrument index has been developed.

Labor was developed as a combination of labor unit rates, direct labor estimates, and allowances where applicable. Labor rates were determined as a function of unit rates and the estimators' experience. Allowances for bulk materials and construction equipment are included for each discipline.

Costs for the Phase 2 and 3 leach pad expansions, as well as the event pond and VCIC expansion were estimated based on the costs developed for the Phase 1 pad and process facility.

The cost estimate summary for the heap leach, ADR and infrastructure is shown below in Table 21-5.

Table 21-5 - Summary of Heap Leach, ADR and Infrastructure Costs

Area	Costs – Phase 1A	Costs – Phase 1B	Costs – Phase 2	Costs – Phase 3	Costs - Total
Leach Pad & Ponds	$18,519,671	$8,325,461	$28,168,000	$16,636,000	$71,649,132
Concrete	$2,412,584		$228,000		$2,640,584
Structural Steel	$1,360,745		$238,000		$1,598,745
Buildings	$3,534,870				$3,534,870
Mechanical	$9,579,951		$1,547,000		$11,126,951
Pumping & Piping	$4,693,047	$2,175,612	$225,000		$7,093,659
Electrical	$7,582,813		$1,497,000		$9,079,813
Instrumentation & Control	$1,314,760		$154,000		$1,468,760
Contractor Mob/Demob	$587,926	$416,273			$1,004,199
Mobile Equipment	$3,188,185				$3,188,185
Subtotal Direct Costs	**$52,774,552**	**$10,917,347**	**$32,057,000**	**$16,636,000**	**$112,384,899**
Indirect Costs:					
Indirect Costs	$20,782,902	$4,512,976	$6,411,400	$3,327,200	$35,034,478
Initial Fills	$600,000				$600,000
Owner's Costs	$3,000,000				$3,000,000
Subtotal Indirect Costs	**$24,382,902**	**$4,512,976**	**$6,411,400**	**$3,327,200**	**$38,634,478**
Subtotal Constructed Cost	**$77,157,454**	**$15,430,323**	**$38,468,400**	**$19,963,200**	**$151,019,377**
Contingency	$14,831,491	$3,086,065	$12,925,382	$6,707,635	$37,550,573
Total Project Cost	**$91,988,945**	**$18,516,387**	**$51,393,782**	**$26,670,835**	**$188,569,949**

The average overall contingency included in the Initial Capex for the heap leach pad, ADR plant and infrastructure is applied at 20% of the constructed cost, excluding Owner's Costs. The contingency is lower for areas having a higher confidence level, such as the leach pad and higher for areas having less design detail at this stage of the project. The average contingency for latter phases of the project (Phases 1, 2, and 3) is 28%, which consists of 20% contingency for Phase 1 and 30% for Phases 2 and 3. This results in an overall contingency of 25% for this scope at this stage of the project.

21.1.2 GRAVITY MILL

Pro Solv developed the capital cost for the crushing and milling circuit. The cost was factored based on a budgetary quote for the major mechanical equipment by Sepro. Indirect costs are included at 30% of the direct cost and a contingency of 20% was included in the estimate. The mill estimate is shown below in Table 21-6.

Table 21-6 - Summary of Mill Costs

Area	Cost (000s)
Direct Costs	$16,499
Indirect Costs	$4,950
Contingency	$4,290
Total Mill Cost	**$25,739**

21.1.3 MINING EQUIPMENT

Mining Equipment required for the NBP was developed from the mine plan tonnage requirements. Loaders and haul truck requirements were developed based on haul profiles developed for each mining area tonnage for each mining period over the mine life. Drill requirements were based on the mine plan tonnage and the drill pattern specified for mineralized material and waste. The capital cost unit rates were based on CostMine 2019 data. Table 21-7 lists the equipment requirement by mining period and Table 21-8 lists the full capital cost requirement by mining period. Corvus plans to finance the mining equipment with 20% down and carry payments for 5 years at a 6% interest rate. The yearly payments are captured in the operating costs.

Table 21-7 - Mine Equipment Units by Mining Period

Mining Equipment Requirement		Units Required by Mining Period					
Equipment List	Equipment Class	-1	1	2	3	4	5-13
Wheel Loaders	20M^3 Bucket	1	2	2	2	2	2
Wheel Loaders	13M^3 Bucket	1	1	1	1	1	1
Haul Trucks	133t	5	9	10	12	12	12
Drills	Rotay crawler, 12.2M	3	5	5	5	5	5
Dozers	447kW, 154,700 #	2	3	3	3	3	3
Motor Graders	216 kW, 16' b;ade	1	2	2	2	2	2
Water Trucks	80t/5000g	1	2	2	2	2	2
Skid steer	3650#	4	4	4	4	4	4
Bulk Anfo truck	Heavy ANFO Blend	2	4	4	4	4	4
Light Plants		6	8	8	8	8	8
Light Duty Pickups	3/4 ton	12	12	12	12	12	12
Field Service Truck	costmine spec	1	1	1	1	1	1
Field Lube Truck	costmine spec	1	2	2	2	2	2
Off-Road Tire Truck	costmine spec	1	1	1	1	1	1

Table 21-8 - Mine Capital Costs by Mining Period

Mining Equipment Capital Costs		From COSTMINE 2019	Capital Costs Required by Mining Period ($'s x 1,000)						
Equipment List	Equipment Class	Unit Price							
			-1	1	2	3	4	5	6
Wheel Loaders	20M^3 Bucket	$5,723,900	$5,724	$5,724	$0	$0	$0	$0	$0
Wheel Loaders	13M^3 Bucket	$5,046,000	$5,046	$0	$0	$0	$0	$0	$0
Haul Trucks	133t	$2,500,800	$12,504	$10,003	$2,501	$5,002	$0	$0	$0
Drills	Rotay crawler, 12.2M	$2,180,000	$6,540	$4,360	$0	$0	$0	$0	$0
Dozers	447kW, 154,700 #	$1,674,900	$3,350	$1,675	$0	$0	$0	$0	$0
Motor Graders	216 kW, 16' b;ade	$848,200	$848	$848	$0	$0	$0	$0	$0
Water Trucks	80t/5000g	$1,010,375	$1,010	$1,010	$0	$0	$0	$0	$0
Skid steer	3650#	$79,800	$319	$0	$0	$0	$0	$0	$0
Bulk Anfo truck	Heavy ANFO Blend	$288,100	$576	$576	$0	$0	$0	$0	$0
Light Plants	0	$15,000	$90	$30	$0	$0	$0	$0	$0
Light Duty Pickups	3/4 ton	$55,000	$660	$0	$0	$0	$0	$0	$0
Field Service Truck	costmine spec	$75,300	$75	$0	$0	$0	$0	$0	$0
Field Lube Truck	costmine spec	$89,800	$90	$90	$0	$0	$0	$0	$0
Off-Road Tire Truck	costmine spec	$177,500	$178	$0	$0	$0	$0	$0	$0
Loader Bucket	20M^3 Bucket	$250,000	$0	$0	$0	$0	$250	$250	$0
Truck Beds	133t	$225,000	$0	$0	$0	$0	$0	$1,350	$900
Dozers Blade	447kW, 154,700 #	$90,000	$0	$0	$0	$0	$90	$180	$0
Light Vehicles	10	$55,000	$0	$0	$0	$0	$220	$330	$0
Total			$37,010	$24,317	$2,501	$5,002	$560	$2,110	$900

21.1.4 MINING INFRASTRUCTURE

Mining infrastructure required for the NBP includes a small maintenance shop, fuel depot and distribution system, equipment wash, shop tools and site communications. The basis for the maintenance shop was CostMine 2019 and costs are listed in Table 21-9, all other items are allowances.

Table 21-9 Mine Infrastructure Costs

		Mine Infrastructure Costs by Period $ x 1,000)					
Assumptions on Mine Infrastructure Costs	Unit Cost	-1	1	2	3	4	5
Mine Shop 40'x50' (Costmine)	$436,000	$436					
Truck Wash Bay w/ recycle	$50,000	$50					
Fuel Storage and Distribution (20K gallons)	$45,000	$45					
Tools	$75,000	$75					
Site Mine Communications	$75,000	$75					
Total		$681	$0	$0	$0	$0	$0

21.2 OPERATING COST ESTIMATES

Total operating cost estimates for the NBP are presented in Table 21-10. The unit operating costs are based on total mined material of 433.4 M tonnes, of which 221.2 M tonnes is mineralized material and 212.0 M tonnes is waste. The estimated mine life is thirteen years plus one year of pre-mining/construction. Processing will continue into Years 14 and 15. The mining costs include waste, mineralized material, stockpile re-handle, dewatering costs, and the interest on the mobile equipment. Operating costs include mill and heap leach processing, as well as the transportation and refining costs. General & Administrative (G&A) costs include all administration and overhead costs, as well as the reclamation costs. The unit cost for mineralized material processed is based on the total tons, including the mill tails reprocessing. The basis for these rates is 232.1 M tonnes.

Table 21-10 Operating Cost Summary

Operating Costs	($M)	$/t-Mineralized Material Processed
Mining	$750.8	$3.23
Processing	$376.4	$1.62
G&A	$102.6	$0.44
Total Operating	**$1,229.7**	**$5.30**

21.2.1 BASIS FOR OPERATING COST ESTIMATES

Mining costs are based on an owner operated fleet. The mining costs were built up using a combination of first principles and factors from CostMine 2019.

Processing labor costs were developed from the anticipated manpower staffing plan. Labor rates were applied with consideration for burden and overtime. The specific rates were based on the estimator's judgment from experience at similar Nevada operations. Power was estimated based on the current utility rate of $0.0972/kWh and the power consumption estimates from the electrical load list. Variable costs for reagents were applied based on laboratory test work, factors, and the estimator's experience. Budgetary quotes for all major reagents were obtained from primary suppliers in the area. Fixed cost elements were derived from factoring and from previous operating budgets at similar facilities.

The supervisory and administrative support staff was sized to accommodate direct front line supervision for operations, as well as provide adequate support personnel for technical services, management, environmental, and administration.

21.2.2 MINING COST ESTIMATES

The mining cost estimate is based on the mine plan tonnage for each mining period. Haul simulations were developed for each area in each mining period and used to calculate haul truck requirements. Detailed operating costs were developed for Year -1 through Year 5. The 5-year average LOM mining cost per tonne ($1.69) was applied to the mined tonnage for Years 6 through 13.

The Table 21-11 shows the cost summary by area through Year 5 and Table 21-12 shows the unit cost by area through Year 5, along with the LOM average.

Table 21-11 - Mining Cost Summary by Area

Mine Cost Summary $'s (x 1,000)						
	-1	1	2	3	4	5
Drilling	$ 5,132	$ 8,748	$ 7,521	$ 9,193	$ 8,255	$ 5,075
Blasting	$ 8,108	$ 12,974	$ 10,788	$ 13,768	$ 12,096	$ 8,005
Loading	$ 3,460	$ 6,423	$ 6,423	$ 6,423	$ 6,423	$ 6,423
Hauling	$ 7,165	$ 12,670	$ 14,197	$ 18,835	$ 17,065	$ 12,905
Mine Support	$ 6,285	$ 7,363	$ 7,363	$ 7,363	$ 7,363	$ 7,363
Maintenance	$ 2,801	$ 2,801	$ 2,801	$ 2,801	$ 2,801	$ 2,801
Technical Services	$ 1,848	$ 2,154	$ 2,017	$ 2,204	$ 2,099	$ 1,841
Total	**$ 34,799**	**$ 53,134**	**$ 51,110**	**$ 60,587**	**$ 56,102**	**$ 44,413**

Table 21-12 - Mining Unit Cost by Area

Unit Cost Summary $'s/ tonne							
	-1	1	2	3	4	5	LOM Avg
Drilling	0.24	0.24	0.25	0.23	0.24	0.24	0.24
Blasting	0.38	0.35	0.36	0.35	0.36	0.38	0.36
Loading	0.16	0.18	0.22	0.16	0.19	0.31	0.20
Hauling	0.34	0.35	0.48	0.48	0.50	0.61	0.46
Mine Support	0.29	0.20	0.25	0.19	0.22	0.35	0.25
Maintenance	0.13	0.08	0.09	0.07	0.08	0.13	0.10
Technical Services	0.09	0.06	0.07	0.06	0.06	0.09	0.07
Total	1.63	1.45	1.72	1.55	1.66	2.11	1.69

Drilling

Drilling costs are based on drilling 10m benches on a 4.3m by 4.3m pattern for waste and a 3.1m by 3.1m pattern in mineralized material. A 1m subdrill was assumed in both mineralized material and waste. A drill penetration rate of 55m per hour, the drill pattern and the mine scheduled tonnage for mineralized material and waste was used to determine equipment hours required. Table 21-13 shows the drilling cost breakdown through period 5.

Table 21-13 - Drilling Cost

Drilling Cost ($s x 1,000)								
	Unit Cost	Units	-1	1	2	3	4	5
Equipment Count			3	5	5	5	5	3
Hours Operating			15,601	26,796	21,767	28,622	24,776	15,365
Fuel	$80	gal/hr	$2,496	$4,287	$3,483	$4,579	$3,964	$2,458
Operators	$80,000	Base $/yr	$1,326	$2,209	$2,209	$2,209	$2,209	$1,326
								$0
Maintenance	$49	$/hr	$764	$1,313	$1,067	$1,402	$1,214	$753
Consumables	$35	$/hr	$546	$938	$762	$1,002	$867	$538
								$0
Total Drilling		$	$5,132	$8,748	$7,521	$9,193	$8,255	$5,075
Unit Cost Drilling		$/tonne	$0.24	$0.24	$0.25	$0.23	$0.24	$0.24

Blasting

Blasting costs are based on utilizing the Ultra-High-Intensity Blasting (UHIB) technique for mineralized material and conventional blasting techniques for waste. Costs are based on crews working day shift only. The powder factors were assumed to be 0.31 kg/t for waste and 1.24 kg/t for mineralized material. Table 21-14 shows the blasting cost breakdown through period 5.

Table 21-14 - Blasting Cost

Blasting Cost ($s x 1,000)								
	Unit Cost	Units	-1	1	2	3	4	5
Equipment Needed	4	bulk truck & skid steer	4	4	4	4	4	4
Blasting Crew Headcount	8	2 crews, days only	8	8	8	8	8	8
Hours Operating	8	hr/day	11,520	11,520	11,520	11,520	11,520	11,520
Fuel	3	gal/hr	$69	$69	$69	$69	$69	$69
Operators	$76,250	Avg. Base $/yr	$842	$842	$842	$842	$842	$842
ANFO Required, tons			9,857	16,931	13,753	18,084	15,654	9,708
ANFO	$460	$/ton	$4,534	$7,788	$6,327	$8,319	$7,201	$4,466
Fuel for ANFO	0.06	% FO in ANFO	$341	$585	$475	$625	$541	$336
Blastholes Required			$69	$119	$97	$127	$110	$68
Blasting Detonation	$27.5	$/hole	$1,907	$3,275	$2,660	$3,498	$3,028	$1,878
Maintenance	$3.5	$/hr	$242	$242	$242	$242	$242	$242
Consumables	$2.5	$/hr	$173	$173	$173	$173	$173	$173
Total Blasting		$	$8,108	$12,974	$10,788	$13,768	$12,096	$8,005
Unit Cost Blasting		$/tonne	$0.38	$0.35	$0.36	$0.35	$0.36	$0.38

Loading

Loading costs are based the calculated equipment hours required for the 20 m^3 primary loading units. The 13 m^3 loader will handle the mill tailings re-handle operation, pit and bench starts, and serve as a backup to the primary loader fleet. Table 21-15 shows the loading cost breakdown through Year 5.

Table 21-15 - Loading Cost

Loading Cost ($s x 1,000)								
	Unit Cost	Units	-1	1	2	3	4	5
Equipment Count, 20M^3			1	2	2	2	2	2
Equipment Count, 13M^3			1	1	1	1	1	1
Loader Hours Required			5,884	10,107	8,210	10,795	9,345	5,795
Operators (All)	$90,000	Avg. Base $/yr	$994	$1,491	$1,491	$1,491	$1,491	$1,491
Fuel,20M^3	70	gal/hr	$791	$1,582	$1,582	$1,582	$1,582	$1,582
Fuel, 13M^3	44	gal/hr	$224	$448	$448	$448	$448	$448
Maintenance,20M^3	87	$/hr	$492	$983	$983	$983	$983	$983
Consumables, 20M^3	90	$/hr	$509	$1,017	$1,017	$1,017	$1,017	$1,017
Maintenance, 13M^3	87	$/hr	$221	$442	$442	$442	$442	$442
Consumables, 13M^3	90	$/hr	$229	$458	$458	$458	$458	$458
Total Loading		$	$3,460	$6,423	$6,423	$6,423	$6,423	$6,423
Unit Cost Loading		$/tonne	$0.16	$0.18	$0.22	$0.16	$0.19	$0.31

Hauling

Hauling costs are based the calculated equipment hours determined by the haul simulations. The initial truck requirement was calculated to be 5 trucks in period -1 with a peak of 12 trucks in period 3. Table 21-16 shows the hauling cost breakdown through Year 5.

Table 21-16 - Hauling Cost

Hauling Cost ($s x 1,000)								
	Unit Cost	Units	-1	1	2	3	4	5
Equipment Count			5	9	10	12	12	9
Hours Operating			29,949	52,578	59,119	81,517	71,111	53,956
Fuel	32	gal/hr	$1,889	$3,316	$3,729	$5,142	$4,485	$3,403
Operators	$75,000	Avg. Base $/yr	$2,071	$3,728	$4,142	$4,971	$4,971	$3,728
			$0	$0	$0	$0	$0	$0
Maintenance	51	$/hr	$1,527	$2,681	$3,015	$4,157	$3,627	$2,752
Consumables	56	$/hr	$1,677	$2,944	$3,311	$4,565	$3,982	$3,022
			$0	$0	$0	$0	$0	$0
Total Hauling		$	$7,165	$12,670	$14,197	$18,835	$17,065	$12,905
Unit Cost Hauling		$/tonne	$0.34	$0.35	$0.48	$0.48	$0.50	$0.61

Mine Support

Support costs include equipment and operators to support road maintenance, dump maintenance, dust control, and pit operations. The support area includes the mine manager and mine foremen. Table 21-17 shows the mine support cost breakdown through Year 5.

Table 21-17 - Support Cost

Support Cost ($s x 1,000)								
	Unit Cost	Units	-1	1	2	3	4	5
Dozer Count	3		3	3	3	3	3	3
Dozer Operating Hours			17,846	17,846	17,846	17,846	17,846	17,846
Dozer Fuel	24	gal/hr	$431	$431	$431	$431	$431	$431
Dozer Operator	$80,000	Avg. Base $/yr	$1,326	$1,326	$1,326	$1,326	$1,326	$1,326
Dozer Maintenance	$40	$/hr	$714	$714	$714	$714	$714	$714
Dozer Consumables	$32	$/hr	$571	$571	$571	$571	$571	$571
Grader Count	2		2	2	2	2	2	2
Grader Operating Hours			5,949	5,949	5,949	5,949	5,949	5,949
Grader Fuel	10	gal/hr	$60	$60	$60	$60	$60	$60
Grader Operator	$80,000	Avg. Base $/yr	$884	$884	$884	$884	$884	$884
Grader Maintenance	$37	$/hr	$220	$220	$220	$220	$220	$220
Grader Consumables	$10	$/hr	$59	$59	$59	$59	$59	$59
Water Truck Count	2		1	2	2	2	2	2
Water Truck Operating Hours			4,957	9,914	9,914	9,914	9,914	9,914
Water Truck Fuel	32	gal/hr	$156	$312	$312	$312	$312	$312
Water Truck Operator	$72,500	Avg. Base $/yr	$400	$801	$801	$801	$801	$801
Water Truck Maintenance	$48	$/hr	$239	$479	$479	$479	$479	$479
Water Truck Consumables	$54	$/hr	$270	$539	$539	$539	$539	$539
Light Vehicle (LV) Count	12		12	12	12	12	12	12
LV Operating Hours	30	usage, min/hr	43,200	43,200	43,200	43,200	43,200	43,200
L V Fuel	2	gal/hr	$86	$86	$86	$86	$86	$86
LV Maintenance	1.5	$/hr	$65	$65	$65	$65	$65	$65
LV Consumables	0.5	$/hr	$22	$22	$22	$22	$22	$22
Light Plant Count	8		6	8	8	8	8	8
Light Plant Operating Hours			22,032	29,376	29,376	29,376	29,376	29,376
Light Plant Fuel	1	gal/hr	$22	$29	$29	$29	$29	$29
Light Plant Maintenance	0.6	$/hr	$13	$18	$18	$18	$18	$18
Light Plant Consumables	0.15	$/hr	$3	$4	$4	$4	$4	$4
Mine Foreman	4	$100,000	$564,164	$564,164	$564,164	$564,164	$564,164	$564,164
Mine Manager	$125,000	$/year	$178,750	$178,750	$178,750	$178,750	$178,750	$178,750
Total Mine Support		$	$6,285	$7,363	$7,363	$7,363	$7,363	$7,363
Unit Cost Mine Support		$/tonne	$0.29	$0.20	$0.25	$0.19	$0.22	$0.35

Maintenance Crew and Equipment

The maintenance crew and equipment cost include maintenance service trucks and maintenance labor, including the maintenance foremen and superintendent. Table 21-18 shows the maintenance crew and equipment cost breakdown through Year 5.

Table 21-18 - Maintenance Crew Cost

Maintenance Crew Cost ($s x 1,000)								
	Unit Cost	Units	-1	1	2	3	4	5
Equipment Count	4	Service, lube & tire truck						
Hours Operating			23,040	23,040	23,040	23,040	23,040	23,040
Fuel	4	gal/hr	$184	$184	$184	$184	$184	$184
			$0	$0	$0	$0	$0	$0
Maintenance	16.5	$/hr	$380	$380	$380	$380	$380	$380
Consumables	8	$/hr	$184	$184	$184	$184	$184	$184
Maint. Laborer	4	$65,000	$367	$367	$367	$367	$367	$367
Maint. Mechanic	8	$75,000	$846	$846	$846	$846	$846	$846
Maint. Planner	2	$75,000	$212	$212	$212	$212	$212	$212
Maint. Shift Foreman	4	$85,000	$480	$480	$480	$480	$480	$480
Maint. Superientendant	1	$105,000	$148	$148	$148	$148	$148	$148
Total Mine Maintenance Crew		$	$2,801	$2,801	$2,801	$2,801	$2,801	$2,801
Unit Cost Maintenance Crew		$/tonne	$0.13	$0.08	$0.09	$0.07	$0.08	$0.13

Technical Services

The technical services cost includes the personnel required to support the mining operation. Table 21-19 shows the technical services cost breakdown through Year 5.

Table 21-19 - Technical Services Cost

Technical Services ($s x 1,000)								
	Unit Cost	Units	-1	1	2	3	4	5
Chief Eng/TS Manager	1	$115,000	$162	$162	$162	$162	$162	$162
Mining Engineers	2	$83,750	$236	$236	$236	$236	$236	$236
Geologists	4	$76,250	$430	$430	$430	$430	$430	$430
Surveyors	6	$70,000	$592	$592	$592	$592	$592	$592
Supplies	0.02	$/tonne	$427	$733	$596	$783	$678	$420
			0	0	0	0	0	0
Total Technical Services		$	$1,848	$2,154	$2,017	$2,204	$2,099	$1,841
Unit Cost Technical Services		$/tonne	$0.09	$0.06	$0.07	$0.06	$0.06	$0.09

21.2.3 PROCESSING COST ESTIMATES

The major processing cost elements include labor, materials, supplies, and consumables, which includes reagents, parts, power, etc., and other minor process cost items. The heap leach and ADR costs were built up as a combination of fixed and variable costs. Components such as labor and supplies are generally fixed during full year operations, while reagents and other consumables are variable, based on the mineralized material processed, gold produced, etc. Table 21-20 below shows the cost breakdown for the heap leach and ADR.

Table 21-20 - Process Cost for ROM Mineralized Material

Item	LOM ($M)	$/t-Mineralized Material
Labor	$34.7	$0.16
Power	$27.8	$0.13
Consumables	$226.6	$1.02
Maintenance Materials	$5.1	$0.02
Other	$20.4	$0.09
Total Processing Cost	**$314.6**	**$1.42[1]**

[1]Includes costs for residual leaching

The costs for the gravity mill are shown below in Table 21-21.

Table 21-21 - Process Cost for Gravity Mill Mineralized Material

Item	LOM ($M)	$/t-Mineralized Material
Labor	$17.3	$1.40
Power	$17.3	$1.40
Consumables	$9.4	$0.77
Maintenance Materials	$3.1	$0.25
Other	$5.4	$0.44
Total Processing Cost	**$52.6**	**$4.26[1]**

[1]Does not include mill tails rehandle

Labor costs are summarized in Table 21-22. The heap leach/ADR and mill will share labor resources for process salaried personnel, as well as control room operators, lab personnel, and maintenance. Dedicated hourly personnel are allocated to each area based on the anticipated staffing requirements. A total of 13 salaried personnel and 50 hourly personnel are envisioned for the project. The process facilities will operate with four rotating crews to provide year-round operations. Including the mill tails reprocessing, the total material processed is 232.1 M tonnes and is used in determining the cost per tonne below.

Table 21-22 - Labor Cost

Item	Number of Persons	Annual Costs ($M)	$/t-Mineralized Material
Management (Staff/Supervisor) Operating	13	$1.3	$0.07
Labor (Including Maintenance)	50	$3.1	$0.16
Total	**63**	**$4.5**	**$0.22**

Power cost estimates are summarized in Table 21-23. Power costs are based on the estimated power consumption as determined in the electrical load list for major mechanical equipment and ancillary loads. The unit costs are based on the total mineralized material processed at 232.1 M tonnes. Power unit rates are the current rates provided by the utility.

Table 21-23 - Power Cost

Item	Utility Power	
	LOM ($M)	($/t-Mineralized Material)
Gravity Mill	$17.3	$0.07
Leaching/ADR/Lab/Admin	$27.8	$0.12
Total Power	**$45.1**	**$0.19**

The two major reagents consumed in the process are lime and sodium cyanide, which account for approximately 91% of the reagent costs. Other costs are estimated, based on the project consumptions for antiscalant, carbon, hydrochloric acid, sodium hydroxide, hydrogen peroxide, and refinery fluxes. Budgetary quotes were obtained for the major reagents. The mill operating cost also accounts for wear items, such as crusher and mill liners and rods for the rod mill.

Maintenance costs were estimated, including all labor and parts, such as pumps, valves, conveyors, agitators, tank and vessel maintenance, as well as electrical and instrumentation expenses. Piping and drip emitter for the leach pad were also estimated and included in the heap leach operating cost.

Other costs include the crusher feed at $0.19/tonne. This assumes a 5 m^3 wheel loader operating approximately 18 hours per day. Allowances have also been provided for other process specific administrative expenses, such as computer hardware or software, travel and training for process personnel, and office supplies. An allowance for contracted service support is also included, which may be used for crane or other equipment rentals or consulting support, as required.

21.2.4 GENERAL ADMINISTRATIVE COST ESTIMATE

General and Administrative costs account for all management and support personnel not directly related to mining or process operations. A total of 16 personnel are assumed for the project (Table 21-24). Costs for all general administrative services related to human resources, health & safety, environmental, site services, such as warehousing, and community relations are included in the G&A costs (Table 21-25). Reclamation costs are also included with G&A.

Table 21-24 - G&A Manpower Costs

G&A Manpower Costs		
Management	Personnel Count	Annual Cost $ x 1,000
General Manager	1	$238
Administrative Superintendent	1	$136
Accounting		
Accountant	1	$109
Accountant's Clerk	2	$109
Payroll Clerk	1	$61
Human Resources		
Human Resources Manager	1	$122
Safety & Environment		
Manager	1	$122
Health & Safety Coordinator	2	$160
Senior Environmental Engineer	1	$88
Junior Environmental Engineer	1	$71
Warehouse Clerk	2	$109
Purchasing, Warehouse, & Site Services		
Purchasing Manager	1	$122
Purchaser	1	$68
G&A Manpower Total	16	$1,516

Table 21-25 - G&A Administration Costs

G&A Administration Costs		
Light Vehicles - G&A		Annual Cost $ x 1,000
Operating and Maintenance Cost		$149
Administration		
Recruitment		$100
Relocation		$100
Contract Services		$200
Company Functions		$25

Advertising		$0
Community Development Programs		$40
Subscriptions & Dues		$10
Communications		$100
Business Permits / Licensing		$100
Legal Fees		$150
Insurance		$250
Water License		$100
Property Tax		$300
Mining Lease		$350
Surface Access Rights		$0
Courier and Hot Shot Services		$30
Public Relations		$50
Sewage/Garbage Collection & Disposal Service		$20
Misc. Services		$24
Reclamation Bonding $/yr		$291
Health & Safety		
HSE Operating Supplies		$70
Employee Medicals and Supplies		$80
Training Supplies		$50
Environment		
Operating Supplies		$75
Sample Analysis		$75
Security		$25
G&A Administration Total		$2,764

22. ECONOMIC ANALYSIS

This PEA is preliminary in nature, it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized. The current basis of Project information is not sufficient to convert the in-situ Mineral Resources to Mineral Reserves, and Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The PEA results are only intended as an initial, first-pass review of the Project economics based on preliminary information.

This PEA is based on technical and economic assumptions which will be evaluated in more advanced studies. The PEA is based on the Mineral Resource Estimate in Chapter 14 (effective date October 7, 2020). The PEA is based on a production plan that includes material in Measured and Indicated classifications from the NBP Mineral Resource model. This report validates that the Project would support an open pit mining operation that recovers gold and silver metals from the ground to be sold at a profit. The methods used to generate this Preliminary Economic Analysis are consistent with good industry practice and meet the required standards for project design and estimation. The financial analysis methodology is consistent with current practice and is consistent with the stage of the NBP development.

The economic analysis of the Project assumed constant 2020 US dollars and was performed on an annual basis beginning at the start of Year -1 when operating permits are assumed to have been issued (although costs for characterization, Feasibility Study, and permitting are included in the economic analysis). Construction was assumed to require 1 year with placement of mineralized material on a heap leach pad and mill processing to start at the beginning of year +1. This PEA assumes a central mill processing facility constructed at the NBP which has a circuit that will be configured to process oxide vein and vein stockwork mineralization from the deposit at NBP. The mill grade mineralization at NBP would be hauled to the mill site where gravity recoverable gold will be collected and leached. Lower grade, oxide mineralization would be processed on heap leach pad. Gold would be captured from the leach solutions using a carbon-in-column adsorption circuit located near the heap leach pad, and the carbon would be hauled to the desorption/refinery facility near the mill for metal stripping and refining.

The PEA estimated mill gold recoveries based on bottle roll testing and gravity concentration testing performed on the YellowJacket vein and stockwork materials which indicate high gold and silver recovery using simple CN leaching. Gold recoveries for the mill are estimated at 45.5% for North Bullfrog mill mineralization, and silver recovery is estimated at approximately 9.5%. Metallurgical testing programs were carried out by McClelland Laboratories, Hazen Research Inc. and Resource Development Inc. on composite samples to provide the basis of the mill circuit configuration. Heap leach recoveries were based on column leach and bottle roll testing data for composite samples constructed from Mayflower, YellowJacket, Savage Valley and Sierra Blanca drilling for North Bullfrog. Recovery estimates for the gravity tails test work were also estimated.

The mill process recovery assumptions reflected overall gold and silver recoveries based on the flow diagram shown in Section 17. The average mill recoveries for gold were 45.5% and 9.5% for silver.

The heap leach process recovery assumptions reflected consideration of the particle size resulting from ultra-high intensity blasting to produce a gradation similar to primary crushing (P80 -84mm) and the leach pad placement schedule. The leach pad production model predicts an average gold recovery of 71.9%, and an average silver recovery of 35.1% of fire assay grade. The production model assumes a 3-year buildup of gold in solution inventory which would require 3 years of rinsing after the final leach pad placement to recover inventory. No cost escalation was included in the calculations, and the cash flows were presented after-royalty and after-tax. A gold price of $1,500 per ounce was assumed for all years (1-15) for the base case. All economic projections were made on an after-royalty and after-tax basis.

The analysis included Measured, Indicated and Inferred Mineral Resources in the mining and economic study. Measured and Indicated Mineral Resources make up 84% of the gold ounces in the total production plan. The remaining 16% of the gold ounces in the total production plan are classified as Inferred Mineral Resources.

22.1 KEY PERFORMANCE PARAMETERS

Mining physicals in the production schedule presented in Table 16-5 were used in conjunction with unit operating cost assumptions to estimate OPEX costs on an annual basis. Estimated capital costs were input on an annual basis from a preliminary schedule that included initial capital associated with pre-mining construction of the Project in year -1 and sustaining capital over the LOM. Mobile equipment was assumed to be financed with 20% down payment and a five-year term at 6% interest. Interest costs were transferred to operating cost.

Key performance parameters are listed in Table 22-1. Tables 22-2 through 22-4 summarize Initial and Sustaining Capital, and Figure 22-1 shows the projected annual gold and silver production from the Project.

Table 22-1 - Projected Key Performance Parameters from the PEA (Constant $, No Escalation, Constant $1,500 per Ounce Gold Price, after-Royalty and after-Tax)

Parameter	Year 1-7 Data Value	LOM Data Value[(4)]
Measured & Indicated Mill Feed (contained oz)	-	12.1M t at 1.47 g/t Au for 575.8k oz
Inferred Mill Feed (contained oz)	-	0.2M t at 1.09 g/t for 7.2k oz
Measured & Indicated Heap Leach Feed (contained oz)	-	149.2M t at 0.21 g/t Au for 983.9k oz
Inferred ROM Heap Leach Feed (contained oz)	-	59.9M t at 0.19 g/t Au for 368.5k oz
Post-Tax and Royalty NPV at 5%	-	$452M
Post-Tax and Royalty IRR	-	47 %
Post-Tax Cashflow		$610M

Pre-tax Cashflow; IRR	-	$763M; 55%
Overall Strip Ratio	1.16:1 (overburden:mineralized material)	0.96:1 (overburden:mineralized material)
Average Annual Payable Gold Production	147 kozs/year	112 kozs/year
Total Payable Gold Produced Years	1,029 kozs	1,467 kozs
Average Gold Recovery - mill	85%	85%
Average Gold Recovery- heap leach	72%	72%
Average Cash Cost(1)	$589/Au Oz	$751/Au Oz
All-in Sustaining Cost (AISC) (2)	$727/Au Oz	$885/Au Oz
Average Silver Recovery – mill	63%	63%
Average Silver Recovery – heap leach	13%	13%
Average Total Mining Rate(3)	89 k tonne/day	85 k tonne/day
Average Mineralized Material Mining Rate(3)	43 k tonne/day	43 k tonne/day

(1)Cash Cost includes mining, processing, site G&A, refining, and royalties.

(2)AISC includes mining, processing, site G&A, refining, royalties, sustaining capital (not initial), and reclamation costs.

(3)Includes mill tails rehandle as well as mining.

(4)Values through Year 15, including 2-year drain down.

In Table 22-1, Average Cash Cost and AISC are for years 1-7, as well as LOM. Average Cash Cost includes mining and processing costs, plus site general and administrative, refining/transport costs, and royalties, along with a credit for the co-product, silver. All-in sustaining cost (AISC) is a non-GAAP production metric and is estimated here to provide additional information only. It should not be considered alone and is included here to provide a comparison to the mining industry. It may aid in the understanding of the comparative economic potential of NBP, but other companies may calculate it differently. A reconciliation of the calculation performed here is listed in Table 22-2.

Table 22-2 Build-up of Estimated All-in Sustaining Cost

Cost Element	Year 1-7 Data Value ($M)	LOM Data Value ($M)
Gold Production (k oz)	1,028.9	1,466.6
Operating Cost ($M)	$651.3	$1,153.3
Royalty ($M)	$6.9	($8.6)
Silver Credit	($52.5)	($60.1)
Cash Cost ($M)	**$605.7**	**$1,101.8**
Cash Cost ($/oz)	***$589***	***$751***
-	-	-
Cash Cost ($M)	$605.7	$1,101.8
Add back Other Costs ($M)	$97.5	$127.9
Total Operating Cost ($M)	**$703.2**	**$1,229.7**
Less Capitalized Mining ($M)	($38.9)	($38.9)
Less Interest Expense ($M)	($13.0)	($13.0)

Plus Royalty ($M)	$6.9	$8.6
Plus Sustaining Capital ($M)	$110.1	$139.2
Plus Initial Fills ($M)	$0.6	$0.6
Plus Working Capital ($M)	$31.1	$31.1
Less Silver Credit ($M)	($52.5)	($60.0)
All-in Sustaining Cost ($M)	**$747.7**	**$1,297.3**
AISC[1] ($/Au oz)	***$727***	***$885***

1-excludes tax

Table 22-3 - Initial Capital Cost Summary

Area	Initial Capital Cost ($M)
Initial Direct Capital Cost	$137.9
EPCM	$8.0
Contingency	$18.5
Owner's Cost	$3.0
Total Initial Capital	**$167.4**

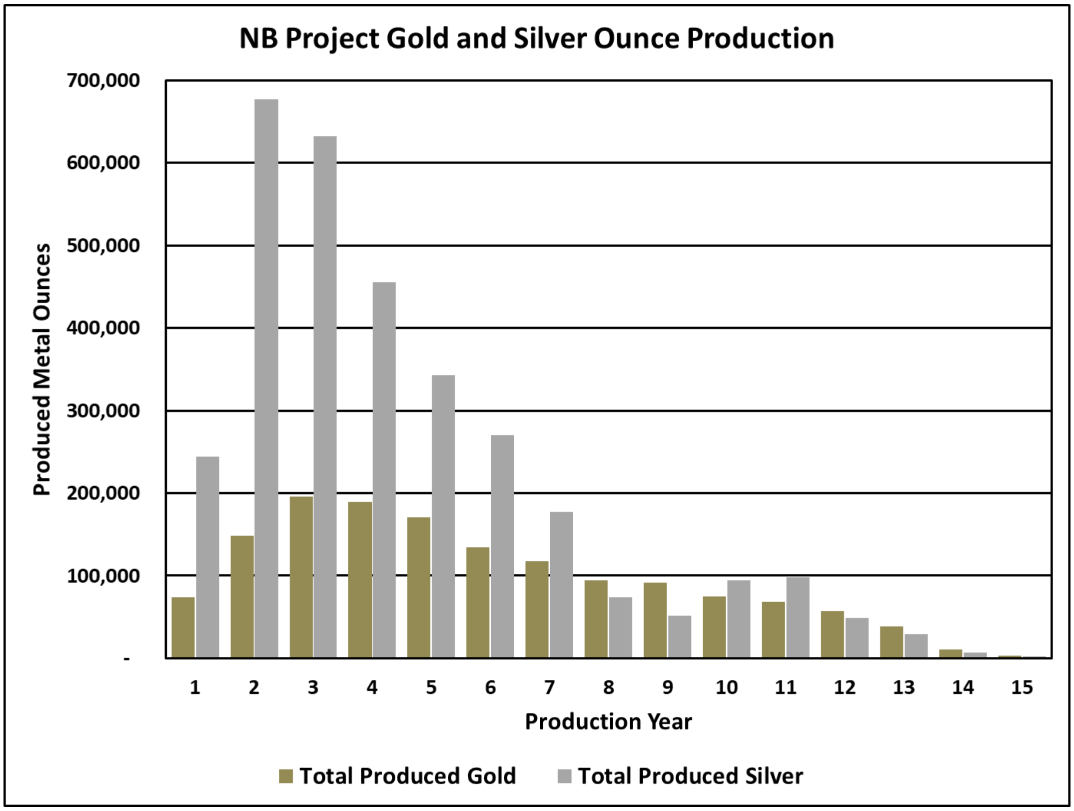
Table 22-4 - Initial Direct Capital Costs

Area	Initial Capital Cost ($M)
Mill	$19.3
Heap Leach	$49.4
Mobile Equipment	$8.0
Infrastructure & Facilities	$22.2
Capitalized Mining	$38.9
Total Initial Direct Capital	**$137.9**

Table 22-5 - Sustaining Capital Costs

Area	Sustaining Capital Cost ($M)
Sustaining Capital	$101.4
EPCM	$8.1
Contingency	$22.7
Total Initial Indirect Capital	**$132.3**

22-1 - Estimated Annual Gold and Silver Production from NBP for Life of Mine (84% Measured and Indicated Mineral Resources, 16% Inferred Mineral Resources)



Estimated physical data for the Project PEA are listed in Table 22-5, for the estimated production mine life of 13 years of active mining.

Table 22-6 - Summary of Physical Data from the NBP Mining Study Production Schedule

Key Physical Data	Units	Value
Heap Leach Feed Mined(2)	M tonnes	209.1
Mill Feed Mined	M tonnes	12.4
Overburden Mined	M tonnes	212.0
Total Material Mined	M tonnes	433.5
Mine Life(1)	Years	13
Contained Gold3	M Oz	1.93
Recovered Gold Payable	M Oz	1.46
Contained Silver	M oz	7.99
Recovered Silver	M Oz	3.20
Average Strip Ratio (mining only)	Overburden/Process Feed	0.96
Average Diluted Gold Grade Heap Leach (incl mill tails grade)	g/t	0.23
Average Diluted Gold Grade Mill	g/t	1.47
Average Gold Recovery	%	75.8
Annual Process Feed Mined	M tonnes/yr	16.7
Annual Gold Produced (LOM)	K Oz/yr	112

1-active mining, excludes pre-production mining or leach pad draindown period at end of mine life

2-Includes mill tails rehandle as well as mining.

3- 84% Measured and Indicated Resource, 16% Inferred Resource

Table 22-7 - Projected LOM Unit Operating Cost and Capital Cost per Process Tonne and per Produced Au Ounce for the Project (Constant 2020 $, No Escalation, Yr 1-15(1)).

Cost Area	Cost per Process tonne ($/tonne)	Cost per Recovered Gold Oz ($/Oz)
Mining	$3.23	$512
Processing	$1.62	$257
Administration	$0.34	$ 53
Reclamation	$0.11	$17
Total Operating Cost	***$5.30***	***$839***
Capital Cost	$1.29	$204
***Projected Total* Cost**	***$6.59***	***$1,043***

(1) Years 14-15 are heap leach draindown/reclamation only.

22.2 CASH FLOW

The projected annual production and cash flow (after-royalty and after-tax) for the NBP are listed in Table 22-7. The estimated payback period assuming the average gold price of $1,500 and average silver price of $18.75 per ounce is 2.1 years

Table 22-8 - Projected Annual Production and Cash Flow (after-Royalty and after-Tax) for the North Bullfrog Project – Base Case (Gold Price $1,500; Silver Price $18.75)

Year	Over- burden Mined (M t)	Process Feed Mined (M t)[2]	Contained Au (k Oz)	Payable Au[1] (k Oz)	Gold Revenue[1] (US $M)	Payable Ag[1] (k Oz)	Silver Revenue[1] (US $M)	Operating Cost (US $M)	Capital Cost (US $M)	Pre-Tax, After Royalty Cash Flow (US $M)	Federal Income Tax (US$M)	Nevada NPT Tax (US$M)	Cash Flow After Tax, After Royalty (US $M)
-1	18.4	2.9	34.5	0	0	0	0	0	-$167.4	-$167.4	$0.0	$0.0	-$167.4
1	23.7	11.2	123.3	73.6	$110.4	244.5	$4.6	-$135.8	-$4.4	-$14.7	-$4.0	-$8.7	-$14.7
2	16.8	11.6	190.5	148.2	$222.3	676.7	$12.7	-$98.7	$20.5	$175.9	-$23.5	-$9.6	$163.2
3	22.2	15.1	253.4	194.6	$291.9	632.6	$11.9	-$129.0	-$30.2	$174.6	-$27.0	-$9.3	$141.6
4	16.3	15.8	244.9	187.3	$280.9	455.4	$8.5	-$120.4	-$12.1	$188.1	-$20.1	-$7.4	$151.8
5	14.4	18.7	216.7	169.6	$254.4	342.3	$6.4	-$136.8	-$69.6	$96.3	-$10.6	-$4.4	$68.8
6	12.0	22.5	155.5	133.9	$200.9	270.0	$5.1	-$136.1	-$6.0	$98.1	-$7.8	-$3.3	$83.2
7	10.5	22.9	158.4	117.1	$175.7	176.8	$3.3	-$124.9	-$1.2	$83.1	-$5.0	-$2.0	$71.9
8	10.0	21.4	130.8	93.3	$139.9	74.2	$1.4	-$108.5	-$0.8	$56.3	-$4.3	-$1.6	$49.2
9	12.0	21.4	130.8	91.2	$136.8	51.9	$1.0	-$119.0	-$29.1	$21.2	-$2.2	-$0.7	$15.4
10	15.0	18.1	88.3	74.4	$111.6	94.7	$1.8	-$110.3	$0.0	$28.6	-$1.2	-$0.2	$25.7
11	15.0	18.0	87.0	68.7	$103.0	97.9	$1.8	-$111.3	$0.0	$17.1	$0.0	$0.0	$15.7
12	13.0	17.0	76.5	56.7	$85.1	49.0	$0.9	-$101.4	$0.0	$4.1	$0.0	$0.0	$4.1
13	7.5	9.8	44.2	38.5	$57.8	29.0	$0.5	-$67.4	$0.7	$4.4	$0.0	$0.0	$4.4
14[3]	0	0	0	10.5	$15.7	6.4	$0.1	-$19.7	$0.0	-$0.5	$0.0	$0.0	-$0.5
15[3]	0	0	0	3.2	$4.9	2.0	$0.0	-$8.2	$0.0	-$2.3	$0.0	$0.0	-$2.3
LOM	**206.8**	**226.6**	**1,934.7**	**1,460.8**	**$2,191.3**	**3,203.5**	**$60.1**	**-$1,527.6**	**-$299.7**	**$763.0**	**-$105.8**	**-$47.2**	**$610.1**
										$574.1	**NPV 5%**		**$452.3**
										55%	**IRR**		**47%**

1-Less royalty ounces
2- Excludes mill tails rehandle
3- Year 14-15 are predominantly heap leach draindown/reclamation

22.3 SENSITIVITY

The sensitivity of the PEA for the Project has been evaluated for variations in the gold price assumption, gold recovery assumption, operating cost and capital cost. These sensitivities are evaluated around the base case price assumptions of an average gold price of US $1,500 per ounce, and the average gold recovery, OPEX and capex price assumptions listed in Tables 22-5 and 22-6. Table 22-8 lists the estimated Net Present Value (NPV) at discount rates of 0%, 5%, 7.5% and 10%, and the estimated Internal Rate of Return (“IRR”) for the gold price assumptions between US $1,200 and $2,000 per ounce.

At the effective date of the PEA the spot price for gold was $1,887 per ounce. The base price for the financial analysis used here was $1,500 per ounce. The range of gold price used to evaluate the sensitivity to price is considered reasonable by the qualified persons responsible for this report. The discount rates employed in the analysis are considered to be appropriate given the current low interest rates available for US Treasury Securities.

Table 22-9 - Projected Sensitivity of Net Present Value and Internal Rate of Return to Variation in Gold Price (after-Royalty and after-Tax)

Gold Price ($/Oz)	Total Cash Flow (US $M)	NPV @ 5% (US $M)	NPV @ 7.5% (US $M)	NPV @ 10% (US $M)	IRR (%)	Payback (years)
$1,200	$238.8	$172.0	$143.5	$117.8	26.9%	2.8
$1,300	$364.2	$266.5	$226.7	$191.8	34.5%	2.5
$1,400	$488.2	$360.1	$309.2	$265.1	41.2%	2.3
$1,500	**$610.1**	**$452.3**	**$390.6**	**$337.5**	**47.5%**	**2.1**
$1,600	$731.6	$544.3	$471.7	$409.7	53.5%	2.0
$1,700	$851.6	$635.2	$552.0	$481.1	59.0%	1.9
$1,800	$970.5	$725.3	$631.5	$551.9	64.4%	1.8
$1,900	$1,089.2	$815.3	$711.0	$622.6	69.6%	1.7
$2,000	$1,207.7	$905.2	$790.3	$693.3	74.7%	1.7

Sensitivity to the proportional change from the base case economic projection, derived at an average gold price of US $1,500 per ounce and gold recovery, OPEX and capex unit costs listed in Tables 22-5 and 22-6, were estimated for a nominal range of + 33% to – 20% from the base case assumptions. The sensitivity is shown graphically for NPV@ 5% and for IRR in Figures 22-2 and 22-3, respectively. This range of sensitivity cost is considered reasonable by the qualified persons for this report considering the accuracy of the cost estimates.

Figure 22-2 - Sensitivity of Estimated NPV @ 5% (after-Royalty and after-Tax) for Changes in Cost, Gold Recovery or Gold Price as a Percent of the Base Case at a Gold Price of $1,500 per Ounce, Gold:Silver Price Ratio of 80.0, 75.8% Gold Recovery and Cost as Defined in Tables 22-5 and 22-6

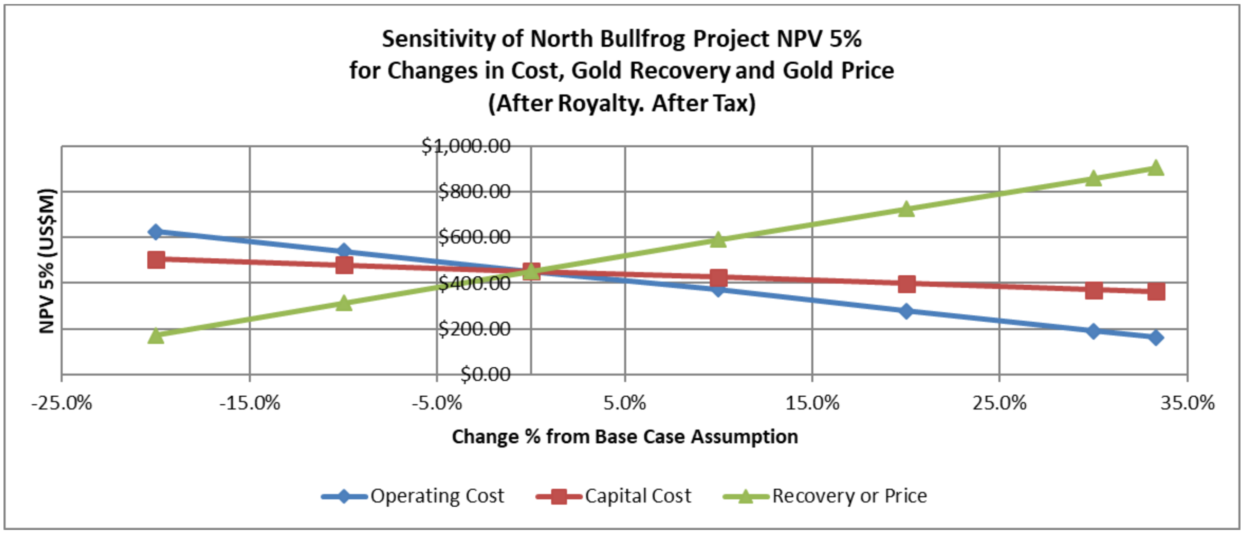
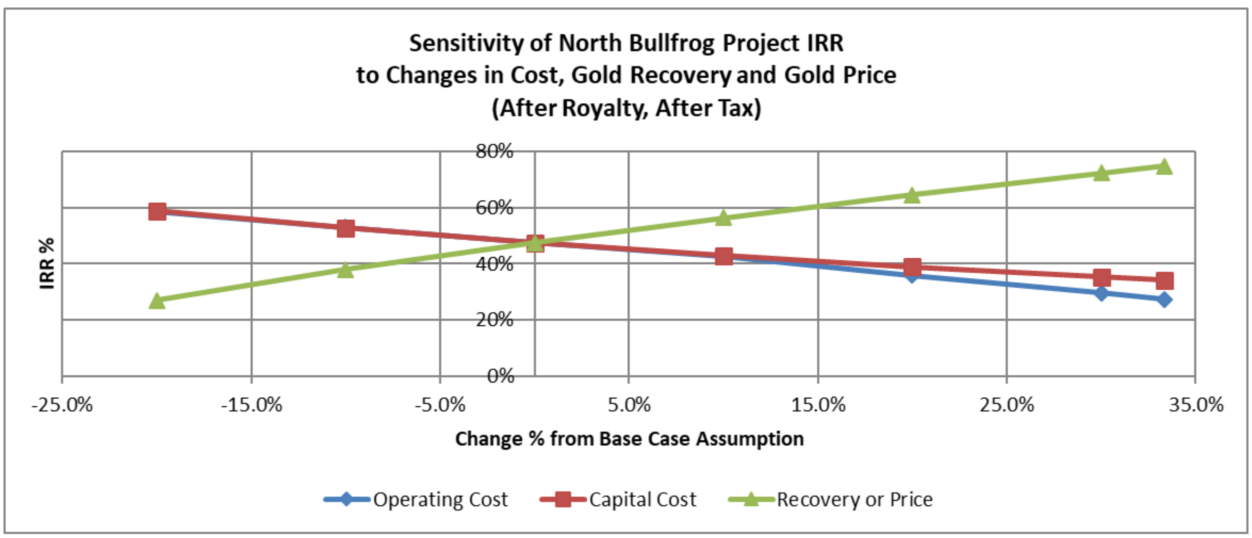


Figure 22-3 - Sensitivity of Estimated IRR (after-Royalty and after-Tax) for Changes in Cost, Gold Recovery or Gold Price as a Percent of the Base Case at a Gold Price of $1,500 per Ounce, Gold:Silver Price Ratio of 80.0, 75.8% Gold Recovery and Cost as Defined in Table 22-5 and 22-6



The sensitivity analysis indicates that the Project would be most sensitive to gold price and gold recovery assumptions. The PEA was less sensitive to changes in cost, with changes in OPEX having a slightly greater effect than changes in capex.

22.4 TAXES, ROYALTIES, AND OTHER INTERESTS

Corvus would be subject to the following taxes as they relate to the Project:

- Nevada Net Proceeds Tax
- Federal Income Tax

Corvus would also be subject to royalties as described in Section 22.4.3.

Estimates of these taxes and royalties were made based on the production schedule in Table 22-7 and operating and capital cost estimates described in Section 21.

22.4.1 NEVADA NET PROCEEDS MINERAL TAX

In Nevada, if the net proceeds of a mine in the taxable year totals $4 million or more the tax rate is 5%. The gross proceeds from the sale of the minerals minus certain allowable deductions were used to estimate the taxable net proceeds. The Nevada net proceeds tax is calculated before deductions of Federal income tax. In general, all operating costs and capital costs directly related to the mining operation are deductible, using Nevada depreciation and depletion schedules.

22.4.1.1 FEDERAL INCOME TAX

Corporate Federal income tax was estimated by computing the regular federal income tax as modified in 2018. Regular tax was estimated by subtracting Nevada Net Proceeds Mineral tax, all allowable operating expenses, overhead, depreciation, amortization and depletion from revenues on an annual basis to estimate the taxable income. The highest effective corporate income tax was 21%.

22.4.2 DEPLETION

Generally speaking, depletion, like depreciation, is a form of cost recovery. Just as the owner of a business asset is allowed to recover the cost of an asset over its useful life, a miner would be allowed to recover the cost of the mineral property. Depletion was taken over the projected period that minerals would be extracted.

For federal income tax purposes, two forms of depletion are allowed: cost depletion and percentage depletion. The taxpayer is required to use the method that will result in the greatest deduction.

A §382 net operating loss limitation in the event of an "ownership change" (as defined under IRC §382(g)) on or before the publishing date of this tax model should not have material impact to the tax model.

22.4.2.1 COST DEPLETION

Cost depletion was estimated based on the adjusted basis of the depletable property multiplied by the units of mineralized material projected to be produced over the production schedule in Table 16-1.

22.4.2.2 PERCENTAGE DEPLETION

Under the percentage depletion method, a flat percentage of 15% of adjusted gross income from gold mining was used to estimate the depletion allowance. However, the deduction for depletion cannot exceed 50% of the adjusted taxable income from the activity. This limitation was computed without regard to the depletion allowance. The amount of the deduction allowable under percentage depletion is not limited by the basis of the property, except for AMT purposes. Thus, even though the basis of the property would be reduced by the amount of depletion taken, if the basis becomes zero, the depletion based on the percentage of adjusted gross income may continue to be claimed for tax purposes.

22.4.2.3 DEPRECIATION

Cost recovery for capital invested was estimated using standard depreciation schedules specified for different types of investment. The estimated cost recovery for calculation of Federal income tax included the 7 years 200% declining balance calculation, an expense 73% with 6% for the next 4 years and the final 3% in the last year calculation, and a units of production depreciation schedule. Both an alternative minimum tax and regular tax depreciation were estimated.

22.4.3 ROYALTIES

The calculation of estimated royalties was based on projected mining production underlying individual leases to which royalties apply. The royalty status of the patented claim blocks at North Bullfrog is discussed in Section 4.1. Where lease agreements provide for royalties and had gold production in the plan (specifically Mayflower and Jolly Jane), annual gold and silver production from those claim blocks has been projected and used to estimate royalty ounces of gold and silver. Total royalty ounces were estimated to be 5,711 Au ozs, or 0.4% of the total estimated gold production. Those royalty ounces were deducted from the produced gold to calculate the payable gold and silver production.

23. ADJACENT PROPERTIES

There are two adjacent properties that are relevant to the NBP; the Sterling Mine, recently acquired by Coeur Mining, Inc. and the Reward Gold Project owned by Waterton Resources Limited (previously owned by Atna Resources, Limited). Both properties have published NI 43-101 technical reports that can be found under the respective owner's profiles on SEDAR.

The qualified persons authoring this Technical Report have been unable to verify the information available with respect to the adjacent properties, and such information is not necessarily indicative of the mineralization at the NBP.

24. OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information for the Project.

25. INTERPRETATION & CONCLUSIONS

This report was prepared by a group of independent consultants, all qualified persons as defined by NI 43-101, to demonstrate the economic viability of open pit mining and processing, based upon the estimated Mineral Resources at North Bullfrog Project (NBP). This report provides a summary of the results and findings to the level that would be expected for a Preliminary Economic Assessment. Standard industry practices and assumptions have been applied in this PEA.

This report is based on all available technical and scientific data as of October 7, 2020, the effective date of this Technical Report. Mineral Resources are considered by the QP to meet the reasonable prospects of eventual economic extraction due to the two main factors of; 1) cutoff grades are based on scientific data and assumptions related to the project and 2) Mineral Resources are estimated only within pit limits derived by the scientific data as well as by using generally accepted mining and processing costs that are similar to many projects in Nevada. Additionally, many of the costs were derived specifically for this Project. Confidence in the Mineral Resource estimate was used to classify Mineral Resources based upon drill hole spacing, geological knowledge of the deposits, metallurgical studies and a proper QA/QC program.

25.1 PRELIMINARY ECONOMIC ASSESSMENT

This PEA is preliminary in nature and is based on technical and economic assumptions which will be evaluated in more advanced studies. The PEA is based on the Mineral Resource estimate in Chapter 14 (effective date October 7, 2020). The PEA is based on a production plan that includes material in Measured, Indicated and Inferred classifications from the NBP Resource model. This report validates that the Project would support an open pit mining operation that recovers gold and silver metals from the ground to be sold at a profit.

Estimated Mineral Resources were assumed to be processed with commonly utilized recovery methods which are in operation throughout the state of Nevada. Processing of lower grade oxide materials on heap leach pads is widely utilized. High grade gravity-separable material would be processed through a 4,700 tpd gravity mill and lower grade oxide material would be shipped as Run-of-Mine (ROM) directly to the heap leach pad. The gravity tail material would be blended with the ROM heap leach feed for final recovery of the contained gold.

Under the base case assumptions for the project, the PEA indicates an undiscounted pre-tax cash flow of $763 M, and a post-tax NPV at 5% discount of $452 M. The resulting post-tax IRR is 47% for an initial capital investment of $167.4 M.

The results of sensitivity analyses of post-tax cash flow and post-tax IRR show that the project is most sensitive to recovery and gold price while the Project is least sensitive to changes in capital costs. An approximately 20% decrease in gold price to $1,200 per ounce results in a positive NPV at 5% discount of $172 M, at a gold price of $1,500 per ounce the NPV at 5% discount results in $452 M, and at a gold price 35% higher ($2,000), the NPV at 5% discount is $905 M. The base case assumptions demonstrate that the Project would produce an average of 147 thousand ounces of gold in years 1 through year 7. The Project would produce 1.47 million ounces total through the entire life of the mine.

25.2 GEOLOGY AND EXPLORATION

During 2012-2104 Corvus delineated the YellowJacket zone of Sierra Blanca. This represented a completely blind discovery of a previously unrecognized style of mineralization at NBP. A 3D IP survey, a gravity survey in the eastern Steam-heated Zone along with more detailed structural/geologic mapping in early 2015 have provided the basis for target generation on the rest of NBP. These structural targets and the general Jolly Jane and Sierra Blanca areas are the priority for future work at the NBP. There are also other alteration and geochemical anomalies throughout in the Eastern Steam-heated Alteration Zone that the Project should evaluate.

Jolly Jane, Sierra Blanca, Yellow Jacket and the Savage Valley areas contain significant low-grade gold-silver deposits. Drilling has shown that the Sierra Blanca Tuff is a primary host rock for broad areas of disseminated semi-stratabound gold mineralization. Much of the mineralization is near surface, oxidized, and metallurgical testing has indicated good heap leaching characteristics.

No additional exploration is required to execute the NBP Phase I project however some condemnation drilling is warranted in the areas of the proposed mine facilities. Exploration potential is still high in the NBP area and will likely add additional mineralized material and mine life.

25.3 METALLURGY

Laboratory testing on higher grade vein and stockwork mineralization in the YellowJacket zone indicated that the preferred process would be a milling system using gravity concentration, intensive CN leaching of the gravity concentrate followed by CN leaching of the combined gravity tail and leached gravity concentrate on a heap leach pad. This became the basis for the process flow sheet in this PEA. The grades and recoveries of the NBP disseminated mineralized material are suitable for heap leach processing and recent laboratory test results indicate good recovery (>70%) of ROM material.

The metallurgical test work performed on the NBP is considered to meet PEA standards.

25.4 MINING

Conventional surface mining methods using surface drill and blast techniques with off highway haul truck and front-end loaders were assumed for this report. Equipment requirements were based on Cat Handbook equipment performance data. Haul truck estimates were based on simulated cycle times derived from haul profiles for each year in each pit's current mining phase.

Pit slope angles were assumed to be 50 degrees for all pits except for Mayflower, which utilized varying pit slopes based on a site specific highwall study. These slope angles may be conservative based on the Yellowjacket highwall study conducted by Knight-Piésold in 2020.

The mine production schedule was developed using the pit bench data from the design pits in years 1-5 and the pit shells for years 6-13. The scheduled production included Measured, Indicated and Inferred Mineral Resources. The mine production schedule focused on opening the YellowJacket pit to provide the annual gravity mill feed of 1.7 Mt. The other pits are sequenced to provide approximately 11.5 Mt of ROM heap leach mineralized material annually for Years 1 through 5, then increasing to 18.7 Mt of ROM material per year through Year 13.

25.5 PROCESSING

25.5.1 GRAVITY MILL

The high-grade mineralization from the YellowJacket vein and vein stockwork deposit would be processed through a mill circuit at a rate of approximately 4,700 tonnes per day. The gravity mill flow sheet was developed based on the positive laboratory test work completed for this study. The coarse grind size (48 mesh) allowed for the gravity mill tailings to be dewatered and placed directly on the leach pad for residual leaching. Blending would occur on the leach pad as the gravity tail and ROM mineralized material was dozed into the 10m lifts.

The basis for the capital estimate for the gravity mill was equipment quotes from the preferred vendor and construction factors for equipment installation. No final engineering design work has been completed for this study therefore some additional work would be required to meet Pre-Feasibility standards.

25.5.2 HEAP LEACH/ADR

Mineralization from the Yellowjacket, Sierra Blanca, Mayflower, Jolly Jane and Savage Valley open pits would be mined and placed onto a heap leach pad. Run of Mine (ROM) mineralized material would be mined at a rate of approximately 44,700 tonnes per day and placed on a conventional heap leach pad in nominal 9.1 m (30 ft) lifts. The heap leach pad and ADR systems were designed to meet the mine production requirements. The leach pad facility was designed to accommodate ~ 244M tonnes of mineralized material.

25.6 Risks and Uncertainties

The current design of these facilities is considered to meet PEA requirements. The qualified persons have reviewed areas critical to a successful project to identify key risks. General risks associated with mining projects include but are not limited to:

- General business, social, political, regulatory and business competition;
- Change in project parameters as development plans are advanced;
- Labor costs and other costs of production;
- Lower gold price;
- Compliance with laws and regulations or other regulatory requirements;
- Availability of management, technical and skilled operations personnel.

Identified Project risks are listed in specific items that follow with a relative risk rating. The identified risk areas are not ordered in rank of importance due to the relative early stage of development:

1. Geology – resource quantity and quality (grade) – considered to be potential upside due to the large, unexplored area;
2. Mining – pit slope geotechnical conditions are not completely defined – considered low risk due to general rock conditions indicated by currently limited evaluation;
3. Processing – data indicate good gold recovery and metallurgical characteristics -considered low risk due to extensive testing;
4. Environmental Permitting – located in a major mining region of the US with good regulatory experience-considered low risk because of the numerous operating mines in Nevada;
5. Construction Schedule – preliminary integration of permitting and construction planning– low risk because of numerous operating mines, well defined regulatory process and project experience;
6. Capital Cost – preliminary designs at this stage – low risk due to applicability of existing technology and equipment in the area;
7. Operating Cost – first principle estimates – low risk, extensive operating experience readily available in Nevada;
8. Land – existing federal mining claims and private land – low risk, Corvus controls the land position, well established public land mining regulations and operating histories;

26. RECOMMENDATIONS

This PEA is preliminary in nature and is based on technical and economic assumptions which will be evaluated in more advanced studies. The detail developed in most areas of the PEA could be considered sufficient as permitting basis, however, some areas will require more advanced design work. Once this additional work (described below) is complete the Project could move forward with permitting for operations.

A key area requiring some additional engineering work is the Gravity Mill Facility. Engineering detail would be needed to show installation details for the equipment and the associated containment of process fluids. A revised capital estimate should be prepared to reflect the engineering detail as well.

The preliminary mine plan was developed based on a nominal 1.7M tpa mill feed for 7 years and the results provide a robust economic project. To achieve this throughput significant waste material was added to the LG pit results for the YellowJacket pit. Further optimization is recommended to determine if a more profitable mining/mill throughput rate is possible.

Significant low-grade mineralization remains in the NBP area. Additional mine planning work should be undertaken to advance the mine design beginning in Year 6 through LOM to the same level of detail included in the design for Year 1 through 5.

The next step for the NBP is to develop an operating mine. In order to achieve this, all the requisite environmental and operating permits will be required. Corvus is currently developing a permitting strategy and will begin collecting time critical data to support the additional baseline work required. It is anticipated that an Environmental Impact Statement (EIS) will be required.

Table 26-1 lists the major task areas required to advance the project to a construction permit.

Table 26-1 - Proposed Work Program to Advance NBP to Construction

Activity	Amount
1 -Condemnation Drilling and Data Management – 2 months, concurrent with Baseline	$ 0.2 M
2 -Baseline Data Collection to support Permit Application – 12 months, concurrent with Feasibility	$ 3.2 M
3 -Feasibility Level Design of Project and Facility - 12 months, concurrent with baseline	$1.2 M
4 -Prepare and submit Plan of Operations and Permit Application – 15 months	$ 1.1 M
5 -Receive Regulatory Permits, 3 months	$ 0.3M
Total – 24 to 30 months	$ 6.0M

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