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JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
604.558.6300
jdsmining.ca

Technical Report
for the Initial Project Feasibility Study on the Back River Gold Property, Nunavut, Canada

Effective Date: September 14, 2015

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Prepared for:

Sabina Gold & Silver Corp.

Suite 202 – 930 West 1st Street

North Vancouver, BC V7P 3N4

Qualified Persons	Company
Gordon Doerksen, P.Eng	JDS Energy & Mining Inc.
Dino Pilotto, P.Eng.	JDS Energy & Mining Inc.
Stacy Freudigmann, P.Eng.	Canenco Canada Inc.
Andrew Fowler, MAusIMM, CP (Geo)	AMC Mining Consultants (Canada) Ltd.
Dinara Nussipakynova, P.Geo	AMC Mining Consultants (Canada) Ltd.
John Morton Shannon, P.Geo	AMC Mining Consultants (Canada) Ltd.
Maritz Rykaart, P.Eng.	SRK Consulting (Canada) Inc.
Robert Mercer, P.Eng.	Knight Piésold

Date and Signature Page

This report entitled "Technical Report for the Initial Project Feasibility Study on the Back River Gold Property, Nunavut, Canada", effective as of September 14, 2015 was prepared and signed by the following authors:

NAME	SIGNING DATE
"Gordon Doerksen" Gordon Doerksen, P.Eng.	October 28, 2015
"Dino Pilotto" Dino Pilotto, P.Eng.	October 28, 2015
"Stacy Freudigmann" Stacy Freudigmann, P.Eng.	October 28, 2015
"Andrew Fowler" Andrew Fowler, MAusIMM	October 28, 2015
"Dinara Nussipakynova" Dinara Nussipakynova, P.Geo	October 28, 2015
"John Morton Shannon" John Morton Shannon, P.Geo	October 28, 2015
"Maritz Rykaart" Maritz Rykaart, P.Eng	October 28, 2015
"Robert Mercer" Robert Mercer, P.Eng	October 28, 2015

NOTICE

JDS Energy & Mining, Inc. prepared this National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Sabina Gold & Silver Corp. The quality of information, conclusions and estimates contained herein is based on: (i) information available at the time of preparation; (ii) data supplied by outside sources, and (iii) the assumptions, conditions, and qualifications set forth in this report.

Sabina Gold & Silver Corp. filed this Technical Report with the Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

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Tables and Figures

1 Executive Summary

This "Technical Report for the Initial Project Feasibility Study on the Back River Gold Property, Nunavut, Canada" (the Initial Project Feasibility Study or the 3KFS) discusses the scope, design features, and economic viability of the Back River Gold Project (Project or Property) located in southwestern Nunavut, Canada. In 2015, Sabina Gold & Silver Corp. (Sabina) commissioned JDS Energy and Mining Inc. (JDS) to lead the FS. The following companies contributed to the study:

- JDS Energy & Mining Inc. (JDS) – Mineral Reserves, mining, on-site and off-site infrastructure, logistics, capital costs, operating costs, financial analysis, and report preparation;
- Canenco Canada Inc. (Canenco) – processing and metallurgy;
- AMC Mining Consultants (Canada) Ltd. (AMC) – geology and Mineral Resources;
- SRK Consulting (Canada) Inc. (SRK) – geochemistry, hydrogeology, overburden geotechnical evaluation, tailings and waste rock management, and water management; and
- Knight Piésold Ltd. (KP) – mining geomechanical evaluation.

1.1 Project Concept

The current Initial Project Feasibility Study was conceived after it was recognized that financing the initial capital for the 6,000 tonnes per day (t/d or tpd) Project, as set out in the June 2015 Feasibility Study (JDS 2015 "Technical Report and Feasibility Study for the Back River Gold Property, Nunavut", June 22, 2015 with an effective date of May 20, 2015), referred throughout this report as the 6KFS, would be challenging under prevailing market conditions. A conceptual study of a smaller throughput, higher grade, and reduced capital cost estimate (CAPEX) option showed the potential for an improved rate of return (IRR) and more importantly, a reduced initial CAPEX. The Project advanced into this current FS and consists of open pit and underground mining at the Goose Site that will only feed a 3,000 t/d whole-ore leach process plant. Mining operations are planned to continue for a 10-year period, while the plant will operate for a further two years, resulting in a total mine life of 12 years. The plan is designed to produce an average of approximately 194,000 oz of gold per year as doré bullion. A total of 12.4 Mt of ore is planned to be mined at a mill head grade of 6.3 g/t and a projected gold recovery of 93%. A total of 2.32 Moz of gold is projected to be recovered over the life of mine. The Project would be built over a 24-month period at an initial capital cost of $415 million. Initially, tailings would be stored in a purpose-built storage facility, followed by deposition into an exhausted open pit.

Although Mineral Resources for both the Goose and George sites are reported, only the Goose Site resources are considered for mining in this FS.

1.2 Project Location and Access

The Project is located in the southwestern part of Nunavut Territory, Canada. It is situated approximately 520 km northeast of Yellowknife, Northwest Territories, 225 km east of the closed Lupin gold mine, 50 km southeast of Glencore Plc's Hackett River Project, 285 km south of TMAC Resources Inc.'s Hope Bay Project (Doris), and 95 km southeast of the southern end of Bathurst Inlet.

The Project is currently accessed and supplied by air, using a combination of both seasonal ice and all-weather airstrips at the Goose Site. During the construction phase and throughout the life of the mine, most equipment, supplies, and fuel will be transported to a Marine Laydown Area (MLA) by ocean-going vessels during the summer open-water season. The MLA will be located on the southern portion of Bathurst Inlet.

Materials will then be transported to the Goose Site by tractor-trailers and road tankers using winter ice roads. Employees will work on a fly-in/fly-out shift rotation basis and be housed in fully catered camps.

1.3 Property Ownership and History

The Property is 100%-controlled by Sabina, and is subject to net smelter return (NSR) royalties on the Goose deposits, payable to various third parties. Additionally, a net profit royalty is payable to the Crown that is deductible from income taxes.

Since exploration began in 1982, the Property has had several owners. Most recently, Dundee Precious Metals Inc. (DPM) conducted operations from 2005, until Sabina purchased the Property in 2009. Prior to that, periods of intensive exploration were conducted by Homestake Mineral Development Company (Homestake Mineral) from 1987 to 1996, Kit Resources Ltd. (Kit) from 1997 to 1998, Kinross Gold Corp. (Kinross), and Miramar Mining Corporation (Miramar) and DPM from 1999 to 2009. Since 2009, Sabina explored the Property with several, multi-faceted campaigns. To date there has been no recorded gold production from any of the Property's deposits.

1.4 Geology and Mineralization

The Property displays gold mineralization that is associated with quartz veins, silicification, and shearing. The gold mineralization occurs within silicified and variably sulphidized iron formation and, to a lesser extent, meta-sedimentary units that commonly have a spatial association with narrow porphyritic felsic dykes and mudstones, wherever these units are present. Gold mineralization is located within two principal areas of the Back River Property: the Goose Site and the George Site. This Initial Project Feasibility Study focuses on advancing the Goose Site only, with no attempt to incorporate the George Mineral Resources.

The Goose Site consists of four main deposits that contain predominantly structurally-controlled gold mineralization: Goose Main, Echo, Umwelt, and Llama. Gold mineralization is predominantly hosted within the Lower Iron Formation (LIF) and, to a much lesser extent, the underlying sediments. The Goose Main, Umwelt, and Llama deposits are associated with anticlinal structures that have been structurally thickened and disrupted, and cut by axial planar felsic dykes, which apparently trace the fluid pathways and are related to mineralization.

The Echo deposit is associated with gentle folding of iron formation and a cross-cutting felsic dyke. Mineralization is spatially associated with the felsic dyke.

The George Exploration Site consists of six main deposits: Locale 1 (Loc1), Locale 2 (Loc2), Slave, GH, LCP North (LCPn) and LCP South (LCPs). Gold mineralization is located within oxide iron formations near the stratigraphic base of this unit. Less significant gold mineralization is also hosted within a silicate iron formation. Gold-bearing zones are associated with sulphide concentrations in the iron formation, and are commonly accompanied by increased quartz veining and attendant alteration of the surrounding rocks.

1.5 Metallurgy

Multiple historical test work programs have been undertaken, including comminution, process mineralogy, ore sorting and gold recovery by gravity concentration, flotation, and cyanidation. Significant mineralogical characterization studies, focusing on gold occurrence in various mineral samples across the deposits, have also been undertaken.

In early 2013, a comprehensive metallurgical test program was conducted to further assess the metallurgical performance of the mineralization to support the Prefeasibility Study (PFS). A subsequent and more detailed test program commenced in late 2013 and concluded mid-2014 to support the FS.

The test work indicated that mineral samples collected from five different geographical zones and five different rock types responded well and showed a high degree of consistency to gravity concentration and cyanidation. The process flow sheet was developed using test results from all of the mineralized zones. Other engineering data was also generated, including tailings settling and viscosity data, and carbon loading response. The 2014 test results were comparable to the results produced from the historical test programs.

Based on the 2014 and historical test results, a combination of gravity separation and cyanide leach processes is proposed for the Project. The concentrate from the gravity separation circuit will be leached separately by intensive cyanide leaching. Although flotation test work demonstrated reasonable recoveries, this process was not selected because of the potential for gold-loss in the flotation tailings.

The 2014 test results are summarized as follows:

- Whole-ore leach showed slightly better metallurgical recoveries when compared to a flotation/regrind/concentrate leach circuit;
- Gold recoveries by gravity concentration ranged from 16 to 76%;
- An average Bond Ball Mill work index (BWi) of 15 kWh/t was determined, indicating moderate hardness in terms of grinding requirements; and
- The optimum grind for the ore was determined to be 50 µm (P80).

Test work results were used to determine the relationship between mill-feed grade and metallurgical recoveries for each of the deposits as shown in Table 1.1.

Table 1.1: Gold Recovery Projections

Mineral Zone	Head Grade (Au g/t)	Estimated Gold Recovery (%)
Umwelt Open Pit	6.49	92.0
Umwelt Underground	7.38	92.0
Llama Open Pit	7.15	91.1
Goose Main Open Pit	5.00	95.0
Life of mine (LOM)	6.30	93.0

Source: JDS 2015

1.6 Mineral Resource Estimate

The Property contains an estimated Measured and Indicated Resource of 28.2 Mt at 5.87 g/t Au, containing 5.33 Moz Au (Table 1.2). Mineral Resources are reported for both Goose and George sites. However, this Initial Project Feasibility Study focuses on advancing the Goose deposits: Llama, Umwelt and Goose Main.

The Mineral Resource for the Goose deposits was reported using a conceptual open pit design at a 1.0 g/t cut-off value and a conceptual underground mine design at a 3.5 g/t cut-off value (except Umwelt, which was designed at a 4.5 g/t cut-off value) assuming a gold price of US$1,500/oz and an exchange rate of C$1.00 to US$1.00.

The Mineral Resource for the George deposits was reported using a conceptual open pit design at a 1.0 g/t cut-off value and a conceptual underground mine design at a 4.0 g/t cut-off value assuming a gold price of US$1,500/oz.

The Mineral Resource estimate is based on geologic block models that incorporated the following:

- 896 drill holes (for a total of 244,853 m and 124,274 assays) at the Goose Site on the Llama, Umwelt, Echo, and Goose Main deposits; and
- 770 drill holes (for a total of 139,695 m and 54,273 assays) at the George Site on the LCPn, LCPs, Loc1, Loc2, GH, and Slave deposits.

Mineralized domains were constructed to constrain the estimates using a 0.3 g/t Au threshold for both the Goose and George sites. Capping was employed where required, and varied by deposit. Data density allowed for Indicated and Inferred resources to be classified at all deposits, with Measured Resources also classified at the Goose Main, Llama, and Umwelt deposits.

Table 1.2: Summary of Estimated Mineral Resources (as of October 21, 2014)

Classification	Tonnes (kt)	Grade (Au (g/t))	Contained Metal (koz Au)
Measured	10,273	5.27	1,740
Indicated	17,969	6.22	3,593
Measured and Indicated	28,242	5.87	5,333
Inferred	7,750	7.43	1,851

Canadian Institute of Mining (CIM) definitions were used for the Mineral Resources.

Ms. D. Nussipakynova, P.Geo. and Dr. A. Fowler, Ph.D., MAusIMM, CP (Geo), both from AMC and Qualified Persons under NI 43-101, take responsibility for the Mineral Resource estimates.

Open pit Mineral Resources are constrained by an optimized pit shell at a gold price of US$1,500 oz. The cut-off grade applied to the open pit resources is 1.0 g/t Au.

The underground cut-off grade is 4.0 g/t Au for all George Mineral Resources (LCPn, LCPs, Loc1, Loc2, GH, and Slave), 3.5 g/t Au for Goose Main, Echo, and Llama, and 4.5 g/t for the Umwelt deposit.

Estimations assumed an exchange rate of C$1.00 to UD$1.00.

The George Mineral Resources were estimated within mineral domains expanded to a minimum horizontal width of 2 m for the underground Mineral Resources.

Drilling results up to December 31, 2013 are included, except for Echo (July 4, 2014) and Loc1 and Loc 2 (July 21, 2014).

George Mineral Resources account for 32% and 53% of Indicated and Inferred gold ounces respectively.

The numbers might not add due to rounding.

Mineral Resources include Mineral Reserves.

Source: AMC Mining Consultants (Canada) Ltd. 2015

1.7 Mineral Reserve Estimate

The Mineral Reserve estimate for the Property is based on the Mineral Resource estimate for the Llama, Umwelt and Goose Main deposits, completed by AMC, with an effective date of October 21, 2014.

The Mineral Reserves were developed by examining each deposit to determine the optimum practical mining method. Cut-off grades were then estimated based on appropriate mine design criteria and the adopted mining method. The mining methods chosen were shovel-and-truck open pit mining at Umwelt, Llama and Goose Main, and underground mining using post pillar cut-and-fill (PPCF) at Umwelt. For the purposes of this FS, no Mineral Reserves can be claimed from the George Site at this time as the George deposits were not part of this study and they are therefore not adequately supported by current economic reserve assumptions.

The estimated Proven and Probable Mineral Reserves total 12.4 Mt at 6.30 g/t Au, containing 2.50 Moz Au (Table 1.3).

Table 1.3: Summary of Estimated Mineral Reserves (as of September 14, 2015)

Area	Classification	Diluted Tonnes (kt)	Diluted Grade (Au (g/t))	Contained Metal (Au (koz))
Total Open Pit	Proven	6,983	5.97	1,340
	Probable	1,885	5.52	335
Total Underground	Proven	20	9.52	6
	Probable	3,471	7.37	822
Total Back River Property	Proven	7,003	5.98	1,346
	Probable	5,356	6.72	1,157

A gold price of US$1,250/oz is assumed.

An exchange rate of C$1.15 to US$1.00 is assumed.

The numbers might not add due to rounding.

Notes for Open Pit:

Dilution and recovery factors are applied as per open pit mining method.

A COG of 2.08 g/t was used for the Umwelt open pit Mineral Reserve estimate.

A COG of 2.14 g/t was used for the Llama open pit Mineral Reserve estimate.

A COG of 2.07 g/t was used for the Goose Main open pit Mineral Reserve estimate.

Notes for Underground:

Dilution and recovery factors are applied as per underground mining method.

A COG of 3.86 g/t was used for the Umwelt underground Mineral Reserve estimate.

Source: JDS 2015

Both the Mineral Resource and Mineral Reserve estimations take into consideration on-site operating costs (e.g., mining, processing, site services, freight, general and administration), geotechnical analysis for both open pit wall angles and underground stope size, metallurgical recoveries, and selling costs. In addition, the Mineral Reserves incorporate allowances for mining recovery and dilution, and overall economic viability.

1.8 Mining Operations

Conventional shovel-and-truck open pits combined with an underground mine are projected to provide the process plant feed at a nominal rate of 3,000 t/d or 1.1 (Mt/a) for a period of 10 years (including the initial pre-production period). Annual mine production of ore and waste is profiled to peak at 13.7 Mt/a from the open pits, with a LOM waste to ore stripping ratio of 10.5. Ore production from underground mining will peak at 0.6 Mt/a and will supplement the feed from the open pits. In order to optimize the Project cash flow, the run of mine ore is planned to be segregated into high, medium, and low-grade stockpiles located adjacent to the processing plant. These stockpiles will also serve to buffer mill processing from mining production. The ore production schedule is shown in Table 1.4.

The mining areas are scheduled to target higher grade material to be delivered from the Umwelt and Llama deposits earlier.

Mining will begin at Goose Site in Year -1 at Umwelt pit to provide waste rock for construction and enable the stockpiling of high grade ore prior to the start of plant processing. Open pit mining will then transition sequentially to the Llama and Goose Main pits. Open pit mining will be completed by Year 8. Underground ore production at Umwelt will begin in Year 3 and will continue until Year 9.

Table 1.4: Run of Mine Ore Production Schedule for Open Pit and Underground Mining

Deposit	Unit	Pre-production	Years	Years	Total
			1 to 5	6 to 10	
Goose Site – Open Pit					
Umwelt	kt	481	2,187	0	2,668
Llama	kt	0	1,749	0	1,749
Goose Main	kt	0	2,043	2,408	4,451
Goose Site – Underground					
Umwelt	kt	0	1,324	2,168	3,492
Overall					
Total Ore Mined	kt	481	7,302	4,575	12,359
Plant Feed	kt	0	5,174	7,185	12,359
Head Grade	g/t Au	0	8.3	4.9	6.3
Recovery	%	0	92.6	93.6	93
Average Annual Recovered Metal	koz	0	274.5	161.5	208.6

Source: JDS 2015

Open pit mining operations will use a fleet comprising 7 m^3 shovels, one 7 m^3 front-end loader, 4 m^3 excavators, and 64 t haul trucks. This fleet will be supplemented by drills, graders, and track and rubber-tire dozers. A 5 m bench height was selected for mining in ore and waste with overall 20 m effective bench heights based on a quadruple-bench configuration.

Underground mining operations will be carried out using PPCF and will use a combination of two-boom jumbos, long-hole production drills, 10 t load-haul-dump (LHD) vehicles, and 30 t trucks.

1.9 Recovery Methods

The 3,000 t/d process plant will be designed to use conventional crushing, grinding, gravity concentration, gold leaching by cyanidation, gold adsorption by carbon-in-pulp (CIP), and gold recovery from loaded carbon and gravity concentrate to produce gold doré. Cyanide destruction of the tailings will be by SO_2. The overall design philosophy uses proven equipment with a simple and conventional single-line process flow that can be operated and maintained effectively in an arctic environment.

The process plant includes the following:

- Three-stage crushing circuit reducing run of mine (ROM) ore to 80% passing (P80) 8.5 mm;
- Fine ore stockpile (feeding the mill) with a live capacity of 3,400 t;
- Grinding and gravity circuit comprising a ball mill (P80 180 µm), a fine grind mill (P80 50 µm), and a single centrifugal gravity concentrator;

- Cyanide leaching and carbon adsorption circuit;
- Carbon stripping and reactivation circuit;
- Gold electrowinning and refining circuit producing bullion; and
- Tailings handling circuit, including cyanide destruction with sodium metabisulphite (SMBS).

1.10 Project Infrastructure

Due to the remoteness of the Property, significant infrastructure is required for freight, power generation, and manpower accommodation. Both the MLA and Goose sites will have bulk fuel storage tanks, laydown yards, diesel power plants, maintenance shops, accommodation camps, water and domestic waste management facilities, and satellite communications. An all-weather airstrip will be located only at the Goose Site. In winter these sites will be connected by a winter ice road. All-weather roads allow for year-round access within each site.

The major infrastructure related to the mining and processing operations at Goose Site includes the process plant, tailings storage facilities, waste rock storage areas (WRSAs), water management drainage and storage ponds, and haul roads and equipment to service the open pit and underground mines. The central administration block will be located at the Goose Site.

The MLA will support the seasonal trans-shipment and staging of construction and operational freight. Because access to the Property is seasonal, the types and capacities of the Project infrastructure need to be able to store and transport the required quantities of equipment, materials, and supplies. Diesel will be received and stored in four 10-ML tanks at the MLA, providing sufficient capacity for peak operating needs of power generation and mobile equipment for one year. Similarly, subsequent years' requirements for consumables, such as processing reagents, maintenance materials, and bulk supplies, will be stored in heated or cold storage warehouses, laydown yards, and sea containers.

The installed power generating capacity will be 15 MW at Goose Site and 1.5 MW at the MLA. Buildings and facilities at the Goose Site will be heated primarily by heat recovered from the power plant. The Umwelt underground mine air will, where required, be heated by a dedicated diesel-fired furnace.

The accommodation complexes will be portable, modular units constructed off-site. The construction phase at the Goose Site will accommodate up to 303 workers. The construction and operation phases at the MLA will require accommodations for up to 94 workers.

The Property is located within the permafrost region; therefore, infrastructure that is particularly sensitive to differential settlement, such as the process plant and fuel storage tanks, will be built on competent bedrock. Less sensitive structures and linear surface elements, such as roads, pipelines, and airstrips, will be built on overburden soils and include an appropriate thermal protection layer.

1.11 Waste Management

1.11.1 Tailings Management

The Project will produce a total of 12.4 Mt / 10.3 Mm^3 of tailings over the LOM. The purpose-built Tailings Storage Facility (TSF), located adjacent to and south of the Goose Main open pit, was designed to contain the first four years of tailings (4.4 Mt / 3.6 Mm^3) behind a frozen foundation dam with an integral liner. The balance (8.0 Mt / 6.7 Mm^3) will be deposited into the mined-out Llama open pit (Llama TF). Ultimately, potential acid generating (PAG) and non-potentially acid generating (NPAG) waste rock will be deposited on the TSF once Goose Main Pit development starts, resulting in a period of co-disposal.

The TSF containment dam will be constructed as a frozen foundation rock-fill dam with a geosynthetic clay liner. The liner will be frozen into the key trench permafrost to seal the upstream slope.

The design and location of the TSF capitalizes on natural topography and its relative proximity to the processing plant. It is outside of the Inuit-owned land which is consistent with the Kitikmeot Inuit Association's (KIA) desire to have no subaerial disposal of tailings on their land. The operational timing and location of the TSF makes progressive reclamation and closure as a tailings and WRSA feasible and desirable.

1.11.2 Waste Rock Management

Over the LOM, a total of 93.1 Mt of mining waste rock would be produced, including unconsolidated overburden. Quarry and waste rock are categorized as being either PAG or NPAG. Geochemical characterization was based on static and humidity cell testing, acid base accounting, and trace element analyses. The acid rock drainage potential was assigned to individual samples on the basis of the neutralization potential/acid generation potential (NP/AP) ratios. The PAG waste rock found on the Property has a slow reaction rate, and will only react in the order of decades.

The tailings have also been characterized as PAG with a similar low reactivity. The NPAG material will leach metals, specifically arsenic, under neutral conditions.

Waste rock will be identified, segregated, and deposited as appropriate during the mining operation. Rock required for constructing pads, roads, and other infrastructure will be sourced from the available NPAG waste rock. The execution plan for Goose is based on sourcing this construction material from the Umwelt pit during the pre-production phase of mining.

Generally, waste rock will be placed in its final location and configuration within WRSA constructed near the source pits. The closure strategy is for the waste rock to freeze; PAG material will be capped with a 5 m thick NPAG cover. This thickness was determined by thermal modelling which estimated an active layer thickness of between 3 and 4 m. The modelling suggested that lift thickness during placement is not critical to ensure timely freeze back (within ten years).

1.12 Water Management

The water management planning covers all phases of the Project from construction through operations to final closure, and accounts for a range of possible climatic and operational conditions. The MLA does not require water management infrastructure beyond best management practices.

Site-wide water and load balances were modelled. These took into account climatic variables, lake dewatering, saline groundwater arising from mining operations, contact water collected from WRSAs and site drainage, and the transfer and treatment requirements of fresh, reclaim, and process water. Various permanent and temporary diversions, holding ponds, and pumping systems will be used to achieve the management objectives throughout the mine life.

Water will be stored in a system of collection ponds depending on water type and timing (see Table 1.5 for Project water management phasing). During construction (Phase 1) Llama Lake will be dewatered and used to store contact water. Contact water will be transferred to the TSF. Intermediate storage ponds will be used in the water management circuits.

Table 1.5: Project Phases as Defined by Tailings and Water Management

Period	Phase	Stage	Project Year	Activity
Construction	1		-3 to -1	Mobilization and Construction
Operations	2	1	1 to 4	Tailings to TSF
		2	4 to 12	Tailings to Llama TF
Closure	3	1	12 to 14	Active Closure
		2	15 to 20	Passive Closure
Post-Closure	4		20 to as required	Monitoring

Source: SRK 2015

Phase 2 will begin with the start of ore processing in Year 1. Tailings will be deposited in the TSF and supernatant water will be reclaimed to the process plant. Saline groundwater will only start to be encountered once Umwelt open pit is completed which will then be used to store saline water. After open pit mining at Llama is complete, it will become a tailings facility (Llama TF) and receive tailings as well as contact water. Similarly, once Goose open pit mining is complete, the pit will become a contact water storage facility. This water will be treated to remove metals before being discharged to Goose Lake as required.

Phase 3 will begin when ore processing ends in Year 12. Treatment of water collected in Goose Main open pit and Llama TF will cease, and ponded water will be treated at the TSF WRSA as part of the closure phase. The TSF WRSA will be frozen by Year 20 and runoff will flow into Goose Main pit and then into Goose Lake. The site will effectively be closed at that point.

1.13 Environmental Studies and Permitting

New and modified mining projects in Nunavut are subject to environmental assessment (EA) and review prior to certification and issuance of permits to authorize construction and operations. The primary environmental review and approval process applicable to the Project is the territorial EA administered by the Nunavut Impact Review Board (NIRB). A Project Certificate, if recommended by NIRB, might be issued by the Minister of Aboriginal Affairs and Northern Development Canada (AANDC) at the conclusion of the EA process. This would represent government approval and allows the Proponent to pursue the necessary regulatory authorizations needed to construct and operate the Project.

In June 2012, Sabina submitted a Project description and various applications to the NIRB, Nunavut Water Board, and AANDC. In January 2014, a Draft Environmental Impact Statement was submitted to the NIRB.

In July 2014, Sabina responded to Project information requests and in October 2014, Sabina responded to agency technical comments. In November 2014, a week-long technical meeting and a pre-hearing conference were held in Cambridge Bay. A Pre-hearing Conference Decision report was produced based on these meetings with the Government of Canada, the Government of Nunavut, the Government of NWT, the KIA, and the general public. This document summarizes Sabina's commitments, and provides further direction for the content of the 2015 Final Environmental Impact Statement (FEIS) which is planned for submission in Q4, 2015.

The design of the Project includes a comprehensive water management plan for construction, operations, and closure. All Project components will be decommissioned and reclaimed according to best industry practices, and territorial and federal regulations. The closure plan uses proven practices that include appropriate long-term management of PAG/metal-leaching materials and any affected waters. The objective of final reclamation for the Project is to return the site to a productive condition after mining activities are completed.

Based on the information available and the proposed design, there appears to be no adverse environmental or socio-economic aspects that could limit the development of the Project.

1.14 Community Sustainability

Sabina is an active member of the Kitikmeot region community with a regional office in Cambridge Bay (established in 2012). Sabina has also actively engaged and consulted local communities through Project planning activities and EA processes. Sabina strives to ensure engagement with all residents of the affected communities and will continue to advance its community engagement program during the EA and permitting process for the Project, and throughout the development and operation of the mine. The results of the programs will be integrated into Sabina's FEIS.

An Inuit Impact Benefit Agreement (IIBA) is required for the Project under the Nunavut Land Claims Agreement as the Property is located on Inuit-Owned Lands. Where possible, Sabina plans to maximize local employment and contracting opportunities and is dedicated to working with community partners on training programs to prepare local residents for employment. Kitikmeot Inuit would be given the first opportunities for Project-related jobs. The total on-site workforce will be 607 people at Goose Site during the operations phase; this excludes drivers and contractors for haulage operations on the winter road. During the most active construction period from Year -2 to Year -1, the on-site workforce will average 202 workers and will peak at 280.

1.15 Capital and Operating Costs

1.15.1 Capital Cost Estimation

The initial capital cost estimate is $415M, as summarized in Table 1.6. Costs are expressed in Canadian dollars with no escalation (Q3-2015 dollars).

Table 1.6: Capital Cost Summary

Capital Cost	Pre-Production $M	Production and Closure Period $M	LOM $M
Mining	46	112	158
On-Site Development	15	1	17
Ore Crushing and Handling	16	0	16
Process Plant	55	0	55
On-Site Infrastructure (Goose)	68	15	83
Off-Site Infrastructure	25	40	65
MLA	26	2	28
Tailings	6	2	8
Indirects	66	0	66
EPCM	30	0	30
WRSA Costs	25	0	25
Contingency	37	13	50
Subtotal	**415**	**185**	**600**
Reclamation	0	64	64
Total Capital	**415**	**249**	**664**

Source: JDS 2015

Preparation of the capital cost estimate is based on the JDS philosophy that emphasizes accuracy over contingency, and uses defined and proven project execution strategies. The estimates were developed using first principles, applying directly-related project experience, and the use of general industry factors. Almost all of the estimates used in this Project were obtained from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in this study.

The initial capital estimates include all pre-production mining activities in Years -3, -2 and -1 and are based on Owner-performed mining. Mobile equipment leases have not been considered in this estimate.

The initial capital estimate is based on the execution plans described in this study. Sunk costs and Owner’s reserve were not considered in the initial capital estimate.

The sustaining capital estimate is based on required capital waste development, mining equipment acquisition and rebuilding, and mining infrastructure installations as defined by the mine plan.

1.15.2 Operating Cost Estimation

The average LOM unit operating cost is estimated at $114.58/t processed and is summarized in Table 1.7.

Table 1.7: Operating Cost Summary

Operating Cost†	Average $M/yr	LOM $M	$/t Processed
Mining*	46	539	43.64
Processing	39	459	37.16
Site Surface	12	137	11.08
Freight	5	55	4.42
G&A	19	226	18.28
Total Operating Costs	121	1,416	114.58

(†): Operating Costs include $47.3M of working capital claimed in the pre-production period and excludes pre-stripping costs.

(*): Average LOM open pit Mining Cost amounts to $3.35/t mined which includes a 10.5:1 strip ratio; average LOM underground Mining Cost amounts to $63.61/t mined.

Source: JDS 2015

The following list summarizes the key Project assumptions used to develop the operating cost estimate:

- Mining operations will be performed by the Owner using Owner-purchased equipment;
- All electrical power will be generated at site using diesel generators with a long-term delivered (to MLA) diesel price of $0.91/L for power generation, and $0.95/L for mobile equipment, yielding an estimated LOM power cost of $0.26/kWh;
- The process plant will process 3,000 t/d (~1.1 Mt/a) of ore; and
- The mine will use a peak total workforce of approximately 607 people, including all contract labour.
- A long-term diesel price of $2.40 / gallon has been applied to the appropriate project operating costs. This price is closer to the long-term historical diesel price and considered more realistic for the duration of the operation. The indicator diesel price as of September 2015 was $1.42 / gallon.

1.16 Project Execution and Development

The project execution plan and general project development schedule consider the seasonality of transporting freight. The procurement and staging of equipment, materials, and fuel at the respective east and west coast ports needs to take place at least 8-12 months before anticipated arrival at the Goose/George sites. The MLA is planned to receive sealift materials in the summer open-water period of August and September. Materials would then be stored until the winter ice road is operational from between January and April. Fixed-wing aircraft landing at Goose Site will support construction and operations activities by delivering passengers and select bulk materials.

1.16.1 Project Execution Schedule

The Project execution schedule includes the following key milestones:

Engineering, environmental approvals, permitting:

• Feasibility Study completion	Q3/Year-5
• FEIS submission	Q4/Year-5
• Basic and detailed engineering	Q4/Year-5
• Project certification	Q2/Year-4
• Type A water licence	Q2/Year-3

Site preparation and pre-construction (Year 1 is first year of full production):

• Initial sealift, mobilization, long lead procurement	Q1-Q2/Year-3
• Construct facilities at MLA	Q3/Year-3 to Q4/Year-3
• Initial winter ice road from MLA to Goose	Q4/Year-3 to Q1/Year-2
• Install construction camp at Goose Site	Q1-Q2/Year-2

- Goose Site: construct first fuel tank — Q2/Year-2

Construction and commissioning at Goose Site:

- Commence open pit mining and TSF construction — Q2/Year-2
- Construct site infrastructure — Q1/Year-2 to Q4/Year-1
- Commission process plant — Q4/Year-1 to Q1/Year 1
- First gold — Q1/Year 1

1.17 Economic Analysis

An engineering economic model was developed to estimate the project value and investment return. Pre-tax estimates of project values were prepared for comparative purposes, and after-tax estimates were developed to better indicate the true investment value. Sensitivity analyses reflecting variations in metal prices, grades, operating costs, and capital costs were performed to determine their relative importance as project value drivers.

This technical report contains forward-looking information resulting from projected mine production rates and resulting forecasted cash flows. The gold grades are based on sufficient sampling that is expected to be reasonably representative of the realized gold grades from actual mining operations.

The following factors could affect the results and cause actual results to differ materially from those presented in this economic analysis:

- Ability to obtain permits so that construction and operations can proceed as planned;
- Ability to secure major equipment and skilled labour; and
- Ability to achieve assumed mine production rates at the assumed grade.

Other economic factors include the following:

- Discount rate of 5% (sensitivities using other discount rates have been calculated for each scenario);
- Costs based on nominal 2015 dollar values;
- No application of inflation values;
- Values are presented on a 100% Ownership basis and do not include management fees or financing costs;
- Exclusion of all pre-development and sunk costs (i.e., exploration and resource definition costs, engineering field work and studies costs, environmental baseline study costs, etc.). Note: pre-development and sunk costs are used in tax calculations;
- Gold price of US$1,150/oz;
- Includes estimated third-party net smelter royalties;
- US$:C$ exchange rate of 0.80;

- Northwest Territories and Nunavut Mineral Royalties (NTNMR) have been estimated and are included with income taxes. The Crown royalty is levied on a mine-by-mine basis and is equal to the lessor of 8% of the net value of mine output during a fiscal year, and an escalating rate from 0% to 14% on incremental levels of net value of the mine output during a fiscal year. NTNMR are deductible from income taxes;
- The Back River Mineral Resources considered in this study are grandfathered properties subject to royalties under the NTNMR;
- Federal tax rate of 15% and a NWT 12% rate were used to estimate future income taxes;
- Canadian Exploration Expense (CEE) and Canadian Development Expense (CDE) tax pools were used with appropriate opening balances to calculate income taxes; and
- Specific capital cost class Capital Cost Allowance (CCA) rates were applied and used to calculate the appropriate CCA the Company can claim during the entire life of the Project.

Pre-tax and after-tax financial performance is summarized in Table 1.8. Pre-tax results provide a point of comparison with similar projects and are not intended to represent a measure of absolute economic value.

Table 1.8: Summary of Economic Results

Category	Unit	Value
Net Revenues	$M	3,202
Operating Costs	$M	1,369
Cash Flow from Operations	$M	1,833
Capital Costs*	$M	664
Cash Cost‡	US$/oz	534
Cash Cost (Incl. Sustaining Capital) º	US$/oz	598
Net Pre-Tax Cash Flow	$M	1,122
Pre-Tax $NPV_{5\%}$	$M	699
Pre-Tax IRR	%	28.2
Pre-Tax Payback	Years	2.9
Break-Even Pre-Tax Gold Price ($NPV_{5\%}$ = 0)	US$/oz	794
Total Taxes	$M	340
Net After-Tax $NPV_{5\%}$	$M	480
After-Tax IRR	%	24.2
After-Tax Payback	Years	2.9
Break-Even After-Tax Gold Price ($NPV_{5\%}$ = 0)	US$/oz	795

(*): Includes pre-production, sustaining, closure and reclamation capital costs

(‡): (Refining Costs + Insurance + Transport Costs + Third Party Royalties + Operating Costs) / Payable Au oz

(º): (Refining Costs + Insurance + Transport Costs + Third Party Royalties + Operating Costs + Sustaining Capital Costs) / Payable Au oz

Source: JDS 2015

A sensitivity analysis was conducted on after-tax net present values ($NPV_{5\%}$) for individual parameters, including the gold price, foreign exchange rate, operating costs, and capital costs. The results are shown in Table 1.9. The Project proved to be most sensitive to changes in the US$:C$ exchange rate, gold price and head grade. The Project showed least sensitivity to operating and capital costs.

The Project was also evaluated using various discount rates to determine the effect on Project NPV. The Project NPV declines as the discount rate increases.

Table 1.9: After-Tax $NPV_{5\%}$ Sensitivity Results

Factor	After-Tax $NPV_{5\%}$ ($M)						
	-15%	-10%	-5%	100%	5%	10%	15%
Metal Price	250	328	404	480	555	631	706
F/X Rate	736	641	556	480	410	347	288
Head Grade	261	336	408	480	552	623	695
OPEX	572	542	511	480	449	418	387
CAPEX	564	536	508	480	452	425	397

Source: JDS 2015

Table 1.10: After-Tax IRR Sensitivity Results

Factor	After-Tax IRR						
	-15%	-10%	-5%	100%	5%	10%	15%
Metal Price	15.90%	18.90%	21.60%	24.20%	26.70%	29.10%	31.40%
F/X Rate	32.30%	29.40%	26.70%	24.20%	21.80%	19.60%	17.30%
Head Grade	16.30%	19.10%	21.70%	24.20%	26.60%	28.90%	31.10%
OPEX	27.10%	26.20%	25.20%	24.20%	23.20%	22.10%	21.10%
CAPEX	29.80%	27.80%	25.90%	24.20%	22.60%	21.10%	19.70%

Source: JDS 2015

1.18 Mine Closure

Mine closure activities will take place immediately following operations and are expected to span six years followed by a post-closure monitoring period of approximately five years but as long as required. Closure was a key consideration in the design of the Project: progressive reclamation mitigates long-term risks and reduces overall costs. This can be achieved by using staff and equipment effectively during operations and avoiding the double-handling of waste rock.

Activities that will be initiated and/or completed during operations include the off-site backhaul of hazardous or recyclable materials and equipment, the capping of the TSF with waste rock, and the backfilling of open pits and underground workings as they become available.

The first two years of active closure after operations will involve the demolition and disposal of structures and equipment that will no longer be used. Open pits and underground mines will be allowed to flood while there will be some selective backfilling underground. The WRSAs and landfills will be covered with 5 m of NPAG and shaped to minimize erosion and maintenance. Roads, pads, and airstrips will be maintained as required, and the natural drainage will be restored as the infrastructure becomes obsolete.

Water management will continue to progress beyond the operational phases with the relocation of water treatment plants, pumps, and piping. As equipment and materials become obsolete, they will also be landfilled. Similarly, water diversion and retention structures will be breached as active closure transitions into passive closure. Finally, the monitoring phase will ensure that all closure objectives are met.

1.19 Interpretation and Conclusions

Based on the findings of the Initial Project Feasibility Study, it can be concluded that the Back River Gold Project will be economically viable under the base case financial parameters.

1.20 Recommendations

It is recommended that the project be advanced to construction through the normal process of permit acquisition, financing detailed engineering and construction. Costs for engineering and construction are included in the capital cost of this study.

2 Introduction

The Back River Gold Project (Project or Property) is an advanced-stage gold exploration Project, located in southwestern Nunavut, Canada. It is located approximately 520 km northeast of Yellowknife, NWT; 50 km southeast of Glencore Canada Corporation's Hackett River Silver-Zinc Project; 285 km south of TMAC Resource Inc.'s Hope Bay Project (Doris); and 95 km southwest of Bathurst Inlet. Sabina Gold & Silver Corp. (Sabina) owns 100% of the Project.

This Initial Project Feasibility Study technical report was prepared for Sabina Gold & Silver Corp. This report was prepared to present a lower throughput, smaller capital cost option for the project that better reflects financing likelihood when compared with the June 2015 JDS report titled "Technical Report and Feasibility Study for the Back River Gold Property, Nunavut", with an effective date of May 20, 2015, also referred within this report as the 6KFS.

The Mineral resource estimate used for this Initial Project Feasibility Study and report was by AMC Mining Consultants Ltd. (AMC) with an effective date of October 21, 2014 as reported herein. The Mineral Resource statement was disclosed publicly by Sabina in a news release on May 20, 2015 for the June 2015 JDS 6KFS report, and is the same as disclosed with the news release on September 14, 2015 for this Initial Project Feasibility Study.

This report was prepared following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1. The Mineral Resource and Mineral Reserve statements reported herein were prepared in conformity with generally accepted CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November, 2003).

2.1 Qualifications and Responsibilities

The results of this technical report do not depend on any prior agreements concerning the arrived-at conclusions, and there are no undisclosed understandings with regard to any future business dealings between Sabina and the Qualified Persons (QPs). The QPs are paid a fee for their work in accordance with normal professional consulting practices.

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing with appropriate professional institutions or associations. The QPs are solely responsible for their specific report sections as shown in Table 2.1.

Table 2.1: QP Responsibilities

Qualified Person, Designation	Company	Years of Relevant Experience	Relevant Experience	QP Responsibility/Role	Report Section(s)
Gordon Doerksen, P.Eng.	JDS	30+	Gord Doerksen is a professional mining engineer with over +20 years of experience on operating mines in Canada, the United States and Zambia and over 10 years of experience as a consultant for mining projects located throughout the world. Gord has led or contributed to dozens of engineering studies, due diligence reviews and Technical Reports. The majority of Gord's experience is with gold and base metal projects but he also has significant coal, trona and aggregate experience. Gord has experience in gold leach plant operations in the north.	Executive Summary, Introduction, Reliance on Other Experts, Mineral Reserves, Mining Methods, Infrastructure, Market Studies & Contracts, Environmental Studies, CAPEX & OPEX, Economic Analysis, Other Relevant Data, Interpretations & Conclusions, and Recommendations	1 to 3, 15, 16, 18 to 23, 25 to 30, except for subsection responsibilities assumed by other QPs
Dino Pilotto, P.Eng.	JDS	20+	Dino Pilotto is a professional mining engineer with over 20 years of production operations and mining engineering experience. He has worked in both open pit and underground mines, including coal and gold. He has previous consulting experience both in mining and in a civil engineering environment. Dino's technical strengths are based on actual operational experience and include mine design, pit optimization and scheduling, equipment selection and economic analysis and practical solutions to operational problems.	OP Reserves, OP Mining Methods, Overall Production Schedule	15.2, 16.2, 21.1.3 and 22.2.1
Andrew Fowler, MAusIMM, CP (Geo)	AMC	11	Andrew Fowler is a professional geologist with 11 years industry experience. He has a structural geology based PhD and has worked as an exploration geologist, underground mine geologist and mineral resource geologist. He has been a consultant for 6 years, estimating many different types of deposits. Andrew's technical strengths, relevant to this report are mineral resource estimate skills.	Executive Summary, Mineral Resource Estimate	1.6, 14
Dinara Nussipakynova, P.Geo	AMC	28	Dinara Nussipakynova is a professional geologist with 28 years of experience in multiple working environments. She has worked in both mineral exploration and mine geology, and specifically Mineral Resource estimation. Dinara's technical strengths are database management, geological interpretation and Mineral Resource estimation and reporting.	Executive Summary, Mineral Resource Estimate	1.6, 14
John Morton Shannon, P.Geo	AMC	40+	Mort Shannon is a professional geologist with over 40 years of experience in mine geology, mineral exploration, and property evaluation. He has worked in base metal and gold mines in Ireland, Zambia, Canada and Papua New Guinea, and has been Chief Geologist on two very large gold mines, in different political and geological environments. He has led or authored parts of multiple Technical Reports.	Executive Summary, Property Description, Accessibility, History, Geology, Deposits, Exploration, Drilling, Sample Preparation, Data Verification, Adjacent Properties	1.3-1.5, 4 to 12, 24
Maritz Rykaart, P.Eng.	SRK	20+	Maritz Rykaart is graduate of the Rand Afrikaans University in 1991 and 1993 with B.Eng. (Civil) and M.Eng. (Civil) degrees respectively. In 2001, he graduated with a PhD (Geotechnical Engineering) from the University of Saskatchewan. He has practiced his profession continuously since graduation in 1993, except for a 3-year break from 1998 to 2001 to complete his PhD. His work experience is related almost entirely to the mining industry, specifically the design, construction, monitoring and closure of mine waste facilities including tailings impoundments.	Geochemistry, Tailings Management, Water Management	1.11, 1.12, 1.19, 18.1.2, 18.1.3, 18.1.4, 18.1.5, 18.4, 18.5, 20.6
Stacy Freudigmann, P.Eng	Canenco	10	Stacy Freudigmann has successfully managed process design consultants, covering the full range of throughputs and developing the process flowsheets and costs associated with the construction and operation of those processing facilities. In the 6KFS, he worked with the Hatch process team and had input into design criteria and parameters on which Hatch developed the process. In the 3KFS, he managed the JDS and Hatch processing and engineering teams for Sabina to develop the process design and associated costs. Stacy continues to manage metallurgical programs and is well-known in the industry as being able to solve "tricky" metallurgical issues, creating value add with recovery improvements and process optimizations.	Metallurgy, Recoveries, Process	13, 17
Robert Mercer, P.Eng	KP	25+	Rob Mercer is a graduate of Queen's University with a PhD in Mining Engineering – Rock Mechanics (1999). He is a Professional Engineer with over 25 years of rock mechanics experience. His recent work ranges from managing geomechanical site investigation programs to providing ongoing rock mechanics support to operating underground and open pit mines. He has worked on over 60 mining and civil projects world-wide and is a licensed Professional Engineer in Ontario, Nunavut and Newfoundland & Labrador. He also has degrees in both Geological and Mining Engineering.	Geomechanical	Provided geomechanical input to JDS on mining section

Source: JDS 2015

2.2 Site Visit

In accordance with National Instrument 43-101 guidelines, Table 2.2 shows the site visit details for the QPs. All QP site visits were led by Wes Carson, Sabina Vice President of Project Development, Jeff Eng, Sabina Director of Engineering and other Sabina employees.

Table 2.2: Site Visit Description

Qualified Person	Company	Date	Accompanied by	Description of Inspection
Gordon Doerksen, P.Eng.	JDS	June 16 & 17,2014	Wes Carson, et. al., Sabina	Helicopter and ground inspection of George and Goose sites. Core review.
Dino Pilotto, P.Eng.	JDS	June 16 & 17,2014	Wes Carson, et. al., Sabina	Helicopter and ground inspection of George and Goose sites. Core review.
Andrew Fowler, MAusIMM, CP(Geo)	AMC	None		Relied on John Morton Shannon for site visit information.
Dinara Nussipakynova, P.Geo	AMC	None		Relied on John Morton Shannon for site visit information.
John Morton Shannon, P.Geo	AMC	Aug, 27-28, 2012	Wes Carson, et. al., Sabina	Detailed site and core review.
Maritz Rykaart, P.Eng.	SRK	June 16 & 17,2014 and September 2014	Wes Carson, et. al., Sabina	Helicopter and ground inspection of George, Goose and MLA sites.
Stacy Freudigmann, P.Eng.	Canenco	None		Lead the 2013/2014 metallurgical test programs.
Robert Mercer, P.Eng	KP	Aug, 27-28, 2012 and September 2013	Wes Carson, et. al., Sabina	Detailed site and core review.

Source: JDS 2015

2.3 Units, Currency and Rounding

The units of measure used in this report are as per the International System of Units (SI) or “metric” except for Imperial units that are commonly used in industry (e.g., ounces (oz.) and pounds (lb.) for the mass of precious and base metals).

All dollar figures quoted in this report refer to Canadian dollars (C$ or $) unless otherwise noted.

Frequently used abbreviations and acronyms can be found in section 29.

This report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a small margin of error. Where these occur, the QPs do not consider them to be material.

2.4 Sources of Information

This report is based on information collected by JDS and individual QPs during site visits, discussions with Sabina personnel, public information and additional information provided by Sabina. The QPs have no reason to doubt the reliability of the information provided by Sabina.

3 Reliance on Other Experts

The QP's opinions contained herein are based on public and private information provided by Sabina and others throughout the course of the study. The authors have carried out due diligence reviews of the information provided to them by Sabina and others for preparation of this report. The authors are satisfied that the information was accurate at the time of writing and that the interpretations and opinions expressed are reasonable, and are based on a current understanding of the mining and processing techniques and costs, economics, mineralization processes and the host geologic setting. The authors have made reasonable efforts to verify the accuracy of the data relied on for this report.

Non-QP specialists who were relied on for specific advice include the following:

- PricewaterhouseCoopers Canada (PwC) for taxation advice; and
- Sabina and ERM Rescan for permitting, socio-economic and environmental considerations.

A summary listing of otherwise unidentified experts associated with areas of contributions to the Report and their qualifications is presented in Table 3.1.

Table 3.1: Other Expert Contributors

Name	Discipline	Years of Relevant Experience	Relevant Experience
QP: Gord Doerksen – Surface Infrastructure			
John Schabert	Logistics	20+	Logistics during construction at Ekati Mine, Diavik Diamond Mine, De Beers Snap Lake, De Beers Gahcho Kue.
Trevor Herd	Infrastructure / Logistics	20	Construction Ekati Mine, Diavik Diamond Mine, Construction Manager (earthworks) Tahoe Resources.
QP: Dino Pilotto – Open Pit Mining			
Jim Robertson	OP Mining	30+	Geological Engineer with over 30 years of experience related to mining and civil engineering projects, involved in projects for mine designs, quarries, tailings dams, buildings and bridges. Accomplished user of Gemcom and Whittle pit optimization software with experience including exploration database management, block modelling, 3D solids modelling, open pit optimization, reserve calculation, computerized mine design and scheduling components of scoping studies, preliminary feasibility studies, feasibility studies and audits.
Tysen Hantelmann	OP Mining	15+	Mine Engineer with underground/open pit mine planning and design using Surpac, MineSight, and Whittle software; Long range underground mine planning/scheduling; Block model manipulation and high-level estimation; Project and resource evaluation; High-level economic reviews; Open pit optimization and detailed mine design
QP: Gord Doerksen – Underground Mining			
Harald Goetz	UG Mining	25	JDS Energy & Mining (Engineering Manager) 2013-Present McArthur River (Various Roles) 2005-2013, BHPBilliton Ekati (Various Roles) 1998-2005 Uranerz Exploration and Mining Ltd (Senior Eng.) 1995-1998 Cameco Rabbit Lake (Ventilation Engineer) 1993-1995 Uranerz Exploration and Mining Ltd (Mining Engineer) 1990-1993
Garrett Whipp	UG Mining	8	JDS Energy & Mining (Mine Engineer) 2014-Present Project Mine Engineer at Tetra Tech 2011-2014, Maptek (Various Roles) 2007-2011.
Guy Hanbury	UG Mining	7	JDS Energy & Mining (Mine EIT) 2014-Present BHPBilliton Olympic Dam (Various Roles) 2008 – 2013
Dariusz Holod	Ventilation	29	Claude Resources 2013-present, Sr. Projects Superintendent. Stantec Consulting (McIntosh Eng) 2001-2003 and 2008-2013, Project Manager. Rio Tinto - Northparkes mine (NSW), 2007-2008, Sr. Mine Engineer. BHP Billiton - Ekati mine, 2003-2007, U/G Projects and Ventilation Eng.. JS Redpath Ltd. 1994-1997 and 2000-2002, various roles. Hurley Ventilation Tech., 1997-2000, Ventilation Eng.. KGHM, mine construction div. 1989-1991, Assist. Project Manager. "PIAST" colliery, 1983-1989, various roles.
QP: Stacy Freudigmann – Metallurgy, Recovery, and Process			
Kelvin Lee	Mineral Processing	6	Kelvin is a Professional Process Engineer with experience in research, field, operations support, construction, and design experience at Hatch. The type of work performed included: development of process flow diagrams and Piping and Instrumentation Diagrams (P&IDs), development of process design criteria, major equipment sizing, managing equipment lists, calculating operating costs, and development financial models.
QP: Maritz Rykaart – Geotechnical, Water, and Waste Management			
Kelly Sexsmith	Geochemistry	25	Kelly provides expertise in the characterization and prediction of acid rock drainage and metal leaching for various new, developed, and closed mining properties. Experience includes design and supervision of geochemical test programs, development of conceptual waste management plans, prediction of water quality from mine components, and mine waste management.
Tom Sharp	Mine Water Management	20	Tom specializes in mine water management, treatment and chemistry working as an operator and technical consultant. He has prepared, designed and evaluated water treatment alternatives, site water and load balances, and water management plans for mine planning, operation and closure.
Sarah Portelance	Hydrotechnical Engineering	6	Sarah specializes in hydrology, hydraulic and water balance modelling. Sarah has experience in baseline hydrology and hydraulic analyses for diversions, culvert and floodplain modelling.
Greg Fagerlund	Hydrogeology	10	Greg experience is focused on groundwater and geotechnical investigations for both underground and open pit mines, and/or managed groundwater components for projects at all levels of engineering studies. He specializes in hydrogeologic data analyses, and developed a solid experience with GIS and groundwater flow models, and has designed several finite element numerical models.
Samantha Barnes	Hydrotechnical Engineering	4	Samantha specializes in hydrology, hydraulics, water and load balance modelling and water management planning. Samantha has performed extensive hydrological analyses throughout North America in the design of mine water management systems and general storm water master planning and permitting.
QPs: Gord Doerksen and Stacy Freudigmann – Plant and Infrastructure Design, Quantity Estimates and Selected Capital Costs			
Esra Akinci	Structural	22	Concrete and structural steel analysis and design.
Charles Harmsen	Mechanical	24	Mechanical engineer active in conceptual design, detailed design, preparation of contract drawings and specifications, project/construction management, site inspection, equipment testing and commissioning for industrial, mining, water and wastewater treatment projects.
Marc Tietz	Electrical	11	Marc is an electrical engineer with extensive experience in the design, engineering, construction, commissioning and operation of mining and mineral processing facilities. He has dealt with budgeting and feasibility studies all the way through to procurement, calculations, design and construction documents. He has a solid technical background relating to the implementation of power systems, including distribution design, protection / coordination schemes, motor control, grid reliability, and lighting systems.
Brent Robinson	Process Plant Layout	5	Brent is a mechanical designer with experience in plant layout for mining and mineral processing projects. Types of projects include conceptual design, basic engineering, detailed design and construction. His roles include preparation of plant layouts in 2 and 3 dimensional space, preparing mechanical take-off quantities, and drafting general arrangement and detail drawings. He also has spent time in the field in both a construction support and brownfield design role.

Source: JDS 2015

The QPs used their experience to determine if the information from previous reports and non-QP specialists was suitable to include in this technical report and updated or adjusted information, as required.

The results and opinions expressed in this report are conditional on the aforementioned information being current, accurate, and complete as of the date of this report, with the understanding that no information has been withheld that would materially affect the conclusions made herein. The authors reserve the right to, but will not be obliged to, revise this report and conclusions if any additional information becomes known to them subsequent to the date of this report.

Neither JDS nor the authors of this technical report are qualified to provide extensive comment on legal issues associated with the Property. As such, portions of section 4 (i.e., mineral tenures and licences, title and interest in the Back River Property, royalties, back-in rights, payments or other agreements and encumbrances to which the Property may be subject) are descriptive in nature and are provided exclusive of a legal opinion.

4 Property Description and Location

4.1 Location

The Property is located in the southwestern part of Nunavut Territory, Canada (107°W longitude and 65°N latitude), as shown in Figure 4.1. It is situated approximately 520 km northeast of Yellowknife, Northwest Territories. Nearby mining operations include the closed Lupin Mine, which is currently owned by Mandalay Resources Corporation, 225 km to the west; Glencore Canada Corporation's Hackett River Silver-Zinc Project, 50 km to the northwest; and TMAC's Hope Bay Project (Doris), 285 km to the north. The Property covers approximately 133,470 acres or 54,040 hectares.

Figure 4.1: Location Map of the Back River Property

Source: Sabina Gold & Silver Corp. 2015

4.2 Mining Rights in Nunavut

Nunavut mining and exploration activities are regulated by Aboriginal Affairs and Northern Development Canada (AANDC). This federal department ensures compliance with the Canada Mining Regulations across the territory. There are three main types of mineral interests under the Canada Mining Regulations: a mineral claim, a prospecting permit, and a mineral lease, also referred to as mining lease.

Under the Nunavut Land Claim Agreement (NLCA) enacted in 1993, the mineral rights for about 2% of the territory have been entrusted to the Inuit. The Designated Inuit Organization (DIO) under the NLCA is Nunavut Tunngavik Inc. (NTI); it negotiates terms and conditions for those blocks that are not under federal jurisdiction. None of the deposits considered in this study fall into the classification of Inuit-Owned Lands (IOL) subsurface rights.

The Property comprises 45 federal mining leases and 19 federal mineral claims covering approximately 133,470 acres or 54,040 hectares. The Property is divided into two sites: Goose and George. There are also four exploration prospects: Boot, Boulder, Del, and Bath. All of the tenure is in good standing; a description of the tenure type, size, and ownership is shown in Table 4.1.

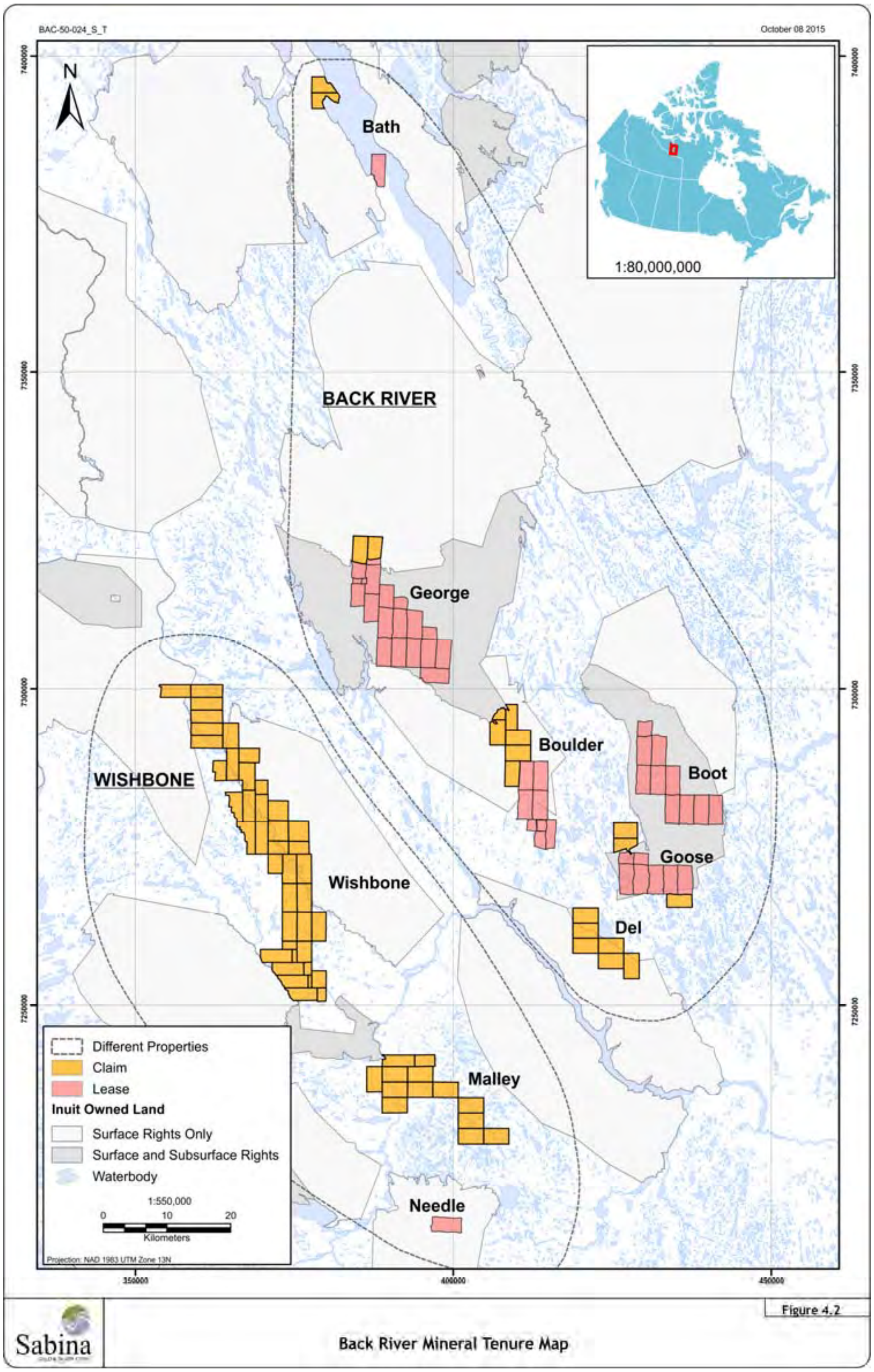
Table 4.1: Land Status (as of September 23, 2015)

Project/ Prospects	Tenure Name	Hectares (ha)	Tenure Type	Registered Ownership as of September23rd, 2015	Expiry / Renewal Date
Goose	3694	417.92	Federal Mining Leases (7)	100% in good standing	16-Oct-2016
	3695	410.27			16-Oct-2016
	3696	1077.71			16-Oct-2016
	3697	1101.80			16-Oct-2016
	3698	1073.66			16-Oct-2016
	3699	1004.00			16-Oct-2016
	3700	1084.59			16-Oct-2016
	K12025	920.36	Federal Mineral Claims (3)	100% in good standing	19-May-2017
	K12026	662.42			19-May-2017
	F94558	800.69			9-Sep-2016
George	3562	69.48	Federal Mining Leases (19)	100% in good standing	9-Nov-2015
	3598	394.16			28-Dec-2015
	3599	821.11			28-Dec-2015
	3600	1008.88			28-Dec-2015
	3601	1097.91			28-Dec-2015
	3602	1027.90			28-Dec-2015
	3603	1078.08			28-Dec-2015
	3604	450.01			28-Dec-2015
	3605	1036.81			19-Dec-2015
	3606	1074.04			19-Dec-2015
	3607	1033.97			19-Dec-2015
	3608	1057.61			19-Dec-2015
	3649	1046.92			19-Dec-2015
	3650	200.08			28-Dec-2015
	3651	1042.07			28-Dec-2015
	3653	1074.85			19-Dec-2015
	3677	536.53			16-Oct-2016
	3729	111.01			16-Oct-2016
	3730	749.88			16-Oct-2016
	F98491	998.04	Federal Mineral Claims (2)	100%in good standing	25-Nov-2015
	F98492	888.29			25-Nov-2015
Boot	3552	1,029.92	Federal Mining Leases (10)	100% in good standing	30-Dec-2015
	3553	1,036.80			30-Dec-2015
	3554	1,093.50			30-Dec-2015
	3555	1,015.17			30-Dec-2015
	3609	1,082.16			30-Dec-2015
	3612	1,080.54			30-Dec-2015
	3613	1,025.06			30-Dec-2015
	3678	1,061.51			16-Oct-2016
	3679	1,002.38			16-Oct-2016
	3724	541.89			16-Oct-2016
Boulder	3466	300.51	Federal Mining Leases (8)	100% in good standing	18-Nov-2015
	3557	1,012.91			30-Dec-2015
	3558	1,052.19			30-Dec-2015
	3559	1,049.36			30-Dec-2015
	3560	1,100.39			30-Dec-2015
	3691	260.01			16-Oct-2016
	3692	456.84			16-Oct-2016
	3693	671.09			16-Oct-2016
	K12027	903.96	Federal Mineral Claims (6)	100% in good standing	4-Oct-2022
	K12028	1,008.86			4-Oct-2022
	K12029	949.73			4-Oct-2022
	K12030	938.79			4-Oct-2022
	K12033	290.79			4-Oct-2022
	K12034	734.27			4-Oct-2022
Bath	5152	983.1375	Federal Mining Lease (1)	100% in good standing	10-Mar-2016
	F94554	650	Federal Mineral Claims (2)	100% in good standing	9-Sep-2016
	F94555	550			9-Sep-2016
Del	K10862	966.74	Federal Mineral Claims (6)	100% in good standing	12-Sep-2018
	K10863	966.74			12-Sep-2018
	K10866	966.74			12-Sep-2018
	K10867	966.74			12-Sep-2018
	K10869	965.52			12-Sep-2018
	K10870	976.46			12-Sep-2018

Source: Sabina Gold & Silver Corp. 2015

Figure 4.2 shows Sabina's claim and lease map of the Property as well as the adjacent Wishbone Property.

Figure 4.2: Sabina Mineral Claim and Lease Map



Source: Sabina Gold & Silver Corp. 2015

4.3 Mineral Tenure

There are six claim groups included in the Property. These are a mix of federal mining leases and federal mineral claims, as shown in Table 4.1 and Figure 4.2. The mining leases have been surveyed by a registered Canadian land surveyor and do not require filing of annual assessment work. However, an annual fee of C$1.00 per acre is required to maintain the existing leases in good standing for the duration of the initial 21-year lease period. As a result of the new Nunavut Mining Regulations, payments increase to C$5.00 per hectare for subsequent 21-year lease renewal periods. The mineral claims have been surveyed and are marked with pickets along claim boundaries and claim posts at the corners of the claims. Also as the result of the new legislation, exploration assessment work, at the rate of C$5.00 per hectare, is required annually on the anniversary of the claim staking to maintain these claims.

All leases and claims are 100% owned by Sabina, and are currently in good standing. Annual reports were delivered to the Kitikmeot Inuit Association (KIA), AANDC, the Nunavut Impact Review Board (NIRB), and the Nunavut Water Board (NWB) as per the terms and conditions of authorizations issued for work done on the Property and on the Wishbone Property.

4.4 Underlying Agreements

The Property is subject to NSR royalties that are payable to various third parties. In 2011, Sabina completed the purchase of some of these royalties on the Property and on the Wishbone Property. The remaining NSR royalties that would apply to the Goose, George, Boulder, Boot, Bath, and Del projects/prospects are:

- Goose Site: 0.7% NSR payable on the first 400,000 oz of gold production, increasing to 4.25% on gold production over 400,000 oz; and
- George Site: 1.15% NSR on the first 800,000 oz of gold production, increasing to 4.25% on gold production over 800,000 oz.

In addition to the described private royalties, under the Northwest Territories and Nunavut Mining Regulations (NTNMR), an annual royalty of up to 13% of the net value of mine production is payable to the federal government for any mine production on mining leases or claims held prior to the NLCA. Under NTNMR, the royalty is based on defined profits multiplied by a royalty rate, which is the lesser of 13% of the net value of mine output during a fiscal year, and an escalating rate from 0 to 14% on incremental levels of the net value of the mine output. The output value is generally the profits from both mining and processing operations, with the deduction of a processing allowance, and deductions for capital and development. The calculations of royalties under NTNMR are not subject to the rules in the *Income Tax Act*; however, any royalties paid are deductible for income tax purposes under the Canadian *Income Tax Act*.

All royalties on the Property are discussed in this section; however, gold production and associated royalties for the George Site are not included in the mine plan or financial model for this Initial Project Feasibility Study. Third-party royalties are estimated to average 3.6% over the life of the mine for the Goose Site and are included in the economic model in this study.

4.5 Permits and Authorizations

Surface rights for IOL are vested in the KIA, which administers the access and management of the lands for the benefit of the Inuit of that region. Access to and use of surface lands requires an Inuit Land Use permit, licence, or commercial lease issued by the KIA. The Mineral Resources used in this study do not have IOL subsurface rights; however, it is overlain by IOL surface lands.

Surface rights on Crown Land are vested in the federal government and in the AANDC. Access to and use of these surface lands requires a land use permit, licence, or commercial lease issued by the AANDC.

Use of water resources in Nunavut requires a Nunavut Water Use permit from the NWB. The NWB and the NIRB are responsible for reviewing and issuing the water licences.

Payments of security deposits are required for water use associated with underground development and exploration, or overburden stripping.

All permits listed in Table 4.3 require an annual work proposal to be submitted to the issuing agency. Leases, claims, and all other permits and authorizations are in good standing and valid with an extension pending for permit N2011F0029 and a renewal for permit 2BEGOO1015.

In accordance with NLCA, at least 180 days prior to the start of major development, negotiations should commence for the purpose of concluding an Inuit Impact and Benefit Agreement (IIBA).

Table 4.3: Summary of Land and Water Access Permits (as of September 30, 2015)

Permit	Expiry (mm/dd/yyyy)	Agency	Description
KTL204C012- Amended	12/13/2015	KIA	Boulder: Staking/prospecting, exploration (ground/air geophysics), geophysical survey, gridding and drilling
KTL204C020- Amended	12/13/2015	KIA	Boot: Exploration (air/ground geophysics), staking, prospecting, fly/survival camp and drilling
KTL304C017- Amended	12/13/2015	KIA	Goose: Staking/prospecting, exploration (ground/air geophysics), drilling, bulk sampling, bulk fuel storage, camp, winter road
KTL304C018 - Amended	12/13/2015	KIA	George: Staking/prospecting, exploration (ground/air geophysics), drilling, bulk sampling, bulk fuel storage, camp, winter road
KTL304F049 - Amended	12/13/2015	KIA	Winter road Bathurst Inlet to Goose and George
KTP11Q001	12/13/2015	KIA	Goose rock quarry
KTP12Q001	12/13/2015	KIA	Goose airstrip borrow area
KTP12Q002	12/13/2015	KIA	George borrow quarry
N2011F0029	12/13/2015	AANDC	Winter road connecting George-Goose
N2010F0017	9/16/2015 (extension pending)	AANDC	Winter road connecting Bathurst Inlet - Back River Project
N2010C0016	10/31/2015 (extension pending)	AANDC	Exploration activities
2BEGOO1520	02/18/2020	NWB	Goose water licence
2BEGEO1520	5/29/2020	NWB	George water licence

Source: Summary by AMC Mining Consultants (Canada) Ltd. based on data provided by Sabina Gold & Silver Corp. 2015

4.6 Environmental Liabilities

There are no known existing environmental liabilities on the Property. In 2009, at the time of the acquisition of the Property, letters of credit (LOCs) were provided to the KIA under the terms and conditions of the transferred land use licences. These LOCs have been subsequently increased as required by the KIA based on-site closure estimates prepared by Sabina.

4.6.1 Inspections and Monitoring

The Property is inspected on an annual basis by the permitting agencies, land owner, and regulators. To date, no orders have been issued. There was one inspection by the regulatory group and one inspection by the land owner, for a total of two inspections in 2015. The results of the inspections were satisfactory and any minor concerns were dealt with at the time of the inspection or within a short period of time from the inspection date.

Surface access and water access authorizations include terms and conditions outlining monitoring and sampling requirements. These requirements include water quality sampling focusing on potable water in camp, water used for drilling, on-ice drilling, and drainage collected in the areas affected by ongoing exploration activities including trenches, drill hole locations, and contact water within secondary containment. Samples are analysed for major and trace metals, major ions, various nutrients, and general water chemistry parameters. This information provides water quality data as part of establishing existing conditions and as part of ongoing progressive reclamation.

Other compliance monitoring applies to solid waste management, incineration, wildlife logs, and archaeological locations.

Sabina's progressive reclamation activities have focused on drill sites and waste disposal. Work that is ongoing includes cutting old drill casings and filling thermokarst permafrost degradation holes in the tundra. Camp waste materials are incinerated or backhauled to Yellowknife.

4.6.2 Environmental Assessment/Permit Process

In late 2011, Sabina began the environmental assessment (EA) and permitting processes for the Property. More recently, the following tasks were completed:

- A 12-volume, Draft Environmental Impact Statement (DEIS) was submitted to the NIRB (January, 2014);
- The DEIS was formerly accepted by the NIRB (February, 2014);
- Interested parties submitted information requests (March, 2014);
- Sabina responded to information requests (July, 2014);
- Sabina received the technical review comments from all federal and territorial interested parties and sent a technical comments response package to NIRB (October, 2014);
- The Technical meetings and Pre-hearing Conference occurred (November, 2014);
- The NIRB Pre-hearing Conference Decision for the Back River Project proposal was received (December, 2014); and
- Initiated the development of the Final Environmental Impact Statement (FEIS) (May, 2015).

5 Accessibility, Climate, Local Resources, Infrastructure & Physiography

5.1 Accessibility

The Goose and George sites are located approximately 520 km northeast of Yellowknife, Northwest Territories and are predominantly accessed by fixed-wing aircraft. The Goose and George sites are shown respectively in Figures 5.1 and 5.2.

During the months when Goose Lake is frozen, access to the existing Goose camp for drilling programs can be via an ice airstrip located on Goose Lake. The ice airstrip is capable of supporting a B-737, Electra, Hercules, DC-3, DC-4, Dornier 228, DASH 7, DASH 8, C-46, or Twin Otter aircraft. An all-weather airstrip, located 600 m southwest of the Goose camp, is capable of supporting a King Air, Buffalo, DC-3, Dornier 228, DASH 7, DASH 8, or Twin Otter aircraft. During the ice-free months, float-equipped Twin Otters and smaller aircraft can land on Goose Lake or on many of the other lakes in the area.

When George Lake is frozen, the existing George camp can be accessed by Electra, Hercules, DC-3, DC-4, C-46, Dornier 228, DASH 7, DASH 8, or Twin Otter aircraft utilizing an ice airstrip. During the summer months, Twin Otters or SkyVans gain access through an all-weather airstrip adjacent to the George camp or, if fitted with floats, by landing on George Lake.

All consumables are currently flown to the Property via fixed-wing aircraft. Personnel are transported to site using Dash 7, Dash 8, Dornier 228, or Twin Otter aircraft. Helicopters are typically on site during camp-based operations and are available to provide transport in the event of a medical or other emergency.

The proposed marine freight laydown site or MLA will be located at Bathurst Inlet. The shipping route would follow the marine waters north of the mainland coast. Sabina has a permitted winter ice road route from Bathurst Inlet to the Goose and George sites.

5.2 Climate and Physiography

The region is characterized by long dark winters and short summers. Typically, the ground is covered in snow from October to June. Lakes are ice-covered from approximately October to June with an ice thickness reaching depths of 3 m. The mean annual temperature is approximately -10.5°C, with a mean temperature of 6°C in summer and -26.5°C in winter. The mean annual precipitation range is 200 to 300 mm (PEG 2010). The average elevation is approximately 288 masl with topography generally characterized by gently rolling landscapes.

The Property lies north of the tree line in the West Kitikmeot region of Nunavut. For vegetation zones, the Goose and George sites lie within the “primarily unvegetated surface zone” and the potential port site area lies within the “tundra-high shrub zone” (PEG 2010). Vegetation is present in the form of low shrubs of willow, birch, Labrador tea, and mountain cranberry. Lichen is also common.

Outcrop distribution is highly variable; it predominantly occurs on the tops or flanks of the hills, while the valleys are commonly filled with glacial overburden. At the Goose Site, overburden depths in drill holes range from 0 to 42 m, with an average of approximately 10 m.

5.3 Local Resources and Infrastructure

The existing camps source goods and services from northern-based suppliers, located mainly in Yellowknife.

The exploration programs employ northern Nunavut residents from the Kitikmeot communities of Gjoa Haven, Taloyoak, Cambridge Bay, Kugaaruk and Kugluktuk who have been employed as geotechnicians, camp labourers, prospectors, core cutters, and site support.

The existing Goose camp contains a 146-person all-season camp consisting of sleeping units, dry and mess facilities, offices, a core processing facility, heavy equipment storage facilities, a warehouse and an engineered bermed fuel farm. The Goose camp is powered by a 400 kW and a 433 kW diesel-powered generator which are backed up by one 175 kW diesel-powered generator. An all-weather road connects the Goose camp to the airstrip and dirt trails are present around the camp. Two maintenance shops are also located at the camp, along with a fuel storage facility of thirteen 75,000 L double-walled enviro-tanks. Figure 5.1 shows the existing Goose camp with the all-weather airstrip in the background.

The existing George camp contains a 58-person all-season camp consisting of sleeping units, dry and mess facilities, a core processing facility, a maintenance shop, a bermed fuel farm, and two pre-fabricated sleeper trailers. A 530 m long gravel airstrip is located at the George camp for use by short take-off and landing aircraft such as Twin Otters. Dirt trails exist at the George camp site. Two 225 kW diesel-powered generators provide power for the site. The bulk fuel storage facility at the George camp utilizes two 75,000 L double-walled steel enviro-tanks. Figure 5.2 shows the George camp with the all-weather airstrip in the foreground.

The camps are operated on a seasonal basis, with exploration programs typically commencing in Q1-Q2 and concluding in Q3-Q4 of each year.

At the start of processing, tailings will be deposited in a constructed Tailings Storage Facility (TSF) located approximately 2 km south of the Goose Main open pit. Once mining is complete at the Llama open pit, tailings will be deposited in it for the balance of the mine life.

Waste rock will be deposited in dedicated Waste Rock Storage Areas (WRSAs) near each source deposit as well as distributed as a cover layer over the filled TSF.

General waste materials will be segregated and managed depending on their nature. Inert garbage materials will be incinerated or stored in landfills. Wastes requiring treatment will be done so in land farms or backhauled off-site for treatment and disposal according regulated practices. Land farms and landfills will be located in dedicated areas within the various WRSAs and TSF and deposited at times coordinated with waste rock disposal operations.

The processing plant site will be located approximately 1 km south of the Umwelt deposit. Ore stockpiles, processing plant, some water treatment facilities, truck shop, administration facility, camp, fuel storage, and laydowns will be located here.

The water demands of the Goose process plant and Goose camp will be served by various sources. As much as practical, water that accompanies the tailings during disposal will be reclaimed for use in the process plant. Supplemental water for processing, potable and fresh water uses on the Goose Site will be drawn from Goose Lake.

Sewage will be collected and treated in biological reactors at Goose Site. The resultant sludge will be incinerated and the treated effluent will be discharged to the tundra.

The proposed MLA will be located approximately 130 km north of Goose at Bathurst Inlet. Seasonal off-loading and staging of construction and operations equipment and materials will be done here. It will have a maintenance shop, camp, fuel storage, and laydown facilities. As there are no processing facilities here, only fresh (and potable) water is required and that will be produced using a desalination plant. Sewage will be incinerated onsite. There will be no landfill at the MLA so general waste will be backhauled to either Goose or off the Property.

Figure 5.1: Existing Goose Camp (Looking West)

Source: Sabina Gold & Silver Corp., 2015

Figure 5.2: Existing George Camp (Looking East)

Source: Sabina Gold & Silver Corp., 2015

6 History

Comprehensive summaries of historical work on the Property were compiled in previous technical reports by WGM (2005), Coffey Mining (2009), and Nakai-Lajoie and Cater (2011). In addition, the SRK 2012 PEA Technical Report presents a complete list of historical work. The following information was sourced from these documents and is summarized in Table 6.1 and Table 6.2.

6.1 Ownership

The Back River Joint Venture (BRJV) owned the Property from 1982-2008. During this time, various companies and individuals acquired and relinquished their interests. At times, these entities were also the operators. A summary of the historical milestones of the Property is presented in Table 6.1.

Table 6.1: Historical Ownership Milestones

Dates	Milestones
1982	BRJV formed.
1985	F.W. Hill (Hill) and Esso Minerals Canada (Esso) was investor and operator.
1985 to 1986	Kerr-McGee Corp. acquired interest in BRJV and was operator in 1986.
1987 to 1996	Homestake Mineral starts to earn into BRJV and becomes operator in 1991.
1997 to 1998	Arauco (later changed name to Kit Resources) acquires BRJV.
1999 to 2002	Kinross starts to earn into BRJV and becomes operator.
2003 to 2004	Miramar starts to earn into BRJV and becomes operator.
2005 to 2008	Dundee Precious Metals (DPM) starts to earn into BRJV and becomes operator.

Source: Summary by AMC Mining Consultants (Canada) Ltd. based on table provided by Sabina Gold & Silver Corp. 2015

Sabina purchased the Property from DPM in 2009.

The Del claims, which are part of the current Property, were not part of the initial Property staked by the BRJV. In 1986, Bow Valley Industries owned the Del claims but dropped them after a small and unsuccessful drilling program (Cater et al., 2009). The area remained inactive until DPM staked 12 claims in 2008. The Del claims were subsequently sold to Sabina as part of the Back River Property in 2009.

6.2 Exploration Work

Table 6.2 summarizes the exploration work carried out by the operators. Trigg, Woollett, Olsen Consulting Limited (TWOCL) was the founder of the BRJV.

Table 6.2: Historical Exploration Summary

Operator	Period	Exploration Completed	Drill Holes Completed	Reports or Studies Completed
TWOCL on behalf of BRJV	1982	Reconnaissance exploration	-	-
Back River Joint Venture	1983-1985	Gridding, geological mapping, sampling, exploration drilling, and aeromagnetic surveys	36	-
Hill and Esso	1985	Airborne magnetics and electromagnetic surveys	-	-
Bow Valley	1986	Soil sampling at Del prospect, trenching, exploration drilling	11	-
Kerr-McGee Corp.	1986	Exploration drilling	31	-
Homestake Mineral	1987-1996	Geological mapping, panel and till sampling, exploration and infill drilling, geochemical study, geophysics, legal surveying	656	Prefeasibility and Feasibility Study (George)
Kit Resources	1997-1998	Geological mapping, sampling, exploration and infill drilling	184	Resource Estimate (George)
Kinross	1999-2002	Spectral induced polarization (IP)/resistivity survey, till sampling, geological mapping, channel sampling, soil sampling, exploration and infill drilling	126	Resource Estimate and Conceptual Study (Goose)
Miramar	2003-2004	Exploration and infill drilling	41	NI 43-101 Report
Dundee Precious Metals	2005-2008	Trench sampling, geological mapping, exploration and infill drilling, structural analysis, airborne magnetic, electromagnetic, and radiometric surveys, geochemistry and rock samples	186	NI 43-101 Report

Source: Summary by AMC Mining Consultants (Canada) Ltd. based on table provided by Sabina Gold & Silver Corp. 2015

6.3 Historical Estimates

A historical Mineral Resource estimate, as defined by NI 43-101 guidelines, is an estimate prepared before the issuer acquiring or entering into an agreement to acquire an interest in the Property.

A number of Mineral Resource estimates were carried out throughout the periods of exploration. The most recent historical Mineral Resource estimate was completed in September 2007 by RSG Global Consulting Pty Ltd. (RSG Global) for DPM, with an effective date of July 30, 2007.

AMC has not done sufficient work to classify the historical estimate as current Mineral Resources, and Sabina is not treating the historical estimate as current Mineral Resources. However, AMC makes the following observations regarding the historical Mineral Resource estimate:

- It was prepared in accordance with the CIM Estimation of Mineral Resource and Mineral Reserves Best Practice Guidelines; therefore, it is a reliable record of the historical Mineral Resources as of July 30, 2007;
- It targeted a high grade, low-tonnage, underground mining operation; and
- It was classified using CIM definitions.

The historical Mineral Resource estimate is summarized in Table 6.3.

Table 6.3: 2007 Historical Mineral Resource Estimate

Resource Category	Tonnes (kt)	Gold Grade (g/t)	Metal (koz Au)
Indicated	3,415	10.9	1,193
Inferred	3,556	10.2	1,162

Notes: CIM definitions were used for the Mineral Resources.
Mineral Resources are reported at a zero cut-off within wireframes determined by a 5.0 g/t Au threshold
Estimate used drilling results to July 30, 2007.
Source: Summary by AMC Mining Consultants (Canada) Ltd. based on table provided in RSG (2007).

6.4 Production

There has been no recorded mining production from any of the deposits on the Property.

7 Geological Setting and Mineralization

7.1 Regional Geology

The Property is situated in the central-eastern portion of the Slave Structural Province of Nunavut Territory (Figure 7.1). The geology of the eastern portion of the Slave Province is predominantly composed of 2.73 to 2.63 billion years (Ga) old greenstones and turbidite sequences (collectively known as the Yellowknife Supergroup) and 2.72 to 2.58 Ga plutonic rocks, mainly underlain by older gneiss and granitoid units. Volcanic-turbidite series (VTS) rocks, the most widespread and abundant supercrustal units in the Slave Province, are dominated by large areas of turbidites that are flanked by narrow volcanic belts and include distinct volcanic complexes. The following two types of volcanic rocks have been identified:

- Hackett River-type volcanic rocks are felsic to intermediate in composition and are of calc-alkaline affinity; and
- Yellowknife-type volcanic rocks are intermediate to mafic in composition and display tholeitic differentiation trends.

The Yellowknife Supergroup in the Hackett River area has been divided into the Hackett River, Beechey Lake, and Back groups. One theory for the evolution of these groups describes volcanism punctuated by erosion and periods of fluvial fan deposition, as indicated by the intimate interbedding of the Hackett River and Back group volcanic rocks with Beechey Lake Group sedimentary rocks.

Iron formations are locally abundant in VTS rocks of the Slave Province. These iron formations host most of the stratiform and stratabound gold prospects in this part of the Slave Province.

The Back River Property, located in the northeastern corner of the Hackett River area, contains oxide and silicate iron formation hosted in Beechey Lake Group turbidites.

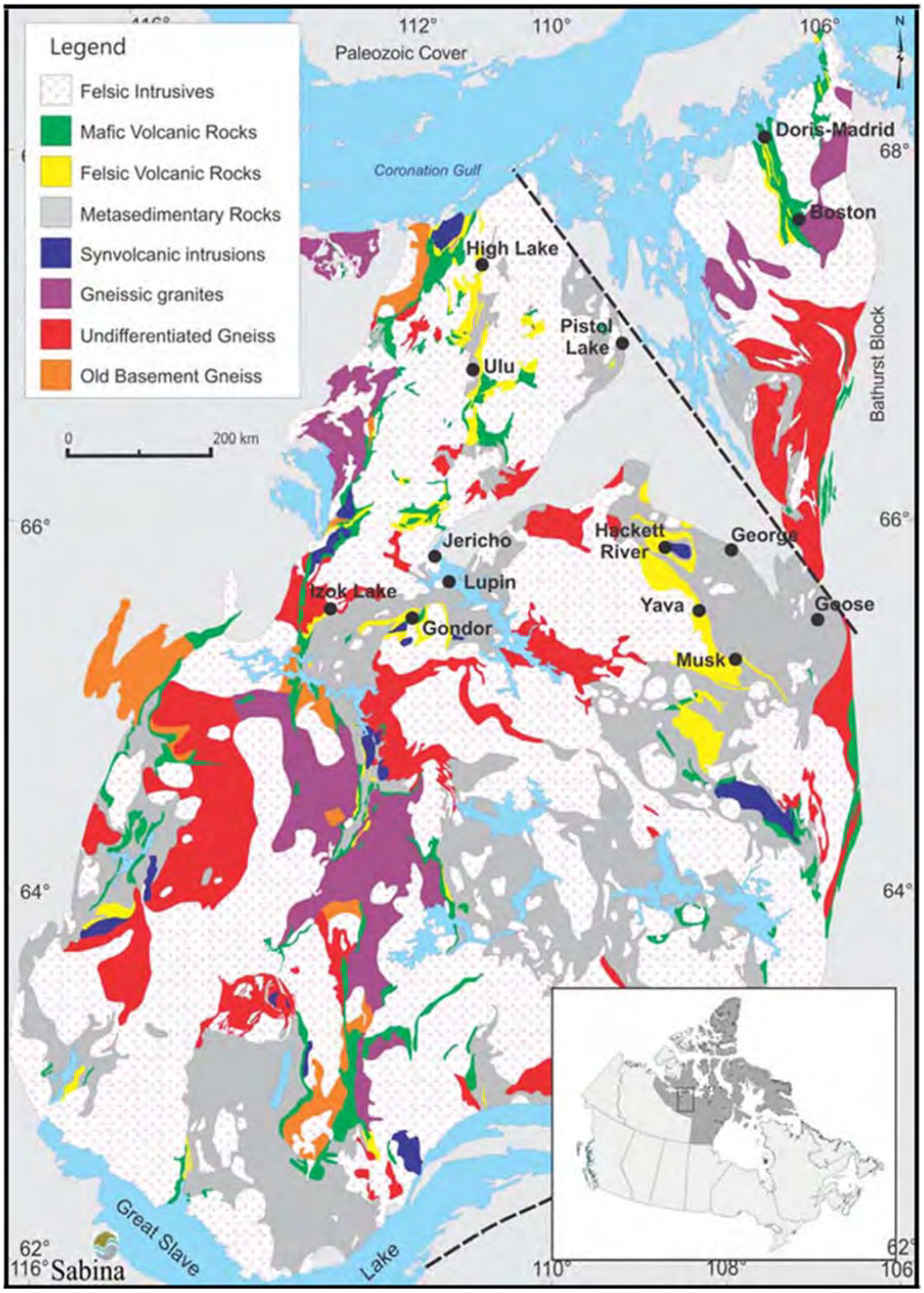
These iron formations are similar in both mineralogy and stratigraphic setting to the Sequence C iron formations of the Back River Volcanic Complex (modified after Henderson, 1993).

During the Archean era, this region underwent the following three periods of deformation:

- Large scale folding and down-warping of volcanic margins (D1);
- Later uplift causing foliation development (D2) when regional thermal metamorphism was at its peak; and
- Formation of a sub-vertical foliation (D3).

Metamorphic grade ranges from greenschist to upper amphibolite facies.

Figure 7.1: Slave Province Geology



Note: Modified from Hoffman and Hall (1993).
Source: Sabina Gold & Silver Corp. 2015

7.2 Property Geology

The Property comprises the Goose Site and the George Site; the geology of each area is summarized in Sections 7.2.1 and 7.2.2, respectively. The geology of the Boot and Boulder prospects is discussed in section 7.2.3 and section 7.2.4, respectively.

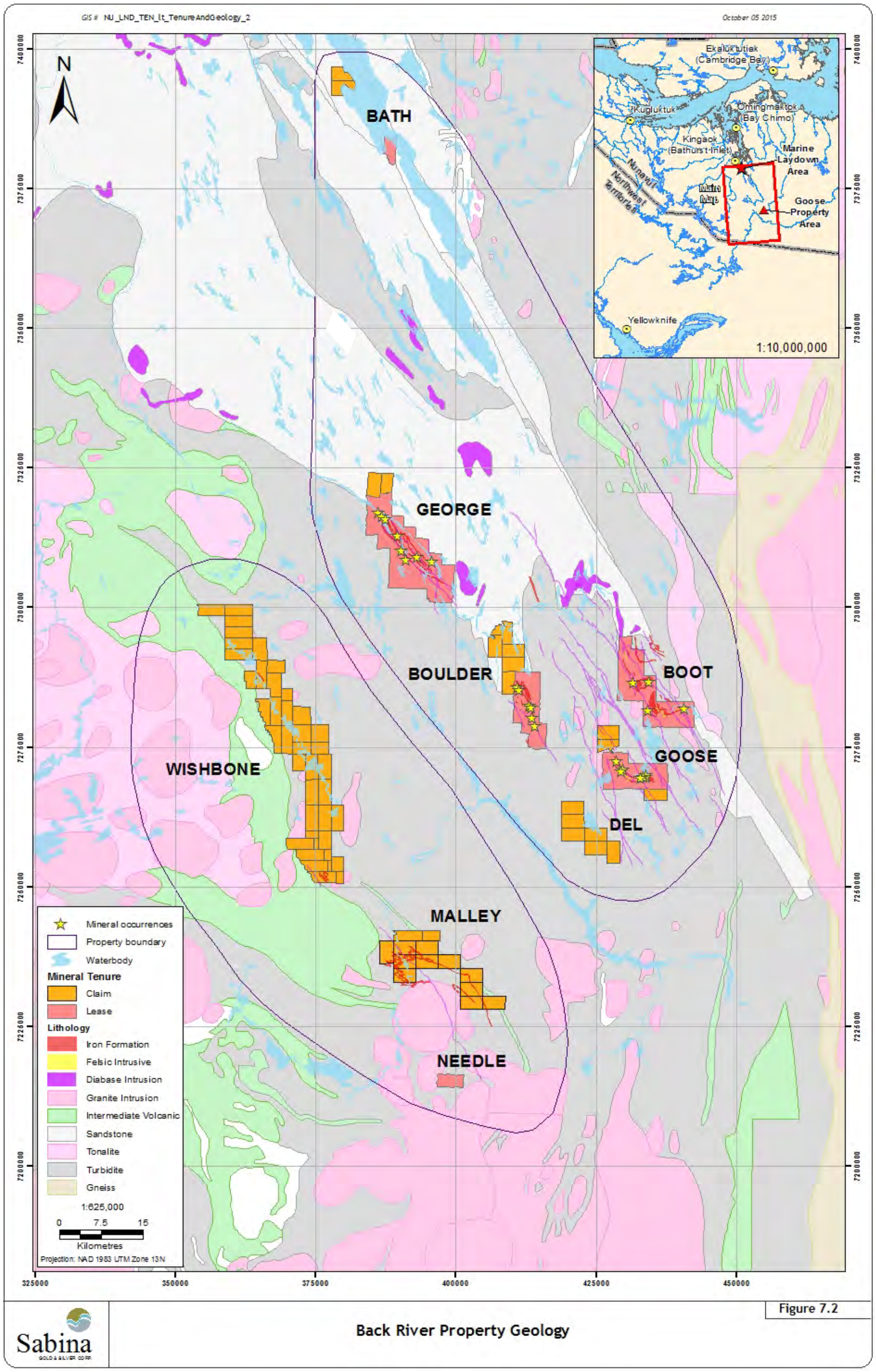
A summary of the Property stratigraphy is shown in Table 7.1. A geological map of the Property is shown in Figure 7.2, where other prospects are also shown.

Table 7.1: Stratigraphy of the Back River Property

Age	Group	Sub-group	Rock Types
Proterozoic	Goulburn	-	Clastic sediments with interbedded carbonates
Archean	Regan Intrusive Suite	-	Granitic to dioritic plutons and dyke equivalents
	Yellowknife Supergroup	Beechey Lake	Turbidite sediments, greywacke, mudstone and iron formation
		Back	Felsic to intermediate flows, tuffs and breccia
		Hackett River	Felsic to mafic volcanic flows, tuffs and chemical sediments

Source: Sabina Gold & Silver Corp. 2015

Figure 7.2: Back River Property Geology

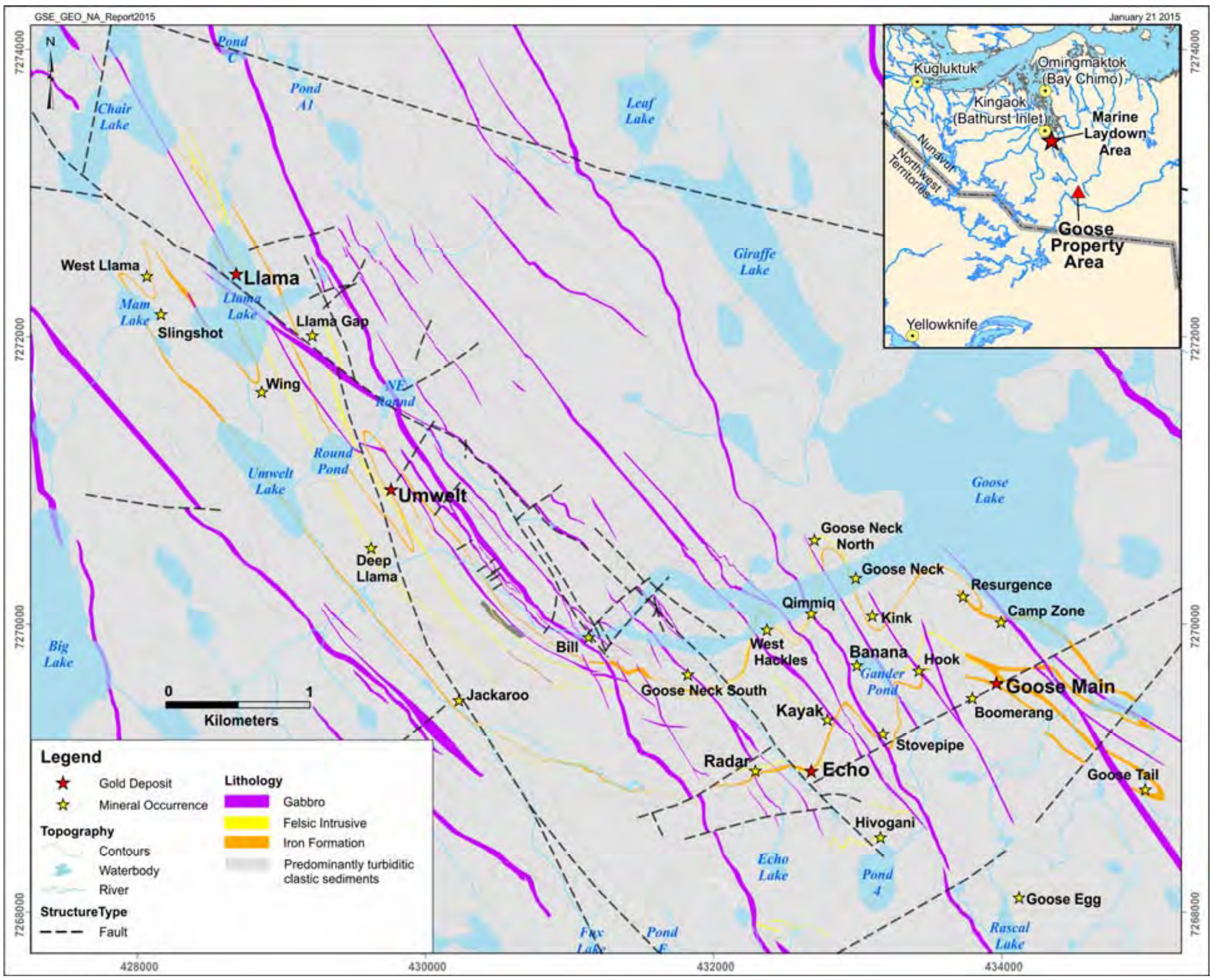


Source: Sabina Gold & Silver Corp. 2015

7.2.1 Goose Site Geology

The Goose Site includes the Llama, Umwelt, Goose Main, and Echo deposits and is underlain by folded Beechey Lake turbiditic meta-sediments, including subordinate oxide and silicate banded iron formation (BIF) (Figure 7.3). This sequence is cut by felsic and gabbroic dykes. From oldest to youngest, the sequence is composed of Lower Greywacke, Deep Iron Formation, Lower Iron Formation, Middle Mudstone, Upper Iron Formation, Phyllite and upper sediments. Gold mineralization tends to be hosted in the Lower Greywacke, Upper Iron Formation and Lower Iron Formation (LIF) units. A narrow (3 to 5 m) felsic dyke is located along the axial plane of the major antiform at the Goose Main deposit. Multiple, thin (0.5 to 5 m) felsic dykes trend nearly parallel to the synform/antiform structures at the Llama and Umwelt deposits. At Echo, a single felsic dyke cross-cuts the mineralization. All units are cross-cut by gabbroic dykes.

Figure 7.3: Goose Site Geology



Source: Sabina Gold & Silver Corp. 2015

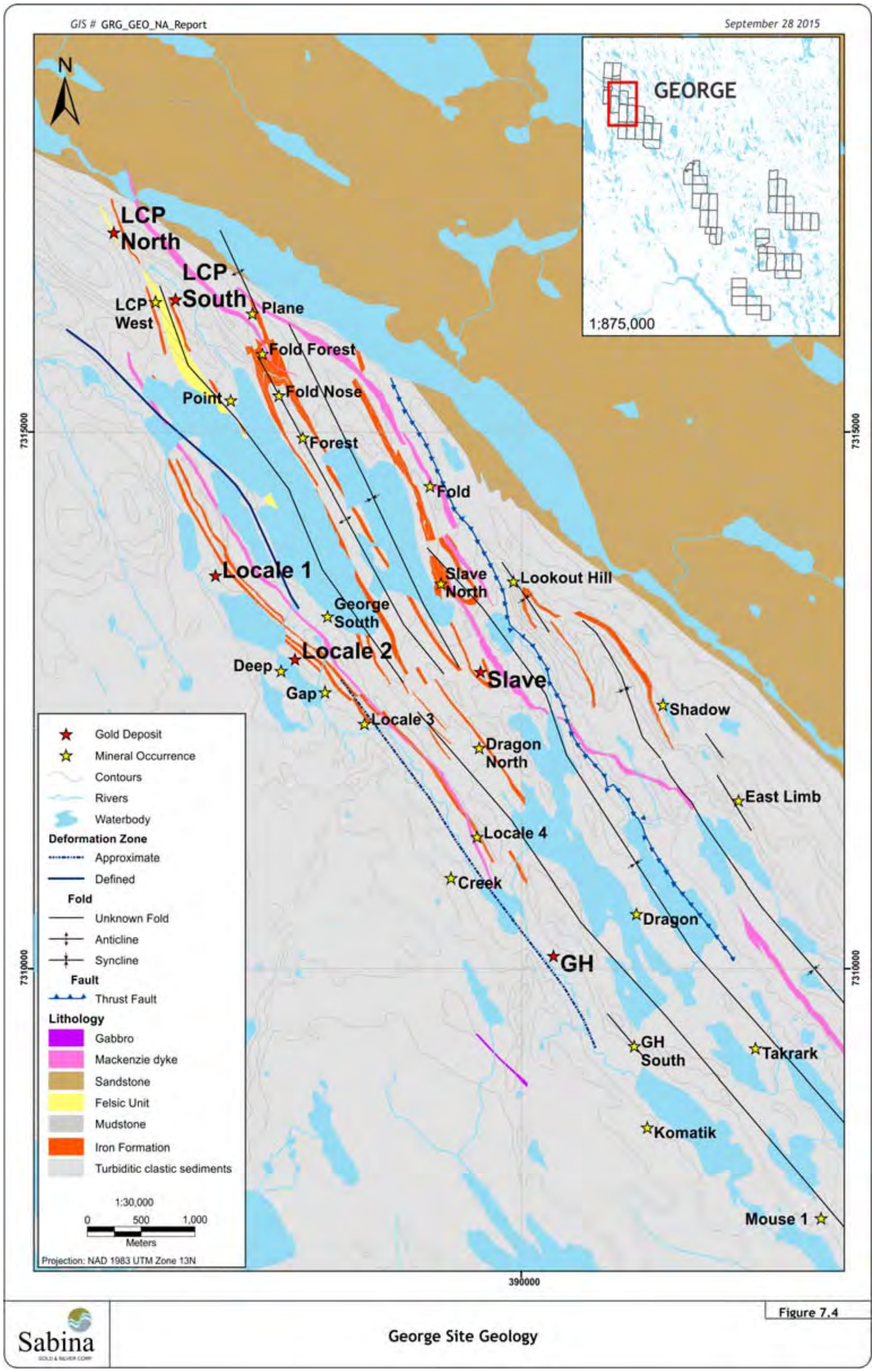
All of the deposits are hosted within a turbiditic sedimentary sequence of locally sulphidized silicate and oxide iron formation interbedded with varying amounts of lesser gold-mineralized greywacke, siltstone, and mudstone. The current interpretation has defined the host stratigraphy as an iron formation and clastic sediment sequence that occurs within a plunging series of folds, which trends northwest to southeast. Gabbro and porphyritic felsic dykes intruding the clastic and chemical sediments are interpreted as favouring pre-existing structural conduits. Airborne and ground magnetic images reveal iron formation fold patterns.

7.2.2 George Site Geology

The George Site includes the Locale 1, Locale 2, LCP North, LCP South, GH, and Slave deposits. The geology for all deposits is presented in this section; however, the reader is reminded that this Initial Project Feasibility Study focuses on developing the Goose Main, Llama and Umwelt deposits. The Beechey Lake Group greywacke, mudstone, and iron formation are the dominant lithologies (Figure 7.4). Beechey Lake Group rocks are cut by felsic to intermediate sills, dykes, and small intrusions, all of which might be the Regan Intrusive Suite. The largest of these intrusions outcrops on the west side and immediately northwest of the George Site as an elongate, sheared quartz-feldspar porphyry. Unconformably overlapping the Archean rocks are Aphebian Goulburn Group sediments, exposed on the north and east margins of the George Site area with scattered remnants of basal unconformity material exposed elsewhere. Northwest-trending gabbroic dykes of Helikian age intrude all of the above-mentioned rocks. A summary of the dominant lithological units is shown in Table 7.1.

Iron formations in the George Site area are dominated volumetrically by oxide facies (magnetite-chert-grunerite) with subordinate silicate facies (chert-grunerite-chlorite). In places, the dominantly oxide facies iron formation grades laterally into silicate facies, primarily as a function of magnetite depletion. Iron carbonate is present in both oxide and silicate facies iron formation. Iron formation occurs in three distinct fold belts named, from west to east, the George Belt, the Fold Nose Belt, and the Lookout Hill Belt (Figure 7.4). The relationship between these spatially separate domains has not been clearly established. However, common stratigraphy within the three belts suggests that they might represent one continuous sequence of iron formation that has been separated, and repeated by faulting and folding. The structural and geologic grain on the George Site is northwest to southeast. The Archean rocks have been affected by at least two early fold deformation episodes with a third fold event also deforming the overlying Aphebian Goulburn Group.

Figure 7.4: George Site Geology



Source: Sabina Gold & Silver Corp. 2015

7.2.3 Boulder Prospect Geology

At the Boulder Prospect, oxide iron formation forms a 10 km linear north-south trend and is hosted by a package of mixed turbidites of the Beechey Lake Group (Figure 7.5).

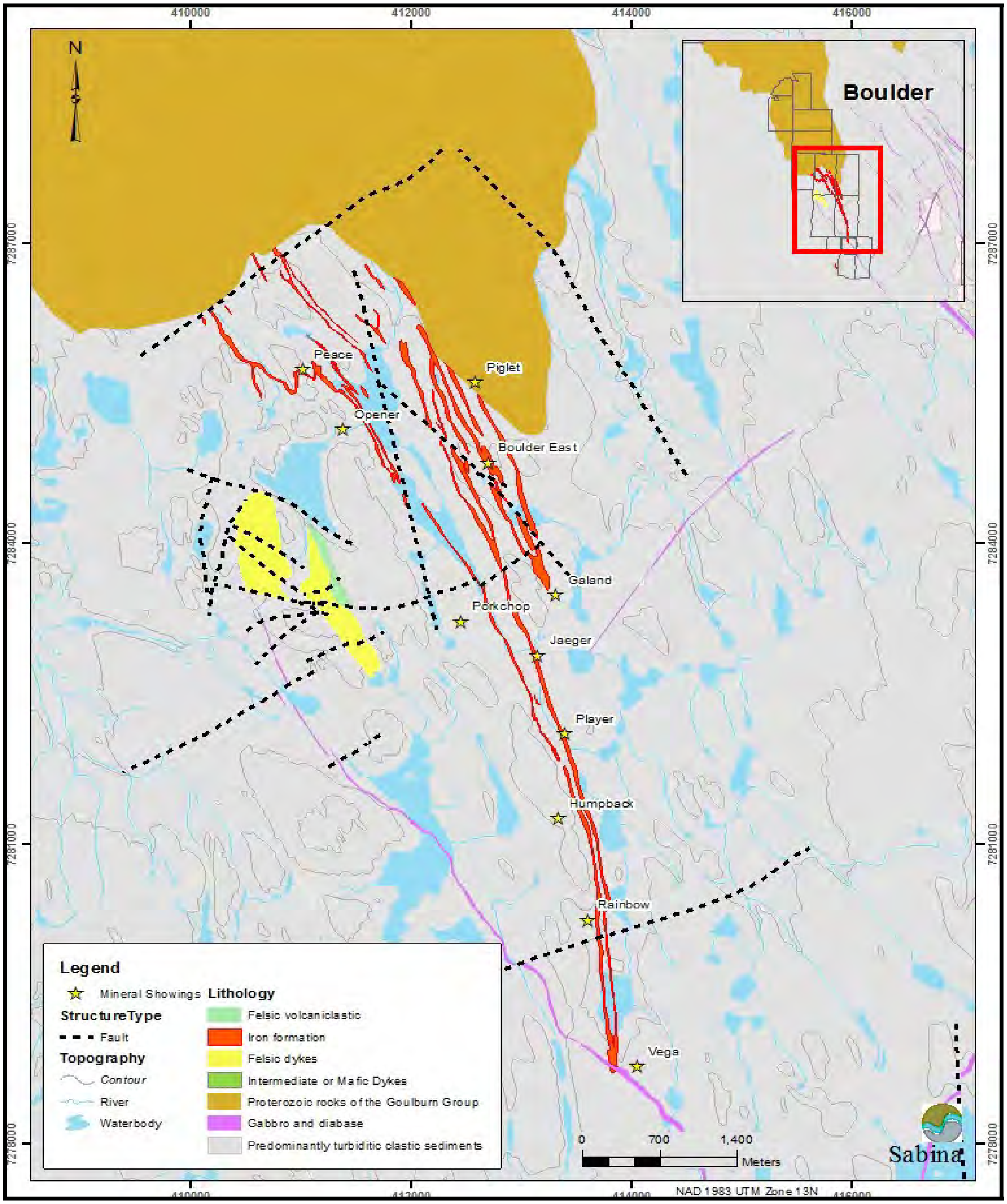
Proprietary and government airborne geophysical surveys suggest that the iron formation on the Boulder Prospect is continuous with the iron formation exposed at the George Site. However, between the two properties, the iron formation is overlain by a thick (up to 300 m) sequence of flat-lying Proterozoic Goulburn Group sedimentary rocks. The main iron formation stratigraphic package (Figure 7.5) is continuous over most of the area; however, it is extremely variable in relative proportions of detrital and chemical components. The main iron formation package contains up to four distinct oxide-rich units interbedded with mudstone, greywacke, and locally intermediate to felsic volcaniclastic units. Mineralogically, the iron formation is relatively consistent throughout the area, both along and across strike. The iron formation is considered to have been deposited under oxide facies conditions and typically consists of magnetite-chlorite-chert and minor hornblende. Subtle variations occur in the relative abundances of these minerals. Grunerite and garnet are noticeably absent from the surface exposures of iron formation at the Boulder Prospect, but are locally present in drill core.

7.2.4 Boot Prospect Geology

As elsewhere on the Property, the Boot prospect comprises multiple-kilometre strike lengths of tightly-folded oxide iron formation within mixed clastic sediments. The main iron formation unit is continuous across the Boot prospect, although the stratigraphy changes strike sharply at junctures in the northwestern, central, and southeastern Boot prospect (Figure 7.6). Volumetrically minor felsic and intermediate dykes and plutons are present; the most significant of these is the Rusty Ring dioritic pluton, which intrudes the central/southwestern part of the Property. Generally northwest-trending gabbroic dykes intrude the Archean stratigraphy, while the much younger (ca. 1.3 Ga) Mackenzie diabases cut all lithologies.

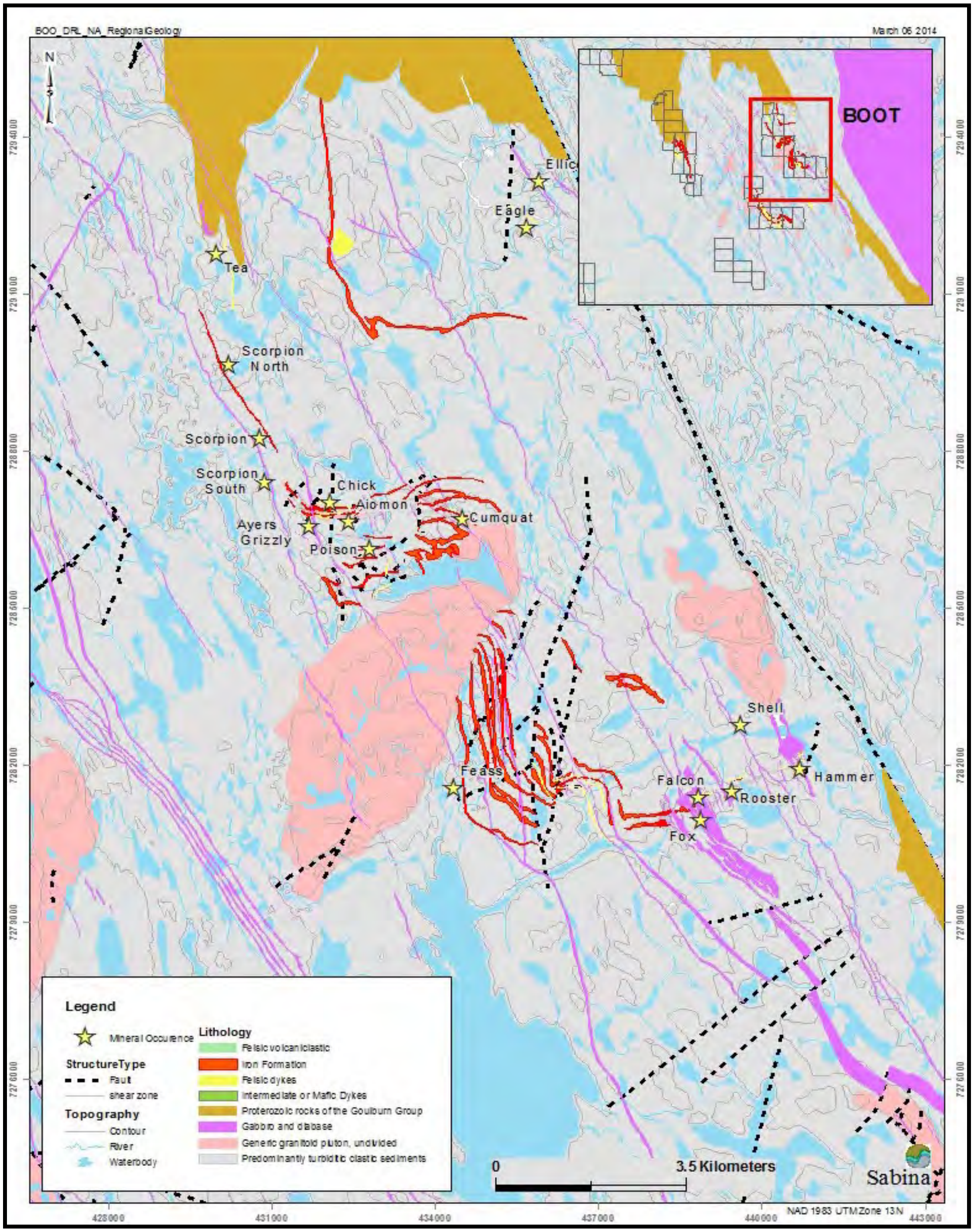
Iron formation units at the Boot prospect are dominated volumetrically by oxide facies (magnetite-chert-grunerite) with subordinate silicate facies (chert-grunerite-chlorite). In many locations, oxide iron formation is heavily sediment-bearing and is debatably similar to the sediment-rich upper iron formation at Goose. The Boot prospect hosts some of the best-exposed and visually well-developed iron formations within the entire Back River Property area. Abrupt transitions between sedimentation styles are noted at the Boot prospect, with thin-bedded sediment sequences capped with iron formation overlain by thick-bedded, coarse sandy sediments.

Figure 7.5: Boulder Prospect Geology



Source: Sabina Gold & Silver Corp. 2015

Figure 7.6: Boot Prospect Geology



Source: Sabina Gold & Silver Corp. 2015

7.3 Property Mineralization

The gold mineralization on the Property is strongly correlated with iron formation, and, as a result, the mineralization geometry is relatively continuous between sections and down- dip. However, within the interpreted mineralized zones, gold grades can be highly erratic and discontinuous. Further details on the mineralization at the Property are provided in the following subsections. Table 7.2 and Table 7.3 show the orientation, length along-strike, average down dip dimension, and mean true thickness of the mineralization by deposit. Generally, the mineralized zones at the Goose Site lie beneath 4 to 10 m of overburden, while all of the George zones outcrop.

Table 7.2: Goose Site Mineralization Summary by Deposit

Deposit	Folded	Trend of Fold Axes (°)	Plunge of Fold Axes (°)	Dip of Fold Axial Plane (°)	Dip	Strike Length (m)	Average Length Down Dip (m)	Mean True Thickness (m)
Goose Main	Yes	285	15	70	W	650	480	15
Llama	Yes	145	20	75	E	1,080	220	12
Umwelt	Yes	135	26	50	E	1,530	240	15
Echo	Yes	145	63	90	-	410	350	6

Source: AMC Mining Consultants (Canada) Ltd. 2015

Table 7.3: George Site Mineralization Summary by Deposit

Deposit	Folded	Dip Direction (°)	Dip (°)	Strike Length (m)	Average Length Down Dip (m)	Mean True Thickness (m)
LCP North	No	238	85	750	220	3
LCP South	No	248	85	525	190	3
Locale 1	No	240	75	1,050	300	3
Locale 2	Yes[1]	220	75	670	350	3
GH	No	230	80	480	200	2
Slave	Yes[1]	230	70	600	180	4

Note: [1] Mineralization folded but modelled as individual planes.

Source: AMC Mining Consultants (Canada) Ltd. 2015

7.3.1 Goose Site Mineralization

7.3.1.1 Llama Deposit

Based on available assay data, gold mineralization is recognized to be hosted in both chemical and clastic sedimentary lithologies as well as quartz ± feldspar porphyry dykes. Late gabbro dykes are known to post-date the timing of gold mineralization and do not host economic concentrations of gold. Banded oxide facies iron formation, consisting of chert + grunerite + magnetite, hosts the majority of the known gold mineralization. Silicate facies iron formation consisting of actinolite + chert + grunerite and locally interbedded clastic sediments, hosts relatively lesser gold mineralization. Clastic sediments consisting of greywacke, siltstone, and mudstone are noted to be mineralized but typically return low levels of gold, with isolated elevated gold assays. In some cases, felsic dykes have been proven to host gold; however, the amount is considered relatively insignificant to date.

Gold mineralization is best characterized as an event of widespread quartz ± carbonate veining and sulphidization related to brittle faulting and folding. Mineralization consisting of pyrite ± arsenopyrite ± pyrrhotite, rare chalcopyrite, and free gold is observed to occur within quartz ± carbonate veining of all lithology types with the exception of gabbro. Gold-mineralized quartz veining occurs commonly within the interpreted structural corridor with local concentrations. Replacement sulphidization of host lithology is also recognized within the Llama Gold Zone, where pyrrhotite ± arsenopyrite (including loellingite) ± pyrite replaces magnetite and grunerite to varying degrees.

Oxide facies iron formation is noted to have the highest level of sulphidization of all lithologies. Sulphidation is most intense proximal to brittle deformation interfaces of the structural corridor. Silicate facies iron formation is noted to have a lesser degree of sulphidization, and similarly correlates with proximity to brittle deformation within the structural corridor. Relatively lower sulphidization of silicate facies iron formation is interpreted as a product of less abundant primary and metamorphic iron-rich minerals. Mineralization in clastic sediment lithologies is limited to dominantly quartz ± carbonate vein-style gold mineralization; however, intervals of silicification with fine-grained disseminated sulphides have been observed. Gold mineralization of this style is best observed proximal to areas of brittle deformation, typically occurring at or near contacts with iron formation.

7.3.1.2 Umwelt Deposit

Gold mineralization has been noted within quartz ± carbonate veining and sulphidized iron formation lithologies, most commonly associated with arsenopyrite ± pyrite ± pyrrhotite. Pyrite and pyrrhotite are the most common sulphides in the Umwelt deposit with pyrrhotite becoming significantly more prominent as the gold-mineralized zone becomes deeper to the south. Arsenopyrite is the most common sulphide associated with gold, occurring as fine- to coarse-grained, euhedral, individual masses of crystals occasionally located preferentially along bedding planes or trailing along fractures, and as vein halos. Pyrrhotite appears to be replacing magnetite in at least some instances; it is present in beds ± magnetite, associated with veins and fractures. Gold-mineralized zones are characterized by sulphide and silica alteration including quartz flooding, accompanied by shearing and veining.

7.3.1.3 Goose Main Deposit

Most of the observed gold mineralization at the Goose Main deposit is associated with quartz veins, silicification, and shearing. Gold mineralization occurs within silicified and variably sulphidized iron formation and, to a lesser extent, mixed iron formation and meta-sedimentary units located in the underlying central greywacke. Observed sulphide minerals include pyrite, arsenopyrite, and pyrrhotite. Sulphide gold mineralization might be associated with accessory chlorite, carbonate, hornblende, and grunerite. Visible gold is locally present, especially when sulphides are greater than 10% and when coarse-grained arsenopyrite is present.

The deposit is located within the Goose antiform structure, which is situated within a greater than 500 m wide corridor of widely spaced, sub-parallel, north to northeast trending, southeast dipping, normal faults that have up to 30 m of left-lateral displacement and a down-dropping of individual fault blocks of up to 75 m.

Approximately 60% of the gold mineralization occurs within the LIF (sulphidized oxide iron formation), and the remaining 40% occurs in the core of the underlying central greywacke. Very minor gold and sulphide mineralization is developed in the upper iron formation. Visible gold is common and typically occurs as sub-millimetre-sized grains, although larger aggregates of up to several millimetres are not uncommon. Visible gold is typically spatially associated with pyrrhotite and/or pyrite in the presence of arsenopyrite. Gold mineralization is more pronounced and of higher grade in areas of brittle deformation, and of lower grade or absent in areas of ductile deformation. Where D2 deformation is absent, gold grades are also low to absent. Late D2 deformation appears to be the key gold mineralizing event, where existing partially or wholly discordant quartz veins acting as fluid pathways are commonly boudinaged, re-oriented parallel to S0 to S1 foliation, and gold-mineralized.

7.3.1.4 Echo Deposit

Gold mineralization at the Echo deposit is defined within, but is not limited to, the contact of the iron formation/interbedded sediments with the regional turbiditic lithologies. Brittle deformation is prominent at the contact; there is also a moderate amount of shearing present locally. A poorly-mineralized quartz-feldspar porphyry dyke intrudes proximal to the structurally-influenced contact and is interpreted to be closely related to the timing of gold mineralization. Alteration consists of varying amounts of grunerite + chlorite + quartz + calcite ± biotite. Mineralization associated with intervals returning higher gold values (up to 120 g/t gold) consists of pyrrhotite + pyrite + arsenopyrite + chalcopyrite occurring with quartz ± calcite veining, as well as replacement of the host rocks. The overall sulphide content ranges from a trace amount up to a maximum of 10% over 0.5 m.

Banded silicate iron formation consisting of actinolite + grunerite + quartz ± tremolite hosts the majority of the known gold mineralization. Oxide iron formation is similarly mineralized but forms only a minor portion of the host. Clastic sediment lithologies appear to be less favourable hosts and appear to be best mineralized within areas of deformation that occur at contacts with iron formation. Because the area is largely covered by overburden, the relationship between the Goose Main deposit and the Echo zone is not well understood. This area has potential for the development of additional gold targets.

7.3.2 George Site Mineralization

Gold mineralization in the Locale 1 and Locale 2 deposits is hosted primarily in the LIF, within 10 to 12 m of the western edge of the western limb of a slightly overturned, tight Locale 1 – Locale 4 syncline. Some gold mineralization is also hosted in the upper iron formation at the Locale 1 and Locale 2 deposits. The gold deposits at LCP North, LCP South, GH, and Slave are located within oxide iron formation in the limbs of tight isoclinal folds. Less significant mineralization is also hosted within silicate iron formation and surrounding sediments.

A spatial correlation exists between ductile shears and Locale 1, Locale 2, LCP North, LCP South, and GH, but the timing relationship with gold genesis has not been determined. There is a close spatial association between gold and iron formation, and other rock types adjacent to mineralized iron formation typically lack gold mineralization. The gold-bearing zones coincide with sulphide-bearing portions of the iron formation. The sulphides typically conform to the banding in the iron formation. The concentrations of gold coincide with sulphide-bearing zones that are associated with late quartz veins cutting iron formation. The sulphide mineralogy associated with the gold comprises pyrrhotite or pyrite in the biotite metamorphic zone, arsenopyrite, loellingite, and minor amounts of chalcopyrite. The sulphides are generally disseminated and might be concentrated in specific bands within the iron formation. Pyrrhotite and pyrite replace magnetite and amphiboles. The predominant amphibole in mineralized iron formation is hornblende, with little or no grunerite. Where both amphiboles are present, grunerite is partially replaced by hornblende. Along-strike from the mineralized zones, the sulphide-bearing, hornblende-rich mineralized iron formation passes into sulphide- and gold-poor unmineralized iron formation that is comprised of grunerite + quartz ± magnetite ± minor amounts of hornblende. These characteristics are also present at the Goose deposits.

At least three gold-mineralized quartz vein sets are present at the George Site. Two of the quartz vein sets are steeply dipping; one set is oriented sub-parallel to the iron formation stratigraphy, and the other is near perpendicular. The third set is sub-horizontal. All three sets of quartz veins are more abundant within the iron formation units than in the surrounding sediments. Not all quartz veins are gold-bearing or associated with sulphides. The quartz veining-associated gold mineralization and alteration occur late in the brittle-ductile shearing structural history.

7.3.3 Boulder Prospect Mineralization

Mineralization at the Boulder Prospect is similar to that identified at the Goose and George sites, occurring as pyrrhotite, pyrite, and arsenopyrite with the arsenopyrite being more closely associated with gold mineralization. Sulphides are typically associated with silicification or quartz veins, and form either massive pods or disseminated euhedral crystals within chloritic vein selvages or within the adjacent wallrock. Additional work is required to further characterize mineralization at the Boulder Prospect.

7.3.4 Boot Prospect Mineralization

Mineralization at the Boot prospect is also similar to that identified at the Goose and George sites. Pyrrhotite, pyrite, and particularly arsenopyrite are associated with gold mineralization. Sulphides are typically associated with silicification or quartz veins, and form either massive pods or disseminated euhedral crystals, within chloritic vein selvages or within the adjacent wall rock. Additional work is required to better describe the nature and range of mineralization styles at the Boot prospect.

8 Deposit Types

This section was taken directly from the NI 43-101 technical report titled “Technical Report and Prefeasibility Study for the Back River Gold Property, Nunavut, Canada”, dated October 9, 2013. This section remains unchanged and is included for completeness and continuity.

The gold deposits at the Property are hosted by sulphidized oxide and silicate iron formation rocks, and clastic sediments that are cut by barren and sulphide-bearing quartz ± carbonate veins. Analogous deposits occurring in this region of the Arctic include the Lupin Mine (located approximately 225 km west of the Property), the Meliadine deposit at Rankin Inlet, and the Meadowbank deposit north of Baker Lake.

BIF-hosted gold deposits mainly occur within Archean-aged greenstone belts, typical of the shield areas of northern Ontario, Quebec, Northwest Territories, and Nunavut. Generally, BIF host rocks are thinly-banded sedimentary rocks with alternating iron-rich and cherty (siliceous) layers.

In BIF-hosted gold deposits, gold mineralization is commonly associated with quartz and iron-carbonate veining, and zones of hydrothermal alteration with iron sulphides (mainly pyrite, pyrrhotite, and/or arsenopyrite). Gold mineralization is mainly located along shear zones associated with tightly-folded and structurally complex BIF horizons that provide favourable chemical and structural traps. This understanding is being applied in the current exploration strategy for the Property.

9 Exploration

This section discusses all exploration and associated work carried out by Sabina, other than drilling. However, it should be noted that the main activity at the Property has been diamond drilling of the known deposits.

9.1 Previous Sabina Exploration

On June 9, 2009, Sabina acquired the Property from DPM. After the acquisition, exploration was initially confined to the Goose Site and focused on finding new gold mineralization away from the existing Goose Main deposit. Sabina's exploration work prior to 2012 is detailed in the SRK 2012 PEA technical report. The 2012 exploration work is detailed in the Tetra Tech 2013 PFS Technical Report. The 2013 exploration work is detailed in the Tetra Tech Technical Report 2014. The 2014 exploration work is detailed in the JDS Feasibility Report 2015. All this activity is briefly summarized in Table 9.1.

Table 9.1: Summary of Sabina's Previous Exploration Work

Year	Location	Exploration Work
2009	Goose Site	Mapping, magnetics, IP and horizontal-loop electromagnetic (HLEM) surveys
2010	Goose Site	Geological mapping and sampling, magnetometer and HLEM ground survey, mineralogical study
2011	Goose Site	Geological mapping and sampling, time-domain electromagnetic (TDEM) and IP ground survey, mineralogical study, TDEM borehole surveys
	George Site	Magnetometer and HLEM ground survey
2012	Goose & George Sites	Grab sample program, metamorphic gold genesis study
	Goose Site	Till orientation study, mafic intrusion geochemistry and structural study, regional mapping*
2013	George Site	Geological mapping, metamorphic grade study, geochemical sampling
	Boot & Boulder	Geological mapping (1:1000 and 1:5000), geochemical sampling
	Regional	Regional –scale work off Property to provide wider geological context for the deposits
2014	Goose Site	IPower 3D® geophysical survey, felsic dyke geochemical characterization study
	George Site	Surface mapping, follow up metamorphic study

Note: *Regional mapping was conducted at the Goose Site and surrounding areas.

Source: AMC Mining Consultants (Canada) Ltd., 2015

9.2 Exploration in 2015

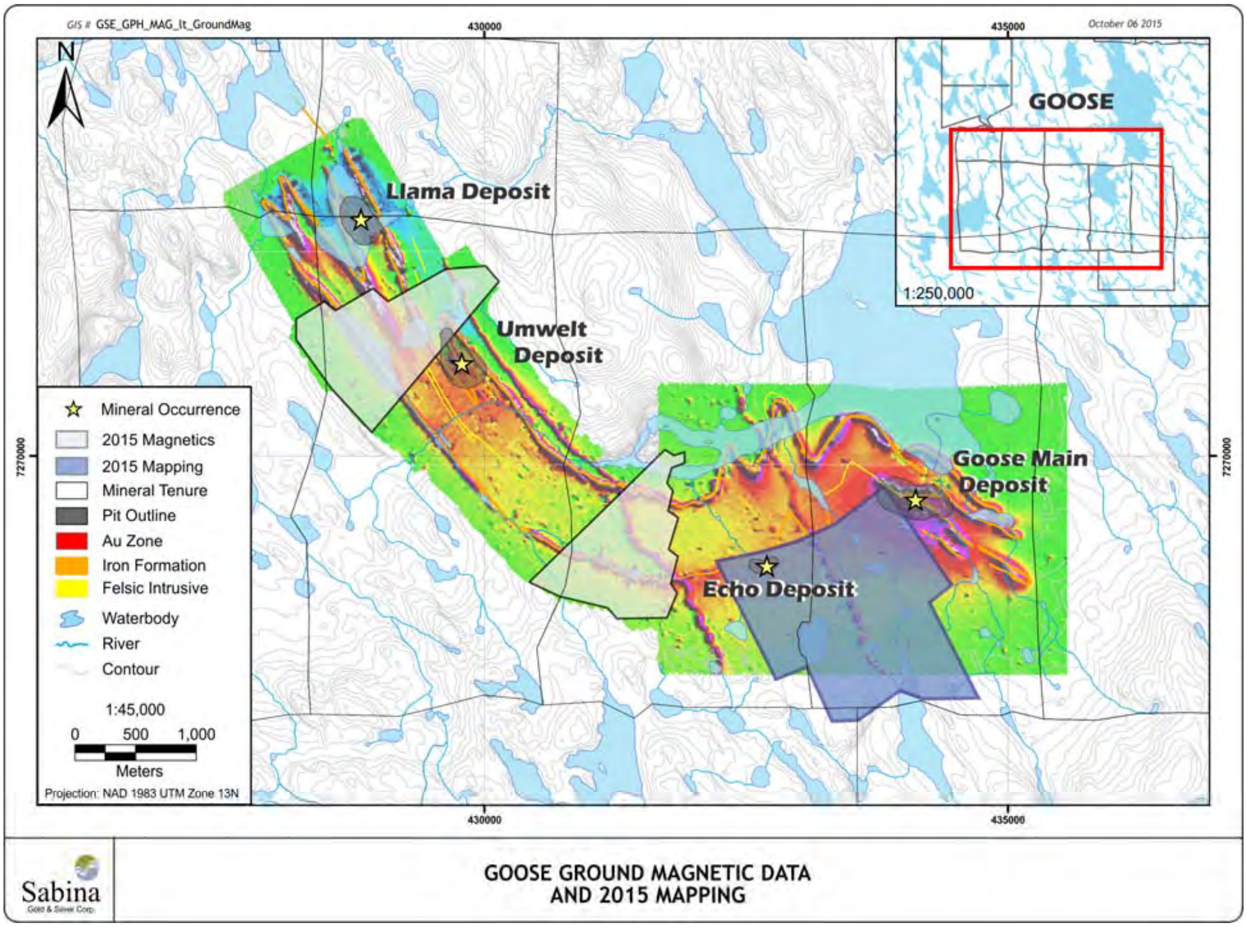
In 2015, Sabina carried out two studies at the Goose Site - a ground magnetics survey and a mapping and sampling program as shown in Figure 9.1. At the Boulder Prospect, a ground magnetics survey was completed as shown in Figure 9.2. Sabina also carried out a geochemistry data collection program from drill core pulp material which covered the Goose Site, George Site and the Boulder Prospect.

9.2.1 Goose Site Exploration

9.2.1.1 Ground Magnetics Survey

In 2015, Sabina contracted Clearview Geophysics Inc. to conduct ground magnetometer surveying by snowmobile support on the areas outlined in black in Figure 9.1. Ground magnetic surveys provide high-resolution data which assists in interpreting geometries in the highly magnetic iron formation. Magnetic responses span white, purple, red, green and blue colours from highest to lowest magnetic intensity. Northwest-southeast trending magnetic gabbroic dykes cross-cut the folded highly magnetic BIF. Surrounding sediments show as green surrounding the iron formation. The survey areas were selected on the basis of infilling data gaps in ground magnetic coverage over the iron formation stratigraphy. A total of 63.4 km of surveying with nominal 50 m line spacing was completed in two areas over a single day and the survey results were integrated with the existing dataset to be used in geological interpretation.

Figure 9.1: Ground Magnetic coverage and field mapping completed in 2015 at the Goose Site



Source: Sabina Gold & Silver Corp. 2015

9.2.1.2 Mapping and Sampling Program

A short field mapping program was undertaken in June 2015 to assess prospective areas along the southern Goose Property as shown in Figure 9.1. Three days were spent in the field.

An induced polarization (IP) ground survey carried out in 2014 led to the identification of a number of chargeability anomalies. One of these, the Hivogani target, was drilled in April 2015 with one drill hole of 140 m length as discussed in section 10. Six other IP anomalies from the 2014 survey were visited in June 2015. All were at least partly exposed in outcrop. Anomalies were explained by low levels of disseminated sulphide minerals (pyrrhotite, pyrite, or arsenopyrite). Increased abundance of quartz veinlets (up to 5%) was also locally coincident with the sulphide mineralization.

Geological mapping and prospecting was extended farther south of the Goose Main deposit through an area known as Goose Egg. A gold-bearing mineralized felsic dyke was identified in the area in 2008. Sabina geologists reinterpreted the direction of strike of the dyke, and successfully traced it through outcrop and subcrop >500 m to the east.

Twenty-nine grab samples were taken in the course of the June fieldwork; all were analysed for gold by fire assay and by a trace element suite by ICP-MS using a 4-acid digest method. This work was carried out by TSL Laboratory Inc. (TSL Labs) in Saskatoon. Significant assay results are summarized in Table 9.2 which shows assay results above 0.1gpt Au from the collected rock samples.

Table 9.2: Grab Sample Assay Results

Sample ID	UTM_X	UTM_Y	Au_ppm	Comments
E620411	433319	7268335	0.13	Weakly recrystallized fine-grained foliated greywacke with rusty spots along sheared quartz veins.
E620416	434408	7267778	0.17	4cm dark grey quartz vein along bedding, approx. orthogonal to the 4a unit.
E620418	434445	7267762	20.78	Coarse quartz-feldspar porphyry w/ clear quartz eyes and slight greenish hue.
E620419	434410	7267777	0.2	Felsic dyke near quartz veining and near contact with greywacke.
E833404	433372	7268390	0.12	Med. grey, fine-grained, highly foliated.

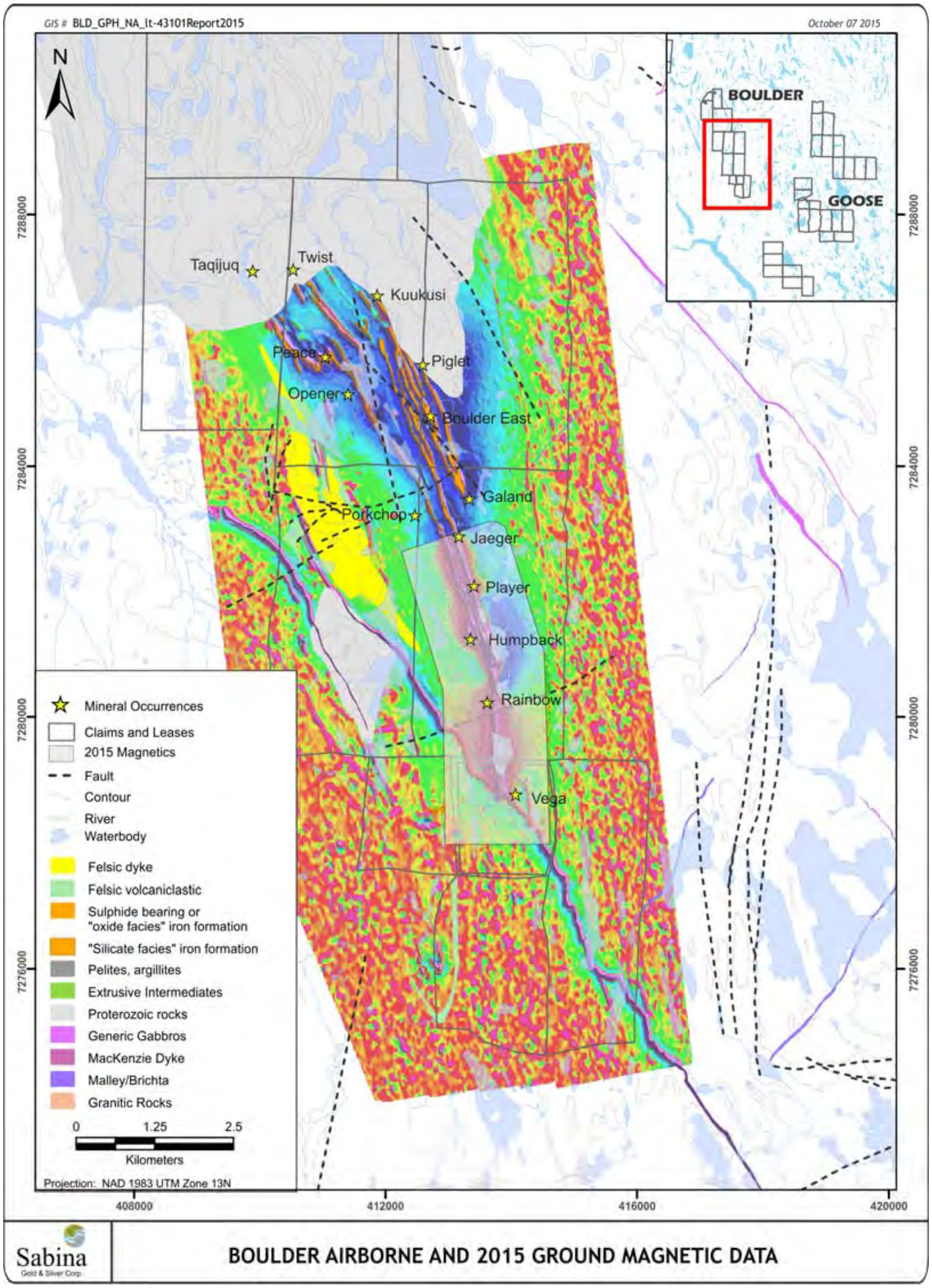
Source: Sabina Gold & Silver Corp. 2015

9.2.2 Boulder Prospect

9.2.2.1 Ground Magnetics Survey

In 2015, Sabina contracted Clearview Geophysics Inc. of Brampton, Ontario, to conduct ground magnetometer surveying by snowmobile support at the Boulder Property on the area outlined in Figure 9.2. Ground magnetic surveys provide high-resolution data which assists in interpreting geometries in the highly magnetic iron formation. Existing lower resolution airborne magnetic data is shown in Figure 9.2. Magnetic responses span white, purple, red, green and blue colours from highest to lowest magnetic intensity. A northwest-southeast trending magnetic gabbroic dyke cross-cuts the folded highly magnetic BIF at its southern end near the Vega showing. Surrounding sediments returned mottled green to purple colours to the east and west of the iron formation. The 2015 survey area was selected on the basis of increasing magnetic detail in a complex area over the iron formation stratigraphy. A total of 179 km of surveying with nominal 50 m line spacing was completed over four days and survey results were integrated with the existing dataset to be used in geological interpretation.

Figure 9.2: Airborne and 2015 Ground Magnetic Coverage on the Boulder Prospect



Source: Sabina Gold & Silver Corp. 2015

9.2.3 Property Exploration

9.2.3.1 Geochemistry Data Collection Program

In an effort to build a comprehensive holistic geochemistry database for the Property, pulp material from historic drill hole assay samples were submitted for trace element analysis. Samples were selected in deposit footprints or in areas of exploration interest, from a suite of drill holes spanning mainly the Goose Site but also George and Boulder areas. Generally, samples were selected at regular intervals approximately every 5 m downhole, although this regular pattern was interrupted in areas of unsampled core (where no assay pulps would be available). The intent was to better characterize element signatures and associations and identify any trace element patterns that might extend beyond deposits, thus assisting in future exploration vectoring.

Approximately 1,560 samples were identified by Sabina in March 2015 and were subsequently retrieved from storage by TSL Labs. Samples were submitted for 60-element analysis by ICP-MS using a 4-acid digest method. These samples are summarized by location in Table 9.3

Table 9.3: Assay pulps for trace element determinations.

Location	Samples Analysed
Goose	1,140
George	3,42
Boulder	79
Total	1,561

Source: Sabina Gold & Silver Corp. 2015

Samples from geotechnical and exploration drilling carried out in 2015 were also analysed. A portion of the remaining pulp material from each sample submitted for gold fire assay at TSL Labs was analysed for the same 60-element analytical suite as the historical pulps, using the same 4-acid digest and ICP-MS protocol. A total of 269 samples from the 2015 Goose Site drilling were selected for this work. Results for both streams of samples have been received but not yet reviewed in detail.

9.3 Sample Collection Procedures

Any sampling carried out in 2015 was by Sabina staff and the sampling procedures are summarized in the following section. Other types of samples that have been taken in previous years include: trench sampling, lake sediment sampling, metamorphic sampling, soil sampling, till sampling, samples for thin section work and surface grab sampling. The trench sampling, soil sampling and lake sediment sampling procedures are summarized in the SRK 2012 PEA technical report. The metamorphic sampling and thin section collection are summarized in the June 2015 JDS Feasibility Study (with an effective date of May 20, 2015).

9.3.1 Surface Sampling

This description refers to the 29 grab samples that were taken on the Property from outcrop and described in section 9.2.1.2. Grab samples were collected from BIF, felsic dykes, mineralized quartz veins, or any other rock types which are potential hosts. The grab samples were not representative of the true width of gold mineralization. The location of each sample was established using a handheld GPS unit with an accuracy of ±7 m. Representative samples were collected for whole-rock analysis in a similar fashion, from the least-altered examples of relevant or uncertain lithologies. Rock samples between 1 and 2 kg were collected and a unique sample number was assigned. The sample number was then recorded on the bag using a waterproof felt marker, and in the sampler's field notebook. A brief description of the sample, including rock type, mineralization, and any other relevant data, was also recorded in the sampler's notebook and on the handheld Trimble Juno 3B mapping computers. Samples were bagged and sent by air to Yellowknife, and then transported by road to TSL Labs, where sample preparation and the appropriate analytical procedures (either ICP or whole-rock) were performed.

9.4 Comments

The ground geophysics surveys at the Goose Site and the Boulder Prospect were successful in adding additional detail to the magnetic data of both areas. This improved data will assist in interpretations of the geometry of the BIF which is highly magnetic.

Mapping and sampling at the Goose Site has supported the identification of anomalies seen in the IPower 3D® System geophysical survey discussed in the 2015 JDS Feasibility Study (with an effective date of May 20, 2015). Samples returned anomalous and significant values within the greywacke package which sits stratigraphically below the iron formation. Further exploration is planned in this area.

Laboratory results from the geochemical data collection continue to be interpreted.

10 Drilling

10.1 Introduction

The Property has been drilled by various operators since 1985. All drilling carried out on the Property is diamond core, which is predominantly NQ diameter, although some drilling of BQ size was completed in 1992. More recently drilling of HQ core has been done for metallurgical and geomechanical purposes. Table 10.1 summarizes the drilling grouped by project/prospect and deposit.

Table 10.1: Drilling Summary by Year

Target	Unit Type	Pre-Sabina	2009-2011	2012	2013	2014	2015	Subtotal Sabina	Total
George Site									
Locale 1	Drill holes	198	-	20	8	-	-	28	226
	Metres	31,659	-	5,518	1,373	-	-	6,892	38,551
Locale 2	Drill holes	186	-	11	20	-	-	31	217
	Metres	43,929	-	3,328	6,323	-	-	9,651	53,580
LCP North	Drill holes	71	-	4	4	-	-	8	79
	Metres	10,456	-	864	311	-	-	1,175	11,631
LCP South	Drill holes	39	-	6	43	-	-	49	88
	Metres	11,356	-	1,336	6,555	-	-	7,891	19,247
GH	Drill holes	69	-	-	-	-	-	-	69
	Metres	10,915	-	-	-	-	-	-	10,915
Slave	Drill holes	39	-	-	-	-	-	-	39
	Metres	5,331	-	-	-	-	-	-	5,331
Other	Drill holes	196	38	21	26	-	-	85	281
	Metres	32,794	8,335	6,216	5,268	-	-	19,818	52,612
George Total	Drill holes	798	38	62	101	-	-	201	999
	Metres	146,440	8,335	17,262	19,830	-	-	45,427	191,867
Goose Site									
Goose Main	Drill holes	294	8	16	12	3	-	39	333
	Metres	73,347	1,476	3,315	2,413	1,251	-	8,454	81,801
Echo	Drill holes	-	35	-	18	25	-	78	78
	Metres	-	6,806	-	3,963	8,832	-	19,600	19,600
Llama	Drill holes	1	89	64	64	-	1	218	219
	Metres	83	25,851	16,598	14,492	-	177.5	57,118	57,201
Umwelt	Drill holes	3	121	49	67	1	-	238	241
	Metres	605	50,282	28,256	9,329	520	-	88,387	88,991
Other	Drill holes	26	71	13	54	5	2	145	171
	Metres	3,988	16,849	5,279	14,873	1,570	380.5	38,951	42,939
Goose Total	Drill holes	324	324	142	215	34	3	718	1,042
	Metres	78,023	101,263	53,447	45,069	12,173	558	212,509	290,532
Boot Prospect									
	Drill holes	100	-	-	29	-	-	29	129
	Metres	12,224	-	-	6,195	-	-	6,195	18,419
Boulder Prospect									
	Drill holes	38	-	10	37	-	-	47	85
	Metres	8,172	-	2,441	11,035	-	-	13,476	21,648
Del Prospect									
	Drill holes	11	-	-	-	-	-	-	11
	Metres	610	-	-	-	-	-	-	610
Grand Total	Drill holes	1,271	362	214	382	34	3	995	2,266
	Metres	245,469	109,597	73,150	82,129	12,173	558	277,606	523,075

Note: All holes are surface diamond drill holes. Holes listed are for exploration, Mineral Resource, geomechanical and metallurgical purposes but do not include geotechnical (site investigation) holes or trenches.
Historical holes have been reassigned to the appropriate deposits and all restarted holes are included in totals.
As discussed in Section 14, the boundary between Locale 1 and Locale 2 has been moved and drill holes reassigned accordingly.
Source: Sabina Gold & Silver Corp. 2015

10.2 Summary of Drilling Activity

A complete description of the drilling carried out in 2009, 2010, and 2011 is available in the SRK 2012 PEA Technical Report and is summarized in Table 10.1 of that report. Highlights of the 2009 to 2011 drilling include the discovery of the Llama, Umwelt, and Echo deposits.

In 2012, Sabina completed 214 diamond drill holes at the Property totaling 73,152 m of core. Drilling was largely focused on the Goose Site, primarily targeting the Umwelt and Llama deposits. Drilling also took place on the George Site and at the Boulder Prospect. A complete description of the 2012 drilling is presented in the Tetra Tech 2013 PFS Technical Report. Cross sections and plan views for each of the main deposits are also detailed in the Tetra Tech 2013 PFS Technical Report.

In 2013, Sabina completed 416 diamond drill holes at the Property totaling 82,593 m of core. Drilling was largely focused on the Goose Site testing Llama, Umwelt and regional targets. Drilling also took place on the George Site, at the Boot prospect and at the Boulder Prospect. A complete description of the 2013 drilling is discussed in detail in the Tetra Tech Technical Report 2014, along with plans and cross sections of the significantly drilled deposits (Llama, Umwelt, Echo, Locale 2, LCP South).

In 2014, Sabina completed 34 diamond drill holes at the Property totaling 12,173 m of core. Drilling was focused on the Goose Site, primarily targeting the Echo deposit. A complete description of the 2014 drilling is discussed in detail in the 2015 JDS Feasibility Study with an effective date of May 20, 2015.

In 2015, Sabina completed 38 diamond drill holes totaling 943 m at the Goose Site. While this program was primarily a geotechnical program for site investigation purposes, two drill holes were exploration holes and one was a geomechanical hole. One exploration drill hole, at Hivogani, was drilled approximately 1,400 m southwest of the Goose Main Deposit. It was targeted outside of the main iron formation which is the host rock to the majority Back River's current resources, on a broad induced polarization (IP) anomaly. The drill hole encountered a broad interval of anomalous gold values over greater than 100 m, hosted in altered, quartz-veined and structurally deformed clastic sedimentary and felsic intrusive rock units. Significant gold values from the hole include 0.82 g/t gold over 13.75 m. The other exploration hole at Nalaot was collared approximately 250 m west of the Echo deposit and was following up an earlier drill hole which returned 4.52 g/t gold over 5.00 m. In addition to the drilling at the Goose Site, 11 diamond drill holes for geotechnical investigation purposes were drilled at the MLA, but these only totaled approximately 16 m and no assaying was carried out on this core.

In 2015, downhole surveys were carried out for holes longer than 20 m using a downhole gyro system. Collars at the Goose Site were surveyed with a DGPS. Collars at the MLA were located with a handheld GPS.

Highlights of the 2012-2015 drilling include the following:

- Infill drilling at the Llama and Umwelt deposits converted Inferred and Indicated Mineral Resources into Indicated and Measured Mineral Resources within the pits;
- Infill drilling at the LCP South deposit extended mineralization to surface and converted Inferred Mineral Resources to Indicated Mineral Resources;
- Extensional drilling at the Umwelt deposit that expanded the Mineral Resources near surface and at depth;
- Extensional drilling at the Llama deposit that expanded the Mineral Resources at depth;
- Infill drilling at the Locales 1 and 2 deposits converted Inferred Mineral Resources to Indicated Mineral Resources;
- Extensional drilling at the Locale 2 deposit resulted in additional Indicated and Inferred Mineral Resources; and
- Exploration and subsequent extensional drilling at the Echo deposit that resulted in Indicated and Inferred Mineral Resources.

Plans and cross sections for the three largest deposits on the Goose Site are provided in the following sections below.

10.3 Goose Site

The Goose Site consists of the Goose Main, Llama, Umwelt, and Echo deposits, as well as the Hivogani, Wing Zone, Goose Neck, Goose Hook, and Camp Zone showings. This Initial Project Feasibility Study focuses on developing the Goose Main, Llama and Umwelt deposits, so plans and cross sections for the Goose Main, Llama and Umwelt deposits are presented below as well as a generalized location plan of the Goose deposits. The drilling in these deposits enables a robust interpretation of the geology and mineralized zones to be carried out. Core recovery for the diamond drilling at the Goose Site is excellent with the average core recoveries over time being 97.4%.

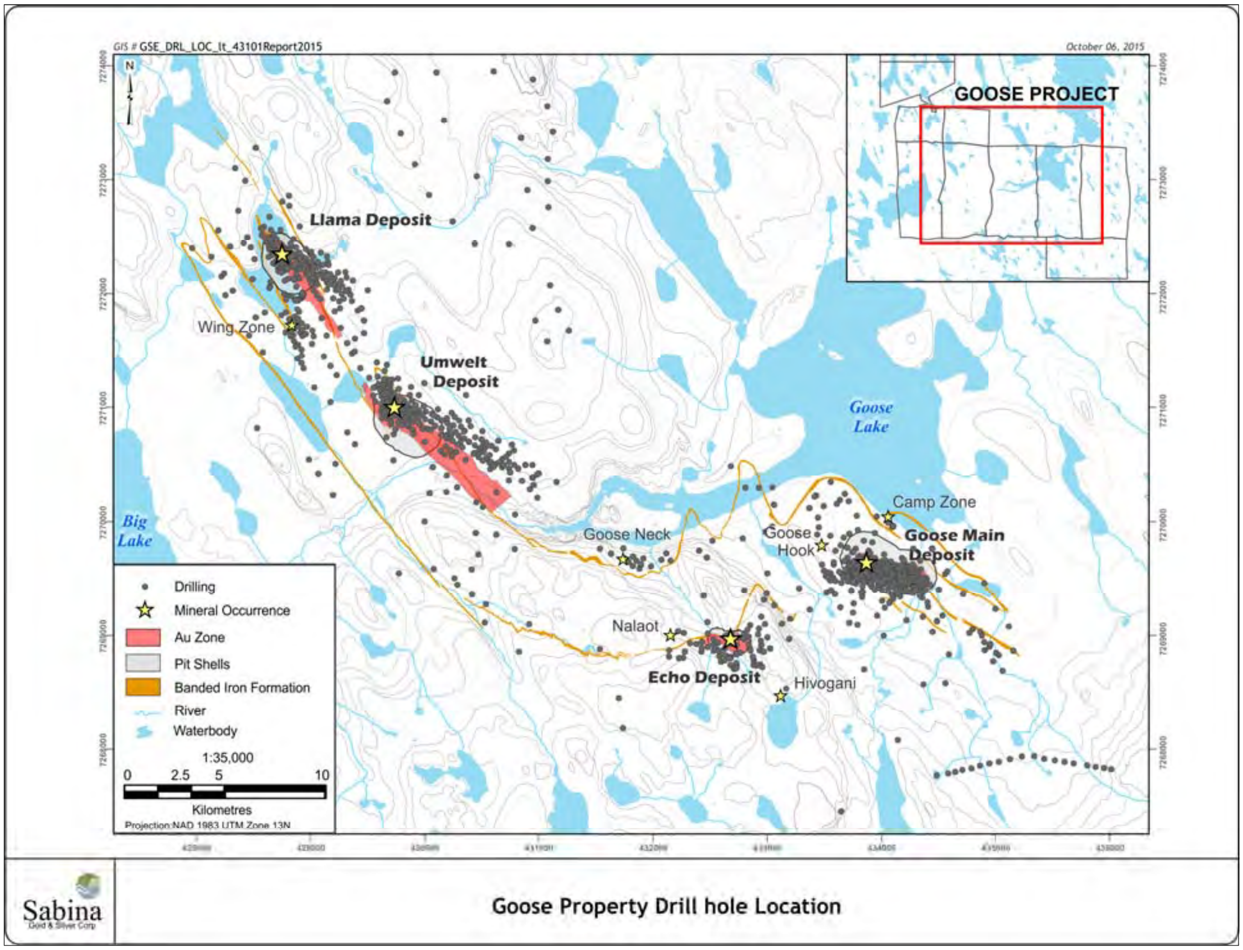
The drill spacing over the deposits at the Goose Site forms a notional grid as follows:

- 15 to 30 m grid north by 30 m grid east over the Goose Main deposit;
- 25 to 50 m grid north by 25 to 50 m grid east over the Llama deposit;
- 25 to 50 m grid north by 25 to 50 m grid east over the Umwelt deposit; and
- 50 m by 50 m over the Echo zone.

Narrative and a complete set of plans and sections are to be found in Tetra Tech Technical Report 2014 and the 2015 JDS Feasibility Study with effective date of May 20, 2015.

Figure 10.1 shows all the drilling carried out on the Goose Site to date on plan with the 2014 and 2015 drill holes shown in bold.

Figure 10.1: Goose Site Diamond Drill Hole Plan

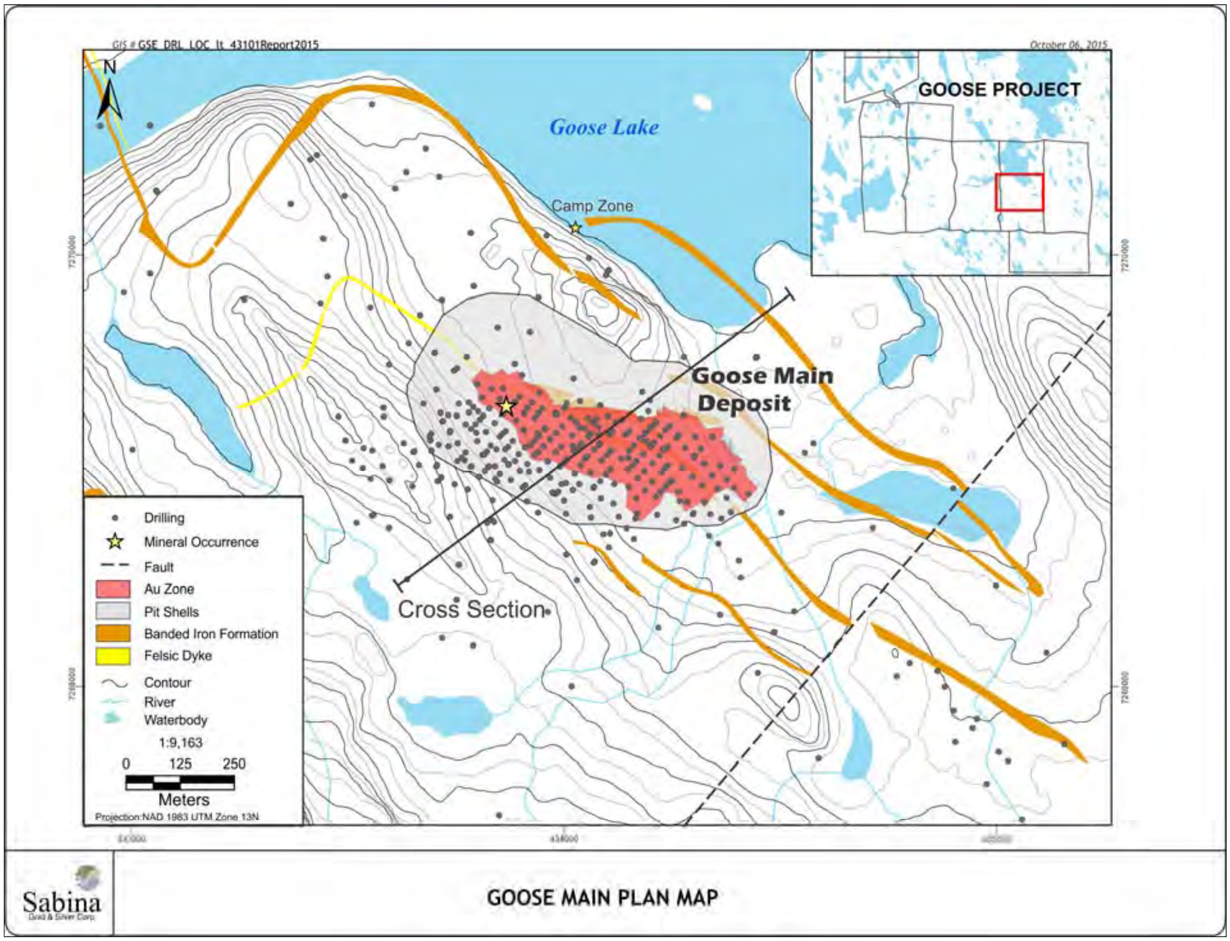


Source: Sabina Gold & Silver Corp. 2015

10.3.1 Goose Main Deposit

Drilling at the Goose Main deposit is comprised of a total of 333 drill holes for a total of 81,801 m. Of that total, 39 drill holes for 8,454 m were completed by Sabina. Collar locations of all drill holes for the Goose Main deposit are shown in plan in Figure 10.2. No drilling occurred on the Goose Main deposit in 2015.

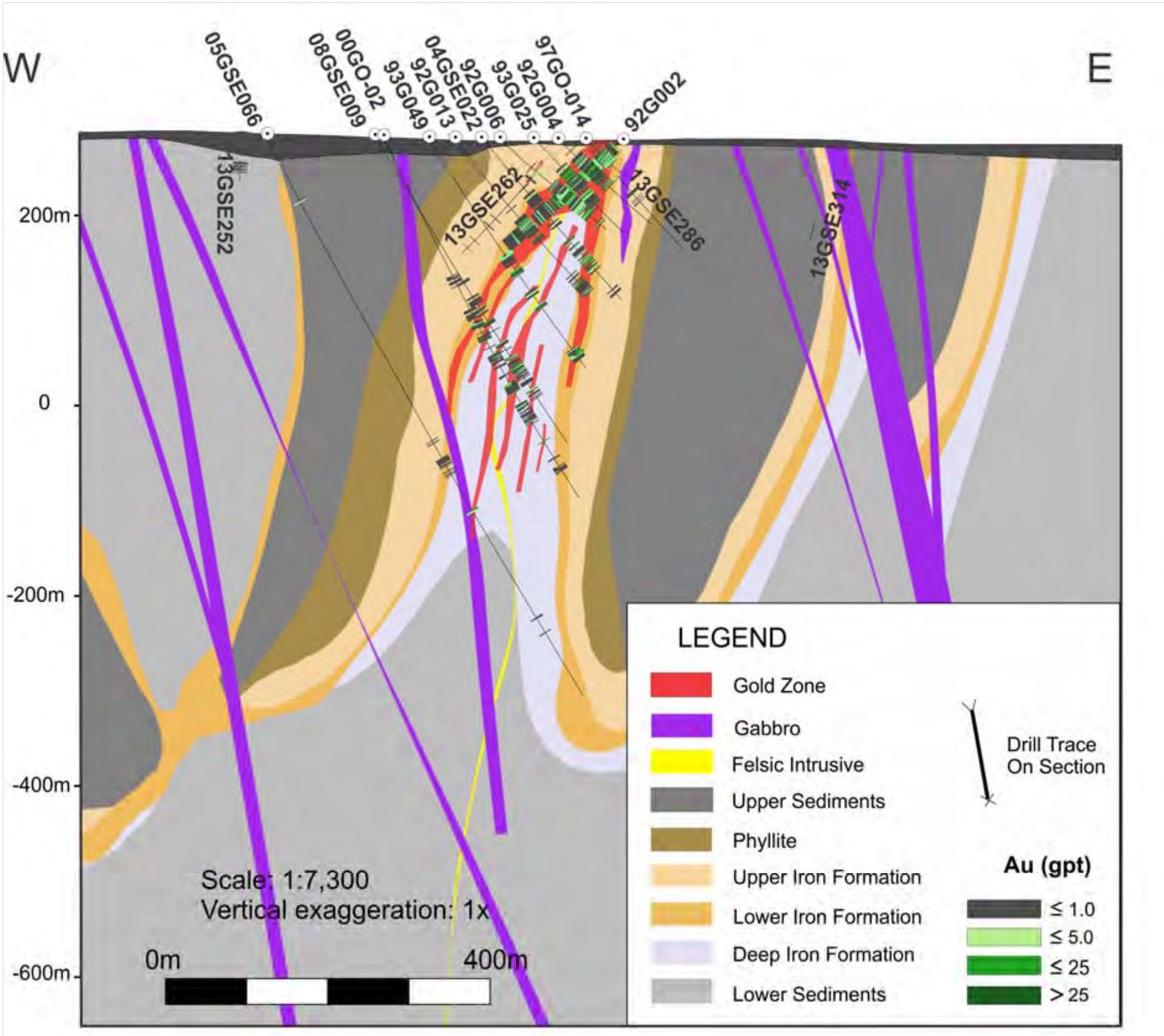
Figure 10.2: Goose Main Diamond Drill Hole Collar Plan



Source: Sabina Gold & Silver Corp. 2015

A representative Goose Main cross-section is included as Figure 10.3, showing the complex geology and the angle of intersection with the mineralization attained from the surface drilling. The gold grades are shown down hole and the hole numbers include the year drilled. Holes post-2008 were drilled by Sabina.

Figure 10.3: Goose Main Section at 5,060 m North (local grid)

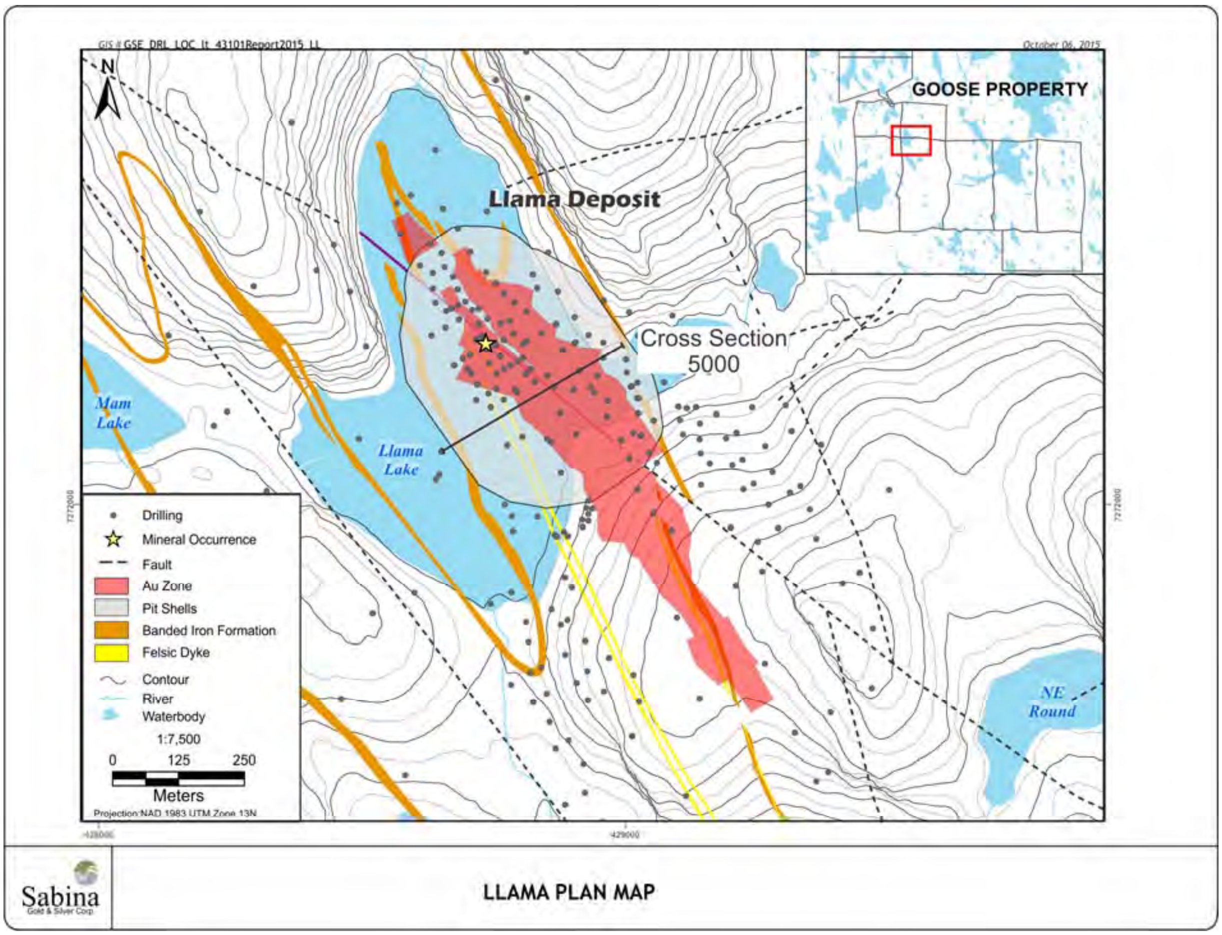


Source: Sabina Gold & Silver Corp. 2015

10.3.2 Llama Deposit

Drilling at the Llama deposit is comprised of a total of 219 drill holes for a total of 57,201 m. Of the total, 218 drill holes for 57,118 m were completed by Sabina. Collar locations of all drill holes for the Llama deposit are shown in Figure 10.4. One geomechanical hole was drilled at the Llama deposit in 2015.

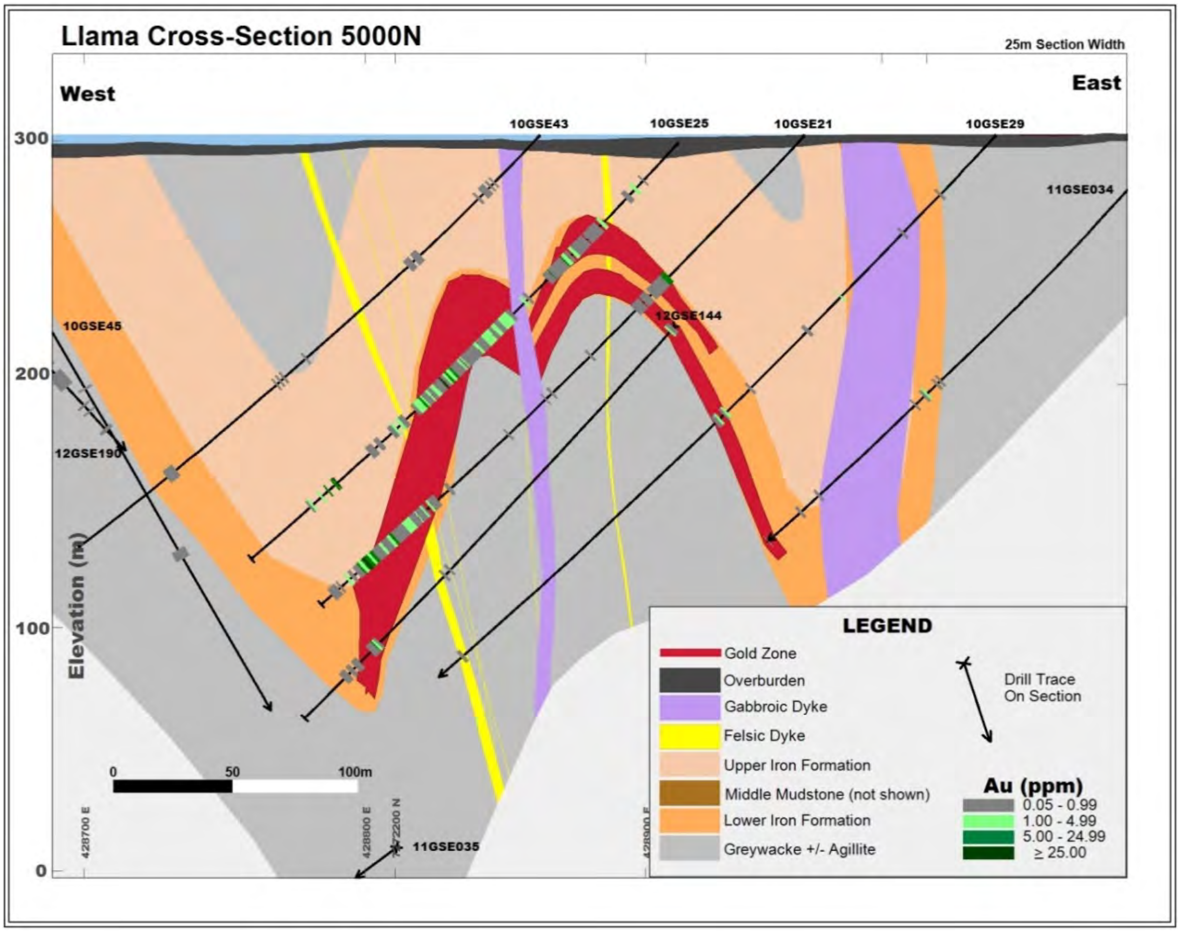
Figure 10.4: Llama Deposit Diamond Drill Hole Collar Plan



Source: Sabina Gold & Silver Corp. 2015

A representative Llama cross-section is included as Figure 10.5, showing the folded nature of the stratigraphy and the angle of intersection with the mineralization attained from the surface drilling. The gold grades are shown down hole and the hole numbers include the year drilled, such that the holes shown were drilled in 2010 and 2011.

Figure 10.5: Llama Section at 5,000 m North (local grid)

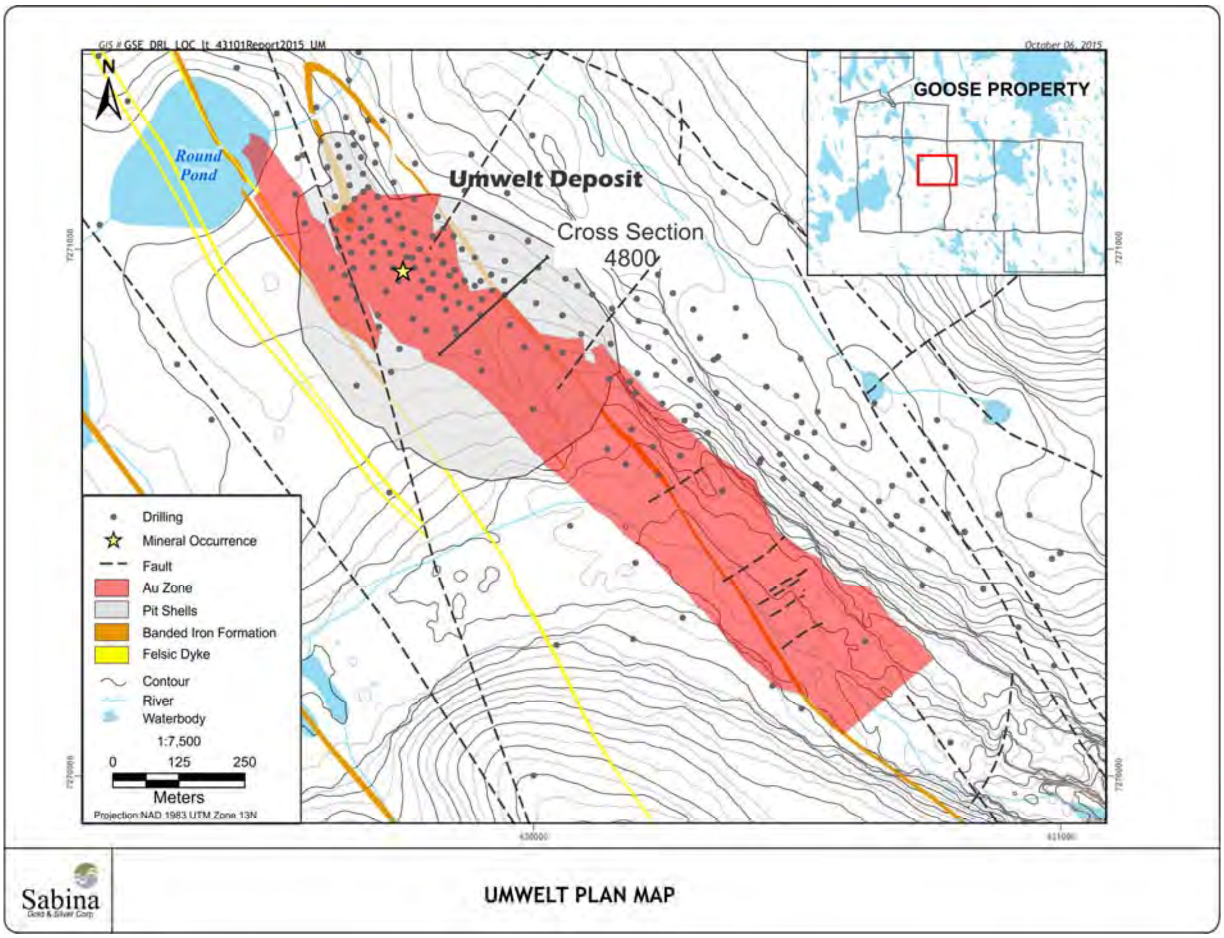


Source: Sabina Gold & Silver Corp. 2015

10.3.3 Umwelt Deposit

Drilling at the Umwelt deposit is comprised of a total of 241 drill holes for a total of 88,991 m. Of the total, 238 drill holes for 83,387 m were completed by Sabina. Collar locations of all drill holes for the Umwelt deposit are shown in Figure 10.6. No drilling occurred on the Umwelt deposit in 2015.

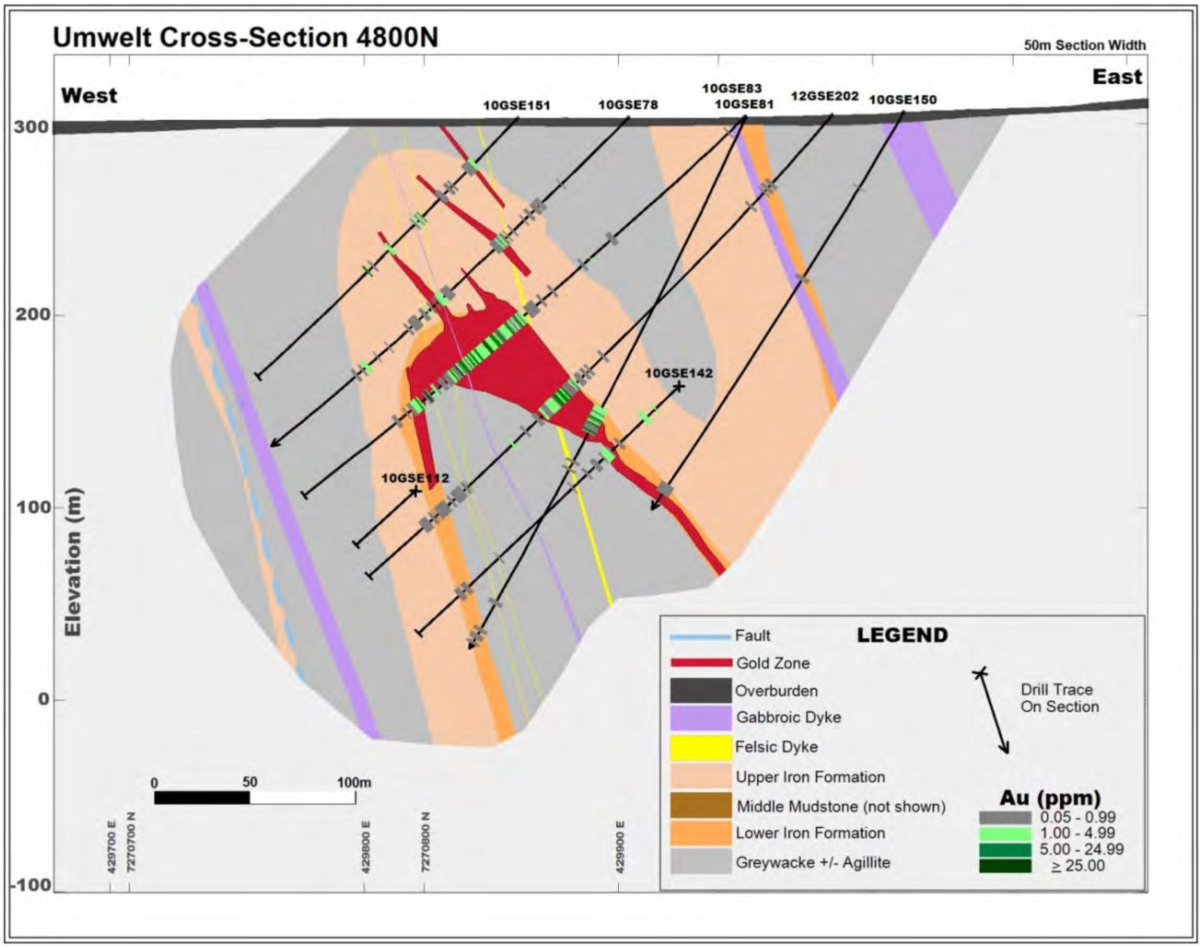
Figure 10.6: Umwelt Deposit Diamond Drill Hole Collar Plan



Source: Sabina Gold & Silver Corp. 2015

A representative Umwelt cross-section is included as Figure 10.7, showing the folded nature of the stratigraphy and the angle of intersection attained from the surface drilling. The gold grades are shown down hole and the holes on this section were drilled in 2010 and 2012. All holes on the section were drilled by Sabina.

Figure 10.7: Umwelt Section at 4,800 m North (local grid)



Source: Sabina Gold & Silver Corp. 2015

10.4 George Site

The George Site consists of the Locale 1, Locale 2, LCP North, LCP South, GH and Slave deposits as well as numerous showings. This Initial Project Feasibility Study focuses on development of the Umwelt, Llama and Goose Main deposits at the Goose Site, so only a location plan and section of the individual George deposits is included here for completeness. Narrative and a complete set of plans and sections are to be found in Tetra Tech Technical Report 2014 and the JDS Feasibility Study dated June 22, 2015, with an effective date of May 20, 2015.

The drilling in these deposits as well as field mapping and geophysical interpretation enables a robust interpretation of the geology and mineralized zones to be carried out. Core recovery for the diamond drilling at the George Site is excellent with the average core recoveries over time being 99.6%.

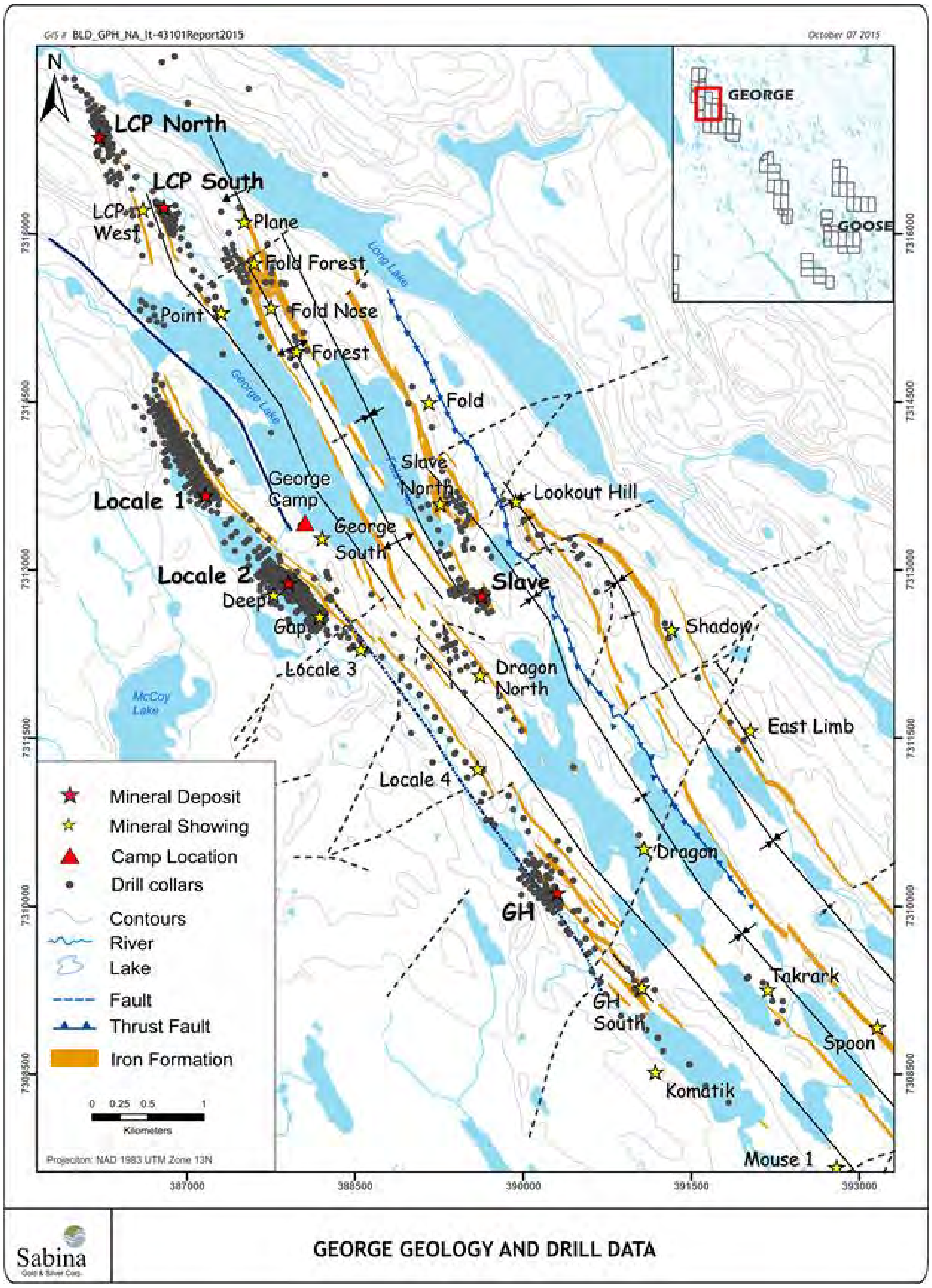
There was no new drilling at the George Site in 2014 or 2015.

The drill spacing over the deposits at the George Site forms a notional grid as follows:

- 30 to 60 m grid north by 30 m grid east over the LCP North (LCPn) deposit;
- 40 to 80 m grid north by 30 m grid east over the LCP South (LCPs) deposit;
- 30 to 60 m grid north by 30 to 60 m grid east over the two Locale deposits;
- 30 m grid north by 30 m grid east over the GH deposit; and
- 30 to 60 m north by 30 m east over the Slave deposit.

The locations of named prospects and drilling locations for the George Site are shown in Figure 10.8. No diamond drilling was carried out at the George Site in 2014 or 2015.

Figure 10.8: George Site Prospects and Drill Hole Locations



Source: Sabina Gold & Silver Corp. 2015

10.5 Diamond Core Sampling and Logging

Diamond core was sampled and logged according to the following process:

1. The diamond core was placed in appropriately labelled wooden core trays at the drill rig prior to transport to the closest camp and core facility;
2. All geological information was captured digitally and updated to a Microsoft Access® database;
3. All diamond core was photographed using a digital camera and the images were stored on the company server;
4. Geological information was captured, including lithology, veining, description of specific structures and alteration styles, along with the width, intensity, and associated mineral assemblage;
5. Rock quality designation (RQD) measurements were undertaken to record the number and nature of natural breaks in the core for subsequent geomechanical assessment. Other data collected included magnetic susceptibility, conductivity and recovery;
6. The core was sampled predominantly on the basis of geological logging with sample intervals ranging from a minimum sample length of 30 cm and a maximum sample length of 150 cm, with an optimal sample length of 100 cm;
7. The orientation of the core in the core box was maintained for sampling consistency;
8. Once all technical data had been derived from the core, selected sections were then halved lengthwise using a diamond saw to consistently cut along the top of the core or along the orientation line (if the core was oriented), before being correctly placed back into the tray;
9. The half core was then sampled, ensuring that the same side was consistently sampled, and placed into bags labelled with the assigned sample number;
10. Two-part sample tickets were used during sampling. One half of the sample tag accompanied the sample, the other half was stapled into the box for reference;
11. The residual half core was catalogued and stored for reference purposes; and
12. The trays were consecutively stacked and clearly re-labelled with the drill hole number, tray number and interval.

AMC considers this drill hole sampling to be representative and not biased. The sampling density was adequate for this type of deposit.

The core processing area is well laid out and equipped (as shown in Figure 10.9 and Figure 10.10) with a good reception area to receive core, a well-lit core logging area, and an efficient process through all of the data capture process, from photography to the cutting area.

Figure 10.9: Core Logging Facility

Source: AMC Mining Consultants Canada Ltd. 2013

Figure 10.10: Photography and Conveyor to Cutting Area

Source: AMC Mining Consultants Canada Ltd. 2013

10.6 Bulk Density Measurements

TSL Labs measured specific gravity by air-weighing the core, then weighing the core in water while suspended beneath scales. Core was not sealed prior to immersion. Previous studies have shown that there is no material difference between the measurements of sealed versus unsealed core (Cater et al., 2009); therefore, AMC considers the specific gravity measurements to be suitable for use as dry bulk density measurements and can be used to generate tonnages in the Mineral Resource estimate.

During the 2011-2014 field seasons, Sabina measured specific gravity on site and in 2012, sent 4% of the specific gravity samples to TSL Labs for comparative test work. Measurements from TSL Labs compared well with those on site, with the correlation coefficients ranging from 0.93 to 0.96 (Pacor, 2012). In 2014, 559 samples were collected for the Echo deposit resulting in a total of 12,820 measurements for the Goose Site; a total of 4,746 measurements have been collected for the George Site.

No specific gravity measurements were collected during the 2015 drilling season due to the holes locations and small number of metres drilled.

Local values were used for applying bulk density measurements to a deposit, provided that the number of samples was greater than 100. This methodology is described in section 14.2.1 for the Goose deposits and in section 14.3.1 for the George deposits.

10.7 Database Review

10.7.1 Re-Compilation of Historical Logged Data by Sabina

In 2013, all 1985 to 2009 drill hole logged data for the Goose Site was re-compiled from various historical digital sources in order to populate the GEOVIA GEMS™ databases with supporting data for ongoing exploration. The compilation of the data is complete for the Goose Site but not carried out for the George Site.

10.7.2 Historical Collar and Down Hole Survey Review by Sabina

For both the Goose and George sites, all survey data from the period 1985 to 2012 was checked by Sabina in 2013. Drill hole survey data was checked against original drill logs, reports and survey files where available. Select collars were resurveyed in the field as a cross-check. Where material errors were found, corrections were applied to the GEOVIA GEMS™ database. Final GEOVIA GEMS™ database correction files for each area were proofed and validated by various Sabina team members.

11 Sample Preparation, Analyses and Security

During 2015, Sabina drilled 38 diamond drill holes totaling 942.7 m, of which two were exploration drill holes, and one a long geomechanical hole. The others were short geomechanical holes drilled for site investigation purposes. This metreage of drilling did not provide enough assayed samples and thus a quantity of quality control samples to merit reviewing in control charts on their own in this report. Thus the data for both the 2014 and 2015 drilling campaigns are discussed here.

11.1 Historical

For the historical datasets, independent reviews were completed by RPA (2011), Coffey Mining (2009), and WGM (2003 and 2005). Based on these reviews, AMC is satisfied that the exploration approach and sample data are of sufficient quality for inclusion in resource evaluation studies. A summary of the historical sample preparation, analyses and security is presented in the SRK 2012 PEA technical report. A summary of the 2012 sample preparation, analyses and security is presented in the Tetra Tech 2013 PFS Technical Report. A summary of the 2013 sample preparation, analyses and security is presented in the Tetra Tech Mineral Resource Update 2014.

11.2 Sample Shipment and Security

Sabina's sample shipment and security procedure involved direct drill and sample management, secure transportation methods, sampling and logging areas, and sample storage facilities.

Sabina continued to use TSL Labs as the primary laboratory for the 2015 drilling programs. Instructions were provided to the laboratory using detailed requisition forms outlining procedures for sample preparation and assay.

Samples were transported by charter aircraft from the Property to Yellowknife. The samples were then transported by either air freight or transport truck from Yellowknife to the laboratory. An established chain of custody was employed to ensure the safe and secure transport and delivery of core samples to the laboratory.

TSL Labs in turn sent notifications of receipt to confirm the arrival of samples at the laboratory.

AMC considers that Sabina's shipment procedures provide adequate security for the samples used in the Mineral Resource estimate.

11.3 Sample Preparation and Analysis

At TSL Labs, samples were prepared using a standard rock preparation procedure (drying, weighing, crushing, splitting, and pulverizing) and assayed for gold by fire assay with gravimetric finish based on a 58.32 g sub-sample. TSL Labs is accredited with International Standards Organization (ISO) 17025 by the Standards Council of Canada for a number of specific test procedures, including the method used to assay the samples submitted by Sabina. TSL Labs also participates in a number of international proficiency tests, such as those managed by CANMET Materials Technology Laboratory, Geostats Pty Ltd., Rocklabs and Ore Research and Exploration Pty Ltd.

The sample preparation procedure was as follows:

- Samples are crushed in oscillating jaw crushers then roll crushers to achieve 95% passing ten mesh (1.70 mm);
- Samples are riffle split; typically, a 1,000-g sub-sample is split off for pulverizing and the remaining coarse material is stored as reject;
- Disc-mill pulverizers ground samples to 95% passing 150 mesh (106 µm);
- Gravimetric gold analysis began with a flux mixture of litharge, soda, borax, silica and fluorspar with further oxidants or reductants added as required;
- Crucibles are placed into trays of 24 and approximately 160 g of flux is added;
- Samples are weighed into crucibles and then mixed;
- After mixing, the samples are removed, inquarted with a silver collection agent, and fused;
- The resultant lead button is then cupelled;
- After cupellation, the subsequent doré bead is flattened, placed in a porcelain cup, and parted with a dilute nitric acid solution; and
- The obtained gold is decanted with de-ionized water, dried, annealed, and weighed on a microbalance.

In addition to fire assay analysis, a selected subset of samples is sent for multi-element ICP-MS analysis with a four-acid total digestion and/or whole-rock analysis. The purpose of these analyses is varied and includes understanding acid rock drainage, alteration vectoring to ore, felsic dyke classification, and identification of altered rocks. This work has been discussed in Section 9 of this and previous reports.

11.4 Quality Assurance and Quality Control

For its 2014 and 2015 exploration programs, Sabina routinely inserted Certified Reference Materials (CRMs) and blanks into each batch. Both standards and blanks are inserted every 20 samples to monitor sample preparation and assay procedures. An additional blank is inserted after each sample in which visible gold is observed. The numbers in the following sentences show the total 2014 / 2015 numbers followed by the 2015 contribution in brackets. TSL Labs ran 336 (25) internal laboratory pulp duplicates to test for repeatability. Sabina also selected 167 (10) pulp duplicate samples and 141 (8) coarse duplicate samples to be re-assayed by TSL Laboratories. External check assays were arranged by sending a selection of samples that returned greater than 0.2 g/t gold to Acme Analytical Laboratories Ltd (AcmeLabs) in Vancouver, now part of Bureau Veritas Commodities Canada Ltd., see discussion below. This resulted in 50 pulp duplicates to test for accuracy of the primary laboratory. Sabina's quality assurance/quality control (QA/QC) program comprised 1,424 (305) samples, of which 613 (20) CRMs and 503 (15) blanks were inserted into the sample stream of 9,111 (270) core samples.

While AcmeLabs has been integrated into Bureau Veritas Commodities Canada Ltd. AcmeLabs continue to operate as part of Bureau Veritas Minerals and the name AcmeLabs continues to be used for business and thus in this report.

AcmeLabs is accredited with the ISO 9001 Model for Quality Assurance and ISO/IEC 17025 General Requirements for the Competence of Testing and Calibration Laboratories.

11.4.1 Assay Results of Certified Reference Materials

A total of 13 different CRMs were used for the 2014 and 2015 drilling; however, only five standards were used per hole. The CRMs were sourced predominantly from Geostats Pty Ltd., with a few CRMs from CDN Laboratories. Their names and recommended values are listed in Table 11.1. along with the number of CRM's inserted by year and site.

Table 11.1: Recommended Values of Certified Reference Materials

CRM	Au (g/t)	Standard Deviation	No. of Assays			Total
			Goose 2014	Goose 2015	George 2014	
G908-2	0.21	0.01	75		40	115
G907-2	0.89	0.06	83		43	126
G310-5	1.01	0.05	4		3	7
CDN GS-1b	1.02	0.04	17		0	17
G300-9	1.53	0.06	16	6	3	25
G307-8	1.99	0.08	18	8	1	27
G308-3	2.5	0.11	65		17	82
G907-3	2.88	0.11	6		2	8
G310-1	4.94	0.22	18	6	0	24
CDN GS-5a	5.1	0.14	9		0	9
G308-4	6.77	0.29	63		17	80
CDN GS-14	7.47	0.16	9		0	9
G910-4	16.92	0.72	67		17	84
Total			**450**	**20**	**143**	**613**

Source: AMC Mining Consultants (Canada) Ltd. 2015

CRMs should be obtained for all economic minerals. For each economic mineral, there should be three corresponding standards:

- At around the expected cut-off grade of the deposit;
- At the expected average grade of the deposit; and
- At a higher grade.

In AMC's opinion, the CRMs should represent approximately 5% of the total samples assayed. CRM results should be reviewed immediately upon receipt of assay. AMC recommends that assay batches with two CRMs outside two standard deviations should be re-run, and that assay batches with one CRM outside three standard deviations should also be re-run. As CRM data accumulates over time, results should be reviewed for biases in the data. Table 11.2 shows the results of Sabina's CRMs.

Table 11.2: 2014 and 2015 Assay Results of Certified Reference Materials for Gold

CRM	Expected Au Value (g/t)	No. of Assays	Warnings	Fails	Mislabeled	True Fails
G908-2	0.21	115	21*	1	1	0
G907-2	0.89	126	0	1	1	0
G310-5	1.01	7	0	0	0	0
CDN GS-1b	1.02	17	0	0	0	0
G300-9	1.53	25	2	0	0	0
G307-8	1.99	27	1	0	0	0
G308-3	2.50	82	0	1	1	0
G907-3	2.88	8	0	0	0	0
G310-1	4.94	24	0	0	0	0
CDN GS-5a	5.10	9	0	0	0	0
G308-4	6.77	80	0	0	0	0
CDN GS-14	7.47	9	1	0	0	0
G910-4	16.92	84	0	0	0	0
Other**	n/a	3	0	0	3	0
Total		**616**	**25**	**3**	**6**	**0**

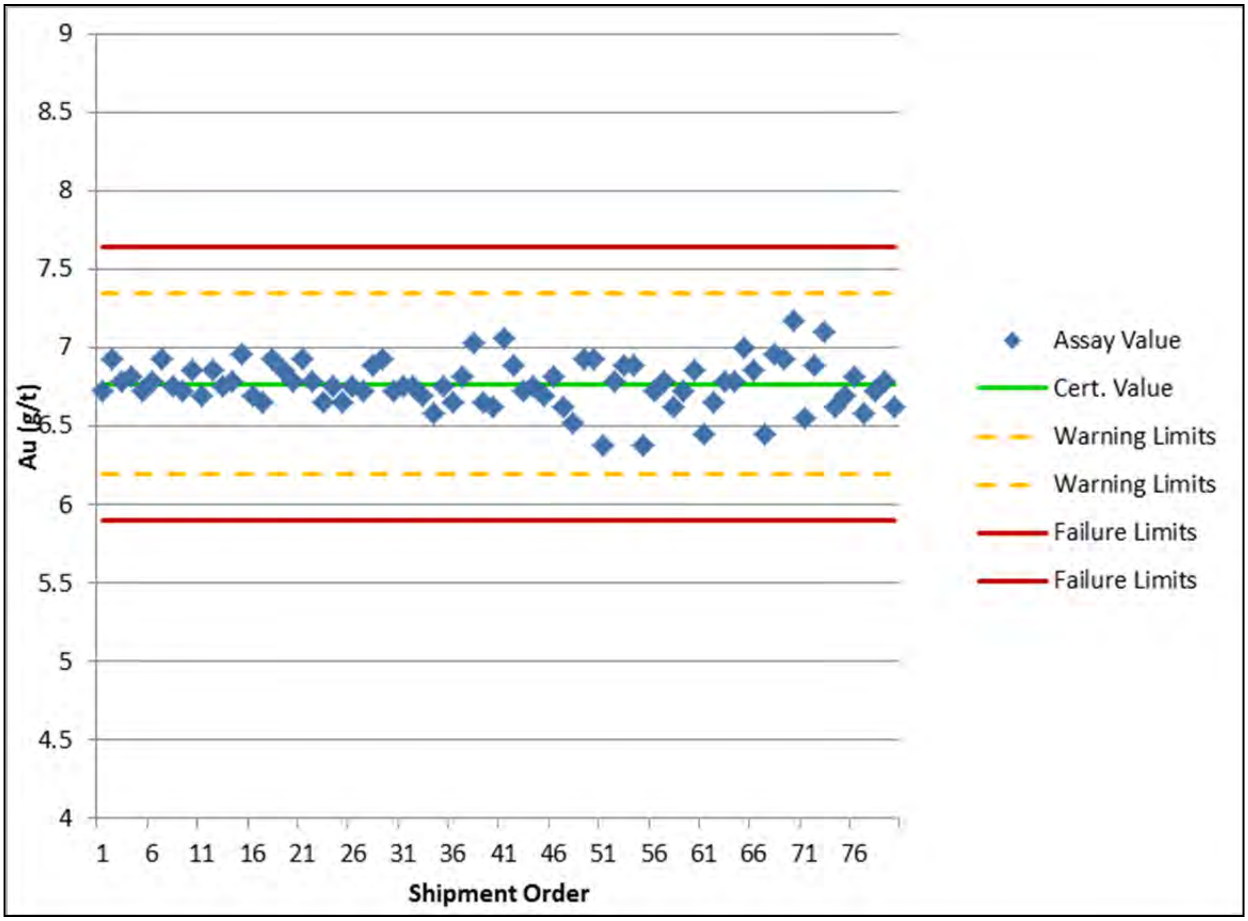
Notes: *Includes instances of consecutive warnings.
**Three standards were not labelled, and not included in the total in Table 11.1.
Source: AMC Mining Consultants (Canada) Ltd.

Sabina has provided CRMs for gold at a variety of values. As the cut-off grade of the Mineral Resource in the open pits is 1.0 g/t (section 14.0), AMC recommends that CRM G908-2 be discontinued, as this CRM is very low grade and is therefore monitoring how accurately the laboratory measures waste. The high number of warnings for this CRM is also noted; this is likely due to the CRM value being near the detection limit for a gravimetric fire assay.

Sabina's CRMs represent over 5% of the total samples assayed, and CRMs are reviewed immediately upon receipt of assay results. Where required, Sabina has re-run assay batches. The rate of failure is low, and represents less than 0.5% of total CRM assays, and with no failures in 2015. Sabina also monitors CRM data accumulated over time.

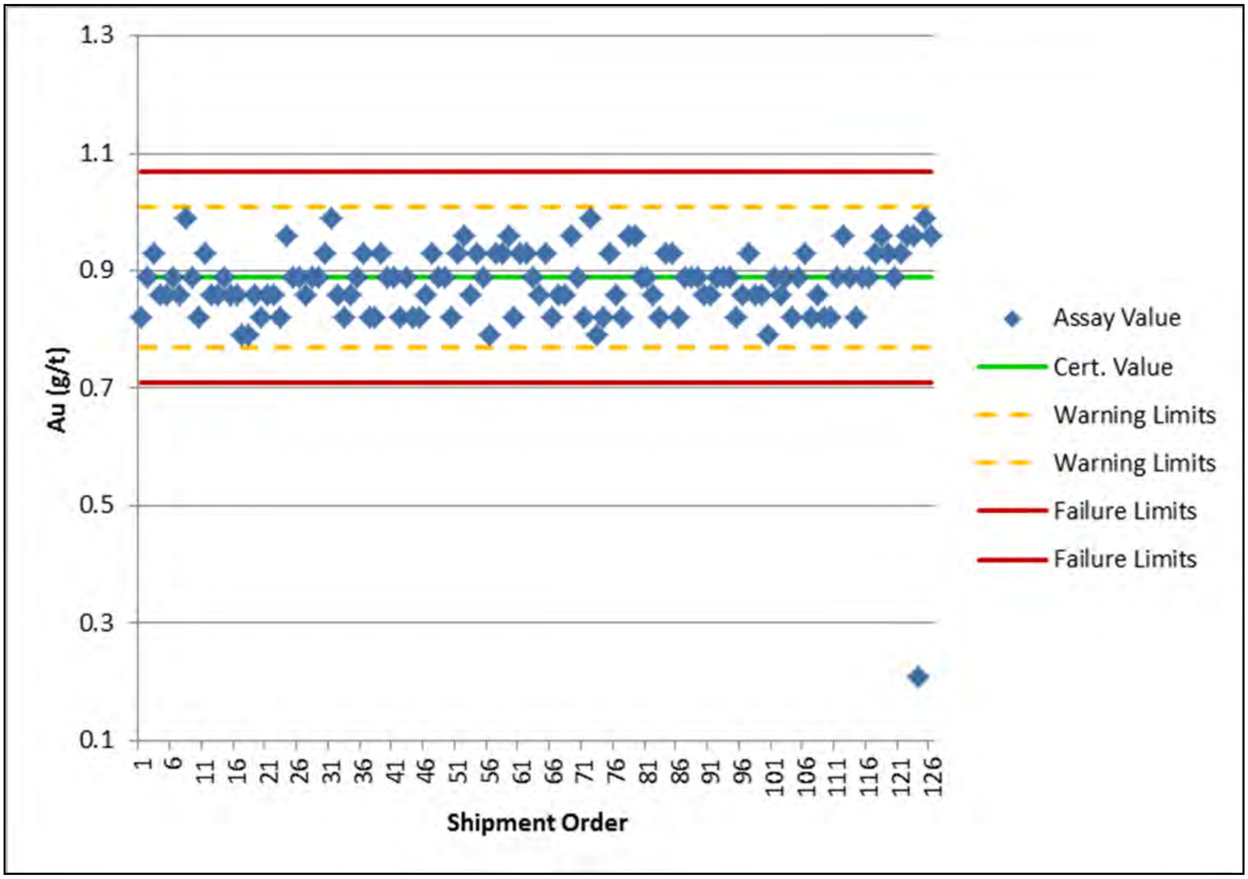
Figure 11.1 and Figure 11.2 show the results of 80 assays on CRM G308-4 and 126 assays on G907-2. The first CRM approximates the average grade of the deposits and the second CRM approximates the cut-off grade of the open pit Mineral Resources.

Figure 11.1: Control Chart for 2014 / 2015 G308-4



Source: AMC Mining Consultants (Canada) Ltd., 2015

Figure 11.2: Control Chart for 2014 / 2015 CRM G907-2



Note: Failed sample is likely incorrectly labelled CRM G908-2
Source: AMC Mining Consultants (Canada) Ltd. 2015

11.4.2 Assay Results of Blank Samples

Coarse blanks test for contamination during both the sample preparation and assay process. Pulp blanks test for contamination during the assay process. Blanks should be inserted in each batch sent to the laboratory with an additional coarse blank inserted immediately after expected high-grade samples. In AMC's opinion, the "pass" requirement for blanks is that 80% of the coarse blank assays should be less than twice the detection limit for that element.

Blanks were taken from unmineralized sections of core (mainly Proterozoic cover or late gabbro dykes) that had been removed from the core boxes and kept in bins in the sampling room.

A total of 503 blank samples were inserted into the sample batches in 2014 and 2015 to monitor possible contamination problems in sample preparation procedures. Two blank samples were detected with anomalous values of gold. As 99.6% of the coarse blank assays were less than twice the detection limit for gold, the assay results of the blank materials are considered acceptable. Table 11.3 shows the assay results for blank material from the Property.

Table 11.3: 2014 and 2015 Assay Results of Blank Material for Gold

	Au (g/t)	No. of Assays			
		Goose 2014	Goose 2015	George 2014	Total
Blank	0	414	15	74	503
Fails	>0.06	2	0	0	2

Note: Detection limit for gold is 0.03 g/t.
Source: AMC Mining Consultants (Canada) Ltd.

Although Sabina did not insert pulp blanks into its sample stream, the low failure rate suggests the laboratory has little or no contamination present in either the sample preparation or assay processes.

Blanks represent over 5% of the total number of samples, which is acceptable.

11.4.3 Assay Results of Duplicates

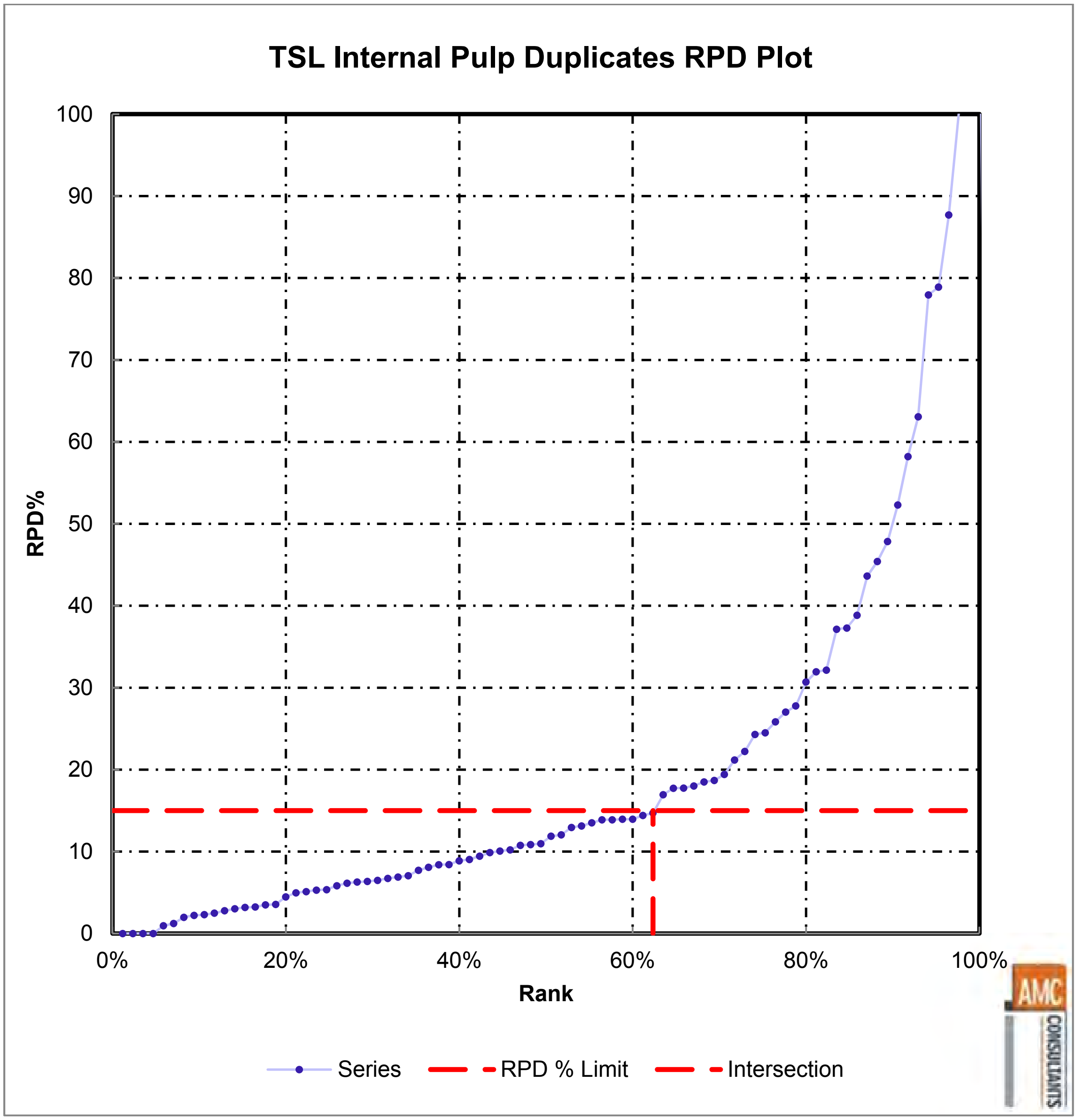
In AMC's opinion, duplicates should constitute around 5% of the samples submitted to the laboratory. Unmineralized samples should not be sent as duplicates because assays near the detection limit are commonly inaccurate. Duplicate data can be viewed on a scatterplot but should also be compared using the relative paired difference (RPD) plot. This method measures the absolute difference between a sample and its duplicate. It is desirable to achieve 80 to 85% of the pairs having less than 15% RPD between the original assay and check assay (Stoker 2006). Sample pairs should be excluded from the analysis if the combined mean of the pair is less than 15 times the detection limit (Kaufman and Stoker 2009). Removing the low values ensures that there is no undue influence on the RPD plots due to the higher variance of grades likely near to the detection limit, where precision becomes poorer (Long et al. 1997).

In 2014 and 2015 TSL Labs ran 336 internal laboratory pulps to test for repeatability. These samples were not based on gold assay results. As a result, only 210 pulp duplicates were above the detection limit. RPD plots are presented in Figure 11.3 for the pulp duplicate datasets.

AMC makes the following observations from the internal TSL Labs duplicate results:

- 64% of the pulp duplicate pairs were less than 15% RPD, which should be investigated
- No significant bias is observed between the original and duplicate assays.

Figure 11.3: Relative Paired Difference Plot for Internal TSL Lab, Pulp Duplicates



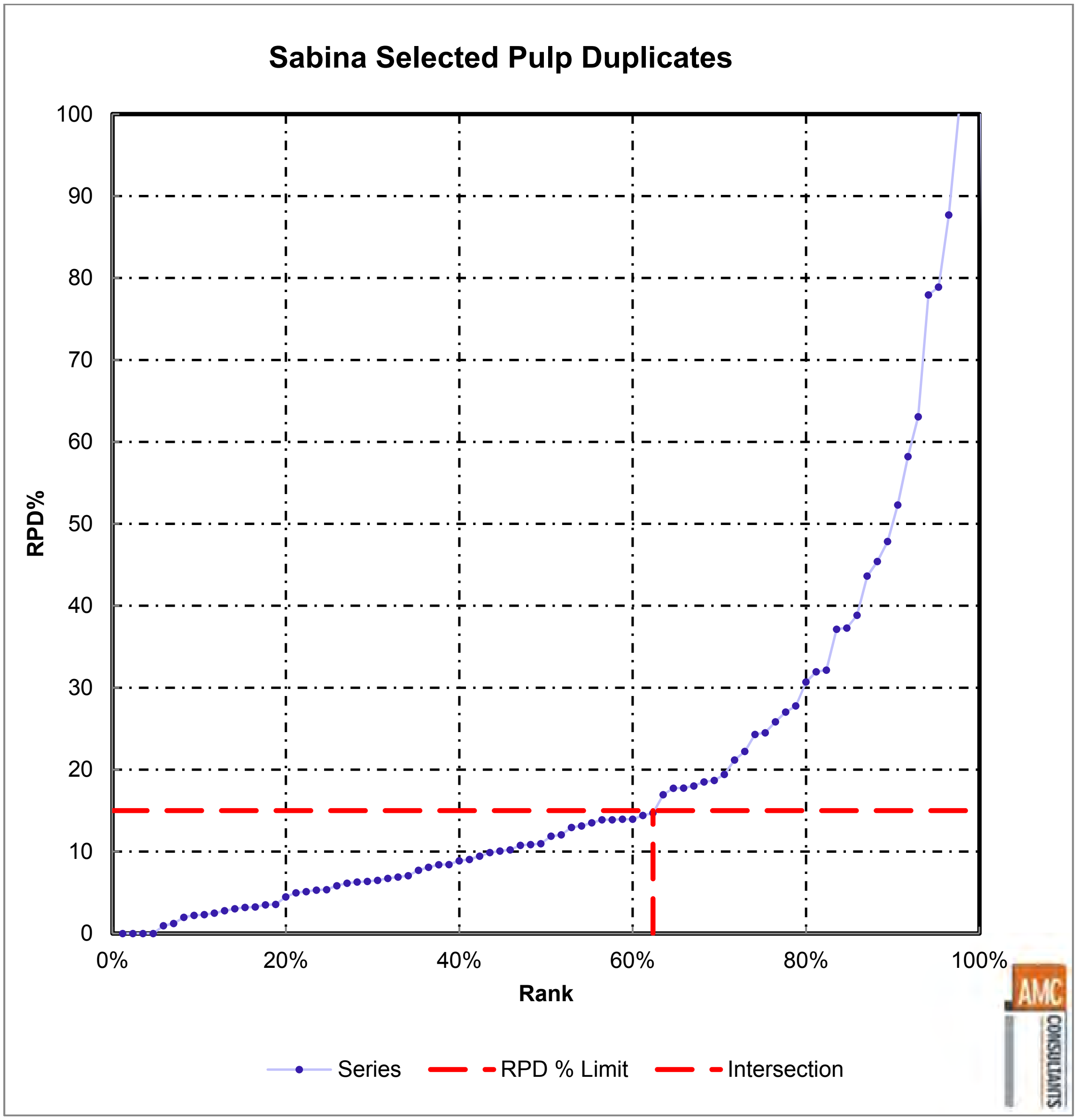
Source: AMC Mining Consultants (Canada) Ltd. 2015

In the same period of 2014 and 2015, Sabina selected 167 pulp duplicates and 141 coarse duplicates to test for repeatability. These samples were not based solely on gold assay results. As a result, only 110 pulp duplicates and 85 coarse duplicates were greater than the 15 times detection limit. RPD plots are presented in Figure 11.4 for the pulp duplicate datasets.

AMC makes the following observations based on the Sabina-selected pulp duplicate results:

- 72% of the pulp duplicate pairs were less than 15% RPD, which, though less than desirable, is an acceptable result;
- No significant bias is observed between the original and duplicate assays; and
- Given the limited number of samples (<100) that were > 15 x detection limit for the coarse duplicates, no conclusions can be drawn from the data.

Figure 11.4: Relative Paired Difference Plot for Sabina-Selected Pulp Duplicates



Source: AMC Mining Consultants (Canada) Ltd.2015

11.4.4 Results of External Check Assays

11.4.4.1 2014 External Check Assays

The purpose of a check laboratory is to increase the confidence in the accuracy of the primary laboratory (Long et al., 1997).

A total of 50 pulps of mineralized core were re-assayed at AcmeLabs in Vancouver as an external check. AcmeLabs is compliant with the ISO 9001 Model for Quality Assurance and ISO/International Electrotechnical Commission (IEC) 17025 General Requirements for the Competence of Testing and Calibration Laboratories. AMC makes the following observations from the results:

- 13 of the 50 samples were greater than 15 times the AcmeLabs detection limit of 0.9 g/t; and
- Given the limited number of samples that were greater than 15 times the detection limit, no conclusions can be drawn from the data.

The low number of umpire samples is due, in part, to the limited drilling in the 2014 field season as well as to the high detection limit of AcmeLabs. In prior years, there were a sufficient number of samples for statistically valid results; however, AMC recommends that in future programs, the laboratory method and detection level for samples sent to an umpire laboratory should be the same as those of the primary lab. The current umpire laboratory is only testing accuracy of samples with > 13.5 g/t Au (greater than 15 times the AcmeLabs detection limit of 0.9 g/t).

At time of writing no external checks had been carried out from the 2015 drilling, (270 core samples only).

11.4.4.2 Reproducibility Investigation

In the Tetra Tech Technical Report 2014, AMC observed no significant bias between the original and duplicate assays overall. However, it was noted that the four highest duplicate pairs suggested a difference in the laboratories. Although not material to the Mineral Resource estimate, AMC recommended the reproducibility between TSL Labs and AcmeLabs be investigated. Sabina investigated this problem in the 2014 data and sent AcmeLabs an additional 100 umpire samples which assayed greater than 20 g/t Au at TSL Labs Sabina also enlisted the assistance of Smee & Associates Consulting Ltd. (Smee). Smee could not offer guidance as to which laboratory was correct because the CRMs were not included in the check sample shipments sent to AcmeLabs (Smee, 2014a). In an effort to correct this, Sabina then resent 96 of the high grade check samples to AcmeLabs with CRMs inserted. The CRMs confirmed that the assay values at AcmeLabs were valid (Smee, 2014b).

Smee suggested that the bias of samples greater than 70 g/t Au towards TSL Labs may be caused by a nugget effect and if so, should not be cause for concern due to the bias being restricted to the high-grade portion of the assays. Sabina interrogated the assays used in the resource estimation and found that only 0.5% of all values lie above the 70 g/t Au threshold at which the bias was seen. In addition, AMC's practice of reviewing that data and capping where required removes many of the affected samples from any resource estimates. As such, the effect of this bias is deemed to be immaterial.

11.5 Conclusions

In the opinion of the QP, the sampling, sample preparation, security, and analytical procedures adopted by Sabina for its exploration programs are rigorous and meet or exceed accepted industry standards. In addition the QA/QC results confirm that the assay results may be relied upon for Mineral Resource estimation purposes.

12 Data Verification

On August 27 and 28, 2012, full-time AMC employee Mr. John Morton Shannon, P.Geo., visited the Property to undertake the following data verification steps:

- Discussions with site geologists regarding:
  - Sample collection;
  - Sample preparation;
  - Sample storage;
  - QA/QC;
  - Data validation procedures;
  - Mapping procedures;
  - Survey procedures;
  - Geological interpretation; and
  - Exploration strategy.
- An inspection of the core shed and drill core intersections from the Goose and George sites.
- For this update, Mr. Shannon reviewed the processes used in the data collection and handling in 2014 and undertook random cross-checks of assay results in the database with original assay results on the assay certificates returned from TSL Laboratories. This verification consisted of comparing 849 of the 7,363 assays for the 2014 drilling at Goose (12%) and 321 of the 1,480 assays for the gap sampling in 2014 at George (22%). No errors were detected.

AMC makes the following observations based on the data verification that was conducted in 2012 and from discussions on the work since:

- Site geologists are appropriately trained and are conscious of the specific sampling requirements of nuggety gold deposits;
- Procedures for data collection and storage are well-established and adhered to;
- QA/QC procedures are adequate and give confidence in the assay results; and
- Cross-checking a sample set of the database with the original assay results uncovered no errors.

The QP considers the database fit-for-purpose and in the QP's opinion, the geological data provided by Sabina for the purposes of Mineral Resource estimation were collected in line with industry best practice as defined in the CIM Exploration Best Practice Guidelines and the CIM Mineral Resource, Mineral Reserve Best Practice Guidelines. As such, the data are suitable for use in the estimation of Mineral Resources.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

This section of the report describes metallurgical test work carried out to support development of the Back River Project process facility. The metallurgical test work forming the basis of this section was supervised by Stacy Freudigmann, P.Eng., Canenco Canada Inc. (Canenco), working in conjunction with the team from Sabina. The mineralogical description in this section of the report was also supplied by Canenco. The results of the test work together with financial evaluation data was used to develop metallurgical design criteria. The majority of this section was previously written by Hatch for the JDS NI 43-101 report “Technical Report and Feasibility Study for the Back River Gold Property, Nunavut” published in June 2015. Canenco has edited that version for this technical report.

As permitted by Item 3 of Form 43-101F1 – Technical Report, published by the Canadian Securities Administrators (“Form 43-101F1”), the Qualified Person (QP) responsible for the preparation of this section has relied upon certain reports, opinions and statements of certain experts who are not qualified persons. These reports, opinions and statements, the makers of each such report, opinion or statement and the extent of reliance are described. The QP authoring this section hereby disclaims liability for such reports, opinions and statement to the extent that they have been relied upon in the preparation of this report, as described herein.

This section of the report contains estimates, projections and conclusions that contains forward-looking information within the meaning of applicable securities laws. Forward-looking statements are based upon the responsible QP’s opinion at the time that they are made but in most cases involve significant risk and uncertainty. Although the QP has attempted to identify factors that could cause actual events or results to differ materially from those described in this report, there may be other factors that cause events or results to not be as anticipated, estimated or projected. There can be no assurance that forward-looking information in this section of the report will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements or information. Accordingly, readers should not place undue reliance on forward-looking information. Forward-looking information is made as of the effective date of this report and the QP does not assume any obligation to update or revise it to reflect new events or circumstances, unless otherwise required under applicable laws.

13.1.1 Mine Plan

The Back River Gold Project contains 12.4 Mt of ore with an average gold head grade of 6.3 g/t (Table 13.1).

Table 13.1: Feed Scheduled to the Process Facility

	Upper Zones			Lower Zones			Total	
Mine	**Reserves (kt)**	**Timing (year)**	**Gold Head Grade (g/t)**	**Reserves (kt)**	**Timing (year)**	**Gold Head Grade (g/t)**	**Reserves (kt)**	**Gold Head Grade (g/t)**
Umwelt	2,668	-1,1,2	6.49	3,492	3-9	7.38	6,160	7.00
Llama	1,749	1-5	7.15				1,749	7.15
Goose Main	4,451	3-8	5.00				4,451	5.00
Total/Average	8,868		5.87	3,492		7.38	12,359	6.30

Source: JDS 2015

The mining section of this report, chapter 16, details the scheduling of open pit and underground reserves.

13.1.2 Testing History

In earlier stages of project development, testing was undertaken on composites from each of the ore bodies designated in the mine plan. More recent test work was carried out to determine the variability of individual samples selected lithologically and spatially that represent the resource. Earlier testing was designed to optimize the Prefeasibility Study (PFS) flowsheet and the resulting flowsheet was then used to determine the metallurgical performance of a number of variability composites.

Most of the metallurgical test work evaluated both silver and gold recovery. Silver has not been included in the mine plan as there was insufficient assaying of the borehole samples. No credit for silver has therefore been claimed in the Project revenue. The main focus in this section is therefore on gold and the optimization of gold recovery.

A substantial amount of testing was done on the Back River Project as summarized in Table 13.2.

Table 13.2: Summary of Test Work Completed

ALS = ALS Metallurgy; Gekko = Gekko Systems Ltd.; Geoscience = Geoscience Laboratories; Hazen = Hazen Research Inc.; PRA = Process Research Associates Ltd.; SGS = SGS Mineral Services; Terra = Terra Mineralogical Services.

Year	Laboratory	Report No.	Deposit/Site	Mineralogy	Comminution	Gravity	Flotation	Cyanidation	Cyanide Destruction	Other
2015	B.C. Mining Research		Goose							Ore Sorting
2014	Tomra		Goose							Ore Sorting
2014	Gekko	T1152	Goose						X	
2014	FLSmidth		Goose			X				
2014	SGS		Goose Main, Llama			X		X		
2014	ALS	KM4361	George, Goose		X	X		X		
2014	ALS	KM4030	George, Goose	X	X	X	X	X		Heap Leach, Settling, Viscosity
2013	ALS	KM3589	George, Goose	X	X	X	X	X		Settling, Viscosity
2011	Terra		Echo, Llama, Umwelt	X						
2010	SGS	12521-001	Goose		X			X		
2009	Gekko	T0439	George, Goose		X	X	X	X	X	Settling
2007	SGS	11320-004	George, Goose	X		X	X			
2006	Geoscience		Goose	X						
1998	PRA	97-080	George, Goose		X	X		X		
1992	Hazen		George		X	X		X		

Source: JDS 2015 FS report

Metallurgical samples were selected from different depths and drill holes to ensure that there was coverage of all the deposits and that recoveries were representative of the mineralization types in those deposits. Additional testing was carried out on samples that represent plant feed being treated early in the mine plan. Consequently, a relatively large number of upper zone and lower zone samples from Umwelt were selected representing both the open pit and underground deposits as indicated.

There has been extensive mineralogical work undertaken on the Back River Property deposits. Since 2010, Giovanni Di Prisco of Terra Mineralogical Services had been contracted to review and provide detailed characterizations and predictive metallurgical assessments of the gold mineralization present in the Goose and George sites. Giovanni produced 13 reports summarizing the Llama, Echo, Umwelt and Goose Main deposits at the Goose Site as well as the Locale 1, Locale 2 and LCP deposits at the George Site.

The reports for studies undertaken by Giovanni Di Prisco are listed here:

- 10AUG-001- Goose Lake- Llama Gold mineralization;
- 11AUG-001- Goose Lake- Echo Gold mineralization;
- 11AUG-001- Goose Lake- Umwelt Gold mineralization;
- 11DEC-004 Umwelt Open Pit Gold Mineralization;
- 11DEC-004 Umwelt Underground Gold Mineralization;
- 12APR - Summary of the Mineralogy Studies of the Gold Deposits from the Goose Lake;
- 12JAN-002 - Goose Lake Gold Mineralization;
- 13JAN-004 - George Lake - L1 Gold Mineralization;
- 13MAR-003 - George Lake - L2 Gold Mineralization;
- 13NOV-001 - Goose Lake Umwelt G2 Gold Mineralization-Edited July 2014;
- 13OCT-002 - Goose Lake Llama Deposit Gold Mineralization-Edited July 29, 2014;
- 13JUN-006 - George Lake - LCPs Gold Mineralization; and
- 13JUN-007 - George Lake - LCPn Gold Mineralization.

13.1.3 Comminution Test Work

13.1.3.1 Test Results

The programs that included comminution test work used for this study are listed in Table 13.3.

Table 13.3: Comminution Test Work Programs

Year	Laboratory	Project Ref.	Deposits/Sites	Comminution Tests
2014	ALS (G&T)	KM4361	Goose Main, Llama, Umwelt	BMWi at 53,106, 212 µm; Ai
2013/2014	ALS (G&T)	KM4030	Goose Main, Llama, Umwelt, George, Echo	BLIi(CRi), BMWi, Ai, JKDropW, SMC
2012/2013	ALS (G&T)	KM3589	Goose Main, Llama, Umwelt, George	BMWi
2010	SGS	12521-001	Echo, Llama, Umwelt	BMWi at 106 & 212 µm
2009	Gekko/Amdel	TO439	George, Goose	BLIi(CRi), BMWi, Ai, VSI, HPGR-PSD
2007	SGS	11320-004	George, Goose	SPI&Ci, BMWi, HPGR-SPT
1998	PRA	97-080	George, Goose	BMWi
1992	Hazen		George	BMWi

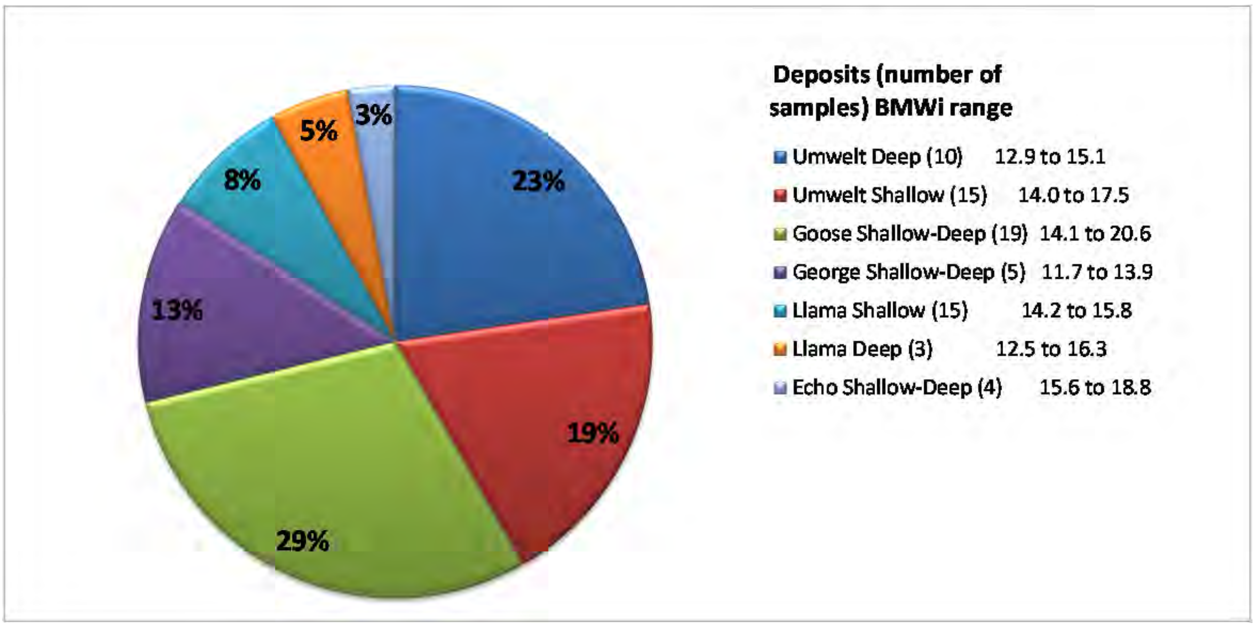
Source: JDS 2015 FS report

Following the review of all these test work reports, the available comminution indices were classified, tabulated and sorted for the calculation of averages, medians, 75th percentile and other statistics. In addition, key comminution indices (e.g. Bond Ball Mill work index (BMWi)), with a large number of samples tested, were sorted and weighted averages calculated. The comminution data analysis undertaken by Hatch for the recent 6KFS was used in whole for inputs to this current 3KFS Study.

13.1.3.1.1 Bond Ball Mill Work Indices

Collectively, from the test work programs listed previously, there are 71 BMWi test results for samples taken from all of the deposits. These 71 tests were completed using 106 µm closing screen size and achieved P80 values in the range of 78 µm to 88 µm. The resulting values of BMWi from these tests range from 11.7 kWh/t to 20.6 kWh/t and demonstrated a degree of variability. The variability within the deposits and their representativeness can be observed in Figure 13.1.

Figure 13.1: Mine Plan Distribution Indicating Number of BMWi Tests and Ranges



Source: JDS 2015 FS report

The selection of the most appropriate BMWi to be used in the circuit design can be described as "partial geometallurgy" analysis which is performed by tabulating the values, weighting based on their contribution to the total reserve, sorting them, and thus determining their weighted average and weighted 75th percentile values.

The weighted average and weighted 75th percentile were calculated as 15.1 kWh/t and 15.8 kWh/t respectively.

Due to the fine grind targeted for the Project, 50 µm P80, the selected comminution circuit has two stages of grinding. The primary grind is achieved with a ball mill in closed circuit with hydrocyclones to produce a P80 of approximately 180 µm. The final P80 is achieved using a stirred-media mill which receives the product from the ball mill circuit. In order to evaluate the power for the different grinding stages accurately, it is important to use data from a similar grind size. The ALS KM4361 test work program included the assessment of BMWi at different grinds and the results are shown in Table 13.4.

Table 13.4: ALS KM4361 BMWI Results at Different Sieves

Sample ID	BMWi kWh/t		
	53 µm	106 µm	212 µm
UM-UG-VS-07	16.7	14.1	14.5
UM-OP-VS-02	16.6	14.1	14.5
LL-OP-VS-12	17.3	14.7	14.3
UM-OP-VS-12	16.3	14.9	14.4
GM-OP-VS-15	17.4	15.8	14.8
GM-OP-VS-11	21	20.6	21.2
Average	17.6	15.7	15.6
BMWi compared to 106µm	112.7%	100%	99.5%

Source: JDS 2015 FS report

From KM4361 test work results, no significant change is observed from resulting BMWi values, using closing screens of 212 µm and 106 µm. However, when the tests are performed with a 53 µm closing screen (P80 ranging from 39 to 43 µm), there is an increase in the BMWi. The average, with 53 µm screens, was 12.7% higher than with 106 µm screens.

Based on these observations, the BMWi values used for the design of the ball mill circuit were based on the weighted 75th percentile determined with the 106 µm tests. For the secondary grinding design, the values used were increased by 12.7% to 17.8 kWh/t.

13.1.3.1.2 JK Drop-Weight and SMC Tests

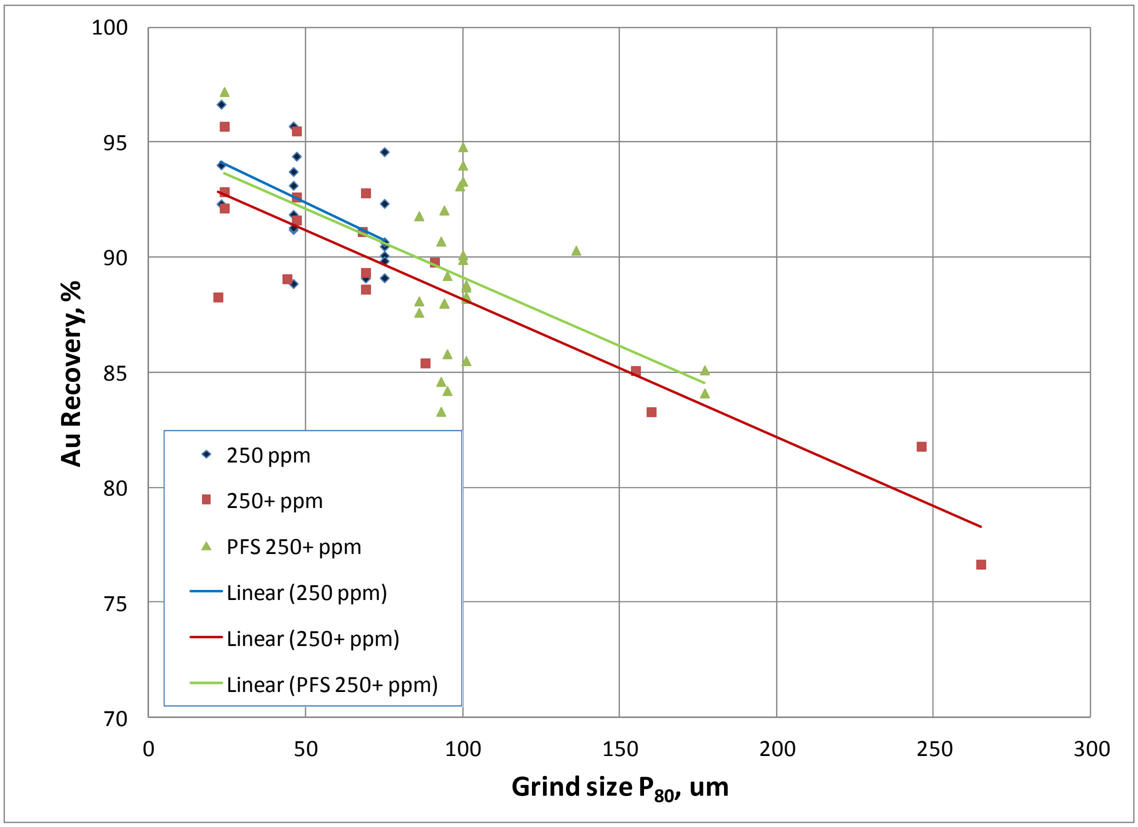
Test work conducted at ALS in 2013 and 2014 under program number KM4030 assessed the comminution characteristics of the ore with regards to coarse grind (SAG/AG milling). In total, 60 tests were completed. Four were JK full range drop-weight type and 56 were SMC type (a simplified version of the drop-weight test). These 60 tests provided the Axb impact breakage parameter along with the other parameters which were required to model and simulate SAG or AG mills using JKSimMet™ software. The resulting values of Axb from these tests range from 17.0 to 47.4 (lower Axb values indicate harder ore). All the values are within the moderate-hard to very-hard classifications ranges. Thirty-two results have Axb values of 30 and below (very-hard) and 21 results are between 30 and 38 (hard).

In a similar manner to the analysis described for the BMW indices, the Axb results were tabulated, applying weighting based on their representativity to the LOM, then sorted, and finally the weighted average and weighted 75th percentile values calculated. This resulted in a weighted average value of Axb of 31.1 and a value of 28.3 for the Axb weighted 75th percentile.

13.1.3.2 Comminution Test Work Interpretation

Most of earlier test work was carried out at a P80 of 75 µm which is typical of many gold leaching operations operating globally. Early in the Feasibility Study, a trade-off study was completed to establish the optimum grind and associated circuit configurations. This was used to direct the optimization test work. Figure 13.2 indicates the recovery relationship as a function of the grind size.

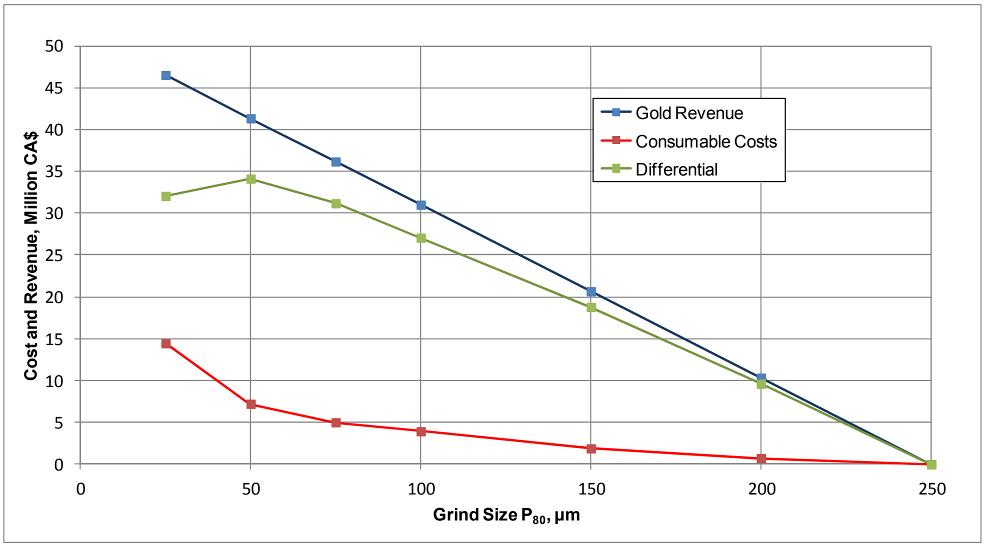
Figure 13.2: Grind Recovery Curve



Source: JDS 2015 FS report

Three different cyanide concentrations were used in the tests depicted in the Figure 13.2 above. Each of these was plotted independently and have very similar slopes, indicating a good relationship between the grind size and recovery. The average relationship was then used to produce Figure 13.3.

Figure 13.3: Optimal Grind Size Determination



Source: JDS 2015 FS report

The revenue line is derived from the recovery curve in Figure 13.2. The cost comparison was limited to consumables such as power, media and wear parts. Power efficiencies reflecting the individual circuit configurations were evaluated. The most economically favourable grind size (i.e., the maximum difference between revenue and cost) is achieved at a P80 of 50 µm, with finer grinds resulting in operating costs that exceed the additional revenue. Test work was only carried out in 25 µm increments so it is possible that the optimum is slightly higher or lower than 50 µm.

13.1.4 Gravity and Cyanidation Test work

13.1.4.1 Gold Recovery Test work Summary

Table 13.5 is a summary of the optimized test work that has been used as the basis of the plant design and financial modelling for this Initial Project Feasibility Study. The gravity and leach recoveries were averaged by deposit from test work and a solution loss of 0.5% applied to account for losses which will typically occur in a plant.

Table 13.5: Gold Recovery by Deposit

Mine	Reserves (kt)	Head Grade (g/t)	Recovery From Test Work (%)	Solution Loss Allowance (%)	Recovery Applied to Financial Model (%)
Umwelt Open Pit	2,668	6.49	92.5	0.5	92.0
Umwelt Underground	3,492	7.38	92.5	0.5	92.0
Llama Open Pit	1,749	7.15	91.6	0.5	91.1
Goose Main Open Pit	4,451	5.00	95.5	0.5	95.0
Total	**12,359**	**6.30**			**93.0**

Source: JDS 2015 FS report

Table 13.6 references and summarizes the specific tests that were used to create the summary above.

Table 13.6: Tests Used to Derive Deposit Recoveries

Sample	Test Program	Test No.	Gold Head Grade	Sulphur Head Grade	Grind P80	Leach Time	Lime (CaO) Added	Cyanide (NaCN) Added	Cyanide (NaCN) Consumed	PbNO3 Added	Gravity Gold Recovered	Leach Recovery	Tails Grade	Recovery Gravity + Leach	Recovery (H-T)/H	Deposit Recovery
			g/t	%	microns	h	kg/t	kg/t	kg/t	kg/t	%	%	g/t	%	%	%
Umwelt																
UM-OP-VS-4	KM4361	29	3.28	2.63	49	48	2.59	2.14	1.42	0.02	25.4	87.8	0.205	90.9	93.75	
UM-OP-VS-6	KM4361	30	8.66	3.19	58	48	2.35	2.06	1.46	0.02	21.7	88.3	0.515	90.8	94.05	
UM-OP-VS-8*	KM4361	31	5.14	3.09	56	48	2.03	2.54	2	0.02	20.2	89.1	0.56	91.3	89.11	
UM-OP-VS-11	KM4361	32	16.36	5.22	55	48	2.31	2.56	2.26	0.02	15.8	90.1	1.14	91.7	93.03	
UM-UG-VS-3	KM4361	33	6.51	5.25	51	48	2.19	2.62	2.26	0.02	23	91.9	0.45	93.8	93.09	
UM-UG-VS-6	KM4361	34	7.46	3.84	54	48	1.35	2.06	1.38	0.02	46.2	89.1	0.22	94.1	97.05	
UM-UG-VS-8*	KM4361	35	6.33		54	48	1.48	1.9	1.2	0.02	28.4	89.5	0.665	92.5	89.49	
Umwelt Average			8.2	3.9	53.9	48	2.04	2.27	1.71	0.02	25.81	89.4	0.536	92.15	92.8	92.5
Llama																
LL-OP-VS-03	KM4030	157	6.33	2.33	63	48	1.68	0.84	0.6		42.64	87.7	0.575	92.9	90.92	
LL-OP-VS-08	KM4030	158	40.06	1.73	53	48	1.88	1.18	0.84		54.06	86	1.85	93.6	95.38	
LL-OP-VS-10	KM4030	175	4.37	2.59	59	48	1.35	1.54	0.98		31.08	92.4	0.23	94.8	94.74	
LL-OP-VS-11	KM4030	176	11.75	3.55	55	48	1.76	1.46	1.02		45.9	81.3	1.165	89.9	90.08	
LL-OP-VS-11	SGS Duplicate		11.3		52	48	0.96	0.92	0.51		47.5	87.3	0.79	93.3	93.01	Note 3
LL-OP-VS-12	KM4030	177	7.9	4.86	61	48	1.9	1.58	1.06		40.96	76	1.05	85.9	86.7	
LL-UG-VS-3	KM4030	162	1.18	0.84	51	48	1.33	0.9	0.62		76.66	91.9	0.08	98.1	93.19	Note 1
Llama Average			15.9	2.7	56.3	48	1.55	1.2	0.8		48.4	86.1	0.82	92.63	92	91.6
Goose Main, Echo, George																
EC-VS-2	KM4030	148	2.87	2.35	51	48	2.45	1.36	1.16		50.9	89.4	0.11	94.8	96.17	
EC-VS-4	KM4030	149	6.46	3.15	62	48	2.3	1.26	1.04		28.5	90.8	0.435	93.4	93.27	
GM-OP-VS-01	KM4030	150	1.94	1.06	54	48	1.19	1.14	0.9		57.2	97.1	0.04	98.8	97.94	
GM-OP-VS-06	KM4030	151	5.98	2.08	53	48	1.88	1.06	0.68		64.3	96.5	0.08	98.8	98.66	
GM-OP-VS-08	KM4030	170	7.18	3.08	37	48	3.29	1.98	0.48		41.3	94.8	0.22	96.9	96.94	Note 2
GM-OP-VS-12	KM4030	153	5.53	0.76	54	48	1.15	1.06	0.78		44	94.5	0.185	96.9	96.65	
GM-OP-VS-14	KM4030	154	10.6	1.56	50	48	1.25	1.06	0.74		27.1	92.1	0.645	94.2	93.92	
GM-OP-VS-14	SGS Duplicate		9.27		62	48	1.08	0.92	0.5		27.8	88.1	0.88	91.4	90.51	Note 3
GM-OP-VS-15	KM4030	155	6.88	2.45	52	48	1.61	1.14	0.94		34	94	0.235	96.1	96.58	
GM-OP-VS-18	KM4030	156	2.71	3.1	56	48	1.93	1.12	0.92		43.6	89.2	0.185	93.9	93.17	
Goose Main, Echo, George.			6.6	2.2	53.1	48	1.81	1.21	0.81		41.9	92.7	0.302	95.52	95.38	95.5
Average - All Deposits			10.23	2.9	54.4		1.8	1.56	1.11		38.7					93.2
Design Criteria			5.22		50	48	1.8	1.38								

Source: JDS 2015 FS report

The variability in total gold recovery across the different samples in the different deposits was not substantial. The Umwelt ore was the most challenging with regard to achieving good recoveries. It required oxygen and lead nitrate to achieve recoveries of over 90%. Two samples were selected (LL-OP-VS-11 and GM-OP-VS-14) and tested at a different laboratory to check for any inherent bias in testing or sampling. These samples resulted in recoveries that were similar to the original samples tested as part of program KM4030. The reagent consumptions for the duplicates were lower, indicating that there may be an opportunity for reduced operating costs if this trend extends across all the deposits. The average grind for all the test work is a P80 of 54 µm and the process has been designed to grind to a P80 of 50 µm. This is slightly conservative as the design grind may provide a slightly improved recovery, or if the circuit is operated at 54 µm, there will be a small power saving.

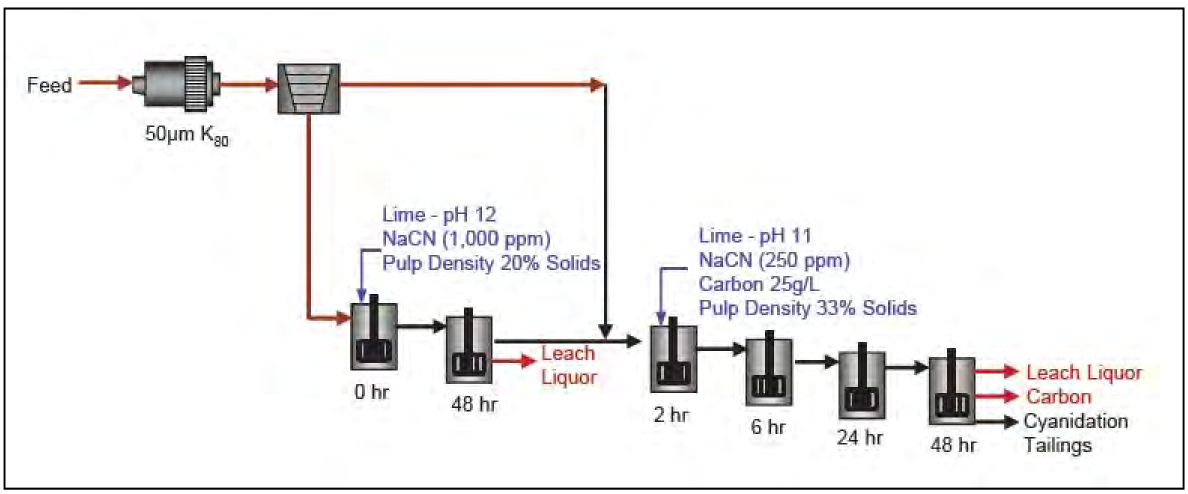
13.1.4.2 Test Conditions

Metallurgical test work for the Feasibility Study commenced in December 2013 at ALS Kamloops under project number KM4030 and this program was completed by May 2014. The test conditions for this test program (Goose Main, George, Echo and Llama) were as follows:

- Primary grind for gravity concentration: P80 of 64-75 µm;
- Regrind prior to carbon-in-leach (CIL): P80 of 50 µm;
- Intensive Leach: 48 hours, pH 12, 1000 ppm NaCN, 20% solids, sparged with oxygen; and
- CIL: 48 hours, pH 11, 250 ppm NaCN, 33% solids, sparged with oxygen.

A schematic representation of the test flowsheet is given as Figure 13.4.

Figure 13.4: Test Program KM4030 Flowsheet

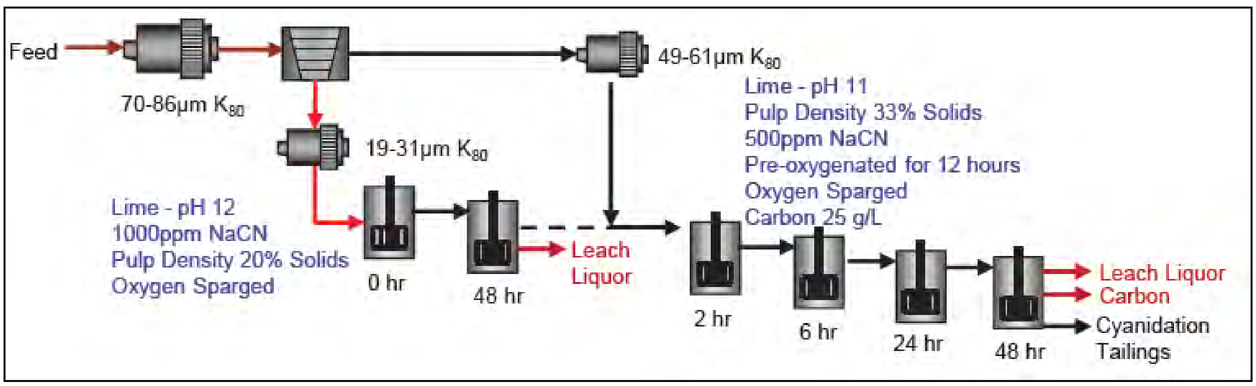


Source: JDS 2015 FS report

An Umwelt optimization program (KM4361) was initiated to improve recoveries for the ore from this deposit. Conditions for this testing are given as Figure 13.5.

- Primary grind for gravity concentration: P80 of 64-75 µm;
- Concentrate regrind for IL: P80 of 40 µm;
- Regrind prior to CIL: P80 of 50 µm;
- Intensive leach: 48 hours leach, pH 12, 1000 ppm NaCN, 20% solids, sparged with oxygen;
- CIL: 12 hours pre-oxygenation, 48 hours leach, pH 11, 20g/t PbNO3, 500 ppm NaCN, 33% solids, sparged with oxygen.

Figure 13.5: Test Program KM4361 Flowsheet



Source: JDS 2015 FS report

13.1.4.3 Test Work Analysis

13.1.4.3.1 Gravity Recovery

It can be seen from Table 13.6 that gravity recovery varied widely from approximately 16% to 76% with an average of 39%. The test work was designed to replicate the plant design in that the gravity product was intensively leached and the gold in solution measured. The leach tails were then added back to the slurry which was then leached in the presence of carbon, resulting in a final solid tail that was then representative of what would be seen in the plant.

The flowsheet was designed with two stages of grinding with gravity concentration on the primary grinding stage. This allows coarse gravity recoverable gold to be extracted early without overgrinding, and finer liberated particles to be extracted in the second stage of grinding. The likely regrind mill that was selected operates in open circuit with hydrocyclone classification. This makes it impractical to recover gravity gold on the second grinding stage cyclone underflow.

In the test work, the gravity product was reground prior to intensive leach. This will not occur exactly the same way in the commercial operation since coarser gold will report to the intensive leach reactor. After the concentrate has been leached, it will be added back to the circuit where it will again be ground, and will likely have a final grind similar to the one tested. The selected intensive leach process operates well on coarser gold particles as it relies on separation of the solution and solids. It was therefore considered to be a more robust design to have the gravity concentrate report directly to intensive leach, and reintroduce the intensive leach tailings into the grinding circuit. This allows for an additional opportunity to recover liberated gravity gold.

13.1.4.3.2 Leach Retention Time

The grind size had been optimized at a P80 of 50 µm. The leach testing was carried out over a range of times up to 48 hours. To produce an average leach curve, the samples with a cyanide concentration of 250 ppm were selected. The results were averaged and are shown in Table 13.7 and Figure 13.6.

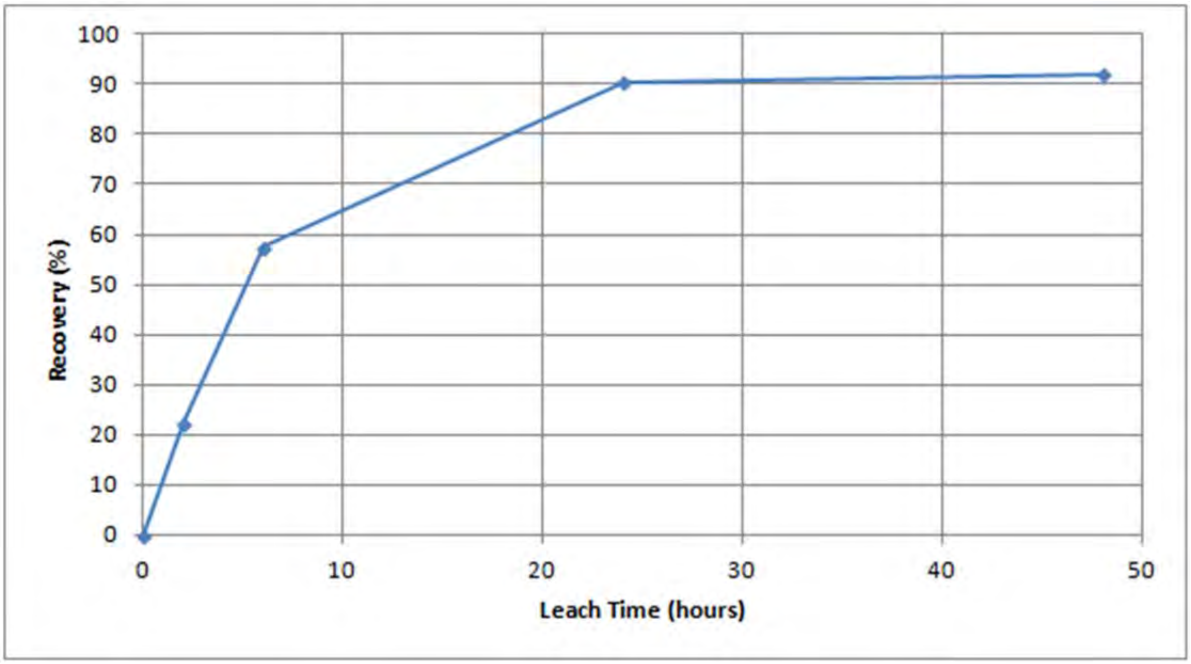
Table 13.7: Recoveries for Different Leach Times

Test Number	Grind Size	Cyanide Concentration	Leach Recovery (%)				
			0h	2h	6h	24h	48h
105GCN	69	250	0	36.48	72.73	88.95	89.09
109GCN	47	250	0	21.93	59.22	94.22	94.39
111GCN	75	250	0	29.71	77.97	88.71	90.48
112GCN	75	250	0	32.51	77.11	91.74	92.34
113GCN	75	250	0	27.56	60.13	91.91	94.59
114GCN	46	250	0	10.34	59.46	92.26	93.72
115GCN	46	250	0	16.96	66.48	91.45	91.86
116GCN	46	250	0	21.74	61.6	94.87	95.7
117GCN	23	250	0	2.82	9.34	73.62	92.32
118GCN	23	250	0	3.41	42.19	94.26	94.01
119GCN	23	250	0	8.04	45.08	96.76	96.65
135GCN	75	250	0	31.39	62.81	88.93	90.67
136GCN	75	250	0	28.41	57.96	92.62	90.08
137GCN	75	250	0	30.54	52.07	91.53	89.84
138GCN	46	250	0	25.33	61.66	93.15	93.12
139GCN	46	250	0	22.78	52.17	87.54	91.2
140GCN	46	250	0	14.18	41.83	85.85	88.86
141GCN	75	250	0	31.55	71.35	88.47	89.11
142GCN	46	250	0	28.04	65.71	92.77	91.28
Average			0	22.3	57.73	90.5	92.07
Incremental Recovery (%)				22.3	35.43	32.77	1.56

Source: JDS 2015 FS report

These recoveries are graphed in Figure 13.6.

Figure 13.6: Recoveries at Different Leach Times



Source: JDS 2015 FS report

From the table and graph above, it can be seen that most of the leaching takes place within 24 hours. The graph indicates that approximately 60% of the leaching is undertaken in the initial eight hours, with considerable leaching occurring up to 24 hours. From 24 to 48 hours, the average additional leach recovery was calculated to be 1.6%.

From 24 to 36 hours, it is estimated that there would be 1% additional recovery, with approximately 0.6% additional recovery between 36 and 48 hours. The incremental costs for tanks and agitators to achieve 36 and 48 hours residence in the previous feasibility was estimated to be C$4.4 million and C$8.8 million, with paybacks of 1.2 and 1.6 years respectively. Based on this calculation, a 48 hour leach circuit was selected and also used as the design input for this Initial Project Feasibility Study.

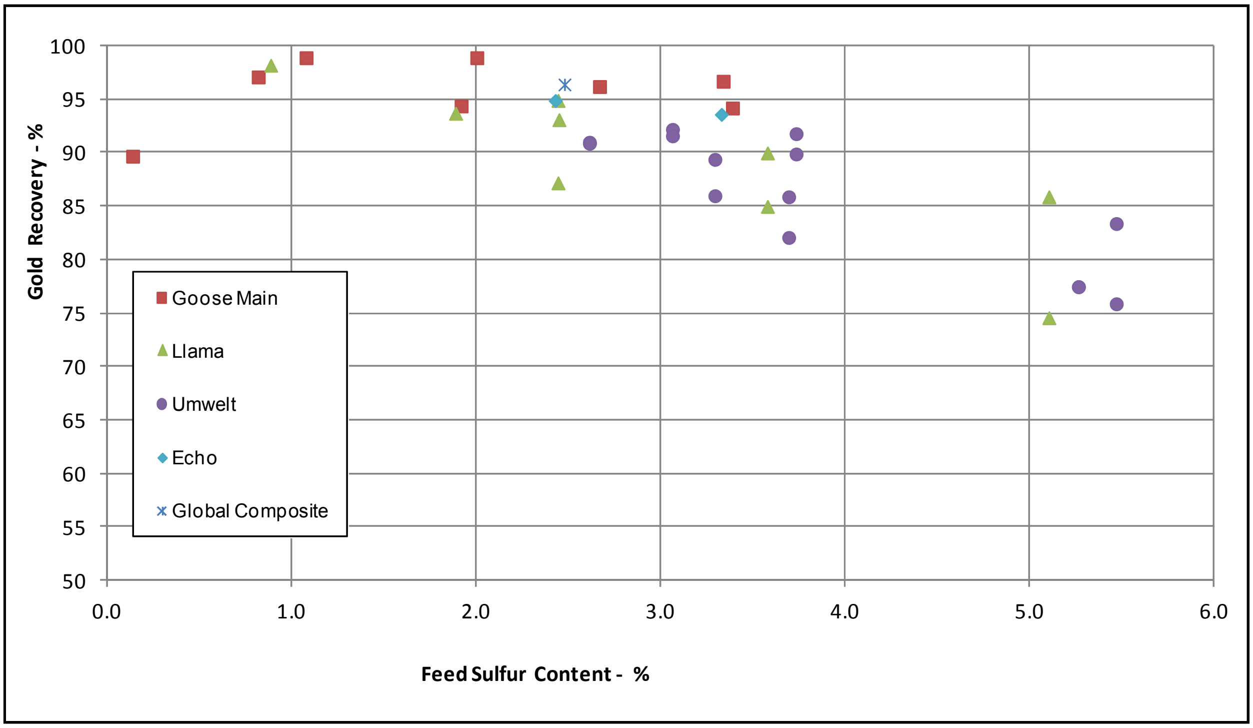
It was noted that all of the leach testing was carried out at a density of 33% solids. While this is fairly standard for bottle rolls, it is not a practical density for operating a full-scale circuit, as it will be difficult to keep the carbon in suspension. A more practical operating density range is approximately 45 - 50% (w/w), which is very similar to the wet density of carbon, and will facilitate good mixing and adsorption. It was felt that even though the actual operation will not be at the tested density, the recoveries will be the same as tested as long as the aeration, retention time and cyanide concentrations are maintained.

13.1.4.3.3 *Recovery*

At the conclusion of test program KM4030, it was observed that numerous independent samples taken from Umwelt, representing a portion of the early production, had recoveries below 90%. There was also a smaller number of tests from the Llama deposit where recoveries were below 90%. The recovery responses from Goose Main, Echo and George deposits were higher.

The data in Figure 13.7 shows a correlation between sulphur levels and recovery, with recovery starting to decrease at sulphur grades of 3-6%.

Figure 13.7: Effect of Sulphur Grade on Recovery



Source: JDS 2015 FS report

A relationship between sulphur grade, gold grade and recovery for the tests in Table 13.6 could not be established. It can be seen that the sulphur grades for Umwelt in both Table 13.6 and Figure 13.8 are the highest, contributing to the lower recoveries.

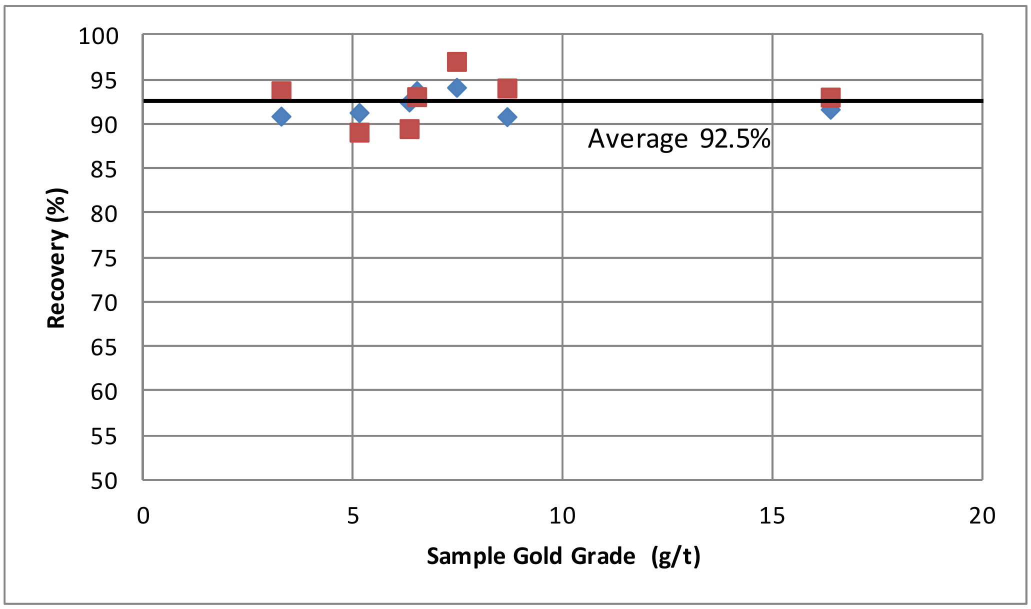
Additional testing of Llama samples showed an improvement with oxygen sparging. However there was limited recovery improvement of the Umwelt samples. An Umwelt optimization program was initiated in July 2014 at ALS Kamloops under project number KM4361. It should be noted that flotation had been tested in earlier programs and this was not reconsidered.

The testing was only a refinement of the whole-ore leach flowsheet to provide improved conditions where these were shown to have an economic benefit.

This program and data analysis was completed in mid-August 2014. With the addition of pre-oxygenation, increased cyanide dosage and lead nitrate, the recoveries improved to above 90%.

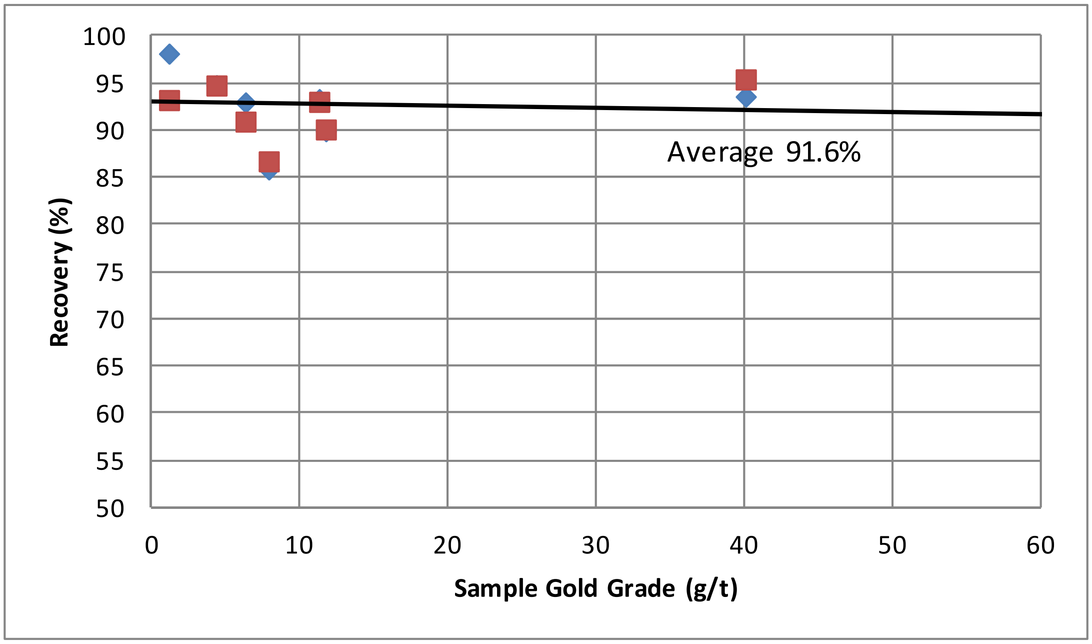
The recoveries for Umwelt, Llama and Goose Main are shown in Figure 13.8, Figure 13.9 and Figure 13.10.

Figure 13.8: Umwelt Recoveries as a Function of Head Grade



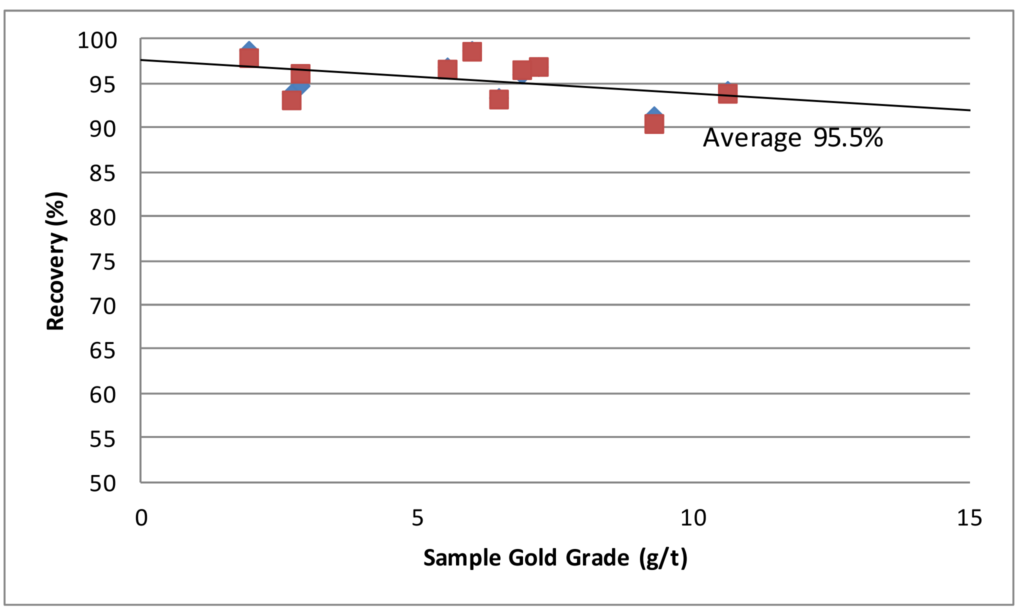
Source: JDS 2015 FS report

Figure 13.9: Llama Recoveries as a Function of Head Grade



Source: JDS 2015 FS report

Figure 13.10: Goose Main, Echo and George Recoveries as a Function of Head Grade



Source: JDS 2015 FS report

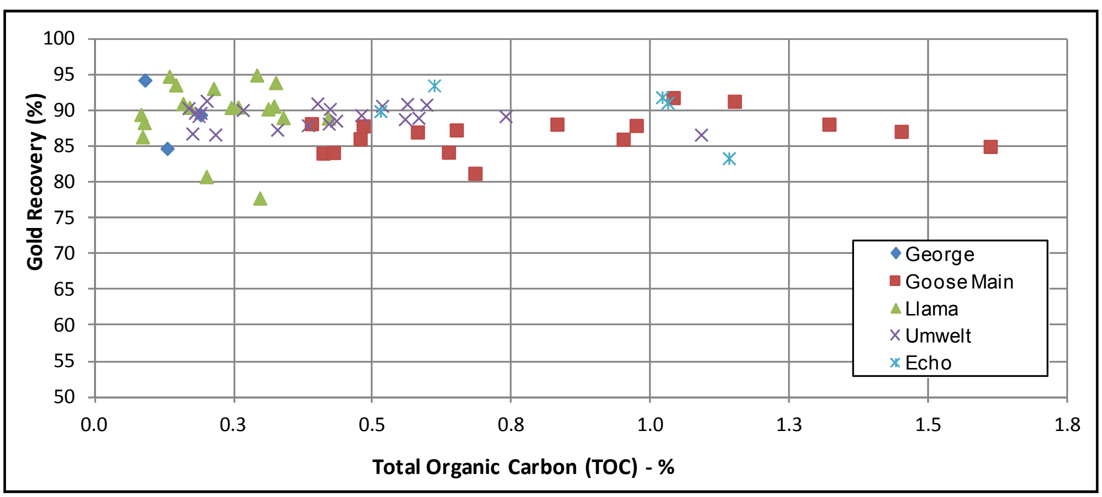
For earlier revisions of the mine plan, equations were developed that related recovery to the gold head grade. At the completion of the variability testing presented in the figures above, there was not a strong enough relationship between head grade and recovery, even over a wide range of grades. It was therefore considered to be more accurate to calculate an average recovery for each deposit which was then used in the financial model to calculate revenues.

During the additional testing of Llama and Umwelt, ALS reported that the bottles in some of the roll tests "collapsed during leaching due to the increased consumption of oxygen". An analysis of data from the decreased recovery tests indicated a strong correlation of decreasing recovery with increased levels of sulphur. Discussions with geologists revealed that both Llama and Umwelt contain pyrrhotite, which is known to be an oxygen consumer and that in Umwelt in particular, pyrrhotite was observed to increase with depth. This hypothesis was tested by repeating these underperforming tests with additional oxygen sparging during the first six to eight hours of leaching to combat the aggressive oxygen consumption. Llama results improved with the additional oxygen sparging and these results were used to create the Llama recovery relationship shown above.

13.1.4.3.4 Total Organic Carbon (TOC)

The carbonaceous content of the ores was measured in test program KM4030. Recoveries at various TOC levels are plotted in Figure 13.11 below. It can be seen that there is not a major decrease in recovery with increase in TOC content in the range of 0 - 1.8%. The test work indicated that there is potential for a leach reduction of up to 2% in some ores. The flowsheet has been designed as a carbon-in-pulp (CIP) rather than a CIL circuit. Both gravity and carbon in columns (CIC) prior to the leach tanks will extract a substantial amount of gold, so the fraction of gold reporting to leach, and the potential for preg-robbing, is reduced.

Figure 13.11: Effect of Total Organic Carbon on Gold Recovery



Source: JDS 2015 FS report

The circuit has been designed to grind in cyanide, which offers associated advantages such as reduced cyanide reporting to the destruction circuit and more flexibility in process water management. As a result, there will be some cyanide at the front of the circuit with some potential for preg-robbing. At the beginning of the leach circuit, the oxidants (oxygen and lead nitrate) will be added. Once the ore has been oxidized, cyanide will be dosed with additional aeration.

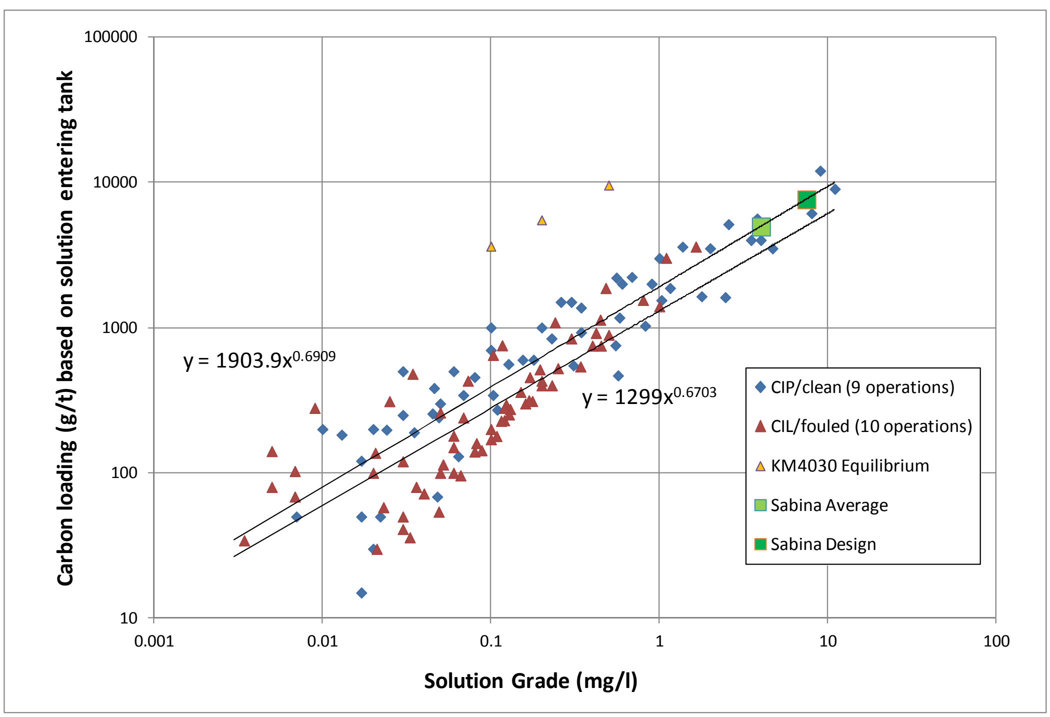
After leaching, the gold will be adsorbed onto carbon in the CIP circuit. The flowsheet has been designed to allow for carbon to be pumped from the CIP circuit further up into the leach circuit so that it can flow back with the slurry to the first CIP tank, where the inter-stage screens will prevent it flowing further. This will allow flexibility to have carbon present during leaching should there be periods when preg-robbing carbonaceous material is present.

13.1.5 Carbon Adsorption

Carbon loading test work was carried out as part of program KM4030.

While the test work showed that adsorption in the laboratory worked well, it is difficult to predict plant conditions accurately. The carbon residence time was designed to be 12 days. There is good empirical data available from many operating plants that provide a good design basis for gold loading. This data was collected from circuits that experience fouling of carbon (e.g., where flotation reagents load prior to adsorption) and clean circuits and is shown as Figure 13.12.

Figure 13.12: Carbon Loading Prediction



Source: JDS 2015 FS report

There were no separate metal loadings calculated for CIC and CIP. The solution tenors are calculated on the assumption that all the gold reports to the CIP circuit as it is possible to operate the circuit in this way. The CIC carbon will typically load higher than CIP carbon for the same solution grades, so the average loading and carbon processing for CIC and CIP should be similar to the average calculated for CIP overall. The strip circuit is designed for one strip per day but it will be possible to strip twice a day should the metal loadings prove to be lower.

13.1.6 Thickening Test Work

Thickening test work was carried out as part of test program KM4030.

Three flocculant types were tested to determine the optimal flocculant to be used for the leach tailings. These were Superfloc A130, Superfloc C496, and Magnafloc 333. It was determined that Magnafloc 333 (non-ionic) is most optimal from the flocculant screening tests.

A suite of static settling tests were conducted on samples from Echo, Goose Main, Llama and Umwelt.

The test work has shown that the ore can be thickened consistently to above 60% solids.

A 60% design factor is added to the average calculated thickener size. This is 10% more area than the worst predicted settling rate. The static test is simplistic and on purchasing the thickeners, the area should be confirmed with the thickener vendors using a dynamic test.

13.1.7 Cyanide Destruction Test Work

Cyanide destruction of the CIP tails was carried out by Gekko. The aim of the test work was to determine the amenability of the sample to the SO_2/O_2 cyanide destruction process to achieve a primary target of less than 10 mg/L weak acid dissociable cyanide (CN_{WAD}) with less than 20 mg/L cyanide total (CN_{TOTAL}). A secondary target of less than 15 mg/L CN_{WAD} with less than 25 mg/L CN_{TOTAL} and a final target of less than 1 mg/L CN_{TOTAL} were stipulated. Table 13.8 and Table 13.9 indicate the successful and unsuccessful tests respectively.

Table 13.8: Successful Conditions for Cyanide Destruction

Test	Conditions						Product Solution Analysis					
	SO_2: CN_{WAD}	Cu addition	Retention Time	pH	Lime/ SMBS	% solids	CNP	Cu	Ni	Fe	Zn	CN_{TOT}
	g/g	ppm	min		g/g		ppm	ppm	ppm	ppm	ppm	ppm
Feed							90.8	7.5	<0.1	1.9	0.3	96
2	6.2	20	81	8.5	0.7	38.6	0.3	1.3	-	0.7	-	2.3
4	6.3	10	95	8.5	0.34	38.6	0.5	0.6	-	0.3	-	1.3
5	6.5	16	88	8.6	0.4	38.6	0.9	1.8	<0.1	0.6	<0.1	2.5
7	5.2	10	87	8.5	0.91	38.6	1.1	1.9	<0.1	1.2	<0.1	4.2
8	5.4	12	88	8.5	0.45	45.2	0.5	1.9	<0.1	1.5	<0.1	4.7
9	5.1	10	87	8.5	0.36	58	1.6	0.9	<0.1	0.3	<0.1	2.3
Confirm.	5	15-Oct	85	8.5	0.36	38.6	0.1	0.88	<0.1	1.4	<0.1	5.2

Table 13.9: Unsuccessful Conditions for Cyanide Destruction

Test	Conditions						Product Solution Analysis					
	SO_2: CN_{WAD}	Cu addition	Retention Time	pH	Lime/ SMBS	% solids	CNP	Cu	Ni	Fe	Zn	CN_{TOT}
	g/g	ppm	min		g/g		ppm	ppm	ppm	ppm	ppm	ppm
1	6.8	5	83	8.6	0.33	38.6	20.3	11.7	<0.1	1.4	<0.1	24.2
3	6	5	109	8.5	0.32	38.6	4.9	7	-	0.6	-	6.6
6	6.2	16	88	8	0.76	38.6	21.9	18.3	<0.1	1.5	<0.1	25.9
6R	6.2	16	86	8	0.1	38.6	2	0.4	<0.1	0	<0.1	2.1
10	4.1	10	87	8.5	0.95	38.6	22.5	19.4	<0.1	1.5	<0.1	26.7
11	5	10 (Zn)	87	8.5	1.05	38.6	32.5	12.4	<0.1	1.9	2.9	37.8

Source: JDS 2015 FS report

A feed slurry containing 90 mg/L CN_{WAD} and 96 mg/L CN_{TOTAL} was effectively treated and resulted in a stable final CN_{WAD} of less than 0.2 mg/L and CN_{TOTAL} of less than 5.2 mg/L, satisfying the primary and secondary targets, under reaction conditions of 38.6% solids, 5.0 g SO_2/g CN_{WAD}, pH 8.5, 15 mg/L copper addition and 85 minutes retention time.

The test work showed that this ore has a strong copper dependency when destructing cyanide. The cyanide destruction process was more sensitive to copper availability than any other parameter. Additional copper catalyst was necessary despite the presence of copper in the tailings. Zinc was tested as a low cost substitute for the copper catalyst but was not successful.

The SO_2/O_2 cyanide destruction process did not achieve any of the target CN_{WAD} levels if the g SO_2:g CN_{WAD} ratio was reduced to below a 4:1 ratio, or the copper addition was reduced to less than 15 mg/L. Increasing the pulp density to 58% was not detrimental to the cyanide destruction process.

An additional repeat continuous bulk test was carried out to generate samples and solution for ongoing tailings testwork. In this test, the leach tail was detoxified using the same ratios of 5g SO2/g CN_{WAD} and15mg/L Cu2+. The cyanide detoxification was successful in achieving the final CN_{TOTAL} target and the resulting product contained less than 0.2ppm CN_{WAD} and less than 0.5ppm CN_{TOTAL}.

13.1.8 Ore Sorting

Two high level studies were conducted to assess the sortability of the Back River deposits. TOMRA Systems ASA (TOMRA) investigated rock specimens for their amenability for detection by four different types of sorting technologies. A review of that testwork along with mineralogical/geological reports was then undertaken by Bern Klein from B.C. Mining Research Ltd. at the University of British Columbia and selected drill core samples were examined. A followup test program was executed based on the latter work.

Mineralogical information for the Back River deposits indicated that while there are variations, there are significant and important similarities related to gold and silver mineralization that are relevant to sorting. It is therefore expected that an effective sorting system for one deposit would be applicable to others. Examination of drill core showed distinct contacts between ore and waste, as well as significant extents of ore versus gangue lithologies implying that sorting should be possible.

The TOMRA test program assessed heterogeneity based on gold and sulphur grades of selected rocks and then evaluated sorting using a range of sensor technologies including: Visible Spectrum (colour), Dual Energy X-Ray Transmission (DEXRT), Near Infrared Spectroscopy (NIS), Electromagnetic (EM), and Optical Sorting with UV illumination. DEXRT was found to be the most promising as it demonstrated good sorting results for sulphide minerals. However, the correlation between gold and sulphur was inconsistent so DEXRT was not able to achieve the same results for gold. Upon review, it was determined that the initial scoping high level study conducted by TOMRA was not comprehensive enough with respect to characterizing heterogeneity of ore and waste, as well as assessing sensor systems. A followup program undertaken by BC Mining Research Ltd. was developed to assess the following sorting options:

1. Analysis of sensor response signatures, (rather than relying on a single threshold levels);
2. Combining sensor responses to improve discrimination;
3. Analysis of proxies for target metals;
4. Assessing heterogeneity of gangue phases for rock rejection;
5. Applying regression analysis to sensor signals; and
6. Assessing rock size heterogeneity.

In this program, the pulverized assay pulp samples (120 samples) from each rock that was used in a study to assess sortability in the initial study, were subjected to X-ray Fluorescence (XRF) and Electromagnetic (EM) sensor analysis. The assayed gold grades for each sample were fitted to the XRF and EM responses using multi-variable linear regression.

Based on threshold gold grades, the regression models were used to sort the rock samples to assess their suitability as an algorithm for ore sorting. A stepwise regression approach was followed which allowed determination of the significant variables and elimination of variables that were not significant.

The regression also included second order interaction effects. The regression model descriptions along with the reported coefficients of determination are listed as follows:

1. Model using XRF and EM responses (combining sensor responses) R2 = 0.63;
2. Model using XRF and EM responses with interaction effects R2 = 0.92;
3. Model using XRF responses only R2 = 0.44;
4. Model using XRF responses only with interaction effects R2 = 0.70;
5. Model using XRF and EM for samples with Au grades > 0.126ppm R2 = 0.98;
6. Model using XRF only for samples with Au grades > 0.126ppm R2 = 0.79;
7. Model using XRF variables from section 4.6 with interaction effects to generate the model R2 = 0.72; and
8. Model that included all responses from XRF, EM and Tomra responses with interaction effects between individual responses R2 = 0.90.

For each model, threshold gold grades were set with the objective of achieving greater than 95% gold recovery. Results for each of the eight models along with predicted sorting product grades, recoveries and weight percentage rock rejection are shown in Table 13.10. The calculated weighted grade of a composite containing all of the 120 samples is 0.336 g/t Au.

Table 13.10: Results from Ore Sorting Testwork by Regression Model

Model	Threshold Au g/t	Sorting Concentrate			
		Weight %	Au g/t	Au Rec %	Rejection Weight %
1	0.03	64.9	0.49	95.2	35.1
2	0.01	73.1	0.40	91.8	26.9
3	0.075	75.3	0.43	95.6	24.7
4	0.14	79.5	0.41	97.4	20.5
5	0.20	77.2	0.42	95.8	22.8
6	0.60	78.8	0.41	96.9	21.2
7	0.10	85.5	0.37	94.4	14.5
8	0.05	80.2	0.49	95.3	19.8

Source: Canenco 2015

At a target recovery of greater than 95%, up to 35% of the rock can be rejected. It should be noted that the low grade of the composite sample has a negative consequence on the calculated recoveries. If the head grade is increased, the recoveries will increase. The regression models would indicate which XRF elements at EM frequency responses are most significant. Ultimately these parameters must relate to the mineralogy of the valuable and non-valuable constituents of the ore. Also, the XRF and EM analyses were performed on pulverized assay pulps, which may respond differently to individual rocks.

Therefore, it is necessary to conduct a similar study using rock samples. Overall, the regression analysis approach to developing a discrimination algorithm shows encouraging results with respect to the sortability of the Back River mineralization. To verify and develop a sorting system for this deposit, additional test work is recommended.

13.1.9 Future Testing

Substantial testing has been done on the Back River Property ore. Additional testing that may be beneficial to the Project includes:

- Coarse beneficiation. Ore sorting testwork performed based on differentiation with colour, dual energy x-ray transmission, near-infrared spectroscopy, x-ray fluorescence and electromagnetic sensor analysis showed some potential and may warrant additional testing. However, upgrading of the gold content may also be possible through simple screening of the coarser fraction produced from blasting and/or crushing. Heterogeneity studies on available samples should be conducted to assess how mineralization relates to comminution particle size.
- Silver. Considerable silver testing has been done, and silver will have some contribution to the Project revenue. It was not deemed to be cost-effective to have all drill core assayed for silver at this stage.
- Reagent and grind size optimization. ALS test program KM4361 was focused on improving gold recovery, primarily for Umwelt. The conditions have resulted in a circuit that has a fine grind (50 µm), pre-oxidation and a long leach time (48 hours). The optimum grind size was calculated to be 50 µm, but this was developed in increments of 25 µm. It is possible that a slightly coarser or finer size could improve the economics. There is likely some potential to optimize the cyanide addition and grind size while maintaining the other parameters, although it is suggested that there is little value added to the Project until the plant is operating and these parameters optimized.
- Variability test work. This was focused on the ores that make up the largest part of the deposit reserve base and that are treated early in the mine plan in order to provide the best confidence in the financial model. There should be some additional variability testing, including tailings mineralogy, undertaken to confirm recoveries and reagent consumptions on areas within the deposit where the mine plan has been adjusted.
- Salinity. Water management at Back River is complicated by the sequencing of the multiple pits and underground mine. The groundwater is hypersaline (up to 60,000 mg/L TDS). Even

though it is planned to store the saline water at surface and then potentially return it to the underground workings, it may be worthwhile testing different concentrations (e.g., 10, 20, 30% of equivalent seawater) to assess recovery and reagent requirements, so that this impact is understood if ever saline water needs to be brought into the process circuit.

- Settling. Thickeners have been sized on static settling tests. It would be beneficial to have dynamic testing done with selected vendors to confirm the sizing prior to placing orders.
- Secondary Grinding and Media Consumption. The flowsheet includes a vertical grinding mill for the second stage of grinding. It would be beneficial to get the vendor to carry out tests to confirm mill suitability for the secondary grinding duty, energy and media consumptions for this unit operation.
- Bulk Material Handling. Industry standards were used for calculating volumes in stockpiles and bins. These could be refined further using samples to calculate angles of repose and other physical characteristics.

13.2 Design Criteria and Flowsheet Development

A table of the summary design criteria is provided in section 17.

14 Mineral Resource Estimate

14.1 Introduction

The Mineral Resources for the Goose Main, Echo, Umwelt, and Llama deposits have been estimated by Ms. D. Nussipakynova, P.Geo., of AMC who takes responsibility for these estimates.

The Mineral Resources for the LCP North, LCP South, Locale 1 and Locale 2 deposits within the George Site have been estimated by Dr. A. Fowler, MAusIMM, CP(Geo), of AMC who takes responsibility for these estimates.

The Mineral Resources for the GH and Slave deposits within the George Site have been estimated by Mr. C. Zamora, MAusIMM, of AMC under the supervision of Dr. Fowler. Dr. Fowler takes responsibility for these estimates.

AMC is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other similar factors that could materially affect the stated Mineral Resource estimates.

These estimates are dated October 21, 2014 and supersede the previous estimates outlined in the “Mineral Resource Update for the Back River Gold Property, Nunavut, Canada” dated March 4, 2014 (Tetra Tech Technical Report 2014). The background to these previous estimates is shown in Table 14.1.

Table 14.1: Previous Mineral Resource Estimates for the Back River Property

Deposit	Mineral Resource Effective Date	QP	Company	Cut-off Date of Data
Goose Deposits	28-Feb-14	D. Nussipakynova	AMC	31-Dec-13
George Deposits	28-Feb-14	Dr. A. Fowler	AMC	31-Dec-13

Source: AMC Mining Consultants (Canada) Ltd.2015

The data used in the October 21, 2014 estimates include results of all drilling carried out on the Property to December 31, 2013 for all deposits except Echo and the Locale deposits where new drilling (Echo) and new sampling of existing core (Locale 1 and Locale 2) took place. The data used in the Echo estimate includes assays to July 4, 2014 and the data used in the Locale deposits includes assays to July 21, 2014. There is no new drilling or data for the GH and Slave deposits. Previous estimates are re-stated here. The remaining deposits underwent a new geological interpretation.

The results of the estimates are summarized in Table 14.2, and expanded in Table 14.3 in the same format as the previous technical report (Tetra Tech, 2014). It is important to note that, in all Mineral Resource tables in this section, the Mineral Resources are inclusive of any Mineral Resources converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The effective date of the Mineral Resource estimate is October 21, 2014.

Table 14.2: Summary of Mineral Resources as of October 21, 2014

Resource Classification	Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)
Measured	10,273	5.27	1,740
Indicated	17,969	6.22	3,593
Measured and Indicated	28,242	5.87	5,333
Inferred	7,750	7.43	1,851

Notes: CIM definitions were used for the Mineral Resources.
Refer to the footnotes in Table 14.3 for prices and cut-off grades applied to each deposit.
Drilling results up to December 31, 2013, are included except for Echo (new drilling to July 4, 2014) and Loc1 and Loc2 (new sampling to July 21, 2014).
The numbers may not add due to rounding.
Source: AMC Mining Consultants (Canada) Ltd. 2015

Table 14.3: Mineral Resources as of October 21, 2014 by Area

Resource Classification	Deposit	Open Pit / Underground	Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)
Measured	Goose Main	Open Pit	4,478	4.32	621
		Underground	110	6.24	22
	Llama	Open Pit	1,874	5.86	353
		Underground	110	5.72	20
	Umwelt	Open Pit	3,699	6.07	722
		Underground	1	9.21	0.3
Measured Total			**10,273**	**5.27**	**1,740**
Indicated	Goose Main	Open Pit	2,877	4.19	388
		Underground	853	7.32	201
	Echo	Open Pit	321	6.07	63
		Underground	596	6.17	118
	Llama	Open Pit	821	6.01	159
		Underground	752	8.72	211
	Umwelt	Open Pit	1,963	5.38	340
		Underground	3,387	8.92	972
	George	Open Pit	4,321	5.04	700
		Underground	2,079	6.62	443
Indicated Total			**17,969**	**6.22**	**3,593**
Inferred	Goose Main	Open Pit	215	3.2	22
		Underground	429	6.83	94
	Echo	Underground	71	5.91	14
	Llama	Underground	295	6.77	64
	Umwelt	Open Pit	121	2.29	9
		Underground	1,788	11.59	667
	George	Open Pit	929	4.75	142
		Underground	3,902	6.69	840
Inferred Total			**7,750**	**7.43**	**1,851**

Notes: CIM definitions were used for the Mineral Resources.
Open pit Mineral Resources are constrained by an optimized pit shell at a gold price of US$1,500 /oz gold with a 1.00 $US:$CD exchange rate.
The cut-off grade applied to the open pit resources is 1.0 g/t gold. The underground cut-off grade is 4.0 g/t gold for all George Mineral Resources (LCP North, LCP South, Locale 1, Locale 2, GH, and Slave), 3.5 g/t gold for Goose Main, Echo and Llama, and 4.5 g/t for the Umwelt deposit.
The George Mineral Resources were estimated within mineral domains expanded to a minimum horizontal width of 2 m for the underground Resources.
Drilling results up to December 31, 2013, are included except for Echo (new drilling to July 4, 2014) and Locale 1 and Locale 2 (new sampling to July 21, 2014).
The numbers may not add due to rounding.
Source: AMC Mining Consultants (Canada) Ltd. 2015

14.2 Goose Site

The Goose Site consists of the Goose Main, Echo, Llama, and Umwelt deposits. All Mineral Resources for the Goose Site are presented in this section; however, this Initial Project Feasibility Study focuses on developing the Goose Main, Llama and Umwelt deposits. The lithological domains for these deposits were provided by Sabina. The domains were reviewed and accepted by AMC. Building of mineralization domains was also carried out by Sabina using a gold threshold of 0.3 g/t gold. Mineralization domains were reviewed and accepted by AMC with no changes.

All estimations were carried out in Datamine™ software. To account for the folded nature of the Goose deposits, the dynamic anisotropy option in Datamine™ was used for estimating, which allows the orientation of the ellipsoid to be defined individually for each block in the model. Interpolation was carried out using ordinary kriging (OK) for all the deposits except two small gold zones at Llama where interpolation was carried out using inverse distance squared (ID2).

14.2.1 Data Used

14.2.1.1 Drill Hole Database

The data used in the estimate consisted of surface diamond drill hole data held in a Microsoft Access® database, which was provided to AMC as Microsoft Excel® files. The data type and number of holes are shown in Table 14.4.

Table 14.4: Goose Site Drill Hole Data used in the October 2014 AMC Resource Estimate

Deposit	Year	No. of Drill Holes	No. of Assays	Metres Drilled (m)
Goose Main	1992 to 2002	197	22,856	42,602
	2004 to 2005	77	15,559	23,125
	2008	20	6,264	7,618.80
	2010	5	699	906
	2011	4	308	594
	2012	16	2,307	3315
	2013	17	1,540	2,508
	Total	336	49,533	80,669
Echo	2009	11	834	2,651
	2010	24	936	4,154
	2013	18	1,896	3,963
	2014	22	4,360	7,922
	Total	75	8,026	18,690
Llama	1997	1	71	83.46
	2009	1	65	269
	2010	58	6,064	15,391
	2011	30	4,576	10,191
	2012	64	10,773	16,598
	2013	64	9,188	14,492
	Total	218	30,737	57,023
Umwelt	1997	1	40	109
	2005	2	466	496
	2010	58	6,787	16,155
	2011	63	10,550	34,127
	2012	49	11,186	28,256
	2013	67	6,949	9,329
	Total	240	35,978	88,471

Note: All drill holes are surface diamond drill holes.
Drill data to 31 December 2013 for all deposits except Echo, which is to 4 July 2014.
Source: Sabina Gold & Silver Corp. 2015

14.2.1.2 Bulk Density

The collection of bulk density measurements is described in section 10. Mineralization at the Property is hosted within competent rock that contains minimal voids, pits and oxidized surfaces. Previous operators undertook a comparative study between specific gravity (SG) and bulk density measurements. As the overall difference between the two determinations was negligible (less than 1%), SG measurements are considered to be a good approximation of bulk density (Cater et al. 2009).

The SG values used for the main rock types are shown in Table 14.5. The majority of mineralization is hosted within the iron formation at the Llama, Echo and Umwelt deposits. At the Goose Main deposit, mineralization also occurs in the greywacke and mudstone.

For the October 21, 2014 estimates, local SGs were derived using the wireframes of each rock type. In the case of the Echo deposit, when the number of samples was less than 100 per rock type, the SGs of the rock types from the adjacent Goose Main deposit were applied.

Table 14.5: Mean Specific Gravity Values for the Goose Site (t/m^3)[1]

Stratigraphy	Goose Main	Echo	Llama	Umwelt
Overburden	1.80	1.80	1.80	1.80
Gabbroic Dyke	3.00	3.00	3.01	2.99
Felsic Dyke	2.69	2.69	2.73	2.70
Upper Iron Formation	2.93	2.98	2.95	2.93
Lower Iron Formation	3.03	3.03	3.14	3.15
Deep Iron Formation	2.82	N/A	2.76	2.79
Phyllite	2.83	2.83	N/A	N/A
Lower Greywacke	2.78	2.75	2.77	2.76
Upper Greywacke	2.77	2.77	2.83	2.83
Middle Mudstone	N/A	N/A	N/A	2.91

[1]An overburden bulk density value of 1.80 t/m^3 was assigned to all deposits.
Source: AMC Mining Consultants (Canada) Ltd. 2015

14.2.2 Domain Modelling

The geology model for all the deposits is composed of six main rock types:

- Upper Greywacke;
- Lower Greywacke;
- Upper Iron Formation;
- Lower Iron Formation;
- Felsic dykes; and
- Gabbroic dykes.

Other stratigraphic units have been recognized at the various deposits, many of which have been modelled. At the Goose Main and Echo deposits, Sabina modelled a phyllite unit. At the Goose Main, Llama and Umwelt deposits, a deep iron formation was modelled. A middle mudstone was modelled at the Umwelt deposit. The stratigraphic package is folded at these locations.

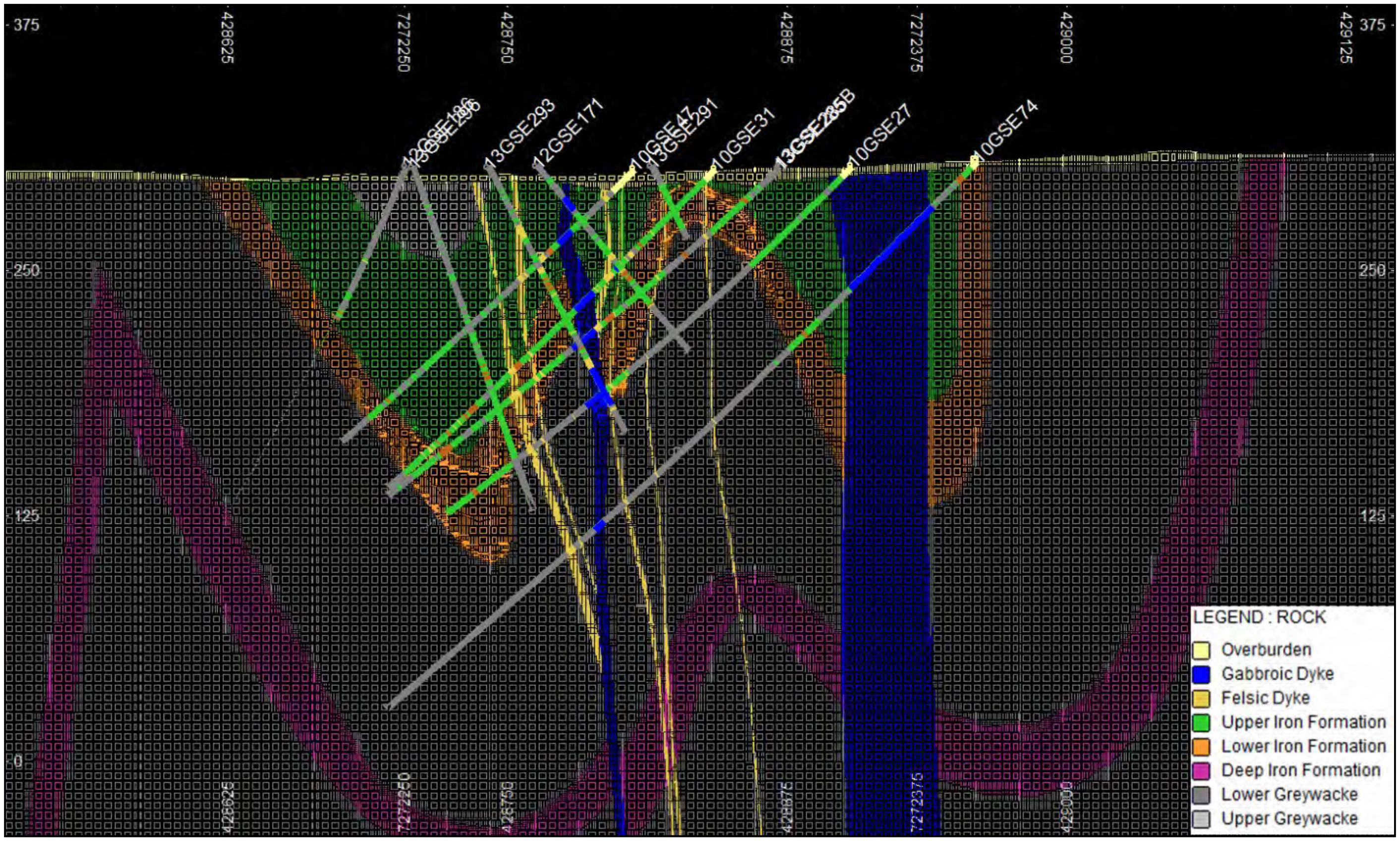
As previously mentioned, building of mineralization domains was also carried out by Sabina, using a gold threshold of 0.3 g/t gold. The mineralization domains were built on the understanding that mineralization occurs predominantly in the Lower Iron Formation and is cross-cut by gabbroic and felsic dykes. Mineralization domains were reviewed and accepted by AMC with no changes.

The blocks inside the block models are coded by different geological units, bulk density values and estimated gold values. Figure 14.1 shows an example of the folded nature of the stratigraphic units and mineralization at the Goose Site.

The number of mineralization domains varied between the deposits. There were two mineralization domains at the Echo deposit, four at the Llama deposit, 13 at the Umwelt deposit, and 24 at the Goose Main deposit. The higher number of mineralization domains at the Goose Main deposit reflects the more complex geometry of the stratigraphic units and mineralizing system at this deposit.

On completion of the domain modelling, visual checks were carried out to ensure that the constraining wireframes respected the raw data.

Figure 14.1: Cross-Section of Llama Deposit Mineralization Looking Northwest



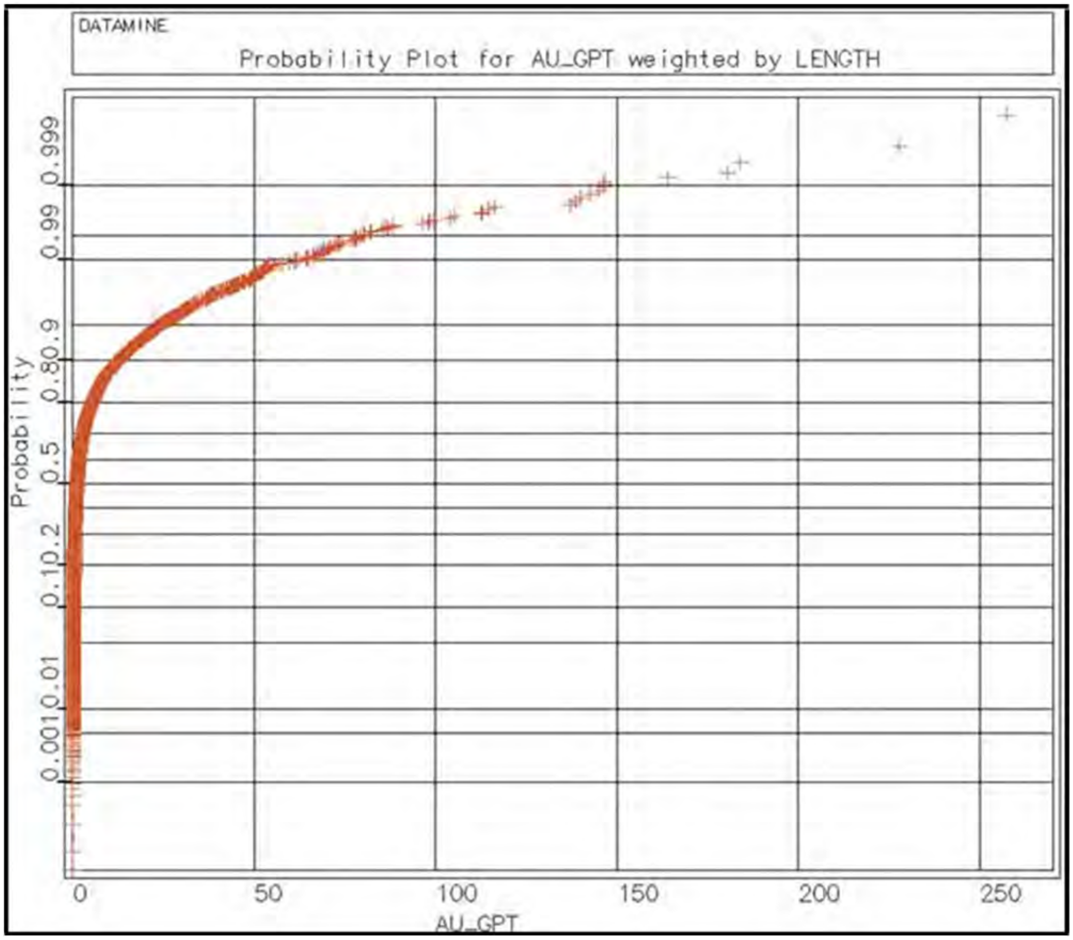
Source: AMC Mining Consultants (Canada) Ltd. 2015

14.2.3 Statistics and Compositing

Within the mineralization domains at the Goose Main deposit, not all of the lower grade material was sampled in the older drilling and locally, some of this unsampled material was included. Unsampled material was given a gold value of 0 g/t. The effect of this in the largest mineralization domain (Domain 1) was an increase in the number of samples from 8,182 to 8,276, and a decrease in the mean grade from 3.96 g/t gold to 3.87 g/t gold (Table 14.6). Unsampled material within mineralization domains was not an issue at the Echo, Llama or Umwelt deposits as there was no unsampled material within the mineralization domains.

The gold assay data sets for all four deposits were viewed on log probability plots. Based on the plots, raw gold assay data was capped at 80 g/t for Umwelt and Echo, and 100 g/t for Llama. Goose Main was divided into an east and west zone. Goose Main West was capped at 150 g/t and Goose Main East was capped at 50 g/t. As an example, Figure 14.2 shows the log probability plot of the Umwelt deposit.

Figure 14.2: Log Probability Plot of Raw Gold Assay Data for the Umwelt Deposit



Source: AMC Mining Consultants (Canada) Ltd. 2015

A review of sample lengths for each deposit showed that the majority of samples have a length of 1.0 m. As such, a composite length of 1.0 m was chosen for all deposits.

Table 14.6 and Table 14.7 show the statistics of raw, capped, and composite gold assay data from the main mineralization domains for each deposit.

Table 14.6: Statistics of Raw, Capped, and Composite Gold Assay Data – Goose Main and Echo

	Goose Main				Echo		
	Raw	Zero Values1	Capped	Comp.	Raw	Capped	Comp.
No. of Samples	8,182	8,276	8,276	6,454	453	453	378
Minimum (Au g/t)	0.00	0.00	0.00	0.00	0.00	0.00	0.015
Maximum (Au g/t)	727.00	727.00	150.00	150.00	119.70	80.00	66.48
Mean (Au g/t)	3.96	3.87	3.69	3.69	3.89	3.75	3.92
SD (Au g/t)	15.56	15.37	10.61	9.26	10.20	8.78	7.67
CoV	3.92	3.97	2.88	2.51	2.62	2.34	1.96

Notes: 1Zero values replaced unsampled intervals within the mineral domains.
SD = Standard Deviation; CoV = Coefficient of Variation.
Source: AMC Mining Consultants (Canada) Ltd. 2015

Table 14.7: Statistics of Raw, Capped and Composite Gold Assay Data – Llama and Umwelt

	Llama			Umwelt		
	Raw	Capped	Comp.	Raw	Capped	Comp.
No. of Samples	**2,107**	**2,107**	**2,025**	**4,170**	**4,170**	**3,964**
Minimum (Au g/t)	**0.03**	**0.03**	**0.03**	**0.00**	**0.03**	**0.03**
Maximum (Au g/t)	**393**	**100**	**100**	**257**	**80**	**80**
Mean (Au g/t)	**5.08**	**4.76**	**4.76**	**5.84**	**5.61**	**5.61**
SD (Au g/t)	**16.53**	**12.31**	**11.04**	**13.81**	**11.80**	**10.68**
CoV	**3.25**	**2.58**	**2.32**	**2.37**	**2.10**	**1.90**

Notes: SD = Standard Deviation; CoV = Coefficient of Variation.
Source: AMC Mining Consultants (Canada) Ltd. 2015

14.2.4 Block Model

14.2.4.1 Block Model Parameters

Parent blocks of 5 m by 10 m by 5 m were used in the Goose Main, Llama and Umwelt block models with sub-blocking using a split of "2". The block model dimensions and rotation are shown in Table 14.8. The models are rotated counter-clockwise around the Z-axis.

Table 14.8: Block Model Parameters – Goose Main, Llama and Umwelt

	Goose Main			Llama			Umwelt		
Parameter	**X**	**Y**	**Z**	**X**	**Y**	**Z**	**X**	**Y**	**Z**
Origin (m)	434,320	7,269,040	-230	428,985	7,271,330	-250	430,270	7,269,780	-600
Rotation Angle (deg)	0	0	-60	0	0	-35	0	0	-35
No. of Blocks	140	104	112	172	130	117	160	169	185

Source: AMC Mining Consultants (Canada) Ltd. 2015

Parent blocks of 10 m by 5 m by 5 m were used in the Echo block model with sub- blocking using a split of five in the X direction, and ten in the Y and Z directions. The block model dimensions are shown in Table 14.9. The Echo model is not rotated.

Table 14.9: Block Model Parameters – Echo

	Echo		
Parameter	**X**	**Y**	**Z**
Origin (m)	432,200	7,268,750	-150
Rotation Angle (deg)	0	0	0
No. of Blocks	80	80	110

Source: AMC Mining Consultants (Canada) Ltd. 2015

14.2.5 Variography and Grade Estimation

Variography was carried out on all domains that had sufficient sample density. The purpose of the variograms was to produce inputs for the OK estimates. In order to model the variograms more accurately, the Datamine™ software "unfold" option was used.

The OK interpolation method was used for the estimates for Goose Main, Echo, Umwelt, and the majority of gold zones in the Llama deposit. The ID2 method was used for two small zones in the Llama deposit. The dimensions of the search radius for each deposit are shown in Table 14.10. To account for the folded nature of the deposits, the "dynamic anisotropy" option in Datamine™ was used for estimating the Goose deposits, allowing the orientation of the search ellipsoid to be defined individually for each block in the model.

A number of passes were employed, each using different search distances and multiples as follows:

- Pass 1 = 1 x search distance
- Pass 2 = 2 x search distance
- Pass 3 = 3 or 4 x search distance, depending on the domain

The third pass was completed to fill the wireframes to depth. The search distances are shown in Table 14.10 along with the minimum and maximum number of samples used for each pass.

At the Goose Main deposit, 71% of the composite samples were contained within Domains 1 and 15. The remaining 22 domains were interpreted from fewer drill holes. As such, less strict parameters were used to populate these domains.

Table 14.10: Minimum and Maximum Sample Parameters – Goose Site

Deposit	Pass	X (m)	Y (m)	Z (m)	Min No. of Samples	Max No. of Samples	Min No. Of Drill Holes
Goose Main Domain 1 and 15	1	40	40	5	4	16	2
	2	80	80	10	4	16	2
	3	120	120	30	2	16	1
Goose Main All Other Domains	1	40	40	5	4	16	1
	2	80	80	10	4	16	1
	3	120	120	30	2	16	1
Echo	1	50	50	6	8	20	3
	2	100	100	12	6	20	2
	3	150	150	18	2	20	1
Llama	1	40	40	5	4	16	2
	2	80	80	10	4	16	2
	3	120	120	30	2	16	1
Umwelt Domain 1	1	60	60	10	4	20	2
	2	120	120	20	4	20	2
	3	180	180	30	2	16	1
Umwelt All Other Domains	1	60	60	10	4	16	2
	2	120	120	20	4	16	2
	3	180	180	30	2	16	1

Source: AMC Mining Consultants (Canada) Ltd. 2015

14.2.6 Resource Classification

Resource classification was completed using an assessment of geological and mineralization complexity, data quality and data density. Classification was carried out using data support as a main criterion, with a manual review creating volumes based on drill hole density and number of samples to inform a block, thus removing outliers.

Parameters were applied as shown in Table 14.11 and Table 14.12 for the Goose Main, Umwelt and Llama deposits. More stringent rules of support were applied to the Goose Main deposit due to the more complex geometry of the mineralization.

Stringent rules of support were also applied to the Echo deposit, as it is a new discovery and geological understanding will advance with more drilling. At Echo, no Measured Resources were classified, as drill hole spacing is too wide. Indicated Resource classification was based on blocks filled in the first pass of the estimate (see Table 14.10). Inferred Resource classification was based on blocks filled in the second and third pass of the estimate (see Table 14.10).

Table 14.11: Main Criteria for Resource Classification – Goose Main

Resource Classification	Search Distance (m)	Goose Main		
		Minimum No. of Samples	Maximum No. of Samples	Minimum No. of Drill Holes
Measured	25 by 25 by 25	10	32	5
Indicated	50 by 50 by 50	8	32	4
Inferred	100 by 100 by 100	4	16	2

Source: AMC Mining Consultants (Canada) Ltd.2015

Table 14.12: Main Criteria for Resource Classification – Umwelt and Llama

Resource Classification	Search Distance (m)	Umwelt and Llama		
		Minimum No. of Samples	Maximum No. of Samples	Minimum No. of Drill Holes
Measured	30 by 30 by 30	8	16	4
Indicated	60 by 60 by 60	8	16	4
Inferred	120 by 120 by 120	4	16	2

Source: AMC Mining Consultants (Canada) Ltd. 2015

14.2.7 Block Model Validation

The block models were validated in three ways. First, visual checks were carried out to ensure that the grades respected the raw gold assay data and also lay within the constraining wireframes. Secondly, swath plots were reviewed. Lastly, in addition to an OK estimate, Nearest Neighbour (NN), ID2, and Inverse Distance Cubed (ID3) were run in Datamine™ for comparison purposes. These results were statistically compared to the composite gold assay data with satisfactory results.

14.2.8 Mineral Resource Estimates

Mineral Resource estimates consist of open pit and underground Mineral Resources for all four deposits at the Goose Site. Open pit Mineral Resources are reported between a base-of-overburden surface and a conceptual pit shell based on a US$1,500/oz gold price. Assumptions considered for the conceptual pit shell included mining costs, processing costs and gold recoveries obtained from the Tetra Tech 2013 PFS Technical Report. These are summarized below in Table 14.13. A cut-off of 1.0 g/t gold was applied for reporting the open pit Mineral Resources.

Table 14.13: Conceptual Mineral Resource Open Pit Shell Parameters

Item	PFS Open Pit Optimization Parameters	Unit
Gold Price	1,500	US$/oz
Exchange Rate	1.00	US$ to C$
Refining/Transport	7.00	C$/oz
Royalties	3.70	%
Processing Costs	30.90	C$/tonne ore
G & A	14.60	C$/tonne ore
Base Mining Costs	4.00	C$/tonne
Incremental Mining Costs	0.03	C$/tonne/10m bench
Preliminary Overall Slope Angles	47 – 48	degrees
Plant Rate	5	Mtpa
PFS Optimization Metallurgical Recovery	Metallurgical Recovery (%) – 90% above 10g/t, 8.0846 x ln (grade, g/t) + 69.917 below 10 g/t (ln=natural log)	

Source: AMC Mining Consultants (Canada) Ltd. 2015

The results of the Goose Site open pit Mineral Resource estimates are shown in Table 14.14 and Table 14.15 at a range of cut-offs, with the selected cut-off shown in bold.

Table 14.14: Goose Main and Echo Open Pit Mineral Resource Estimates

Resource Classification	Cut-off Grade (g/t Au)	Goose Main			Echo		
		Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)	Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)
Measured	0.00	5,339	3.72	638	N/A	N/A	N/A
	0.50	5,035	3.92	635	N/A	N/A	N/A
	0.75	4,789	4.09	630	N/A	N/A	N/A
	1.00	4,478	4.32	621	N/A	N/A	N/A
	1.25	4,160	4.56	610	N/A	N/A	N/A
	1.50	3,824	4.84	595	N/A	N/A	N/A
	1.75	3,509	5.13	579	N/A	N/A	N/A
	2.00	3,235	5.40	562	N/A	N/A	N/A
Indicated	0.00	3,665	3.41	402	321	6.07	63
	0.50	3,353	3.70	399	321	6.07	63
	0.75	3,107	3.95	394	321	6.07	63
	1.00	2,877	4.19	388	321	6.07	63
	1.25	2,638	4.47	379	317	6.12	62
	1.50	2,406	4.77	369	315	6.15	62
	1.75	2,191	5.08	358	311	6.21	62
	2.00	2,007	5.37	347	307	6.27	62
Inferred	0.00	314	2.41	24	N/A	N/A	N/A
	0.50	294	2.55	24	N/A	N/A	N/A
	0.75	268	2.74	24	N/A	N/A	N/A
	1.00	215	3.20	22	N/A	N/A	N/A
	1.25	170	3.75	20	N/A	N/A	N/A
	1.50	137	4.33	19	N/A	N/A	N/A
	1.75	124	4.61	18	N/A	N/A	N/A
	2.00	110	4.96	17	N/A	N/A	N/A

Notes: CIM definitions were used for the Mineral Resources.
Open pit Mineral Resources were constrained by an optimized pit shell at a gold price of US$1,500/oz gold.
Drilling results are included up to December 31, 2013 for Goose Main, and July 4, 2014 for Echo.
Source: AMC Mining Consultants (Canada) Ltd. 2015

Table 14.15: Llama and Umwelt Open Pit Mineral Resource Estimates

Resource Classification	Cut-off Grade (g/t Au)	Llama			Umwelt		
		Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)	Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)
Measured	0.00	2,315	4.88	363	4,312	5.29	734
	0.50	2,233	5.04	362	4,055	5.61	731
	0.75	2,059	5.41	358	3,860	5.86	727
	1.00	1,874	5.86	353	3,699	6.07	722
	1.25	1,716	6.30	347	3,559	6.27	717
	1.50	1,594	6.68	342	3,394	6.51	710
	1.75	1,490	7.03	337	3,243	6.73	702
	2.00	1,401	7.36	331	3,065	7.02	691
Indicated	0.00	1,086	4.70	164	2,591	4.22	352
	0.50	1,005	5.05	163	2,355	4.61	349
	0.75	913	5.49	161	2,150	4.99	345
	1.00	821	6.01	159	1,963	5.38	340
	1.25	748	6.49	156	1,821	5.71	334
	1.50	693	6.89	154	1,698	6.03	329
	1.75	642	7.31	151	1,588	6.33	323
	2.00	602	7.67	148	1,494	6.61	318
Inferred	0.00	N/A	N/A	N/A	269	1.40	12
	0.50	N/A	N/A	N/A	234	1.56	12
	0.75	N/A	N/A	N/A	192	1.77	11
	1.00	N/A	N/A	N/A	121	2.29	9
	1.25	N/A	N/A	N/A	82	2.84	8
	1.50	N/A	N/A	N/A	63	3.29	7
	1.75	N/A	N/A	N/A	54	3.56	6
	2.00	N/A	N/A	N/A	49	3.73	6

Notes: CIM definitions were used for the Mineral Resources.
Open pit Mineral Resources were constrained by an optimized pit shell at a gold price of US$1,500/oz gold.
Drilling results up to December 31, 2013 are included.
Source: AMC Mining Consultants (Canada) Ltd. 2015

The underground Mineral Resources were reported between the base of the conceptual pit shells and the base of the Inferred Resource at each deposit. No allowances were made for crown pillars. The cut-off applied to the underground Mineral Resources was 3.5 g/t gold for all deposits except Umwelt, which was 4.5 g/t gold. Work to date has indicated the need for a higher cost underground mining method at Umwelt than at the other deposits.

Assumptions to derive a cut-off grade included mining costs, processing costs and recoveries were obtained from the Tetra Tech 2013 PFS Technical Report.

The results of the Goose Site underground Mineral Resource estimates are shown in Table 14.16 and Table 14.17 at a range of cut-offs, with the selected cut-offs shown in bold.

Table 14.16: Goose Main and Echo Underground Mineral Resource Estimates

Resource Classification	Cut-off Grade (g/t Au)	Goose Main			Echo		
		Tonnes (kt)	Grade (g/t Au)	Metal(koz Au)	Tonnes (kt)	Grade (g/t Au)	Metal(koz Au)
Measured	2.0	198	4.64	30	N/A	N/A	N/A
	3.0	131	5.76	24	N/A	N/A	N/A
	3.5	110	6.24	22	N/A	N/A	N/A
	4.0	90	6.80	20	N/A	N/A	N/A
	4.5	76	7.28	18	N/A	N/A	N/A
	5.0	60	7.95	15	N/A	N/A	N/A
	5.5	44	8.92	13	N/A	N/A	N/A
	6.0	37	9.53	11	N/A	N/A	N/A
	7.0	28	10.51	10	N/A	N/A	N/A
Indicated	2.0	1,579	5.19	263	1,071	4.62	159
	3.0	1,065	6.51	223	722	5.66	131
	3.5	853	7.32	201	596	6.17	118
	4.0	698	8.12	182	500	6.65	107
	4.5	564	9.05	164	424	7.08	97
	5.0	451	10.12	147	366	7.45	88
	5.5	376	11.09	134	297	7.96	76
	6.0	326	11.91	125	257	8.31	69
	7.0	247	13.62	108	176	9.15	52
Inferred	2.0	869	4.70	131	187	3.88	23
	3.0	546	6.06	106	96	5.21	16
	3.5	429	6.83	94	71	5.91	14
	4.0	365	7.38	87	63	6.17	13
	4.5	272	8.48	74	42	7.13	10
	5.0	220	9.36	66	39	7.34	9
	5.5	198	9.81	63	35	7.57	9
	6.0	163	10.67	56	28	8.02	7
	7.0	115	12.43	46	19	8.66	5

Notes: CIM definitions were used for the Mineral Resources.
Drilling results are included up to December 31, 2013 for Goose Main, and July 4, 2014 for Echo.
Source: AMC Mining Consultants (Canada) Ltd. 2015

Table 14.17: Llama and Umwelt Underground Mineral Resource Estimates

Resource Classification	Cut-off Grade (g/t Au)	Llama			Umwelt		
		Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)	Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)
Measured	2.0	247	4.01	32	7.6	3.46	0.8
	3.0	146	5.10	24	2.4	5.86	0.4
	3.5	110	5.72	20	1.7	6.84	0.4
	4.0	88	6.22	18	1.4	7.51	0.3
	4.5	65	6.89	15	0.9	9.21	0.3
	5.0	51	7.52	12	0.8	9.96	0.3
	5.5	43	7.95	11	0.7	10.36	0.2
	6.0	38	8.26	10	0.5	12.86	0.2
	7.0	28	8.87	8	0.4	14.28	0.2
Indicated	2.0	1,256	6.30	254	6,718	6.05	1306
	3.0	891	7.86	225	5,141	7.14	1180
	3.5	752	8.72	211	4,452	7.74	1108
	4.0	647	9.52	198	3,865	8.35	1037
	4.5	573	10.21	188	3,387	8.92	972
	5.0	506	10.93	178	2,969	9.51	908
	5.5	456	11.56	169	2,630	10.06	851
	6.0	412	12.18	161	2,332	10.62	796
	7.0	336	13.48	145	1,829	11.76	691
Inferred	2.0	449	5.36	77	4,127	6.78	900
	3.0	332	6.38	68	3,034	8.33	813
	3.5	295	6.77	64	2,513	9.39	758
	4.0	254	7.25	59	2,086	10.54	707
	4.5	211	7.86	53	1,788	11.59	667
	5.0	172	8.56	47	1,557	12.61	632
	5.5	143	9.22	42	1,363	13.67	599
	6.0	126	9.69	39	1,221	14.59	573
	7.0	94	10.77	32	1,014	16.25	530

Notes: CIM definitions were used for the Mineral Resources.
Drilling results to December 31, 2013 are included.
Source: AMC Mining Consultants (Canada) Ltd. 2015

14.3 George Site

The George Site consists of the Locale 1, Locale 2, LCP North, LCP South, GH, and Slave deposits. All Mineral Resources for the George Site are represented in this section; however, this Initial Project Feasibility Study focuses on developing the Goose Main, Llama and Umwelt deposits at the Goose Site. The lithological domains for these deposits were provided by Sabina. The domains were reviewed by AMC and generally accepted. Building of mineralization domains was also carried out by Sabina using a gold threshold of 0.3 g/t gold. Mineralization domains were reviewed and accepted by AMC with only minor changes.

All estimations were carried out in Datamine™ software. Due to the narrow width of mineralization at the Locale 1, Locale 2, GH, and Slave deposits, a 2D accumulation method was used to estimate the Mineral Resource. In this method, the gold accumulation (gold grade multiplied by horizontal thickness) and the horizontal thickness are estimated into a 2D block model, which is required to correctly assign weights to samples of different lengths during estimation. The estimated block grade is then back-calculated by dividing estimated block gold accumulation by estimated horizontal thickness of the block. With this method, one dimension of the parent block is the horizontal thickness of the mineralization. The other parent cell dimensions are determined by the average drill hole spacing in longitudinal section. The parent block size varies between deposits with the smallest block size being 20 m by 10 m and the largest at 25 m by 25 m; most commonly, these are in the Y and Z direction respectively.

Interpolation was carried out for all deposits using the OK method.

14.3.1 Data Used

14.3.1.1 Drill Hole Database

The data used in the estimate consisted of surface diamond drill hole data held in a Microsoft Access® database, which was provided to AMC as Microsoft Excel® files. The data type and number of holes are shown in Table 14.18.

Table 14.18: George Site Drill Hole Data used in the October 2014 AMC Resource Estimate

Deposit	Year	No. of Drill Holes	No. of Assays	Metres Drilled (m)
Locale 1	1986 to 1997	184	6,678	28,595
	2005	2	968	1,036
	2008	9	1,461	1,522
	2012	20	1,791	5,518
	2013	8	478	1,373
	Total	223	11,376	38,044
Locale 2	1985 to 1997	226	12,115	39,882
	2005	4	1,697	2,102
	2006	1	413	703
	2008	10	2,123	2,189
	2012	11	2,066	3,328
	2013	20	3,412	6,323
	Total	272	21,826	54,527
LCP North	1985 to 1997	69	1,720	9,693
	2007	2	728	763
	2012	4	173	864
	2013	4	-	311
	Total	79	2,621	11,631
LCP South	1988 to 1997	3	72	270
	2006	6	770	1,323
	2007	30	9,440	9,763
	2012	6	329	1,336
	2013	43	2,613	6,555
	Total	88	13,224	19,247
GH	1986 to 1997	69	3,096	10,915
Slave	1986 to 1997	39	2,130	5,331

Note: All drill holes are surface diamond drill holes.
Numbers may not directly correlate to those in the Tetra Tech 2014 Technical Report as drill holes were reassigned to deposit areas in August 2014.
Drill hole data is to 31 December 2013 for all deposits except for Locale 1 and Locale 2, for which some existing core was re-sampled up to 21 July 2014.
Source: Sabina Gold & Silver Corp.2015

14.3.1.2 Bulk Density

The collection of bulk density measurements is described in the section 10. Mineralization at the Property is hosted within competent rock that contains minimal voids, pits, and oxidized surfaces. Previous operators undertook a comparative study between SG and bulk density measurements. As the overall difference between the two determinations was negligible (less than 1%), SG measurements are considered to be a good approximation of bulk density (Cater et al. 2009).

The SG values used for the main rock types of each deposit are shown in Table 14.19. For each deposit, the local specific gravities were derived using the wireframes of each rock type where possible. In the case of the GH and Slave deposits, where the number of samples was less than 100 per rock type, the SGs of rock types from the entire George data set were applied. In the case of the other deposits, if there were fewer than 100 samples per rock type, data from the adjacent deposit was added to the local values.

Table 14.19: Mean Specific Gravity Values for the George Deposits (t/m3)1

Stratigraphy	Locale 1	Locale 2	LCP North	LCP South	GH	Slave
Upper Greywacke	N/A	N/A	2.76	2.76	N/A	N/A
Lower Greywacke	2.76	2.76	2.77	2.77	2.76	2.76
Upper Iron Formation	3.04	3.04	N/A	N/A	3.07	3.07
Lower Iron Formation	3.12	3.12	3.00	3.00	3.07	3.07
Deep Iron Formation	2.77	2.77	2.77	2.77	N/A	N/A
Phyllite	2.81	2.81	2.81	2.81	N/A	N/A
Pelite	N/A	N/A	N/A	2.81	2.822	N/A
Intermediate Dyke	2.76	2.76	2.81	N/A	N/A	N/A
Felsic Dyke	N/A	N/A	N/A	2.69	2.67	N/A
Gabbro	N/A	2.92	N/A	N/A	2.89	N/A

1. An overburden value of 1.80 t/m3 was assigned to all deposits.
2. Value is for the specific gravity of the mudstone at the GH deposit.
Source: AMC Mining Consultants (Canada) Ltd. 2015

The majority of mineralization is hosted within the iron formation at the George Site. Occasionally, mineral domains were expanded beyond the iron formation to a minimum width of 2 m. These extensions were considered waste material and assigned the bulk density of the greywacke.

14.3.2 Domain Modelling

Sabina provided all geology models. The geology models for the George Site all consisted of iron formation and greywacke solids. Faults (Locale 1, Locale 2, Slave), felsic or intermediate dykes (all deposits except Slave), and mudstone or pelite and phyllite solids (all deposits except Slave) were provided where necessary.

As previously mentioned, building of mineralization domains was carried out by Sabina, using a gold threshold 0.3 g/t gold. Mineralization domains were accepted by AMC with only minor changes.

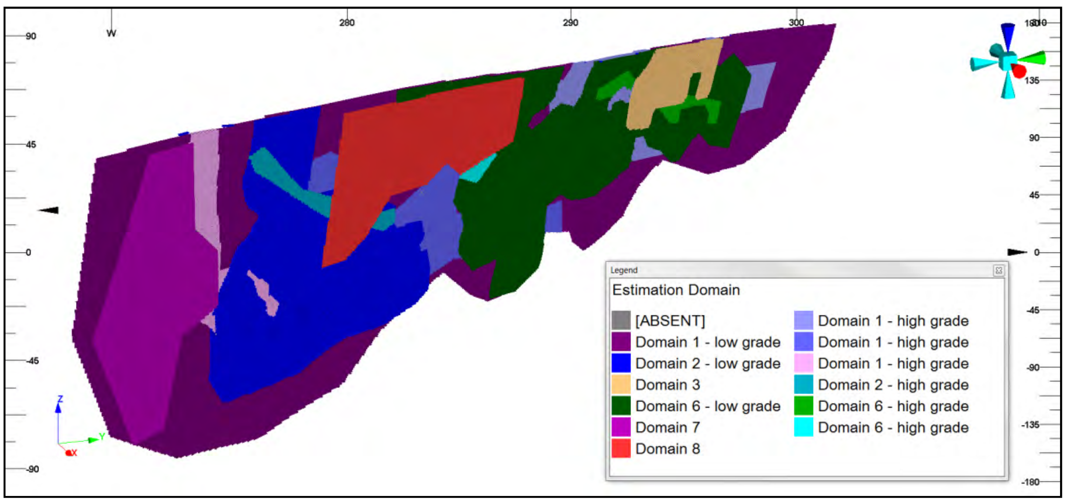
The blocks inside the block models are coded by different geological units, bulk density values and estimated gold values. Figure 14.3 shows the mineralization domains at the Locale 1 deposit.

The number of mineralization domains varied among the deposits. There were two mineralization domains each at the LCP North and LCP South deposits, five each at the GH, and Slave deposits, six at the Locale 1 deposit (three of these zones were subdivided into low and high-grade domains before estimation, as shown in Figure 14.3), and 19 at the Locale 2 deposit. The reason for the higher number of mineralization domains at Locale 2 was the numerous footwall zones.

The previous Locale 1 Mineral Resource estimate contained domains 4 and 5. These domains now form part of the Locale 2 Mineral Resource. This is in addition to the change in mineralization interpretation for Locale 1 and 2.

On completion of the domain modelling, visual checks were carried out to ensure that the constraining wireframes respected the raw data.

Figure 14.3: 3D View of Locale 1 Mineralization Domains Looking Northwest



Note: Different colours represent individual mineral domains including high-grade sub-domains.
Source: AMC Mining Consultants (Canada) Ltd. 2015

14.3.3 Statistics and Compositing

The LCP North and LCP South deposits were estimated into a normal block model; as such, AMC selected a compositing interval of 1 m, which is the median sample length. This length also gave the appropriate selectivity for the narrow-vein style of this mineralization. Composited gold assay data was then examined on a probability plot, and gold grades were capped at 40 g/t gold, affecting less than 1.5% of the samples. The raw, composited, and capped gold assay data of the LCP North and LCP South deposits is shown in Table 14.20.

Table 14.20: Statistics of Raw, Composited and Capped Gold Assay Data – LCPn & LCPs

	LCP North			LCP South		
	Raw	Composite	Capped	Raw	Composite	Capped
No. of Samples	601	425	425	695	707	707
Minimum (Au g/t)	0.02	0.02	0.02	0.00	0.01	0.01
Maximum (Au g/t)	232.84	73.79	40.00	71.33	71.33	40.00
Mean (Au g/t)	4.94	4.81	4.55	5.90	5.80	5.70
Standard Deviation (Au g/t)	13.40	9.03	7.58	8.93	8.22	7.60
Coefficient of Variation	2.71	1.88	1.67	1.51	1.42	1.33

Note: Statistics are shown for the largest domain of each deposit.
Source: AMC Mining Consultants (Canada) Ltd. 2015

The Locale 1, Locale 2, GH, and Slave deposits were estimated by the 2D accumulation method. As such, the full width of mineralization was composited in preparation for the 2D accumulation method. True and horizontal thicknesses were calculated using the orientation of each drill hole and the average vein orientation.

The statistics of the raw and composited gold assay data are presented for the main domain of each of the deposits in Table 14.21. True thickness is presented in Table 14.21 as this is a more geologically meaningful value than horizontal thickness. However; it should be noted that all gold accumulation, thickness estimation and dilution post-processing was completed on the basis of horizontal thickness to simplify the estimation process, and ease of use of the block model.

Table 14.21: Statistics of Locale 1, Locale 2, GH, and Slave Deposits

Deposit and Domain	Data Type	Drill Holes	Composites		
	Variable	Au (g/t)	Au (g/t)	True Thickness (m)	Au Accumulation
	Weighted by	Length	True Thickness	Declustered Weights	Declustered Weights
Locale 1 Domain 1	No. of Samples	2,080	202	202	202
	Minimum	0.005	0.13	0.36	0.17
	Maximum	88.3	24.2	18.35	263.44
	Mean	4.95	4.78	5.04	24.94
	SD	8.83	4.31	3.19	33.36
	CoV	1.78	0.90	0.63	1.34
Locale 2 Domain 1	No. of Samples	1,878	183	183	183
	Minimum	0	0.04	0.971	0.03
	Maximum	102.14	36.04	15.55	161.23
	Mean	4.39	4.30	4.57	19.91
	SD	8.75	4.20	2.88	26.20
	CoV	1.99	0.98	0.63	1.34
GH Domain 1 IF	No. of Samples	294	53	53	53
	Minimum	0.01	0.0	0.2	0.0
	Maximum	96.7	14.7	12.0	93.1
	Mean	4.3	4.2	2.7	11.4
	SD	6.7	3.1	2.5	18.8
	CoV	1.56	0.7	0.9	1.6
Slave Domain 2	No. of Samples	207	21	21	21
	Minimum	0.01	0.1	0.5	0.2
	Maximum	64.2	8.6	26.4	143.5
	Mean	4.5	4.5	5.3	27.8
	SD	7.4	2.7	5.3	41.5
	CoV	1.64	0.6	1.0	1.5

Notes: Statistics are for the largest domain in each deposit based on number of samples/domain.
SD = Standard Deviation; CoV = Coefficient of Variation.
Source: AMC Mining Consultants (Canada) Ltd. 2015

After analysis, capping of gold values was not employed for the Locale 1, Locale 2, GH, and Slave deposits.

14.3.4 Block Model

14.3.4.1 Block Model Parameters

The parent block size for the LCP North and LCP South deposits was 5 m by 10 m by 10 m with sub-blocking employed in both. Sub-blocking resulted in minimum cell dimensions of 0.25 m by 1 m by 1 m.

The 2D accumulation method was used for the other George deposits where one dimension of the parent block is the horizontal thickness of the mineralization. However, the other parent cell dimensions were determined by the average drill hole spacing in longitudinal section. This cell dimension was 20 m by 20 m for Locale 1 and Locale 2, and 25 m by 25 m for the GH and Slave deposits.

The block model dimensions and rotations used for the estimates are shown in Table 14.22. The models were rotated counter-clockwise around the Z-axis.

Table 14.22: Block Model Parameters – George Site

Deposit	LCP North			LCP South		
Parameter	X	Y	Z	X	Y	Z
Origin (m)	386,300	7,315,320	-30	386,650	7,315,550	-70
Rotation Angle (deg)	0	0	-30	0	0	-24
No. of Blocks	300	198	43	120	75	100
Deposit	**Locale 1**			**Locale 2**		
Parameter	X	Y	Z	X	Y	Z
Origin (m)	387,500	7,311,000	-340	387,500	7,311,000	-340
Rotation Angle (deg)	0	0	-30	0	0	-30
No. of Blocks	1	220	36	1	220	36
Deposit	**GH**			**Slave**		
Parameter	X	Y	Z	X	Y	Z
Origin (m)	390,390	7,309,460	-130	389,480	7,312,470	-10
Rotation Angle (deg)	0	0	-38	0	0	-25
No. of Blocks	1	50	30	1	20	18

Note: These are the block model origins used for the estimates and do not correspond to the origins of models that have been regularized for engineering purposes.
Source: AMC Mining Consultants (Canada) Ltd. 2015

14.3.5 Variography and Grade Estimation

Variography was carried out on all domains that had sufficient sample density. If a domain did not have enough sample density for variography, the adjacent domain was used. The exception to this was the Slave deposit, which had insufficient sample density for variography in all domains; therefore, the variography from the GH deposit was used instead.

Interpolation was carried out using the OK method. A number of passes were employed that used increasing search ellipsoid distances. The objective of the search strategy was to capture approximately two lines of drilling in the first pass, and then expand the search in the second and third passes to allow the estimate to capture the minimum number of samples required.

A fourth pass was employed with a large search distance and relaxed search criteria to fill blocks that were un-estimated after three passes. There is low confidence in the grade of the fourth pass blocks, and the majority of them are excluded from the Mineral Resource.

The search parameters are provided in Table 14.23. For the Slave and Locale 2 deposits, the parameters for the largest domain are shown.

Table 14.23: Minimum and Maximum Sample Parameters – George Site

Deposit	Pass	Search Distance Major Direction (m)	Search Distance Semi-major Direction (m)	Search Distance Across Strike (m)	Min. No. of Samples	Max. No. of Samples	Min. No. of Drill Holes
LCP North	1	30	20	5	4	16	2
	2	60	40	10	4	16	2
	3	100	65	15	4	16	2
	4	300	195	45	1	16	2
LCP South	1	30	20	5	4	16	2
	2	60	40	10	4	16	2
	3	100	65	15	4	16	2
	4	300	195	45	1	16	1
Locale 1	1	60	60	N/A	6	16	6
	2	90	90	N/A	6	16	6
	3	120	120	N/A	6	16	6
	4	200	200	N/A	1	16	1
Locale 2 (Domain 1)	1	120	100	N/A	6	16	6
	2	180	150	N/A	6	16	6
	3	200	200	N/A	2	16	2
	4	200	200	N/A	1	16	1
GH	1	65	40	N/A	2	4	2
	2	130	80	N/A	2	4	2
	3	195	120	N/A	2	4	2
	4	260	160	N/A	2	4	2
Slave (Domain 2)	1	60	50	N/A	2	3	2
	2	90	75	N/A	2	3	2
	3	120	150	N/A	1	3	1

Source: AMC Mining Consultants (Canada) Ltd. 2015

14.3.6 Resource Classification

AMC classified the Mineral Resource with consideration of the narrow-vein style of mineralization, the observed gold grade continuity and the drill hole spacing. The nominal drill hole sample spacing in the Indicated Mineral Resource in longitudinal projection was approximately 30 m by 30 m, while the nominal drill hole sample spacing in the Inferred Mineral Resource in longitudinal projection was approximately 50 m by 50 m. Currently, there is insufficient knowledge of the structural style and continuity of the mineralization to support a Measured Resource category for the George deposits.

14.3.7 Block Model Validation

The block models were validated by visual checks, statistics and swath plots.

14.3.8 Mineral Resource Estimates

Mineral Resource estimates consisted of underground and open pit Mineral Resources for all the George deposits.

Open pit Mineral Resources were reported between a base-of-overburden surface and a conceptual pit shell based on a US$1,500/oz gold price. Assumptions considered for the conceptual pit shell included mining costs, processing costs and gold recoveries obtained from the Tetra Tech 2013 PFS technical report (assumptions are shown in Table 14.13). A cut-off of 1.0 g/t gold was applied for reporting the open pit Mineral Resources.

The results for the George Site open pit and underground Mineral Resource estimates are shown in Table 14.24 to Table 14.29 at a range of cut-offs, with the selected cut-off shown in bold.

Table 14.24: LCP North and LCP South Open Pit Mineral Resource Estimates

Resource Classification	Cut-off Grade (g/t Au)	LCP North			LCP South		
		Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)	Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)
Indicated	0.00	507	5.92	96	1,122	5.29	191
	0.50	507	5.92	96	1,060	5.59	190
	0.75	491	6.08	96	1,039	5.69	190
	1.00	489	6.11	96	1,013	5.81	189
	1.25	472	6.29	95	992	5.91	188
	1.50	455	6.48	95	963	6.05	187
	1.75	448	6.55	94	934	6.18	186
	2.00	440	6.63	94	906	6.32	184
Inferred	0.00	27	4.51	4	9	5.54	2
	0.50	25	4.93	4	9	5.58	2
	0.75	20	6.04	4	9	5.58	2
	1.00	20	6.04	4	9	5.58	2
	1.25	20	6.04	4	9	5.58	2
	1.50	18	6.31	4	9	5.59	2
	1.75	17	6.62	4	9	5.59	2
	2.00	16	6.97	4	9	5.59	2

Notes: CIM definitions were used for the Mineral Resources.
Open pit Mineral Resources were constrained by an optimized pit shell at a gold price of US$1,500/oz gold.
Drilling results up to December 31, 2013 are included.
Source: AMC Mining Consultants (Canada) Ltd.2015

Table 14.25: Locale 1 and Locale 2 Open Pit Mineral Resource Estimates

Resource Classification	Cut-off Grade (g/t Au)	Locale 1			Locale 2		
		Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)	Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)
Indicated	0.00	1,798	4.87	282	754	3.98	97
	0.50	1,798	4.87	282	753	3.98	97
	0.75	1,796	4.87	281	753	3.98	97
	1.00	1,773	4.93	281	752	3.99	97
	1.25	1,731	5.02	279	744	4.02	96
	1.50	1,693	5.10	278	731	4.06	96
	1.75	1,625	5.25	274	724	4.09	95
	2.00	1,505	5.51	267	717	4.11	95
Inferred	0.00	281	2.48	22	51	2.08	3
	0.50	281	2.48	22	51	2.08	3
	0.75	261	2.61	22	51	2.08	3
	1.00	242	2.75	21	50	2.12	3
	1.25	188	3.22	19	47	2.19	3
	1.50	159	3.56	18	44	2.24	3
	1.75	151	3.66	18	37	2.36	3
	2.00	135	3.88	17	29	2.50	2

Notes: CIM definitions were used for the Mineral Resources.
Open pit Mineral Resources were constrained by an optimized pit shell at a gold price of US$1,500/oz gold.
Drilling results to 31 December 2013 and re-sampling results to 21 July 2014.
Source: AMC Mining Consultants (Canada) Ltd. 2015

Table 14.26: GH and Slave Open Pit Mineral Resource Estimates

Resource Classification	Cut-off Grade (g/t Au)	GH			Slave		
		Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)	Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)
Indicated	0	150	2.41	12	202	4.09	27
	0.5	148	2.44	12	191	4.32	26
	0.75	131	2.68	11	170	4.77	26
	1	124	2.77	11	170	4.77	26
	1.25	119	2.84	11	165	4.88	26
	1.5	109	2.98	10	165	4.88	26
	1.75	102	3.07	10	165	4.88	26
	2	101	3.09	10	165	4.88	26
Inferred	0	529	5.65	96	150	3.55	17
	0.5	525	5.69	96	146	3.65	17
	0.75	507	5.87	96	119	4.35	17
	1	495	5.99	95	114	4.51	16
	1.25	460	6.36	94	109	4.65	16
	1.5	453	6.44	94	109	4.65	16
	1.75	426	6.74	92	109	4.65	16
	2	420	6.81	92	105	4.75	16

Notes: CIM definitions were used for the Mineral Resources.
Mineral Resources were estimated within mineral domains expanded to a minimum width of 2 m.
Drilling results up to December 31, 2013 are included.
Source: AMC Mining Consultants (Canada) Ltd. 2015

Table 14.27: LCP North and LCP South Underground Mineral Resource Estimates

Resource Classification	Cut-off Grade (g/t Au)	LCP North			LCP South		
		Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)	Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)
Indicated	2.0	350	5.32	60	246	6.55	52
	3.0	256	6.39	52	204	7.38	48
	4.0	203	7.13	47	173	8.08	45
	4.5	179	7.52	43	162	8.35	43
	5.0	157	7.89	40	145	8.75	41
	5.5	137	8.29	37	134	9.04	39
	6.0	126	8.52	35	124	9.29	37
	7.0	98	9.10	29	99	9.96	32
Inferred	2.0	205	3.95	26	190	6.34	39
	3.0	121	4.99	19	141	7.65	35
	4.0	87	5.51	15	124	8.23	33
	4.5	68	5.83	13	114	8.60	31
	5.0	47	6.32	10	97	9.27	29
	5.5	35	6.69	8	88	9.67	27
	6.0	25	7.06	6	80	10.09	26
	7.0	10	7.93	3	65	10.85	23

Notes: CIM definitions were used for the Mineral Resources.
Mineral Resources were estimated within mineral domains expanded to a minimum horizontal width of 2 m.
Drilling results up to December 31, 2013 are included.
Source: AMC Mining Consultants (Canada) Ltd. 2015

Table 14.28: Locale 1 and Locale 2 Underground Mineral Resource Estimates

Resource Classification	Cut-off Grade (g/t Au)	Locale 1			Locale 2		
		Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)	Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)
Indicated	2.0	1,011	4.67	152	2,225	4.71	338
	3.0	764	5.39	132	1,580	5.61	286
	4.0	543	6.20	108	1,073	6.63	229
	4.5	500	6.38	103	902	7.09	206
	5.0	460	6.53	97	750	7.56	183
	5.5	410	6.68	88	596	8.14	157
	6.0	295	7.06	67	537	8.41	145
	7.0	145	7.66	36	350	9.48	107
Inferred	2.0	1,771	5.88	335	6,167	4.01	796
	3.0	1,450	6.64	310	3,426	5.25	579
	4.0	1,272	7.09	290	1,920	6.67	413
	4.5	1,078	7.60	263	1,596	7.18	369
	5.0	914	8.11	238	1,326	7.67	328
	5.5	773	8.64	215	1,105	8.15	290
	6.0	685	9.01	198	937	8.58	259
	7.0	433	10.51	146	631	9.62	196

Notes: CIM definitions were used for the Mineral Resources.
Mineral Resources were estimated within mineral domains expanded to a minimum horizontal width of 2 m.
Drilling results up to 31 December 2013 and re-sampling results to 21 July 2014.
Source: AMC Mining Consultants (Canada) Ltd. 2015

Table 14.29: GH and Slave Underground Mineral Resource Estimate

Resource Classification	Cut-off Grade (g/t Au)	GH			Slave		
		Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)	Tonnes (kt)	Grade (g/t Au)	Metal (koz Au)
Indicated	2.0	98	3.55	11	131	3.77	16
	3.0	46	4.92	7	99	4.14	13
	4.0	31	5.65	6	56	4.80	9
	4.5	21	6.23	4	41	5.06	7
	5.0	20	6.34	4	36	5.11	6
	5.5	20	6.34	4	N/A	N/A	N/A
	6.0	14	6.60	3	N/A	N/A	N/A
	7.0	0	7.76	0.1	N/A	N/A	N/A
Inferred	2.0	323	4.53	47	461	4.64	69
	3.0	218	5.58	39	396	4.97	63
	4.0	172	6.14	34	327	5.28	56
	4.5	155	6.35	32	254	5.59	46
	5.0	114	6.95	25	219	5.71	40
	5.5	107	7.08	24	157	5.89	30
	6.0	97	7.22	23	24	6.86	5
	7.0	18	9.71	6	10	7.14	2

Notes: CIM definitions were used for the Mineral Resources.
Mineral Resources were estimated within mineral domains expanded to a minimum horizontal width of 2 m.
Drilling results up to December 31, 2013 are included.
Source: AMC Mining Consultants (Canada) Ltd. 2015

14.4 Comparison with Previous Resource Estimates

The previous Mineral Resource estimates on the Property were published in the Tetra Tech 2014 Technical Report. Changes to the Mineral Resource estimates in this report are due predominately to:

- New drilling at the Echo deposit which converted Inferred Resources into Indicated Resources;
- New sampling of existing core at the Locale 1 and Locale 2 deposits and subsequent re-interpretation of the mineralization zones. This resulted in smaller pits at both deposits and hence more mineralization being accounted for in the George underground resource; and
- Recognition of additional cross-cutting (unmineralized) gabbroic dykes at the Goose Main deposit, thus decreasing Measured Resources.

Less significant changes to the Mineral Resource estimates resulted from:

- Development of a more detailed geology model resulting in additional geological units and a subsequent refinement of SGs;
- Two small (low-grade) mineralization domains assigned to the Locale 1 deposit in the February 2014 estimate were reassigned to the Locale 2 deposit in the October 2014 estimates. This had a small effect on the size of the Locale 2 pit;
- Smaller sub-blocks at Llama deposit were used to better define the updated geology; this impacted the volume of the model; and
- Inclusion of the gold assay results of geomechanical samples from drill holes for Locale 1 and Locale 2.

Table 14.30 shows a comparison between the estimates.

Table 14.30: Comparison of Mineral Resource Estimates

Deposit	AMC February 28, 2014			AMC October 21, 2014		
	Tonnes	Grade	Metal	Tonnes	Grade	Metal
	(kt)	(g/t Au)	(koz Au)	(kt)	(g/t Au)	(koz Au)
Measured						
Goose Main Open Pit	4,627	4.29	638	4,478	4.32	621
Goose Main Underground	107	6.17	21	110	6.24	22
Llama Open Pit	1,886	5.85	355	1,874	5.86	353
Llama Underground	128	6.1	25	110	5.72	20
Umwelt Open Pit	3,697	6.07	722	3,699	6.07	722
Umwelt Underground	1	9.21	0.3	0.9	9.21	0.3
Total Measured	10,446	5.24	1,761	10,273	5.27	1,740
Indicated						
Goose Main Open Pit	2,896	4.2	391	2,877	4.19	388
Goose Main Underground	863	7.44	206	853	7.32	201
Echo Open pit	230	6.87	51	321	6.07	63
Echo Underground	377	6.75	82	596	6.17	118
Llama Open Pit	864	5.94	165	821	6.01	159
Llama Underground	750	8.76	211	752	8.72	211
Umwelt Open Pit	1,974	5.38	341	1,963	5.38	340
Umwelt Underground	3,377	8.93	969	3,387	8.92	972
George Open Pit	4,891	4.79	753	4,321	5.04	700
George Underground	1,686	6.76	367	2,079	6.62	443
Total Indicated	17,907	6.14	3,536	17,969	6.22	3,593
Total Measured & Indicated	28,354	5.81	5,297	28,242	5.87	5,333
Inferred						
Goose Main Open Pit	217	3.19	22	215	3.2	22
Goose Main Underground	432	6.84	95	429	6.83	94
Echo Open pit	49	5.43	9	0	0	0
Echo Underground	502	7.37	119	71	5.91	14
Llama Open Pit	14	5.86	3	0	0	0
Llama Underground	294	6.65	63	295	6.77	64
Umwelt Open Pit	120	2.29	9	121	2.29	9
Umwelt Underground	1,784	11.59	665	1,788	11.59	667
George Open Pit	985	5.52	175	929	4.75	142
George Underground	3,782	6.32	769	3,902	6.69	840
Total Inferred	**8,179**	**7.33**	**1,927**	**7,750**	**7.43**	**1,851**

Notes: The numbers may not add due to rounding.
CIM definitions were used for the Mineral Resources.
Open pit Mineral Resources are constrained by an optimized pit shell at a gold price of US$1,500/oz gold with a 1.00 $US:$CD exchange rate.
The cut-off grade applied to the open pit resources is 1.0 g/t gold in both the February and October estimates. The underground cut-off grade is 4.0 g/t gold for George deposits and 3.5 g/t gold for all the Goose deposits except Umwelt, which is 4.5 g/t gold.
The George underground Mineral Resources (LCP North, LCP South, Locale 1, Locale 2, GH, and Slave) were estimated within mineral domains expanded to a minimum width of 2 m.
Drilling results up to December 31, 2013, are included except for the October Echo estimate (July 4, 2014) and the October Locale 1 and Locale 2 estimates (July 21, 2014).
Source: AMC Mining Consultants (Canada) Ltd. 2015

In the Measured category, there has been a decrease of 21,000 contained gold ounces. In the Indicated category, there has been an increase of 57,000 contained gold ounces. In the Inferred category, there has been a decrease of 76,000 contained gold ounces.

15 Mineral Reserve Estimates

15.1 Introduction

The Mineral Reserve documented in this section was estimated based on CIM guidelines that define Mineral Reserves as “the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.”

Mineral Reserves are those parts of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Qualified Person(s) making the estimates, is the basis of an economically viable Project after taking account of all relevant processing, metallurgical, economic, marketing, legal, environment, socio-economic and government factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the treatment plant or equivalent facility. The term ‘Mineral Reserve’ need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

To convert Mineral Resources to Mineral Reserves, estimates of gold price, mining dilution, process recovery, refining/transport costs, royalties, mining costs (both open pit and underground), processing, and general and administration costs were used to estimate cut-off grades for each deposit. Along with geotechnical parameters, the COGs formed the basis for the selection of economic mining blocks. For the Umwelt deposit, where a combination of open pit and underground mining methods was considered, crossover analyses was performed to determine the elevation at which the change from open pit to underground mining is optimal.

The QPs have not identified any known legal, political, environmental, or other risks that would materially affect the potential development of the Mineral Reserves, except for the risk of not being able to secure the necessary permits from the government for development and operation of the Project. The QPs are not aware of any unique characteristics of the Project that would prevent permitting.

A summary of the Mineral Reserves for the Project are shown in Table 15.1. The effective date for all of the Mineral Reserves contained in this report is September 14, 2015.

Table 15.1: Summary of Mineral Reserves

Site/ Reserve Category	Mining Method	Diluted Ore ('000s tonne)	Diluted Gold Grade (Au g/t)	Contained Gold ('000s ounce)
Goose Deposits				
Proven	Open Pit	6,983	5.97	1,340
Probable	Open Pit	1,885	5.52	335
Proven	Underground	20	9.52	6
Probable	Underground	3,471	7.37	822
Proven	Open Pit & Underground	7,003	5.98	1,346
Probable	Open Pit & Underground	5,356	6.72	1,157
Total Proven and Probable	**Open Pit & Underground**	**12,359**	**6.30**	**2,503**

Notes: A gold price of US$1,250/oz is assumed
An exchange rate of C$1.15 to US$1.00 is assumed
Mineral Reserves are based on Measured and Indicated Mineral Resources only
Marginal cut-off grade based on optimization design criteria
Source: JDS 2015

The Mineral Reserve estimations take into consideration on-site operating costs (e.g., mining, processing, site services, freight, general and administration), geotechnical analysis for both open pit wall angles and underground stope size, metallurgical recoveries, and selling costs. In addition, the Mineral Reserves incorporate allowances for mining recovery and dilution, and overall economic viability.

15.2 Open Pit Mineral Reserves

15.2.1 Open Pit Mineral Reserve Basis of Estimate

The Open pit reserves were calculated using Datamine NPVS™ (NPVS) software to optimize the Mineral Resource block model using COGs and mining factors to obtain economic mining shapes or shells. Only Measured and Indicated Mineral Resources were included in the optimization process. Inferred resources were considered as waste. A thorough analysis of the optimized shells was then conducted (including open pit/underground crossover analyses where appropriate) in order to select the shells to be used as guides to detailed design.

15.2.2 Mining Method and Mining Costs

The deposits at the Goose Site were tested for amenability to extraction by both open pit and underground methods. At this preliminary crossover analysis stage, mining costs of \$3.60/t mined for the open pit and \$55.00/t for the underground were assumed. The open pit and underground mining cost estimates were generated from first principles and by benchmarking comparable Canadian operations in similar northern locations. Open pit and underground crossover shells were developed for the deposits at Goose.

The open pit and underground crossover optimizations resulted in open pits at each of the three deposits at the Goose Site (Umwelt, Llama, and Goose Main) and provide the basis of estimation for the open pit Mineral Reserves.

15.2.3 Dilution

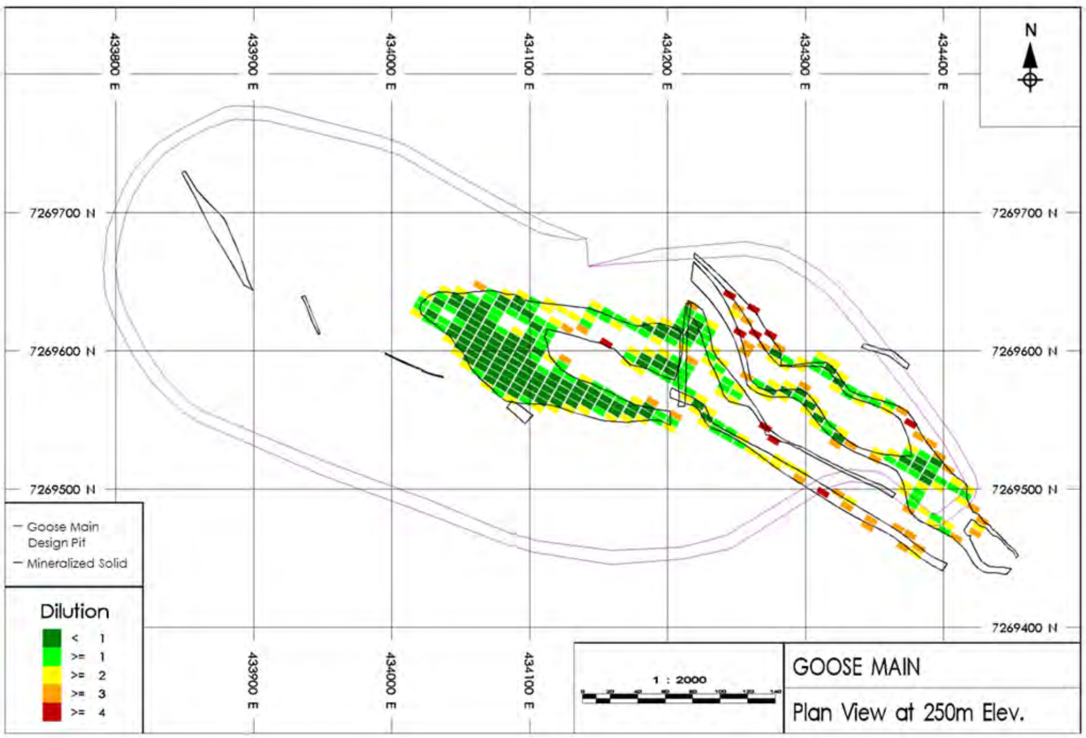
As input to the initial pit limit optimization and subsequent mine scheduling, and to reflect the selectivity of the mining method chosen when compared to the block model parameters, an external mining dilution was calculated and applied to the various deposits.

This external mining dilution was based on a calculation of the number of waste blocks adjacent to an ore block in the mineral inventory block model (using Hexagon Mining MineSight™ "four side contact routine"). Only blocks which were contained within a given zone (in this case a resource classification of Measured or Indicated), and above a given gold cut-off grade, were considered as ore blocks.

The waste block edges for each block were calculated on each horizontal plane in the model. This is shown in Figure 15.1 for a typical bench in Goose Main, where the number of waste edges can vary from zero (i.e., block is surrounded by all ore blocks) to four (i.e., block is surrounded by all waste blocks). Dilution was estimated using the number of waste edges for each block, an assumed grade of zero for all waste, and a dilution width of 0.3 m for each edge.

The results of the above analyses are summarized, by deposit, in Table 15.2. As a result of the analyses, external dilutions of 9%, 7%, and 10% were applied to the Llama, Umwelt, and Goose Main deposits, respectively.

Figure 15.1: Goose Main Plan View at 250 masl Elevation



Source: JDS 2015

Table 15.2: Open Pit Dilution Estimates by Deposit

# of Waste Edges	Llama				Umwelt				Goose Main			
	# of Blocks	Distribution (%)	Dilution Applied (%)	Contribution to Total External Dilution (%)	# of Blocks	Distribution (%)	Dilution Applied (%)	Contribution to Total External Dilution (%)	# of Blocks	Distribution (%)	Dilution Applied (%)	Contribution to Total External Dilution (%)
0	1,969	20	0	0	10,037	35	0	0	4,009	19	0	0
1	3,183	32	6	1.9	8,336	29	6	1.7	5,994	28	6	1.7
2	2,937	29	12	3.5	6,946	24	12	2.9	6,966	32	12	3.9
3	1,487	15	18	2.7	2,798	10	18	1.7	3,542	16	18	2.9
4	445	4	24	1.1	880	3	24	0.7	1,157	5	24	1.3
Total	**10,021**	**100**		**9.2**	**28,997**	**100**		**7.1**	**21,668**	**100**		**9.7**

Source: JDS 2015

15.2.4 Geotechnical Considerations

Knight Piésold carried out field investigations and analyses designed to characterize geotechnical and hydrogeological conditions required for feasibility-level open pit designs. The various pit slope design parameters, including geotechnical considerations, are discussed in detail in section 16.

Based on the location and characteristics of the geomechanical domains and the pit shells, design sectors were identified for each of the proposed pits. Slope stability analyses were undertaken on each sector to define achievable slope configurations. These analyses included kinematic and limit-equilibrium analyses. The results from these analyses provided guidance regarding achievable bench faces, and inter-ramp and overall slope angles for each design sector as shown in Table 15.5.

The results of the Knight Piésold analyses and a review of precedent practice suggest that the recommended geometries are reasonable and appropriate. To achieve these angles, the design assumes that controlled blasting and proactive geotechnical monitoring would be undertaken, along with an ongoing commitment to geomechanical data collection and analyses during future stages of design. Maintaining flexibility in the mine plan would be important to accommodate any slope stability issues.

15.2.5 Lerchs-Grossman Optimization

The sizes and shape of the ultimate open pits were obtained using the optimizing Lerchs- Grossman (LG) algorithm as implemented in Datamine NPVS software. Key inputs used for the LG runs are shown in Table 15.3.

It must be noted that the key open pit optimization input parameters may differ from the final parameters used in the project economic model. JDS has reviewed the differences and has concluded that the variances are immaterial to the pit designs.

Table 15.3: Optimization Parameters

Item	Unit	Optimization Assumption
Revenue, Smelting & Refining		
Gold Price	US$/oz	1,250
Payable metal	%Au	100
Refining/transport	US$/oz	7
Royalties % of NSR	US$/oz	45.99
Exchange rate	C$:US$	1.15
Net Return	C$/oz	1,377
	C$/g	44.26
OP OPEX estimates		
OP Mining Cost- Goose Pits	C$/t mined	3.6
Goose Pit strip ratio (estimated)	t:t	6.8
OP Mining Cost- Goose Pits	C$/t milled	28.08
Processing Cost - Goose Deposits	C$/t milled	33.72
G&A - Goose Deposits	C$/t milled	21.78
Surface Services - Goose Deposits	C$/t milled	9.03
Freight (excluding fuel)	C$/t milled	14.52
Total OPEX estimate (excluding mining) - Goose	C$/t milled	79.05
UG OPEX estimates		
UG Mining Cost - Umwelt	C$/t milled	55
Process and Mining Losses		
Process Recovery		
Umwelt	%	92
Goose Main	%	95
Llama	%	91.1
External Mining Dilution - OP		
Umwelt	%	7
Goose Main	%	10
Llama	%	9
Geotechnical/Processing Parameters		
Slope Angles (Overall)	°	40 to 50
Working Bench Height	m	5
Mill throughput	tpd	3,000

Source: JDS 2015

A separate series of pit optimization runs was completed for each deposit. Open pit and Underground crossover optimizations were undertaken where consideration to underground mining was taken into account in determining the final open pit shapes.

In all cases, the selected shell corresponding to the US$1,250/oz Au price (Revenue Factor 1) was selected (i.e. where maximum values are reached) as a guide for detailed mine design.

15.2.6 Cut-Off Grade and Resource Classification Criteria

Once pit shapes were established, marginal cut-off grades were used to determine the total amount and grade of ore in each pit. The marginal, or incremental, COG is specific to the mining method and is defined as the minimum grade at which mineralized material, already located at the pit rim (i.e., contained within the pit and already mined), pays for all additional costs incurred if it is sent for processing. According to this definition, the marginal COG corresponds to a break-even grade that excludes mining costs. The open pit Mineral Reserves comprise all mineralized material with grades equal to or above this marginal COG. Incremental COGs are shown in Table 15.4.

Table 15.4: Incremental COGs

Item	Unit	COG
Incremental Cut-off Grade - OP		
Umwelt Pit	g/t Au	2.08
Goose Main Pit	g/t Au	2.07
Llama Pit	g/t Au	2.14

Source: JDS 2015

15.2.7 Mine Design

Detailed pit design involves the conversion of the optimized pit shells into an operational open pit mine design, which is discussed further in section 16. Table 15.5 shows the main geometrical parameters used in the pit design.

Table 15.5: Pit Design Parameters

Description	Value
Ultimate Pit Design Parameters – All Pits	
Bench Height	5 m (single, working)
Face Angle	65° to 75° (double bench, final pit)
Berm Width	8.6 m to 10 m
Inter-ramp Angle (IRA)	48° to 55°
Ramp Width – Double Lane	20 m (total excavation)
Ramp Width (Single-lane -lower benches)	15 m
Ramp gradient	10 %
Overall Angle (OSA)	40° to 50

Source: JDS 2015

15.2.8 Open Pit Mineral Reserves Estimate Statement

The Back River open pit Mineral Reserves are shown in Table 15.6.

Table 15.6: Back River Open Pit Mineral Reserves Estimate

Deposit	Category	Total Ore	Gold Cut-off Grade	Gold Grade	Contained Gold
		('000's t)	(g/t)	(g/t)	('000's oz)
Goose Deposits					
Umwelt	Proven	2,348	2.08	6.68	504
	Probable	320		5.15	53
Llama	Proven	1,369	2.14	7.05	310
	Probable	380		7.52	92
Goose Main	Proven	3,266	2.07	5.27	526
	Probable	1,185		4.89	190
Total Open Pit Mineral Reserves	Proven	6,983		5.97	1,340
	Probable	1,885		5.52	335
	Total	8,868		5.87	1,675

Notes: A gold price of US$1,250/oz is assumed
An exchange rate of C$1.15 to US$1.00 is assumed
Mineral Reserves are based on Measured and Indicated Mineral Resources only
Marginal cut-off grade based on pit optimization design criteria
Source: JDS 2015

15.3 Underground Mineral Reserves

Based on open pit/underground crossover analysis, the Umwelt deposit has sufficient resources identified below the planned open pit to justify underground operations. For the other deposits at Back River, the underground resource base is either too small or does not align with project objectives. Deposit geometry, geotechnical and hydrogeological constraints, and production capacity were considered when selecting the mining method.

The Umwelt underground mine is planned to be separated from the open pit by a crown pillar. The crown pillar thickness of 60 m was provided by Knight Piésold based on geomechanical considerations (see Section 16.3.1.2). The Umwelt pit is planned to be used for tailings storage upon completion of open pit mining. Mineral Resources contained in the crown pillars are deemed non-mineable and are not included in the Mineral Reserves.

Umwelt underground is planned to be mined using post pillar cut-and-fill (PPCF) method. PPCF is a highly mechanized and systematic mining method. It is a version of room and pillar mining combined with drift and fill (DF). Structural load-bearing rock pillars are left behind to support the back of the stopes.

15.3.1 Cut-Off Grade and Stope Optimization Parameters

The underground cut-off grade for Umwelt was determined to be 3.86 g/t Au.

The reserve inventory contains some mineralized material in development drifts that is below COG, but it must be excavated to access stopes above COG. The marginal grade of this mineralized material pays for all non-mining costs incurred if it is sent for processing.

The COG and subsequent stope optimization analysis was estimated based on the assumptions shown in Tables 15.7 and 15.8. The assumptions used in the COG estimation may vary slightly from the final numbers in the project economic model but is not material to the reserve estimate.

Table 15.7: External Economic Inputs

Item	Unit	Economic Inputs
Revenue, Smelting & Refining		
Gold Price	US$/oz	1,250
Payable Metal	%Au	100
Refining/Transport	US$/oz	7
Royalties % of NSR	US$/oz	45.99
Exchange Rate	C$/US$	1.15
Net Return	C$/oz	1,377
	C$/g	44.26

Source: JDS 2015

15.3.1.1 Cost Estimates

Non-mining cost estimates for processing, general and administration, ore haulage, site services, and freight were based on a process plant feed rate of 3,000 t/d.

Underground ore is planned to be hauled with open pit equipment from the portal area to a central run of mine (ROM) ore stockpile. These surface ore haulage costs were estimated separately from the underground mining costs and included in the COG calculation.

Underground mining costs were established from first principles as well as relevant benchmarking with comparable Canadian underground operations in northern locations. Based on the deposit geometry and selected mining method, sustaining access development costs were estimated for the underground operation.

Estimated costs for calculating COGs are summarized in Table 15.8.

Table 15.8: Cost Estimates Applied to Underground Cut-off Grade Estimation

Item	Unit	Costs
Umwelt – PPCF mining cost	C$/t mined	54.88
Goose Deposits Processing Cost	C$/t milled	33.72
Goose Site Ore Haulage Costs	C$/t milled	0.8
G&A	C$/t milled	21.78
Surface Services	C$/t milled	9.03
Freight	C$/t milled	14.52
Sustaining Access Development Costs Umwelt (PPCF)	C$/t milled	12.24
Total	**C$/t milled**	**146.97**

Source: JDS 2015

15.3.1.2 Preliminary Dilution Estimate

Typical stope and pillar dimensions were determined in consultation with Knight Piésold for the Umwelt deposit; these were based on average widths of the ore zones, mining method and geotechnical conditions. External mine dilution was then determined by using estimated over break values. For Umwelt, an average dilution factor of 7% was estimated for input into the COG calculation.

15.3.1.3 Metallurgical Recovery Estimate

Metallurgical bench-scale test results for Umwelt deposit were used for the gold recovery of 91.7% used in the COG calculation.

15.3.2 Stope Optimization

Maptek's Vulcan grade shells were used to produce economic stope shells by applying COGs and deposit-specific stope dimensions to Measured and Indicated resource blocks. The stope shells were reviewed by mine planners to include development expenditures and verify the economic viability of each stope. At the same time, stope shells located within permanent pillars (crown pillar or sill pillar) were removed from the stope inventory.

Stope optimization at Umwelt started at the deposit level to determine the best location of sill pillars in the mine. Two sill pillars were designed to split the Umwelt deposit into three mining blocks (Zones A, B, and C), allowing increased production rates and optimized ore production grades from three independent mining horizons. Based on geotechnical analysis, Knight Piésold recommended sill pillars with a minimum 15 m vertical thickness.

The block model was analysed by comparing the tonnes, grades, and contained metal on 5 m cuts. The three cuts containing the least amount of metal at the lowest grades between Zones A and B were aggregated together to become a sill pillar. The process was then repeated to identify the second sill pillar. Due to an extended area of low-grade mineralization between Zones B and C, a 20 m sill pillar was left in this location.

For Umwelt, a crown pillar with a vertical thickness of 60 m was recommended by Knight Piésold based solely on geomechanical considerations. The final open pit design shell was offset 60 m vertically downwards and any stope shells located within this crown pillar were removed from the inventory.

Pillar locations for the PPCF method were planned on a regular “checkerboard pattern,” but the optimum location of the pillars was determined to minimize metal loss in pillars. Pillars with a footprint of 5 m by 5 m extending from the footwall to the hanging wall were designed in Vulcan.

The pillar dimensions were selected based on analyses by Knight Piésold. Pillars were truncated at the sill pillars and crown pillar. Nine different checkerboard patterns were evaluated by calculating the contained metal lost in pillars for each pattern.

The pillar pattern with the least contained metal was selected for the mine design. The process was performed for each of the three mining blocks.

Pillar locations were not consistent between the different mining blocks, since the sill pillars ensure vertical separation.

In the final step, Vulcan grade shells were used for each 5 m cut to create economically mineable stope shells, while honouring geotechnical constraints. The hanging wall near the Middle Mudstone contact was identified by Knight Piésold as an area of concern for increased dilution. Therefore, stopes with more than 20% mudstone content were removed from the stope reserve inventory.

15.3.3 Reserve Estimation

External dilution consists of wall dilution and floor dilution. Floor dilution, from mucking some backfill with the ore, carries zero grade and was estimated to be 0.2 m. Wall dilution for PPCF, due to overbreak or hanging wall sloughage, was estimated to be 0.25 m for each exposed wall. Wall dilution shells were created by expanding the stope shells near the expected overbreak and calculating incremental dilution tonnes and grades based on the block model.

The average calculated external dilution for Umwelt is 10% based on wall and floor dilution. This is slightly higher that the estimated dilution used for the COG calculation, but based on more refined stope and pillar shapes.

During the mining process, incremental ore losses are expected to occur due to under-break in the blasting process and mucking losses to the floor. To account for these losses a 95% mining recovery factor was applied for all the reserve calculations.

Mining dimensions for PPCF at Umwelt were defined using 5 m by 5 m pillars with a 10 m span between pillars. This mining geometry results in a resource tonnage recovery of 89%. Stopes were designed on a cut-by-cut basis, by tracing the grade shells around the designed post pillars.

15.3.4 Underground Mineral Reserves Statement

The Back River Underground Mineral Reserves were based on the assumptions, optimization results, and detailed planning explained in the preceding sections. The Mineral Reserves estimate is shown in Table 15.9.

Table 15.9: Back River Underground Mineral Reserves Estimate

Deposit	Category	Total Ore	Gold Cut-off Grade	Gold Grade	Contained Gold
		('000's t)	(g/t)	(g/t)	('000's oz)
Goose Deposits					
Umwelt	Proven	20	3.86	9.52	6
	Probable	3,471		7.37	822
Total Underground Mineral Reserves	Proven	20		9.52	6
	Probable	3,471		7.37	822
	Total	3,491		7.38	829

Notes: A gold price of US$1,250/troy ounce is assumed.
An exchange rate of C$1.15 to US$1.00 is assumed.
Mineral Reserves are based on Measured and Indicated Mineral Resources only.
Marginal cut-off grade based on underground design criteria.
Source: JDS 2015

16 Mining Methods

16.1 Introduction

The Back River Gold Project comprises multiple deposits at the Goose and George sites. Although Mineral Resources for both the Goose and George sites are reported, only the Goose Site resources are considered for mining in this Initial Project Feasibility Study. The Umwelt deposit is planned to be extracted using both open pit and underground mining. The Llama and Goose Main deposits are planned to be extracted using only open pit mining. The proposed underground mining methods is post pillar cut-and-fill (PPCF) and the proposed open pit mining method is shovel and truck.

Open pit/underground crossover analysis was performed to determine the optimal split between open pit and underground resources. The analysis and subsequent detailed mine design estimated 8.9 Mt and 3.5 Mt of ore will be mined by open pit and underground respectively. The average underground ore gold mill head grade is estimated to be 7.38 g/t, and the total open pit ore gold mill head grade is estimated to be 5.9 g/t. The proposed open pit and underground mines deliver 1.7 Moz and 0.8 Moz respectively for a total of 2.5 Moz. The mine plan and reserves do not include the crown pillar that separates the open pit and underground mining areas at Umwelt.

Table 16.1 LOM Tonnes and Grade by Mining Method

Mining Method	Deposits	Ore tonnes	Diluted Gold Grade	Contained Gold
		(M tonnes)	(Au g/t)	(M ounce)
Underground	Umwelt	3.5	7.4	0.8
Open Pit	Umwelt + Llama + Goose Main	8.9	5.9	1.7
Total		**12.4**	**6.3**	**2.5**

Source: JDS 2015

Industry-standard mining methods, equipment, dilution calculations, and production rates were used throughout the planning process.

16.2 Open Pit Mining

16.2.1 Open Pit Planning

Industry-standard methodologies were adopted for pit limit analysis, mining sequence, cut-off grade optimization design, and detailed design.

The following main steps were part of the planning process:

- Assignment of economic criteria to the geological resource model;
- Definition of optimization parameters, such as gold price, preliminary operating cost estimates, pit wall angles, preliminary dilution and metallurgical recovery estimates;
- Calculation of economic ultimate pit limits for the various deposits using the Datamine NPV Scheduler™ (NPVS) software. (This software applies the Lerchs-Grossman algorithm to define optimal mining shells. A series of nested envelopes was produced within a range of economic conditions, including crossover analysis for underground mining);
- Development of the economic scheduling sequence using the NPVS series of optimum nested pits as guides;
- Development of the operational designs for the ultimate pits using Geovia Surpac (Surpac) and Mintec MineSight™ (MineSight) software;
- Development of operational designs for mining phases using the Datamine NPV Scheduler™ (NPVS) software;
- Determination of incremental (or mill) cut-off grade based on economic parameters;
- Determination of external mining dilution based on mineral inventory block model;
- Creation of LOM production schedule to maximize economic return, while satisfying the plant feed and mine production constraints and considering the underground mine production;
- Development of waste rock storage area (WRSA) designs and volume estimates;
- Calculation of hauling distances, per bench and phase, according to the LOM plan for each of the deposits and the defined haulage network; and
- Estimation of equipment fleet requirements from the LOM production schedule, and performance and operational characteristics of the proposed primary open pit fleet using Runge Limited's TALPAC® software. Creation of a spreadsheet model to estimate operating hours and number of units required. Application of this model to calculate equipment procurement schedules, workforce requirements, capital expenditures and operating costs.

16.2.1.1 Topographic and Resource Model Description

Mine topography, including the WRSAs and Tailings Storage Facility (TSF) areas, was provided digitally by Sabina in UTM WGS-84, Zone 13 W coordinates. Topography was supplied as 1 m-level LIDAR contour data and was used for all pit design calculations and engineering estimates. Volumetric estimates were calculated using design surfaces that intersected these topographic contours.

AMC Mining Consultants (Canada) Ltd. (AMC) prepared the 3D resource block models for the various deposits used in this study; these models are explained in detail in section 14 Mineral Resource Estimate. The models contain lithology, in-situ density, ore and waste types, resource classification, ore and waste percentage, and gold grades (g/t Au).

16.2.2 Open Pit Optimization and Sensitivity Analysis

16.2.2.1 Objective and Scope

OP optimization was undertaken to generate a series of nested pit shell surfaces to guide the design of the open pits. The Lerchs-Grossman algorithm in the NPVS software package was used for the optimization work and accompanying sensitivity work. The resulting nested pit shells were generated by varying the revenue factor applied to the base case values. Crossover optimization runs were also performed on the Umwelt deposit, to estimate the optimum open pit/underground mining interface.

16.2.2.2 Material Characteristics and Plant Constraints

In-situ density values, used to report reserves and develop the LOM plan, were defined and incorporated into the various block models. The bulk density values assigned to the resource blocks were either estimated or derived from an average of all core sample data in mineralized and unmineralized areas.

A 30% swell factor was assumed for all materials in the loading and hauling calculations as well as for WRSA designs. An average plant capacity of 3,000 dry metric tonnes per day for 365 days per year was selected, after allowing for equipment availability.

16.2.2.3 Optimization Parameters

The optimization parameters used are shown in Table 16.2 and were based on preliminary estimates and differ slightly from final estimates used in the economic model, however, overall differences between the preliminary and final numbers are considered insignificant.

Table 16.2: Optimization Parameters

Item	Unit	Optimization Assumption
Revenue, smelting & refining		
Gold Price	US$/oz	1,250
Payable metal	%Au	100
Refining/transport	US$/oz	7.00
Royalties % of NSR	US$/oz	45.99
Exchange rate	C$:US$	1.15
Net Return	C$/oz	1,377
	C$/g	44.26
OP OPEX estimates		
OP Mining Cost- Goose Pits	C$/t mined	3.60
Goose Pit strip ratio (estimated)	t:t	6.8
OP Mining Cost- Goose Pits	C$/t milled	28.08
Processing Cost - Goose Deposits	C$/t milled	33.72
G&A - Goose Deposits	C$/t milled	21.78
Surface Services - Goose Deposits	C$/t milled	9.03
Freight (excluding fuel)	C$/t milled	14.52
Total OPEX estimate (excluding mining) - Goose	C$/t milled	79.05
UG OPEX estimates		
UG Mining Cost - Umwelt	C$/t milled	55.00
Process and Mining Losses		
Process Recovery		
Umwelt	%	92.0
Goose Main	%	95.0
Llama	%	91.1
External Mining Dilution - OP		
Umwelt	%	7.0
Goose Main	%	10.0
Llama	%	9.0
Geotechnical/Processing Parameters		
Slope Angles (Overall)	°	37 to 55
Working Bench Height	m	5
Mill throughput	tpd	3,000

Source: JDS 2015

16.2.2.4 Open Pit Optimization Results

It was decided that both underground and open pit mining methods appear viable at Umwelt and suitable for the Project, (i.e., where, given the depth of the resources and the Project economic assumptions, a portion of the resources are mined more economically using underground methods). Crossover pit optimizations were conducted where the estimated additional cost of underground mining and associated recoveries were included in the optimization input parameters to determine the open pit/underground interface.

The results of the pit optimization analysis for various deposits in the Project are summarized in Table 16.2.

Table 16.2: Pit Optimization Results*

Deposit	Pit Optimization Results					
	Mineralized material	Diluted Gold Grade	Contained Gold	Waste	Total Material	Strip Ratio
	('000s tonne)	(Au g/t)	('000s ounce)	('000s tonne)	('000s tonne)	(t:t)
Goose Deposits						
Umwelt	2,919	6.55	614	13,862	16,781	4.7
Llama	1,758	7.26	410	27,737	29,494	15.8
Goose Main	4,428	5.10	726	47,173	51,601	10.7
Total All deposits	**9,104**	**5.98**	**1,751**	**88,773**	**97,877**	**9.8**

*Pit optimization results are based on shells and ARE NOT RESERVES.
Source: JDS 2015

16.2.3 Open Pit Design Criteria

16.2.3.1 Geotechnical and Hydrogeological Characterization

Knight Piésold's (KP) recommendations for slope angles, based on its analyses of geomechanical and hydrogeological conditions, are described in section 15.2.4 and were used in the open pit mine planning.

16.2.3.2 General Design Parameters

The general open pit design parameters used in detailed design are as follows:

- Pit Walls
  - Bench height, single bench mining, 5 m;
  - Height between catch benches, 20 m;
  - Bench face angle, 55° to 75° (variable as per KP geomechanical guidance); and
  - Berm width, 8.6 to 10 m (variable).
- Haul Roads (in and out of pit)
  - Total road width allowance, 20 m;
  - Running surface on final two-way roads, 15 m;
  - Berms, 3.8 m wide (40° slope);
  - Ditch, 1.0 m wide;
  - Ramp grades, 10% standard and 12% for pit bottom access; and
  - Single-lane road allowance, 15 m.
- Operations Road (general light vehicle traffic and occasional heavy equipment)
  - Width, 8 to 10 m; and
  - Maximum gradient, 8%.
- Mining
  - Minimum pushback operating width, 50 to 60 m;
  - Minimum pit bottom width, 25 to 30 m; and
  - Pit bottom sub-out depth, 5 m.

Primary haulage roads are required between the various open pit deposits and the primary ore crusher, WRSA, construction areas and maintenance facilities. Roads are planned to be, to the extent possible, constructed using cut-and-fill techniques to achieve design alignment and grade. Roads within the ultimate WRSAs are designed to be all-fill construction. Roads are proposed to be constructed of NPAG material generated from the open pits. Dust on the roads is planned to be controlled using water trucks or, possibly, chemical suppressants as needed.

16.2.3.3 Open Pit Designs

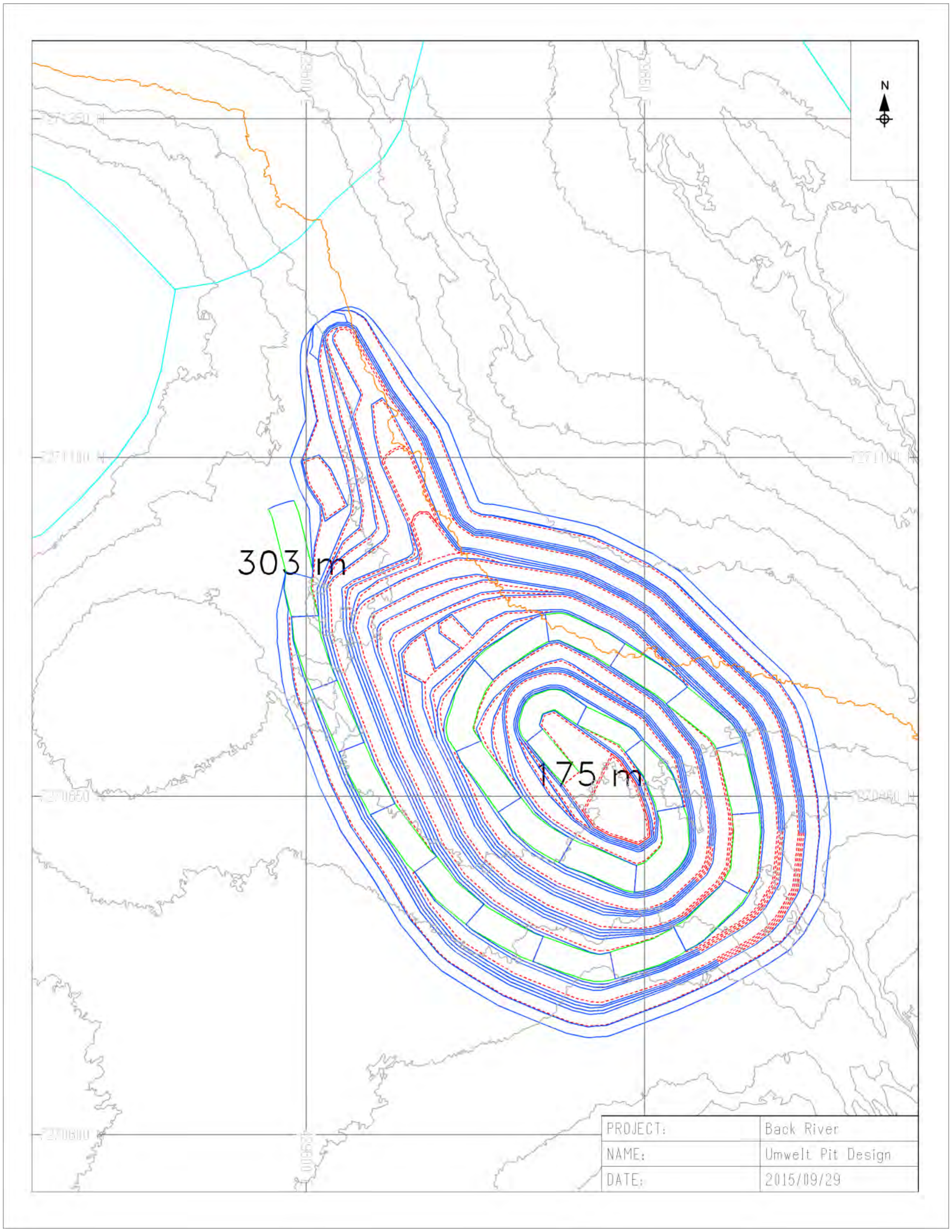
Detailed mine designs were undertaken on all three proposed open pits using the NPVS shells as guidance. The pit shape dimensions are summarized in Table 16.3. Plan views of each Open pit design are shown in Figures 16.1 to 16.3. Note that for the Goose Main deposit, an initial phase was designed in addition to the ultimate pit.

Table 16.3: Open Pit Dimensions

Open Pit	Length (m)	Width (m)	Depth (m)
Umwelt	600	350	135
Llama	550	420	200
Goose Main	850	400	180

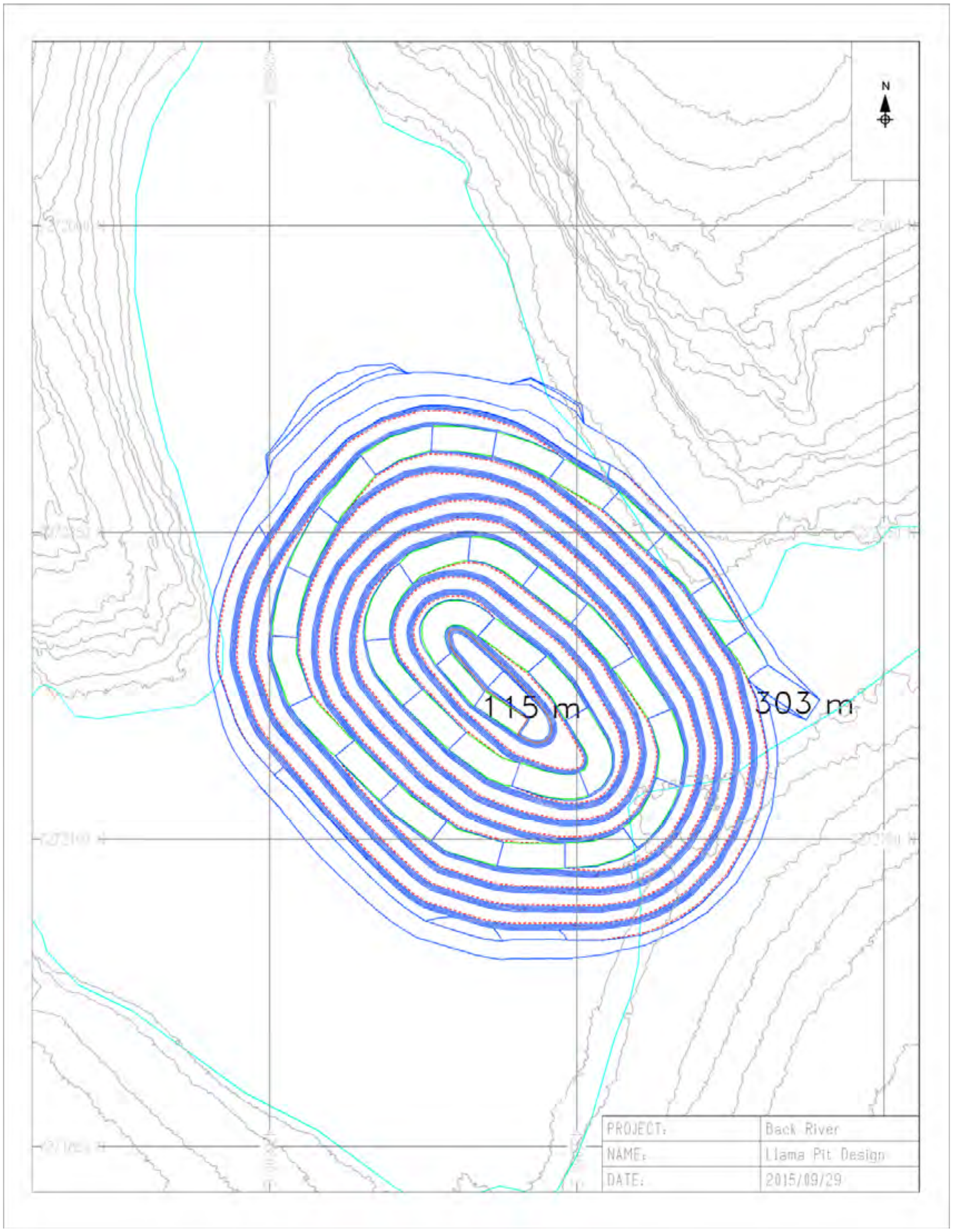
Source: JDS 2015

Figure 16.1: Umwelt Pit Design



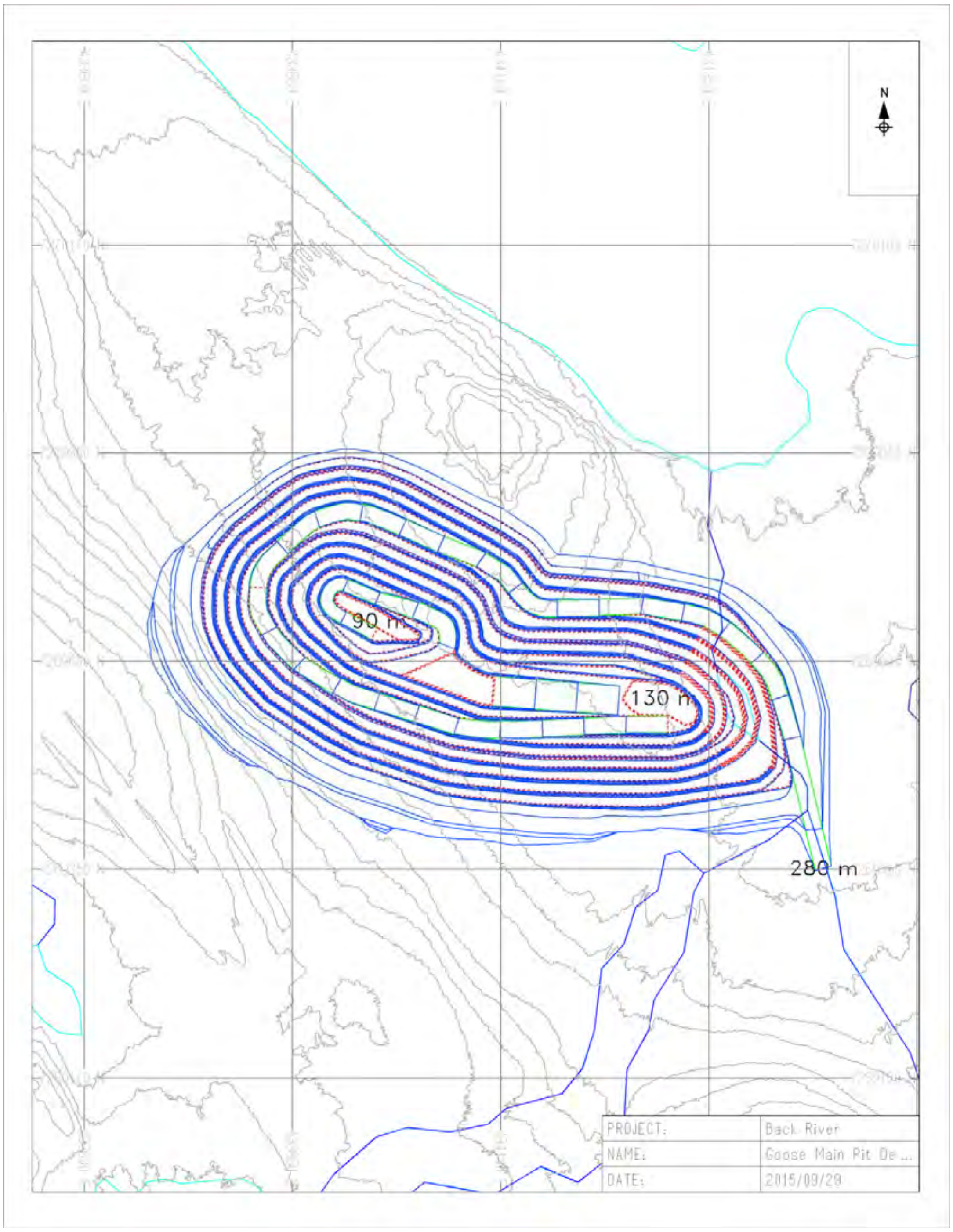
Source: JDS 2015

Figure 16.2: Llama Pit Design



Source: JDS 2015

Figure 16.3: Goose Main Pit Design



Source: JDS 2015

The detailed pit designs, combined with calculated cut-off grades, determine the Mineral Reserve estimate for each deposit as summarized in Table 16.4 and Figure 16.9. These values were used in the LOM schedule and economic model.

Table 16.4: Back River Project Open Pit Mineral Reserve Estimate

Deposit	Category	Total Ore ('000's t)	Gold Cut-off Grade (g/t)	Gold Grade (g/t)	Contained Gold ('000's oz)
Goose Site					
Umwelt	Proven	2,348	2.08	6.68	504
	Probable	320		5.15	53
Llama	Proven	1,369	2.14	7.05	310
	Probable	380		7.52	92
Goose Main	Proven	3,266	2.07	5.27	526
	Probable	1,185		4.89	190
Total Open Pit Mineral Reserves	**Proven**	**6,983**		**5.97**	**1,340**
	Probable	**1,885**		**5.52**	**335**
	Total	**8,868**		**5.87**	**1,675**

Notes: A gold price of US$1,250 ounce is assumed

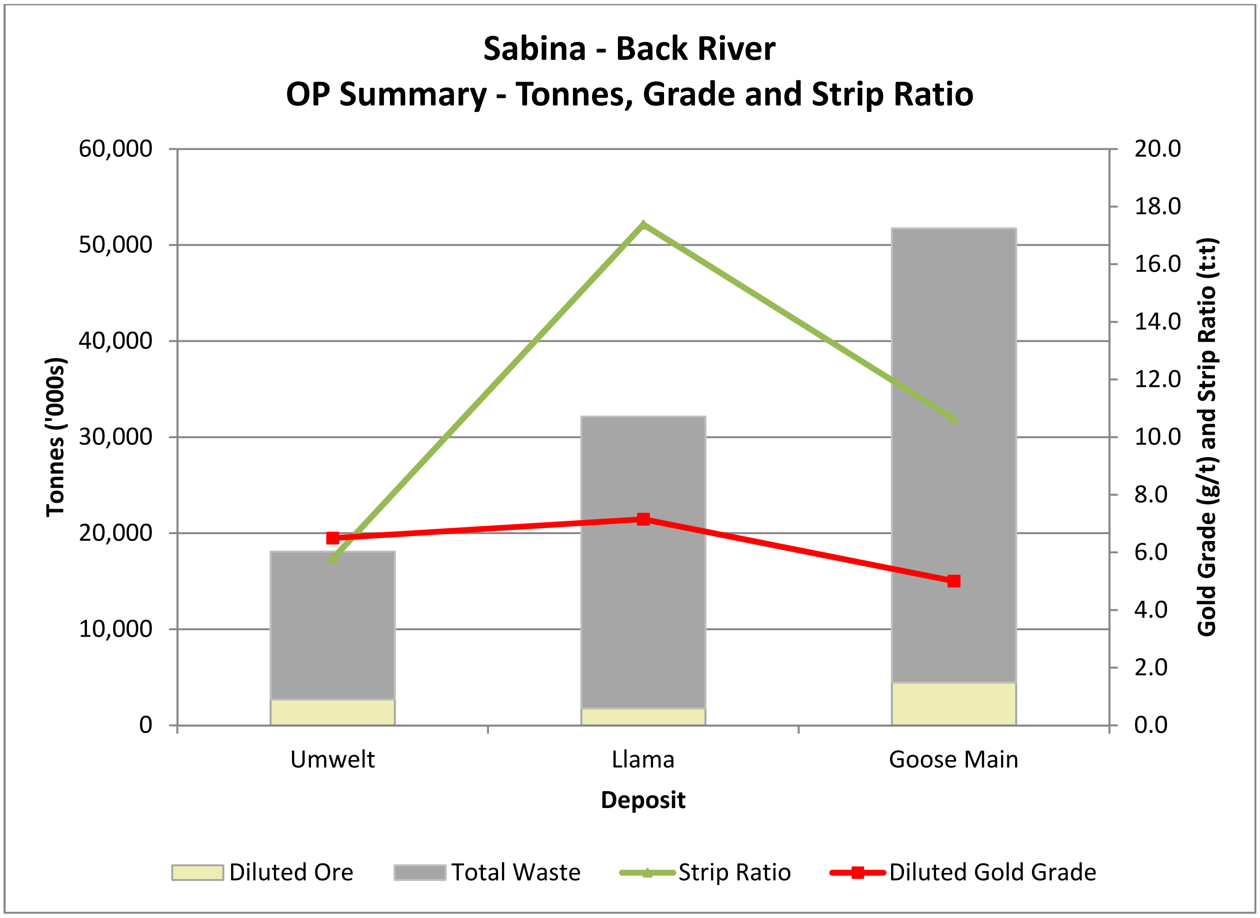
An exchange rate of C$1.15 to US$1.00 is assumed

Mineral Reserves are based on Measured and Indicated Mineral Resources only

Marginal cut-off grade based on pit optimization design criteria

Source: JDS 2015

Figure 16.4: Open Pit Reserve Summary



Source: JDS 2015

16.2.4 Open Pit Mine Production Schedule

16.2.4.1 Summary

The Project open pit mines are designed to produce a total of 8.4 Mt of ore and 86.2 Mt of waste rock over an 8-year open pit mine operating life (excluding the initial construction/earthworks and pre-production period which will produce 0.5 Mt of ore and 6.9 Mt of waste, respectively), yielding an overall open pit strip ratio of 10.3:1 (t:t).

The open pit and underground mine production schedule includes only the deposits at Goose (Umwelt, Llama and Goose Main). Due to the limited production capacity of the Umwelt underground mine, specifically its inability to produce the full 3,000 t/d mill feed, the open pit mines will supplement ore feed. The mine schedule was developed using the following basic criteria:

- Maximize Project economics;
- Maintain plant throughput at a net yearly production rate of 1.1 Mt/a (3,000 t/d) of ore;
- Ensure suitable and adequate quantities of waste material is produced from the open pits for construction during the pre-production period;
- Use ROM stockpiles and accelerated open pit mining methods to maximize the mill head grade during the early years of the operation;
- Ensure at least two open pits are actively producing at any given time (deposits are most economical when the open pit mines and the underground workings are mined concurrently);
- Maximize the pit production rate per period according to the geometry of the phases and the number of shovels that can work within that geometry. Note: The maximum yearly open pit production is 13.7 Mt (LOM average is 6.0 Mt/a);
- Capitalize pre-stripping tonnage (7.3 Mt of total material to be mined in initial construction/ earthworks and pre-production period using Owner-operated equipment and resources, including 1.5 Mt of waste material used for construction purposes);
- Establish run of mine (ROM) stockpiles with different gold cut-off grades so that stockpiled material can be blended to maximize mill head grades; and
- Operate 355 days per year (allowance made for ten weather days per year).

16.2.4.2 Open Pit Phase Design

Given the relatively small footprints of the open pits at Llama and Umwelt, no additional pushbacks or phases were designed for the mine plan development. At the larger Goose Main open pit, the deposit will be mined in two phases.

16.2.4.3 Open Pit Mining Sequencing

Pit sequencing focuses on achieving the required plant feed production rate, mining higher grade ore and stockpiling lower grade mill feed (to maximize mill head grades) early in the mine life, while balancing grade and strip ratios, and taking into account underground production.

The open pit mining sequence begins with the Umwelt open pit, followed by Llama and Goose Main.

All process plant feed material would be hauled directly to the ore stockpiles near the crusher at the process plant site.

16.2.4.4 Pre-production Development Schedule

Pre-production covers the period prior to first commercial gold production. Open Pit mining activities during this period are scheduled to provide sufficient ore exposure for plant start-up and commissioning, which takes place in the first quarter of Year 1. Mining also focuses on providing sufficient waste rock for the construction of site roads, laydown areas, the TSF dam, etc. Ore mined during the pre-production period is planned to be stockpiled and re-handled to the mill during operations. Mining in the pre-production period would create substantial high grade stockpiles to maximize mill head grades in the early part of the production schedule.

A total of 6.9 Mt of waste and 0.5 Mt of ore are scheduled to be mined from the Umwelt open pit in this period using the mine production fleet. This includes approximately 1.5 Mt of waste for initial construction/earthworks that is not reflected in the mine schedule, but has been allocated to infrastructure capital costs.

16.2.4.5 Mine Plan and Open Pit Production Schedule

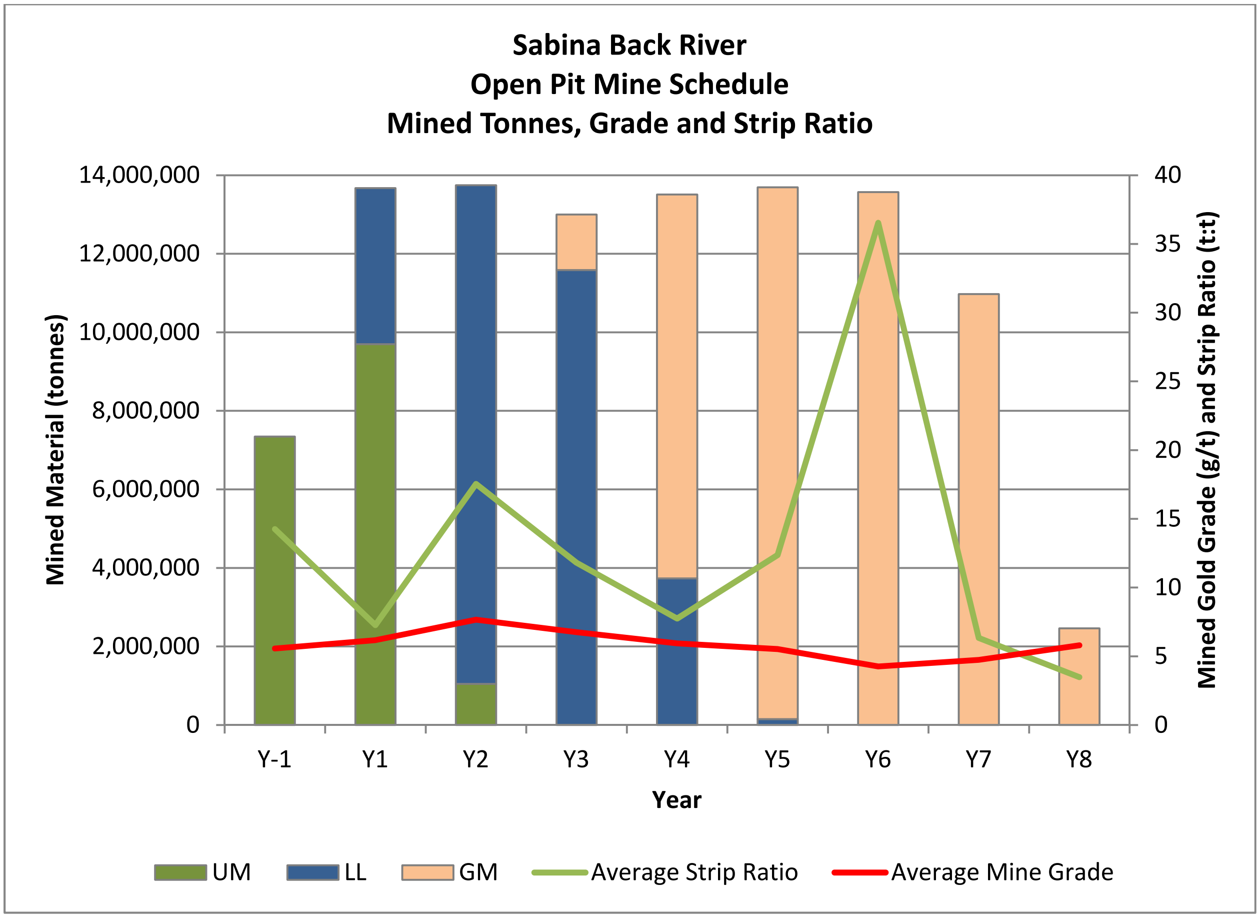
ROM stockpiles were designed to allow for ore blending at the mill and to maximize mill head grades in the early part of the Project. Three stockpiles are planned:

- Low grade (cut-off grade to 4 g/t Au);
- Medium grade (4 g/t to 6 g/t Au); and
- High-grade (greater than 6 g/t Au).

The open pit mining fleet was sized to maintain these ore stockpiles during production.

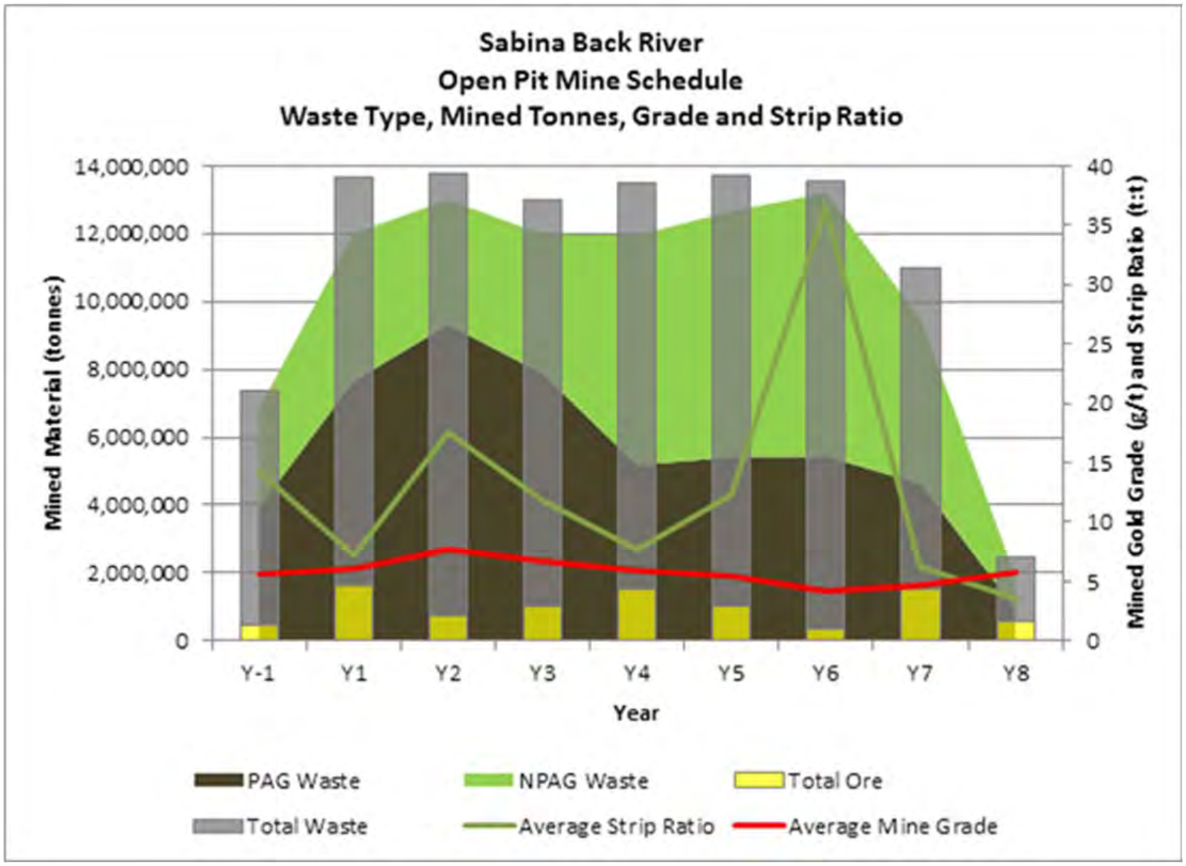
The Project deposits are most economic when the open pits are mined concurrently. Figure 16.5 and Figure 16.6 summarize ore/waste tonnages and grade by period. Figure 16.7 shows the quarterly bench advance rates from each open pit mine.

Figure 16.5: Total Ore and Waste Tonnages, Gold Grade and Strip Ratio



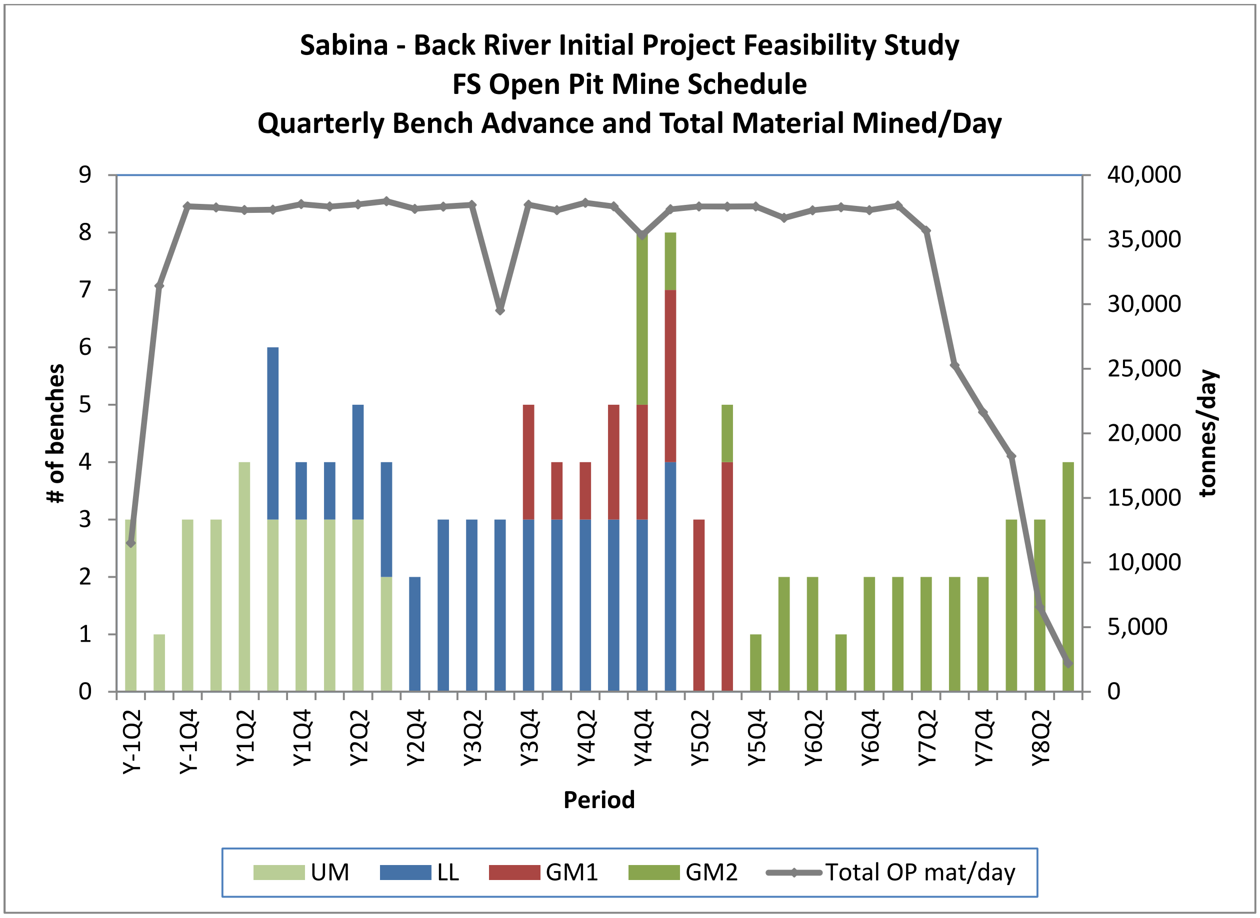
Source: JDS 2015

Figure 16.6: Ore and Waste Tonnages, Grade and Strip Ratio



Source: JDS 2015

Figure 16.7: Open Pit Bench Advance



Source: JDS 2015

16.2.4.6 Open Pit Development

Year -2: Earthworks/construction begins at the Goose Site. Appropriate (NPAG) waste rock is planned to be used to construct roads, laydown areas, and for site preparation. Approximately 1.5 Mt of waste is expected to be required (and produced) in this period.

Year -1: Pre-production continues with mining at the Umwelt open pit. Total ore to be stockpiled is 0.5 Mt, with 5.4 Mt of waste mined.

Year 1: Plant ramp-up commences in the first quarter and full-scale production is expected to be attained in the final quarter of this year. Open pit mining at Umwelt continues and mining at Llama commences. Approximately 1.7 Mt of ore is scheduled to be mined in this period. Mill head grades for the year are expected to average 9.5 g/t Au. Approximately 12.0 Mt of waste is expected to be produced at a strip ratio of 7.3:1.

Year 2: Umwelt open pit mining is scheduled to be completed in this period while mining in Llama open pit continues. Average mill head grade over the period are expected to be 7.4 g/t Au at a steady production rate of 3,000 t/d (1.1 Mt/a). A total of 0.7 Mt of ore and 13.0 Mt of waste is planned to be mined from the open pits.

Years 3-5: Goose Main open pit mining commences in Year 3 and mining in Llama open pit is completed early in Year 5 . Average mill head grade is expected to be 8.23 g/t Au. The waste produced over this 3-year period is planned to total 36.6 Mt; total ore is planned to be 3.6 Mt.

Years 6-8: Goose Main open pit is completed in Year 8. A total of 2.4 Mt of ore is planned to be mined over the 3-year period. Average plant head grade is estimated to be 6.9 g/t Au. Total waste tonnage produced from the open pits is estimated to be 24.6 Mt.

16.2.5 Waste Rock Management

Open pit waste rock was categorized into potentially acid generating (PAG) and non-potentially acid generating (NPAG). These categories were based on the geochemical characterization of the material and its acid generating potential. For mine planning and the purposes of the Project, it was assumed that the PAG and NPAG could be identified during mining and separated. The NPAG rock is planned to be used for road construction, laydown areas, base for ROM stockpiles, and as a final cover on the TSF and WRSAs.

Table 16.5 summarizes the open pit waste material by type, where overburden (OVB) is included in the NPAG totals.

Table 16.5: Open Pit Waste Rock Summary - Goose Property

Scenario		Quantity ('000s t)			
		PAG	NPAG	OVB	Total All Waste
75% of NPAG recovered except 0% in LIF and 50% in Umwelt/Llama UIF	Umwelt	10,105	5,319	1,177	16,601
	Llama	21,047	9,342	1,278	31,667
	Goose Main	19,271	28,015	4,019	51,305
	Total	**50,423**	**42,676**	**6,474**	99,573

Source: JDS 2015 (LIF – Lower Iron Formation; UIF – Upper Iron Formation)

16.2.5.1 Waste Rock Scheduling and Storage Area Design

Waste rock from the Project is planned to be deposited in engineered WRSAs adjacent to each of the pits. In addition, suitable waste rock is planned to be used for road construction, laydown areas, TSF dam construction, and capping of the TSF.

Each WRSA is planned to be constructed by placing material at its natural angle of repose (approximately 1.5H:1V) with 17 m wide safety berms spaced vertically every 10 m, resulting in final slopes of 3:1. A 30% swell factor is assumed.

Significant amounts of overburden are not expected during open pit mining. Segregation of the various waste material types, if deemed necessary, can be easily managed given the various WRSA designs.

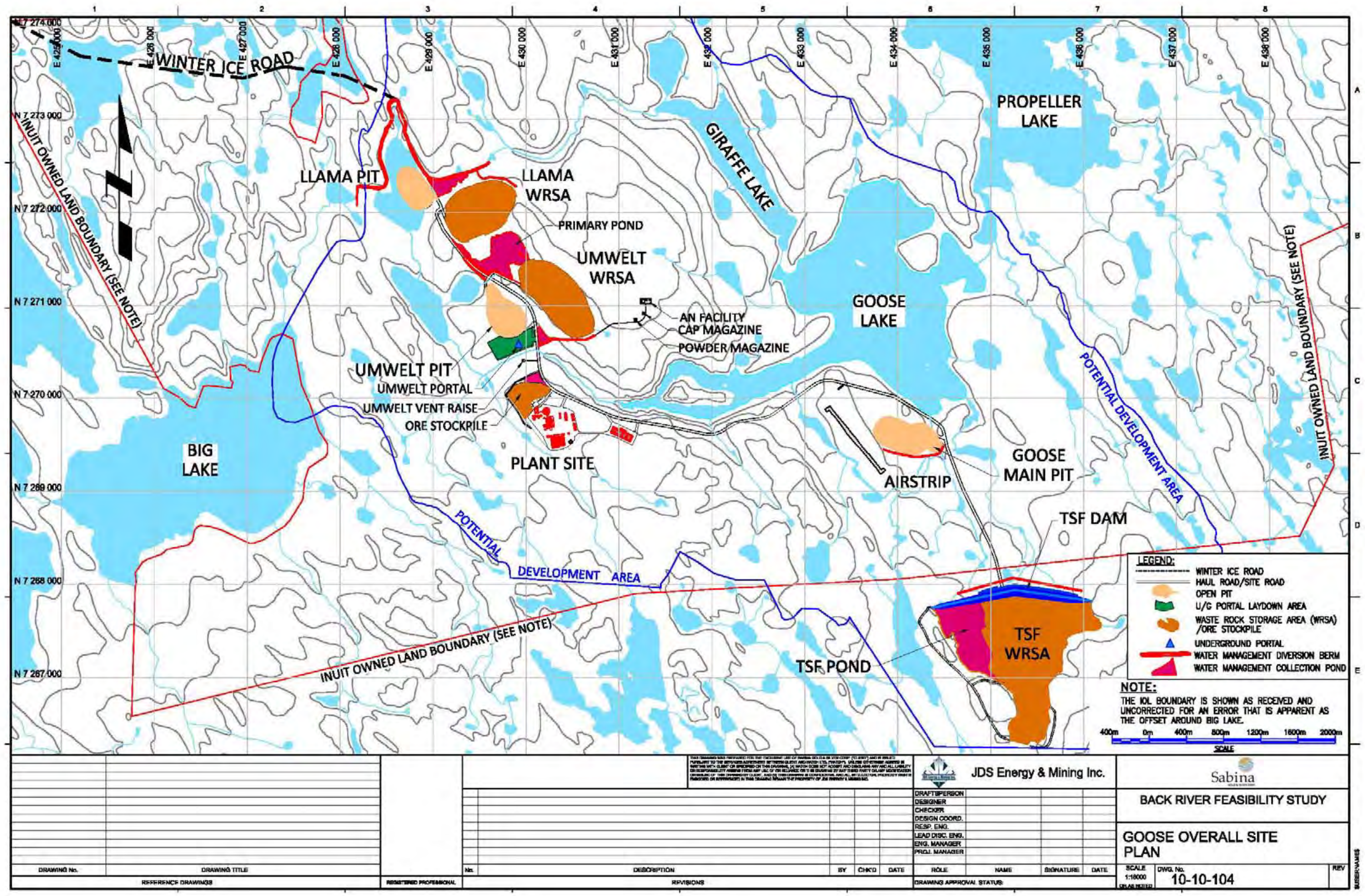
The Umwelt WRSA is planned to have an ultimate crest elevation of 345 masl (approximately 35 m WRSA height) and would be located east of the Umwelt pit limits. An access ramp along the west side of the WRSA is designed to allow access as placement of material advances. Umwelt WRSA has a planned design capacity of 7.0 Mm^3 (loose cubic metres based on 30% swell factor).

The Llama WRSA would be built to a planned final crest elevation of 355 masl (approximately 45 m WRSA height) and is planned to be located southeast of the proposed Llama pit limits. Llama WRSA has a planned design capacity of 14.4 Mm^3 (loose cubic metres based on 30% swell factor).

Waste material from Goose Main is planned to be placed on the TSF (TSF WRSA) and would ultimately cover the TSF dam. The TSF WRSA is located due south of the Goose Main open pit and would be built to a planned final crest elevation of 330 masl with a design capacity of 22.2 Mm^3.

Figure 16.8 shows the Goose Property overall site plan that includes the locations and ultimate designs of these proposed waste rock facilities.

Figure 16.8: Goose Property Overall Site Plan



Source: JDS 2015

16.2.6 Mine Equipment Requirements

16.2.6.1 Introduction

The open pit mining activities for the Project were assumed to be undertaken by an Owner-operated fleet. The equipment was selected based a standard open pit mining operation with conventional drill, blast, load and haul activities. Selection also considered bulk excavation of waste using hydraulic excavators, and bulk-selective loading of ore using a front-end loader or smaller hydraulic backhoes. Given the overall scale of operations and equipment requirements, a diesel-powered only fleet was selected.

Any reference to a specific supplier or piece of equipment should not be seen as an endorsement; this information is provided for reference purposes only. Additional analysis regarding equipment selection is planned to be carried out at the engineering and procurement stages of the Project.

The annual open pit mining equipment requirements are shown in Table 16.6.

Table 16.6: Open Pit Mine Primary Equipment Requirements

Type	Units	Total	Year								
			-1	1	2	3	4	5	6	7	8
MD 6240 Drill (152 - 270 mm)	#	2	2	2	2	2	2	2	2	2	2
MD 5125 Drill (89 – 152 mm)	#	1	1	1	1	1	1	1	1	1	1
Cat 775G Truck (64 t)	#	16	10	11	10	13	14	16	16	16	11
Cat 6015 Shovel (7 m³)	#	2	2	2	2	2	2	2	2	2	1
Cat 390 Excavator(4 m³)	#	2	1	2	2	2	2	2	2	2	1
Cat 988 Wheel Loader(7 m³)	#	1	1	1	1	1	1	1	1	1	1
Cat D8 Track Dozer	#	3	2	3	3	3	3	3	3	3	2
Cat 824 Wheel Dozer (4.2-m blade)	#	2	2	2	2	2	2	2	2	2	1
Cat 14M Grader	#	3	3	3	3	3	3	3	3	3	2

Source: JDS 2015

16.2.6.2 General Operating Parameters

The open pits are designed with 5 m benches in both the waste and ore headings with adequate phase geometry to achieve a maximum production level of 13.8 Mt/a. The design is to accommodate a maximum annual mill throughput of 1.1 Mt of ore. Mining is scheduled to advance sequentially through the pits, with up to two pits active at any time. Given the required production rate and pit geometries, vertical advance rates were limited to three benches per quarter, resulting in frequent ramp development and new bench construction.

Time definitions, work regime structure, and standard standby and delay parameters were applied to the mine equipment section.

Estimates for effective utilization of major equipment were based on vendor recommendations, cost services, factors and JDS experience. Initially, effective utilizations of 65% for the drilling equipment, 52% for the loading equipment, 59% for the hauling equipment, and 65% for support and auxiliary equipment were assumed. For Year 5 and beyond, a reduction in mechanical availability of 5% has been applied causing a reduction in effective utilization in the later part of the mine life. See Table 16.7.

Table 16.7: Availability, Target Use of Availability, and Effective Utilization of Major Equipment

Open Pit Equipment	Mechanical Availability (Yr 5 +) (%)	Use of Availability (%)	Effective Utilization (%)
250 mm dia. Rotary, Crawler Drill	85	90	65
Diesel, 7-m³ Front Shovel	85	96	52
Diesel, 7-m³ Wheel Loader	85	96	55
64-t Haul Truck	85	96	59

Definitions:

Mechanical availability: measure of maintenance down time which is (total available hours less mechanical downtime) divided by total available hours. For Year 5 and beyond a reduction in mechanical availability of 5% has been applied.

Use of availability: operational hours divided by total available hours.

Effective utilization: product of mechanical availability, utilization, operator efficiency and operational losses.

Source: JDS 2015

16.2.6.3 Blasthole Drilling and Blasting

Based on the selected bench height (drilling is planned to occur on 10 m high benches) and the production schedule, a 250 mm diameter production drill was selected. Drill pattern details are shown in Table 16.8.

Table 16.8: Ore and Waste Drilling Parameter Assumptions

Item	Unit	Value
Diameter	mm	250
Dry density (in-situ)	t/m³	2.9
Drill bench height	m	10
Burden	m	6.3
Spacing	m	7.8
Sub-drill	m	2
Total hole length	m	12
Stemming	m	3.1
Tonnes/hole	t	1,420
Drilling factor	t/m	118
Penetration rate	m/hr	11.2

Source: JDS 2015

To achieve the recommended bench face angles (BFAs) and inter-ramp angles (IRAs) within the stronger rock mass units, it was assumed that 3% of the total drilled material would be pre-split and conducted with a smaller drill. Operating costs were included to cover the additional cost of this small-diameter (115 mm) drill.

Based on these parameters, annual production capacity was estimated for each type of drill for each period of the mine plan. The blast design used ammonium nitrate fuel oil (ANFO) as the main explosive for blast holes. Given the relatively dry climatic conditions at the Property, emulsion was not included in the cost estimate. Table 16.9 shows the planned blasting parameters for both ore and waste.

Table 16.9: Estimated Blasting Parameters

Item	Unit	Value
Column charge	m	8.9
Column charge	kg	372
Powder factor	kg/t	0.26

Source: JDS 2015

Wet rock conditions are assumed to be minimal given the climatic conditions at site. However, if any water is encountered, blast holes would need to be dewatered and/or bag liners used. An explosives supplier is planned to be contracted to mix ANFO and provide blasting accessories. The Owner would supply ammonium nitrate (AN), fuel oil, explosives magazines and delivery trucks. Owner personnel are assumed to be responsible for loading and pattern tie-ins.

16.2.6.4 Loading

The main criterion for the selection of loading equipment is the ability to mine selectively given the nature of the ore bodies and pit design configurations.

Primary loading is planned to be performed by diesel hydraulic front shovels with a 7 m³ bucket. A wheel loader with a 7 m³ bucket and a 4 m³ bucket excavator would be used for secondary loading, rehandle and shovel support.

Operating hours for the loading fleet were estimated by calculating the amount of material required to be moved within a given period with appropriate productivity factors applied. Fleet size was then calculated using total operating hours for the period and the operating hours per unit within the period.

Productivities showing the number of passes and fill factors are summarized in Table 16.10 for both waste and ore. In addition to the loading time, the loading unit productivities include estimates for waiting, maneuvering time and unproductive time. Based on these parameters, the annual production capacity was estimated for each type of loading unit for each period of the mine plan.

Dig rates reflect the selective nature of the mining operation.

Table 16.10: Loading Unit Productivity Assumptions

Item	Units	Ore		Waste	
		7 m^3 Excavator	7 m^3 FEL	7 m^3 Excavator	7 m^3 FEL
In-situ material density	t/m^3	3.0	3.0	2.9	2.9
Material swell factor	loose:bank	1.4	1.4	1.4	1.4
Loose material density	t/lcm	2.1	2.1	2.1	2.1
Bucket size	m^3	7	7	7	7
Bucket fill factor	%	90	85	90	85
Tonnes per bucket	t	13.5	12.8	13.0	12.3
Size of truck to load	t	64	64	64	64
Theoretical buckets to load	#	4.5	4.8	4.7	5.0
Average buckets to load	#	5	5	5	5
Average loading cycle time	sec	40	60	40	60
Average spot time between loads	sec	45	45	45	45
First bucket time	sec	15	15	15	15
Total time to load truck	min	3.7	5.0	3.7	5.0
Theoretical loading time per day	min	1,212	1,212	1,212	1,212
Theoretical avg. truck loads per day	#	248	182	248	182
Truck load factor	%	95	95	95	95
Average truck load	t	61	61	61	61
Estimated loading productivity	t/day	15,100	11,000	15,100	11,000
Estimated loading productivity	t/ophr	746	547	746	547

Source: JDS 2015

16.2.6.5 Hauling

The truck haulage fleet for the Project was selected to match the selected loading fleet; this resulted in the selection of trucks with a payload of 64 t. Haulage profiles were estimated for the mine plan for every bench of the Project pits in the different years of the mine life and for each material type (waste/ore). Separate values were calculated for haulage within the pits (between the bench and the pit exits) and outside of the pit limits (between the pit exit and the final destination, e.g., primary crusher/stockpile or WRSAs). The distances were split between ramp and horizontal haulage.

Runge TALPAC® software was used to determine truck requirements and productivities. Table 16.11 summarizes the haul cycle parameters used to calculate truck productivities. Truck performance was calculated for every loading unit and period of the plan, with allowance for the travel time and other fixed times of the cycle such as loading. This varies according to the loading equipment used, dumping, waiting and spot times.

Table 16.11: Haulage Cycle Parameters

Description	Unit	Value
Rated payload	tonnes	64
Fill factor	%	95
Adjusted average payload	tonnes	61
Dump time at crusher/stockpile	min/load	1.5
Dump time at crusher/stockpile	min/load	1.5
Dump time at WRSA	min/load	1
Stopped time (non-hauling)	% of net operating hours	10
Effective utilization	%	59

Source: JDS 2015

16.2.6.6 Support/Ancillary Equipment

The selection of auxiliary and support equipment was based on the size and type of the primary loading and hauling fleet, the geometries of the various open pits, and the number of roads and WRSAs that would be in operation at any given time.

The selection of the type of equipment was based on vendor recommendations as well as JDS experience in similarly sized operations. The auxiliary equipment fleet is planned to be composed of one type of track dozer (Cat D8-class), one type of wheel dozer (Cat 824-class), one type of grader (Cat 14M-class), and one size of water truck (55 m^3).

The major tasks to be completed by the support equipment include the following:

- Bench and road maintenance;
- General maintenance;
- Reclamation support;
- Tailings dam support; and
- Shovel support/cleanup.

The primary support equipment unit functions are as follows:

- Cat D8 Track Dozer – primarily used for shovel support/cleanup, WRSA maintenance, road construction, high-wall cleaning and other projects as needed.
- Cat 824 Wheel Dozer – used to support WRSA maintenance, drill pattern cleanup, and support for shovel floor maintenance.
- Cat 14M Grader – primarily used for road maintenance and pit and WRSA floor maintenance, road construction and service road maintenance.

The following items were also included as support equipment:

- Drill (115 mm) for secondary blasting and pre-split drilling;
- Fuel trucks to supply diesel fuel to all the hydraulic diesel excavators, dozers, drills, as required;
- Lube truck to supply lubricants, hydraulic fluids, and cooling water to all open pit equipment;
- Mobile mechanical trucks, equipped with tools, welding machine, worktable with press, and replacement parts, to provide preventative and corrective maintenance in the field;
- Small excavator (3 m^3) for road and pit maintenance;
- Low-boy transporter truck (100 t) to transport dozers, drills, small backhoe and major components;
- Tire manipulator for tire maintenance;
- Mobile crane (65 t) for field maintenance; and
- Mobile lights to illuminate waste dumps and construction areas.

16.2.7 Mine Maintenance

The key elements provided by maintenance to satisfy the requirements of open pit mine production are equipment safety, availability, reliability, and operability.

The strategy for repair and maintenance of the open pit mobile equipment fleets for the Project is planned to be a balance between minimizing risk and minimizing costs to Sabina. All on-site maintenance would be carried out by Sabina personnel using Sabina's own installations. On-site work would consist of mainly preventative maintenance and major-component exchange. Given the estimated mine life, no major rebuilds are anticipated; however, if required, they would be performed on site by contractors.

16.2.8 Mine Personnel and Organization Structure

16.2.8.1 Basis

The work schedule assumes a 24 hours/day, 7 days/week and 355 days/year mining operation (10 days of non-production have been assumed to account for adverse weather conditions). Operations and mining personnel would work on two 12-hour shifts per day. Production, maintenance and technical personnel are planned to be primarily on a 2-week-in / 2-week-out rotation.

With the exception of the blasting crew, all hourly labour and supervisory personnel would rotate between day and night shifts. Management and technical staff would work the day shift only, with the exception of ore control technicians who would rotate with the crews.

Equipment operator labour requirements are based on equipment hours calculated from engineering estimates of productivities and activities, quantities of the various materials moved, and hourly equipment operating rates. Other support labour requirements within the open pit mining operation are determined by engineering estimates of activities.

Maintenance labour requirements are based on the number of equipment units to be maintained, estimates of mechanical availability, and maintenance labour intensities for each open pit fleet type.

16.2.8.2 Personnel Levels and Structure

The mining operation is planned to be overseen by the mine manager, who would report to the General Manager. There are four Mining departments planned: mine operations, mine maintenance, engineering, and geology.

Under the direction of the mine superintendent, the mine operations department would be responsible for operator training and the open pit operation. This includes drilling, blasting, loading, and hauling of ore and waste, dump and haul road construction/maintenance, and mine dewatering. Each mine operating crew would be led by a mine shift foreman. The estimated number of operators is based on the annual equipment requirements and the crew schedule.

The mine maintenance department would report to the mine maintenance superintendent. Maintenance crews are planned to work the same shift schedule as the mine operations crews. Each maintenance crew would be led by a maintenance shift foreman. A mine and maintenance general foreman is also planned for the operation. The estimated number of maintenance personnel is based on the annual equipment requirements.

The engineering department would be led by the chief engineer; the department would be responsible for providing daily, weekly, monthly and yearly plans for the open pit operations.

The chief geologist and his team members would be responsible for updating the various resource models, calculating ore reserves, and providing necessary ore grade control.

Staff and labour requirements over the LOM for operations, maintenance and supervisory/technical departments are summarized in Table 16.12. Staff requirements were determined by assessing the LOM plan and scale of open pit operations.

Table 16.12: Yearly Personnel Requirements

Description	Year								
	-1	1	2	3	4	5	6	7	8
Mine Operations									
Mine Shift Foreman	4	4	4	4	4	4	4	4	4
Driller, Blasthole	9	9	9	9	9	9	9	9	5
Blaster	2	2	2	2	2	2	2	2	2
Blasting Helper	4	4	4	4	4	4	4	4	2
Shovel/Loader Operator	12	14	15	14	14	17	13	14	8
Truck Driver	35	37	35	45	49	53	54	55	36
Track Dozer Operator	5	8	8	8	8	8	8	8	5
R.T. Dozer Operator	5	5	5	5	5	5	5	5	3
Grader Operator	7	8	8	8	8	8	8	8	5
Labourer/Trainee	4	4	8	8	8	8	8	8	4
Subtotal	**87**	**95**	**98**	**107**	**111**	**118**	**115**	**117**	**74**
Mine Maintenance									
Heavy Equipment Mechanic	6	7	7	7	8	8	8	8	6
Welder/Mechanic	6	7	7	7	8	8	8	8	6
Electrician/Instrument	3	3	3	4	4	4	4	4	3
Lube/PM Mechanic/Light Duty Mechanic	6	7	7	7	8	8	8	8	6
Subtotal	**21**	**24**	**24**	**25**	**28**	**28**	**28**	**18**	**21**
Technical									
Mine Technicians LTP / STP OP	2	2	2	2	2	2	2	2	2
Surveyor OP	2	2	2	2	2	2	2	2	2
Mine General	4	4	4	4	4	4	4	4	4
Subtotal	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**

Source: JDS 2015

16.2.8.3 Labour Build-up and Initial Training Schedule

Key operations personnel are planned to be recruited before the completion of the construction phase of the Project. The management team, including managers, human resources, health and safety, would be in place before construction begins. Staffing levels would then progressively increase during the construction phase of the Project.

16.3 Underground Mining

The Goose Site has four deposits that have the necessary grade, continuity, and tonnage to be considered for underground mining, namely Umwelt, Llama, Goose Main, and Echo. As described previously in section 16.2.3 of this report, an open pit/underground crossover review was performed for each deposit to determine the most economic depth at which operations can transition from open pit mining to underground mining. Only Umwelt had sufficient economic underground resources to support an underground mining operation.

16.3.1 Underground Mining Context

16.3.1.1 Deposit Geometry

Umwelt is a continuous, massive deposit with a shallow dip of 35°. The deposit extends to a depth of 650 m below surface. It has an average strike length of 170 m and an approximate thickness of 40 m. Umwelt has a large underground reserve inventory; the highest grades are located in the deepest mining block.

16.3.1.2 Geotechnical Considerations

The main lithological units at the Project are as follows:

- Greywacke and Interbedded Sediments: These sediments are the most common rock type at site. These units overlie and underlie the ore deposits;
- Iron Formation: Several BIFs have been identified by Sabina. The Iron Formation is the host rock for the gold mineralization. The Lower Iron Formation (5 to 20 m thick) is separated from the Upper Iron Formation (15 to 30 m thick) by the Middle Mudstone (described below);
- Mudstone: Phyllitic Mudstone overlies the Iron Formation in some areas and is typically 5 to 20 m thick. Middle Mudstone is a layer that is less than 10 m thick; it separates the Lower and Upper Iron Formations. This Mudstone can be strongly foliated; and
- Felsic Dykes, Porphyry Intrusions and Gabbro/Diorite Dykes: These units cross-cut the other lithologies and are the youngest units at site.

Two main fault orientations were recorded: sub-parallel to the deposits (NW-SE to N-S) and perpendicular to the deposits (NE-SW). Faults typically dip steeply, are a few metres thick, contain gouge, and are slickensided.

KP defined the geomechanical domains by lithology. Their characteristics are as follows:

- Greywacke and Interbedded Sediments: These units are generally of GOOD to VERY GOOD quality (RMR values from 60 to 85) with a mean UCS of 95 MPa;
- Lower Iron Formation: This unit is generally of GOOD to VERY GOOD quality (RMR values from 65 to 90) with a mean UCS of 195 MPa;
- Upper Iron Formation: This unit is generally of GOOD to VERY GOOD quality (RMR values from 65 to 85) with a mean UCS of 125 MPa;
- Phyllitic Mudstone: This unit is generally of GOOD quality (RMR values from 60 to 80) with a mean UCS of 55 MPa;

- Middle Mudstone: The rock mass quality of the Middle Mudstone is highly variable and ranges from FAIR to GOOD quality (RMR values from 45 to 75) with a mean UCS of 60 MPa. The Middle Mudstone is generally of lower and more variable quality than the other lithologies; and
- Gabbro Dykes: These units are generally of GOOD to VERY GOOD quality (RMR values from 70 to 85) with a mean UCS of 140 MPa. KP grouped the Felsic Dykes and Intermediate Dykes with their host domain.

KP developed geotechnical and geomechanical underground design recommendations for the Umwelt deposit (KP, 2015). The recommendations were based on the available data and the selected mining methods, including achievable stope spans, expected mining dilution, sill pillar, and crown pillar dimensions, ground support standards, and access development placement.

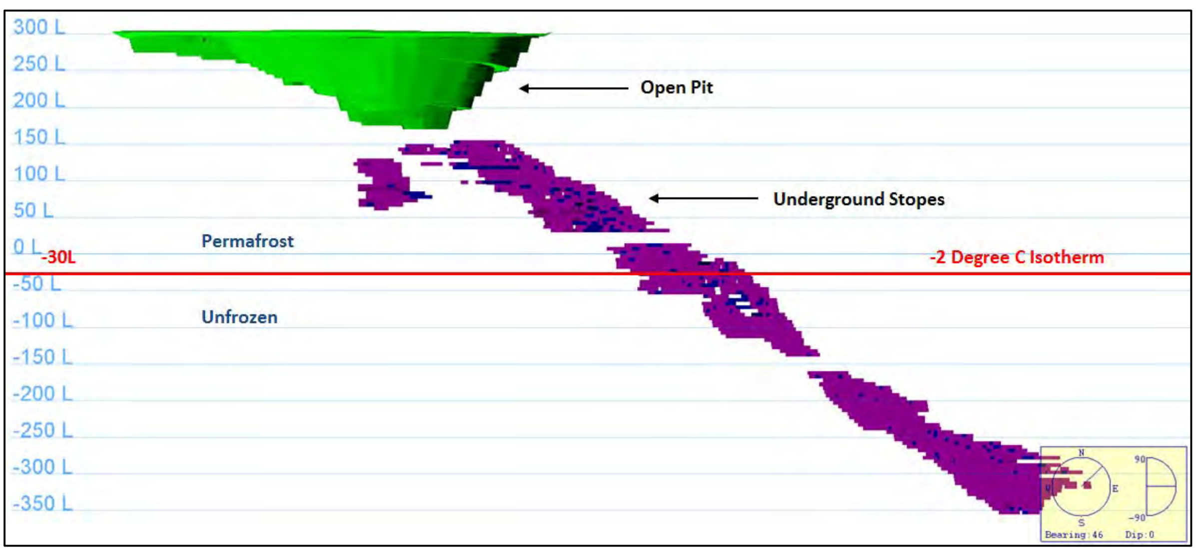
16.3.1.3 Hydrogeological and Permafrost Considerations

SRK conducted a hydrogeological assessment to estimate groundwater inflows for the different mining areas. The Project area is located within the region of continuous permafrost, which is essentially impermeable to groundwater flows; however, localized unfrozen conditions are expected.

SRK predicts deep groundwater flows below the -2°C isotherm, at a depth of -30 masl. The -2°C isotherm is considered to be the limit between frozen and unfrozen ground because of the high salinity of deep groundwater. Portions of Umwelt underground workings are planned to be located below the base of the permafrost (see Figure 16.9).

SRK developed a groundwater inflow model for Umwelt. The predicted average inflow rate at Umwelt is 550 m^3/d. This low inflow rate is considered manageable with conventional mine dewatering techniques.

Figure 16.9: Expected Groundwater Zone at Umwelt



Source: JDS 2015

16.3.2 Underground Mining Method Selection

The underground mining method was selected on the basis of ore body characteristics, such as grade, dilution, dip, continuity, thickness, etc.

Mining methods involving cemented backfill were not considered due to cement freight costs to site. Uncemented rock fill requires an overhand mining strategy. Sill pillars in ore are required to allow concurrent mining of multiple blocks.

The PPCF method was selected for the Umwelt deposit due to the deposit variability, the shallow dip and deposit thickness. Other semi-bulk mining methods, like long-hole stoping, would create unstable hanging wall exposures of the weak Middle Mudstone. PPCF mining limits the hanging wall exposure and allows the installation of ground support. High production rates can be attained with this mining method.

16.3.2.1 Description of Post Pillar Cut-and-Fill Mining

PPCF is a highly mechanized and systematic mining method, which is well suited for massive, shallow dipping and laterally extensive deposits. It is a variation of room and pillar mining, combined with drift and fill.

The mining block would be accessed from the main decline located in the footwall with cross-cut and attack ramps.

Typically, rooms would be excavated in 5 m high cuts, using an overhand mining sequence. Load-bearing rock pillars would be left behind in a checkerboard pattern to support the back of the cut. It is important the pillars are aligned vertically on each cut. The stope design at Umwelt is based on 10 m room spans and pillar sizes of 5 m by 5 m, which would result in an ore recovery rate of 89%.

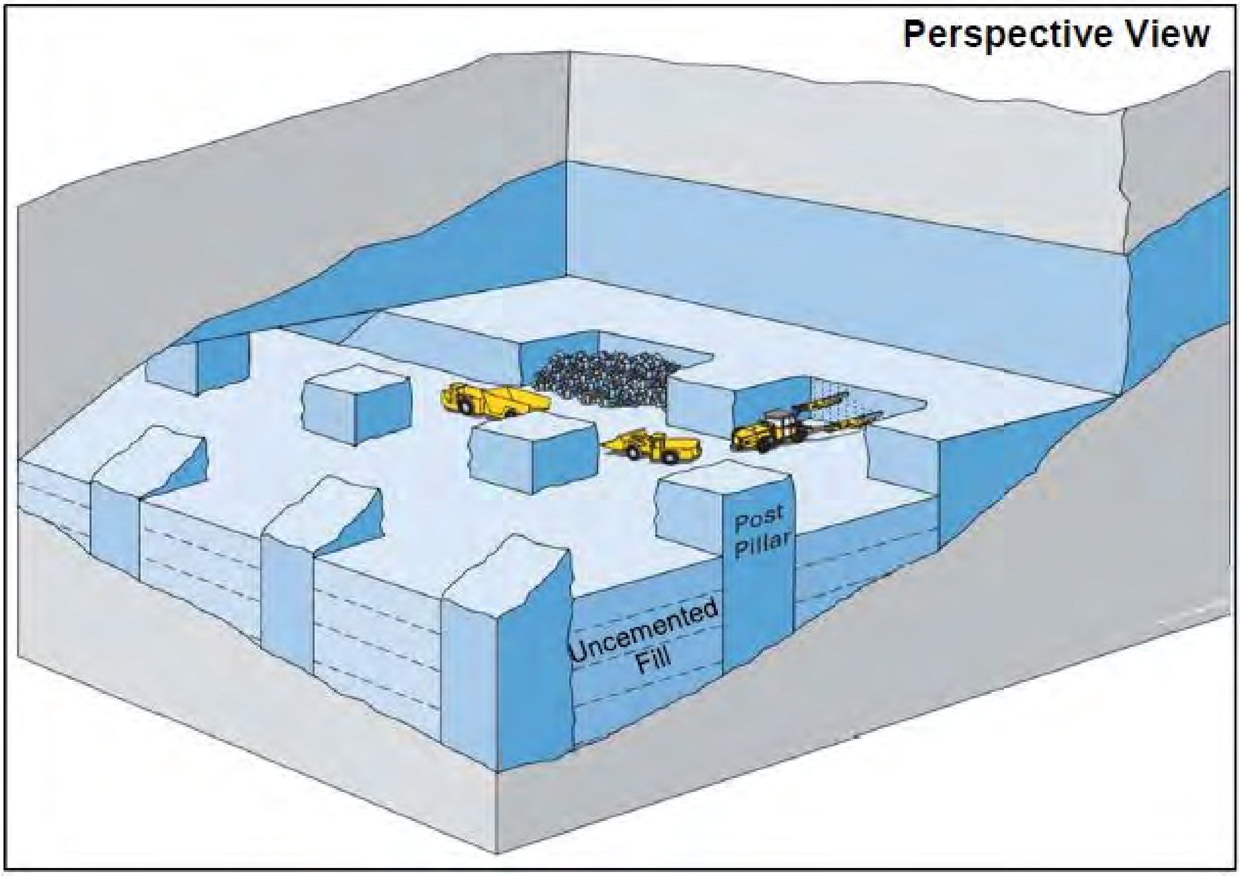
Once the ore has been mined, the stopes are planned to be backfilled with uncemented waste rock obtained from underground development or open pit waste stripping. The backfill would confine the pillars, which results in additional support when the pillars yield.

To start the next cut, the attack ramp would be slashed at an appropriate gradient to gain the required elevation. The slashed waste rock would be left in place and be used as a ramp. Once access to the next lift is established, the PPCF mining cycle would begin again, working on top of the waste backfill.

PPCF is a very productive mining method if multiple headings are available. Maintaining high production rates depends on adherence to an efficient mining cycle of ground support, drilling, blasting, mucking, hauling, and filling. The main disadvantage is that it is an entry method that requires ground support on every cut.

A conceptual view of PPCF is shown in Figure 16.10.

Figure 16.10: Post Pillar Cut-and-Fill Mining



Source: KP 2015, modified from Atlas Copco 2007

16.3.3 Underground Production Capacity and Scheduling

The optimum mine size was estimated to maximize return on investment. During the early mine planning stage, rules-of-thumb formulas (Taylor's Law and Rice's Law) were applied and tonnes-per-vertical-metre were reviewed to arrive at a preliminary mining rate.

During detailed mine planning, Minemax iGantt software was used to arrive at an optimized mining schedule. Each designed object (access drifts and stopes) was assigned a value based on mining cost and revenue generated. The scheduling software targeted high-value mining areas, given constraints such as rates for development, drilling, stoping, and backfilling. The cost of access was also considered.

The following strategy was developed to fit the underground operation into the 10-year overall mine life of the Project:

- Decouple open pit and underground mining by starting the decline adjacent to the pit crest and not at the pit bottom. This also allows the use of the mined-out open pit for tailings or water storage; and
- Schedule development of Umwelt for Year 2 to minimize pre-production capital expenditures, for underground production start-up in Year 3, and finish production by Year 9.

16.3.4 Mine Access Design

The Umwelt deposit is planned to be accessed via decline, based mainly on the depth of the mineralized zones. A decline is scheduled to provide early access to the ore zones, reduce initial capital, allow access to follow the deposit down plunge, and allow the opportunity to carry out infill exploration drilling.

16.3.4.1 Decline

The decline would be used to haul ore and waste and provide access for personnel, equipment, materials, and services. It is also planned to be used as an exhaust airway.

The location of the decline portal was chosen close to the pit access road. Local site conditions were also considered, such as flat topography, to minimize cut-and-fill work for laydown pad construction. Environmental offset limits from streams were maintained. A box cut is planned to be excavated through overburden to solid bedrock to collar the decline.

The size of the decline was selected according to required clearances for the chosen mobile equipment and required ventilation during development and production. It was determined that a 4.5 m wide by 5.0 m high profile would be suitable for a 30 t haul truck. In general, the decline is planned to be driven at a 15% gradient.

The decline is designed to minimize intersections with major faults or the Middle Mudstone rock unit. In addition, the offset of the decline from the open pit was maximized, and a 25 to 30 m offset between decline and stopes was maintained.

Remuck bays are proposed to be located every 150 m along the decline. They are designed to be 15 m long to store two rounds of development muck. Later, the remuck bays would be used for equipment and material storage, or could be converted to sumps, refuge stations or explosives magazines.

16.3.4.2 Level Access and Attack Ramps

Access development at Umwelt underground is planned to consist of a decline from surface to 640 m depth. Level access cross-cuts are designed to be located every 40 to 50 m vertically and include a remuck and sump on each level.

Attack ramps would provide the access to the orebody and would have a maximum gradient of 20 %. Once a cut has been mined and backfilled, the back of the attack ramp is planned to be slashed down (i.e., take down back) and a ramp would be constructed with the slashed rock to access the next cut above.

Level access cross-cuts and attack ramps are planned to be developed off the decline at a 4.5 by 5.0 m profile. All infrastructure development, which would not be used for access, i.e., remucks, ventilation drifts and sumps, were designed at a 4 m by 4 m profile.

16.3.4.3 Ventilation Raises and Drifts

Ventilation raises are planned to be developed from surface for air circulation. Some of the raises would also be equipped with ladderways to serve as secondary egress in case of emergency. The top and longest portion of the ventilation raises are designed to be developed by raiseboring at 4.0 m diameter. The lower portions would be relatively short and would be developed by drop raising on a 4 m by 4 m profile.

Based on ventilation modelling results, a twinned Fresh Air Raise (FAR) system would be required at Umwelt (see section 16.3.7.1). One of the two FAR systems would be equipped with an egress ladderway. The surface legs of each FAR are planned to be mined by raiseboring and the underground portions by drop raising.

Lateral ventilation drifts at 4 m by 4 m profile are planned to follow the dip of the deposit and connect the ventilation circuits to the decline and level access. An Exhaust Air Raise (EAR) system midway down the deposit would be required to keep the air speed at an acceptable level in the decline. The EAR would be developed by raiseboring from surface. Figure 16.11 shows the access design for Umwelt.

Figure 16.11: Umwelt Underground Operation Access Design (Long Section)

Source: JDS 2015

16.3.5 Stope Design

Ore is planned to be mined by excavating rooms in 5 m high cuts in a bottom-up mining sequence. Initially, a 6 m wide pilot drift is planned to be advanced. Deep ground support would then be installed in the back of the pilot drift. The walls would then be slashed to establish the final room span of 10 m. Permanent pillars with a footprint of 5 m by 5 m are planned to remain between the rooms to support the back. Sill pillars with vertical height of 15 m would separate the mining blocks.

16.3.6 Unit Operations

General mine planning criteria are summarized in Table 16.13.

Table 16.13: Mine Planning Criteria

Parameter	Unit	Value
Operating Days per Year	days	365
Shifts per Day	shifts	2
Hours per Shift	hour	11
Work Rotation	weeks in / weeks out	2x2
Ore Density	t/m³	variable, from block model 3.0 average
Waste Density	t/m³	2.89
Swell Factor	%	34
Placed Backfill Density	t/m³	2.16

Source: JDS 2015

16.3.6.1 Development

The development headings are planned to be driven with electro-hydraulic, two-boom jumbos similar to the Sandvik DD421-60. Blastholes with 45 mm diameter would be drilled to a depth of 4.2 m. The advance per round is assumed to be 4.0 m. It is planned that one jumbo can drill between two to three rounds per shift, depending on travel time between faces.

After the round is drilled, the blastholes would be charged with explosives by a MacLean AC2-type ANFO Loader. The holes would be loaded with a blend of ANFO. The lifter holes would be loaded with packaged emulsion because of its water resistance. The round would be tied-in by the blaster and the blast would either occur during a shift, when all personnel are accounted for in a designated safe location, or at a shift change. Blasting is planned to be initiated by non-electric caps.

Following the blast, ventilation would be re-established and the blast smoke would clear. Upon re-entry to the heading, the miners would scale and check the work place for any safety concerns. The muck pile would be washed down to suppress any dust.

In the next stage, the blasted rock would be mucked from the face by a LHD either directly into a truck or into a remuck bay. A Sandvik LH410 was selected as an example unit for the mucking operation. The muck is planned to be hauled from underground to surface by haul trucks like the Sandvik TH430. Additional information on the haulage cycle is provided in section 16.3.6.3.

Once the face is clear of muck and the back, walls, and face of the heading are scaled, ground support would be installed according to geotechnical recommendations. A MacLean 928-type Scissor Bolter was selected for ground support installation.

Typical ground support installation in access development is planned to consist of 2.4 m long Swellex bolts on the back and 1.8 m long Swellex on the walls at a 1.5 m by 1.5 m pattern with 6-gauge welded wire mesh installed within 1.5 m of the floor. For corrosion protection, plastic-coated Swellex bolts and galvanized mesh would be installed in long-term development headings. In areas with poor ground conditions and in intersections, 3.7-m or 4.6-m Super Swellex bolts would be used for deep ground support. Swellex bolts have been selected as the ground temperature is expected to generally be below 0^{0} C.

It was assumed that approximately 1 to 2% of the drifts would require shotcrete to address poor ground conditions. A MacLean SS3 Shotcrete Machine was selected for this task.

A MacLean SL3 Scissor Truck is planned to be used to install services, which would be advanced roughly every 20 m. Service water and dewatering lines are designed to be steel pipes connected by Victaulic-type couplings. Electrical power and leaky feeder communication lines are planned to be advanced at the same time. The scissor truck would also be used to install and maintain auxiliary ventilation fans and ducting.

Access development advance rates are scheduled for single-heading advance at 6.7 m/day and multiple-heading advance at 8.6 m/day.

Ventilation drop raises are planned to be drilled using a Boart Longyear Stopemaster Drill. Drop raises would be developed on a 4 m by 4 m profile and a maximum length of 35 m.

On average, 29 blastholes are planned for a raise. The holes would be loaded with ANFO and blasted in 2.8-m lifts. A sufficient amount of muck is planned to be removed at the bottom of the raise between blasts so that the next blast is not choked off.

Upon completion of blasting, miners would scale and install ground support in the raise by working on top of the muck pile or on top of temporary staging installed in the raise. Galvanized welded wire mesh and 1.8-m plastic-coated Swellex bolts would be installed on a 0.9-m square pattern. In addition to installing ground support, the miners would also install the escape-way ladders. The ground support and ladders would be installed starting at the top of the raise, and muck would be drawn down as the installation process proceeds.

Raiseboring is planned to be performed by a specialized raisebore contractor. Budgetary quotes were obtained for the raisebore program.

16.3.6.2 Stoping

PPCF stoping would use the same equipment as described in the Development section (section 16.3.6.1).

Drilling and blasting is planned to be done with a two-boom jumbo and an ANFO loader. Pilot drifts at 6 m wide by 5 m high profile are planned to advance in 3.85 m rounds. Blasted ore would be mucked by a dedicated LHD from the face to the stope remuck bay in the access cross-cut. Haul trucks would then take the ore through the decline to surface. Bolting is planned using a scissor bolter.

Ground support in the pilot drift is designed to consist of 2.4 m Swellex and 4.6 m Super Swellex bolts in the back, and 1.8 m Swellex bolts in the walls, installed on a 1.2 m square pattern. Mesh would be extended from the back to within 3 m of the floor.

After the pilot drift is finished, the rooms would be slashed out to a final span of 10 m by taking an additional 2 m wall slash on each side. Ground support in the slashed sections would consist of 2.4 m Swellex bolts in the back and 1.8 m Swellex bolts in the walls, installed on a 1.2 m square pattern with mesh installed to within 1.5 m of the floor. It was assumed that approximately 1% of the stope development would require shotcrete due to poor ground conditions.

After the initial cut is finished in each mining block, approximately 80% of the PPCF production rounds would be done on top of backfill. This breasting round would not require a lifter row of blastholes or a cut. Blast patterns for wall slashing would also have a free face, which further reduces the powder factor. The average powder factor for the pilot drift is estimated to be 0.74 kg/t, while slashing would be 0.34 kg/t.

When mining a cut is complete, services and infrastructure would be removed on retreat with a scissor truck and waste backfill would be placed in the stopes.

16.3.6.3 Loading and Hauling

Blasted material from development headings would be mucked by LHD directly to a haul truck or to a remuck bay, located up to 150 m from the face. During the pre-production period, development waste rock would be hauled to surface and stored close to the portal at a temporary waste stockpile. When underground ore production commences, development waste would be transported to mined-out stopes for direct backfilling.

Trucks loaded with ore are planned to drive up the decline to surface and dump the ore in different ore stockpile areas depending on the grade of the material. The same trucks would transport backfill material on backhaul from surface to mined-out stopes when required. A Caterpillar 966H front-end loader would load the underground haul truck with backfill from a waste stockpile at the portal laydown area on surface.

The Sandvik LH410 10 t LHD with a 4.5 m³ bucket is selected to be paired to a Sandvik TH430 30 t haul truck. The LHD would fully load the truck with three passes. The trucks and LHDs would not be overloaded when mucking ore or waste. The trucks and LHDs would have a slightly lower fill factor when filled with waste due to the lower material density.

Haulage profiles for all production levels and material types were generated for the underground operation. These calculations provide accurate equipment hours for both the haul trucks and LHDs.

16.3.6.4 Backfilling

PPCF mining requires backfill to maintain ground stability and to provide a work base for the equipment on the next cut above. The primary source for backfill material is planned to be development waste from the underground mine. A net shortage of backfill material is expected. The shortfall would be made up by using waste rock from the open pit operation. Using waste rock for backfilling would reduce the surface environmental impact because potentially acid generating material would be permanently stored underground. Unconsolidated waste rock is planned to be used to avoid costs of cement.

During pre-production, underground waste rock is planned to be hauled to surface and stored on a temporary backfill stockpile close to the portal. Upon commencement of underground production, development waste would be hauled by the underground trucks directly to the area requiring backfill. When there is no backfill available from development faces underground, the temporary waste stockpile on surface would be used as a backfill source. When all the underground development waste is depleted, open pit haul trucks are planned to transport waste rock from the open pit to this stockpile, as required.

A Caterpillar 966H front-end loader would load underground haul trucks from the temporary waste stockpile with backfill. A grizzly would be used to scalp off oversized rocks from the open pit waste. Underground haul trucks would transport the backfill from surface to the mining areas, where the material would be dumped directly into the mined-out stopes or into a remuck bay.

A smaller Sandvik LH307-type LHD with a 3.7 m³ bucket is planned for tight filling at the PPCF operation when needed. After the backfill is piled-up as close to the back as possible, the bucket would be replaced by a "rammer jammer" attachment. The "rammer jammer" would be used to push backfill tight against the back.

Table 16.14 summarizes the backfill requirements for the Umwelt underground operation.

Table 16.14: Backfill Summary

Deposit	UG Development Waste Generated (m³)	Backfill Required (m³)	Backfill Sourced from Open Pit (m³)
Umwelt	499,622	1,142,574	642,952

Source: JDS 2015

16.3.7 Mine Services

16.3.7.1 Ventilation

The ventilation system for the underground operation is designed to dilute and remove dust, diesel emissions and blasting fumes and maintain compliance with Nunavut mine regulations. A ventilation network was modelled in Ventsim™ software for Umwelt, based on the detailed mine design. The mine design was imported from Vulcan with the proper drift profiles. Industry-standard friction factors and shock losses were included in the model to accurately simulate the ventilation system. Additional resistance for escape ladderways was included in the modelling. The required pressures and flow rates were calculated and used to select primary ventilation fans and estimate the electrical power consumption.

16.3.7.2 Airflow Requirements

Airflow requirements for the underground operation were based on expected diesel emissions of the underground mining fleet. According to the Nunavut mining legislation, “the ventilation quantity shall be at least 0.06 m^3/s for each kW of the diesel-powered equipment operating at the work site” (Mine Health and Safety Act, section 10.62 (2)).

Mobile equipment lists were compiled to determine when the mine would be in full production. The power rating of each piece of equipment was determined, and then utilization factors, representing the equipment in use at any given time, were applied to estimate the amount of air required. Ventilation losses of 20% were applied to determine total ventilation requirements. It was determined that the maximum required airflow for the Umwelt Underground operation is 225 m^3/s.

16.3.7.3 Ventilation Design Concept

The mine is designed to be ventilated by a “push” ventilation system with primary fans located at air intake raises. Fresh air would be directed down through a system of FARs which follow the decline. The individual raises would be connected by ventilation drifts. At each of the levels, a regulator would be installed at the ventilation drift to control the amount of air going into the level or the decline. The return air from the levels is planned to be exhausted up the decline and an EAR. A maximum air velocity of 6 m/s in the decline was used in the design criteria to avoid dust problems.

16.3.7.4 Main Fans

The main fans are designed to be installed on surface at the collars of the FARs. Airflow would be controlled with a variable frequency drive (VFD). The VFD would allow power to be reduced to the fan when the mine is not operating at full capacity.

Fans are planned to be skid-mounted to reduce capital costs for concrete foundations and allow fast installation. Due to similar operating parameters, main fans and motors for Umwelt would be standardized and interchangeable; this would reduce costs for critical spares.

The main fan power, total pressure, and decline air velocity for Umwelt are shown in Table 16.15.

Table 16.15: Primary Ventilation

Deposit	No. of Intake Raises	Main Fan Total Pressure (Pa)	Main Fan Power (kW)	Air Velocity in Decline (m/s)
Umwelt	2	1,311	216	6.0
		1,232	162	

Source: JDS 2015

16.3.7.5 Ventilation During Development

Prior to establishing the primary ventilation system, consisting of FAR and main fan, all the air required for the advance of the decline is planned to be supplied by auxiliary ventilation. Steel ducting with 1.22 m diameter and 150 kW fans would be installed to provide least 38 m^3/s of air to the decline face. Where decline development to the first raise exceeds 250 m, booster fans would be required in the duct line. Once the first ventilation raise is established, the ducting and fans would be stripped out of the decline.

The required number of development fans for the underground mine is shown in Table 16.16.

Table 16.16: Development Phase Ventilation Requirements

Deposit	Development Distance to First Raise (m)	Number of 150 kW Fans
Umwelt	850	3

Source: JDS 2015

16.3.7.6 Auxiliary Fans

Auxiliary fans are planned to be used to ventilate the advancing development and active production levels. Fresh air would be sourced from the ventilation raises and forced using smaller auxiliary fans through ventilation ducting to the active headings. Ducting would be removed on retreat and used again on the next cut or advancing face.

The expected number of auxiliary fans required during production is shown in Table 16.17.

Table 16.17: Auxiliary Fan Requirements

Deposit	Number of 75 kW Fans	Number of 50 kW Fans
Umwelt	6	1

Source: JDS 2015

16.3.7.7 Ventilation Bulkheads

To balance airflows through the mine, ventilation bulkheads with regulators are required at the ventilation cross-cuts on each level; this is where the fresh air circuit connects to the access development. Auxiliary fans would be mounted at these bulkheads. Air-lock access doors would be required at each regulator to facilitate access to the escape ladderway in cases of emergency.

16.3.7.8 Mine Air Heating

At the Umwelt underground operation groundwater inflows are expected. Therefore, intake air would need to be heated during the winter months to prevent ice build-up on roadways and in ventilation raises.

The intake air is planned to be heated to a temperature of +2°C. The mine air heating system at Umwelt would consist of indirect fired diesel heaters.

Heating calculations are based on average site temperatures and modelled intake air flows. Heat generated from underground diesel equipment was not considered in the estimation of mine air heating requirements.

16.3.7.9 Electrical Power Distribution

Electrical power is planned to be supplied from the central power plant by 4.16 kV feeder cables to the Umwelt portal site. A skid-mounted E-House (housing containing electrical switchgear, transformers, motor control systems, etc.) at the portal site would contain switchgear, MCCs and transformers. If the main power line goes down, a diesel-powered standby generator would be used to provide emergency power for dewatering.

High-voltage cable would enter the mine via the decline and would be distributed to electrical substations located close to the main mining blocks. The power cables would be suspended at the back of development headings. The 4.16 kV decline feeder is designed to terminate at the underground substations, where step-down transformers will provide 600 V to various electrical equipment.

The major electrical power consumption in the mine arises from the following:

- Main and auxiliary ventilation;
- Mine dewatering pumps;
- Underground mobile equipment;
- Air compressors; and
- Refuge stations.

Table 16.18 lists the estimated annual electrical power consumption for the underground operation.

Table 16.18: Projected Annual Electrical Power Consumption

Mine Year	Electrical Consumption (MWh/a)
2	2,613
3	9,403
4	11,005
5	10,911
6	11,037
7	10,937
8	10,971
9	9,668
TOTAL	**76,546**

Source: JDS 2015

16.3.7.10 Compressed Air

The Underground mobile drilling equipment such as jumbos, rockbolters, long-hole rigs, and ANFO loaders are designed to have their own compressors. Portable compressors would be used to satisfy compressed air requirements for any secondary pumping, jackleg drilling, and cleaning out longholes.

16.3.7.11 Service Water Supply

Service water for underground is planned to be used mainly for drilling, dust suppression, and washing of development faces. Water would be supplied from a 55,000 L service water tank close to the portal and would be gravity fed to the underground work areas via 100 mm diameter pipelines. Pressure reduction valves would be installed along the decline as needed. The service water tank would be refilled with underground mine water or by a site services water truck. It is estimated that the Umwelt underground operation would consume on average 61,500 L/d of service water.

Face pumps are planned to collect the service water and return it to the nearest sump, where it would be decanted and pumped to surface for reuse. Recycling of water would reduce the overall water consumption at site.

16.3.7.12 Mine Dewatering

Groundwater inflows are expected at Umwelt. Peak and average inflow rates were estimated by SRK and these are shown in Table 16.19.

Table 16.19: Groundwater Inflow Rates

Deposit	Peak Inflow Rate (m^3/d)	Average Inflow Rate (m^3/d)
Umwelt	1,000	550

Source: SRK 2015

To control groundwater inflows, twin 43 kW submersible pumps are planned to be installed at dewatering sumps at 90 m vertical intervals with staged pumping to surface. The second pump at each sump would be used as a backup pump in case of failure and when peak inflows occur. Drain holes are designed to be used in the access cross-cut sumps, draining to the closest dewatering sump. Mine water is planned to be pumped from the portal with a booster pump via surface pipeline to the water treatment facility.

16.3.7.13 Communications

A leaky feeder communication system would be used as the communication system for underground operations. Key personnel, such as mobile mechanics, crew leaders, shift supervisors and mobile equipment operators, would be supplied with an underground radio to contact the leaky feeder network.

Femco mine telephones are planned to be located at key infrastructure locations in the mine, such as refuge stations. The mine telephones would be independently powered and would also operate in the absence of mine power.

16.3.7.14 Explosives Storage and Handling

The primary explosives storage facility is planned to be located on surface. Secondary facilities would be located underground to fulfill explosives need for up to seven days. Day boxes would be used as temporary storage for daily consumption.

Two underground magazines are planned to separately store bulk explosives and detonators. Each magazine would be located in a bay off the decline. Access would be controlled with lockable gates. The magazines would be equipped with fire extinguishers, wooden shelves, and concrete flooring.

ANFO is planned to be used as the main explosive for mine development and stoping. Packaged emulsion would be used as a primer and for loading lifter holes in development headings. Smooth blasting techniques can be used as required in main access development headings; trim powder can be used to load the perimeter holes.

During the pre-production period, blasting in the development headings would be done at any time during the shift when the face is ready for blast. All underground personnel would be required to relocate to a designated safe work area during blasting. During the production period, a central blast system is planned to be used to initiate blasts for all loaded development headings and production stopes at the end of the shift.

Explosives handling, loading and detonation are planned to be carried out by trained and authorized personnel.

16.3.7.15 Fuel Storage and Distribution

A fuel station for underground mobile equipment is planned to be located near the mine portal. Haul trucks, LHDs, and smaller mobile equipment would be refueled at the beginning of each shift at surface. The fuel tank would be refilled by site services on a regular basis.

No permanent underground fuel and lube stations are planned. A Maclean FL3 fuel and lube truck is planned to have a capacity of 4,500 L of fuel and 1,000 L of oil and lube. Mobile equipment such as bolters and jumbos would be refilled by the fuel and lube truck.

Day tanks for diesel fuel are planned to be installed for the mine air heaters. The tanks would be refueled by site services.

Fuel consumption for mobile and stationary underground equipment was estimated based on equipment operating hours, engine fuel consumption, load factors, and utilization. Annual fuel consumption, including fuel for mine air heating, is shown in Table 16.20.

Table 16.20: Annual Fuel Consumption

Deposit	Annual Fuel Consumption (L/a)	
	Average	Maximum
Umwelt	4,563,000	5,645,000

Source: JDS 2015

16.3.7.16 Underground Transportation

All mine supplies and personnel would access the underground work areas via the decline.

Mobile equipment, such as LHDs, haul trucks, and supervisor vehicles, are planned to be parked on surface between shifts. Workers, who are not operating this equipment, would be transported to the underground work areas via a personnel carrier.

Materials and supplies are planned to be delivered to the active underground workings or storage bays by a Maclean BT3 boom truck, a Caterpillar TH407C telehandler, or a utility vehicle operated by the nipper.

16.3.7.17 Surface Infrastructure

Next to the portal, a laydown pad is planned to be constructed; this would be used for short-term storage of underground consumables, mobile equipment parking, and permanent surface infrastructure such as a fuel station, E-House, and an emergency shelter. Temporary ore and backfill waste stockpiles as well as the main ventilation fans and mine air heaters would also be located in this area.

All other major infrastructure are planned to be located at the central truck shop, which includes the mine dry, mine offices, warehouse, and mechanical shops.

16.3.8 Mine Safety

16.3.8.1 Fire Prevention

Fire extinguishers would be provided and maintained in accordance with regulations and best practices at the underground refuge stations, electrical substations, pump stations, fueling stations, explosives magazines, and other strategic areas. Every vehicle would carry at least one fire extinguisher; the correct size and type would depend on the type of vehicle. Underground heavy equipment would be equipped with automatic fire suppression systems.

16.3.8.2 Mine Rescue

A fully trained and equipped mine rescue team is essential to the safe operation of any mine. The mine rescue team would be trained for surface and underground emergencies.

16.3.8.3 Refuge Stations

Self-contained portable refuge stations would be provided in the main underground work areas. The refuge stations are designed to be equipped with compressed air, potable water, and first aid equipment. They would also be supplied with a fixed telephone line and emergency lighting. The refuge chambers would be sealable to prevent the entry of gases. The portable refuge stations are planned to be moved to new locations as the work areas advance; this eliminates the need to construct permanent refuge stations.

16.3.8.4 Emergency Egress

The main decline is planned to provide primary egress from the underground workings. The FAR system with a dedicated manway would provide the secondary egress in case of emergency. The manway would be equipped with steel ladders and platforms.

16.3.8.5 Emergency Stench System

A stench gas system would be installed on each fresh air intake and could be triggered to alert underground personnel in the event of an emergency.

16.3.8.6 Mobile Equipment

The selection of underground mining equipment is based on the mining method, drift and stope dimensions, production rate, and operating and capital costs. Since the overall LOM is nearly ten years, it was assumed that only new equipment would be purchased. Over time, the equipment is planned to be rebuilt or replaced, as recommended by the manufacturers.

A summary of selected mobile equipment for the underground operation is shown in Table 16.21.

Mining equipment makes and models are for reference purposes only and were not part of a comprehensive competitive selection process.

Table 16.21: Mobile Equipment Summary

Description	Make	Model	Overhaul Frequency (hrs)	Replacement Frequency (hrs)	Estimated Mechanical Availability
Haulage Truck (30 t)	Sandvik	TH430	14,000	28,000	85%
LHD 4.5 m³ (10 t)	Sandvik	LH410	14,000	28,000	85%
LHD 3.7 m³ (6.7 t)	Sandvik	LH307	14,000	28,000	85%
Jumbo (2 boom)	Sandvik	DD421-60	12,500	25,000	75%
Production Drill Small	Boart Longyear	Stopemaster	12,500	25,000	75%
Diamond Drill	Boart Longyear	LM55	15,000	30,000	75%
Rockbolter	MacLean	928 Scissor	12,500	25,000	75%
Shotcrete Machine	MacLean	SS3	12,500	25,000	90%
ANFO Loader	MacLean	AC2 ANFO	15,000	30,000	90%
Boom Truck	MacLean	BT3	15,000	30,000	90%
Fuel-Lube Truck	MacLean	FL3	15,000	30,000	90%
Scissor Truck	MacLean	SL3	14,000	28,000	90%
Supervisor Vehicle	Toyota	Landcruiser	15,000	30,000	90%
Electrician Vehicle	Toyota	Landcruiser	15,000	30,000	90%
Personnel Carrier	Toyota	Landcruiser	15,000	30,000	90%
Utility / Nipper	Toyota	Landcruiser	15,000	30,000	90%
Mechanics Truck	Toyota	Landcruiser	15,000	30,000	90%
Portable Welder	Lincoln Electric	Classic 300 he	14,000	28,000	90%
Grader	CAT	12M	14,000	28,000	90%
Forklift/Telehandler	CAT	TH407C	14,000	28,000	90%
FEL (Surface)	CAT	966H	14,000	28,000	90%

Source: JDS 2015

Equipment requirements were developed based on the scheduled quantities of work and estimated from first principles. Operational efficiencies and mechanical equipment availability factors were included in the calculations.

A summary of the LOM equipment requirements is shown in Table 16.22.

Table 16.22: Annual Mobile Fleet Requirements

Year	Units	LOM Total	2	3	4	5	6	7	8	9
Description										
Haulage Truck (30 t)	#	**6**	2	5	6	6	6	5	5	5
LHD 4.5 m³ (10 t)	#	**2**	1	2	2	2	2	2	2	2
LHD 3.7 m³ (6.7 t)	#	**2**	0	1	2	2	2	2	2	2
Jumbo (2 boom)	#	**2**	1	2	2	2	2	2	2	2
Production Drill Small	#	**1**	0	1	1	0	0	0	0	1
Diamond Drill	#	**1**	0	1	0	0	0	0	0	0
Rockbolter	#	**4**	1	4	4	4	4	4	4	4
Shotcreting Machine	#	**1**	1	1	1	1	1	1	1	1
ANFO Loader	#	**2**	1	2	2	2	2	2	2	2
Boom Truck	#	**1**	1	1	1	1	1	1	1	1
Fuel-Lube Truck	#	**1**	1	1	1	1	1	1	1	1
Scissor Truck	#	**1**	1	1	1	1	1	1	1	1
Supervisor Vehicle	#	**1**	1	1	1	1	1	1	1	1
Electrician Vehicle	#	**1**	1	1	1	1	1	1	1	1
Personnel Carrier	#	**1**	1	1	1	1	1	1	1	1
Utility / Nipper	#	**1**	1	1	1	1	1	1	1	1
Mechanics Truck	#	**1**	1	1	1	1	1	1	1	1
Portable Welder	#	**1**	1	1	1	1	1	1	1	1
Grader	#	**1**	1	1	1	1	1	1	1	1
Forklift/Telehandler	#	**1**	1	1	1	1	1	1	1	1
FEL (Surface)	#	**1**	1	1	1	1	1	1	1	1

Source: JDS 2015

A centralized maintenance facility is planned for the Goose Site, which would be shared between the open pit and underground operations. Mechanics' trucks would be used to perform small maintenance and emergency repairs underground. Where further maintenance is required, the equipment would be returned to the central truck shop.

16.3.9 Mine Personnel

The underground mine is planned to operate on two 11-hour shifts (day shift/night shift), 365 days per year, with four crews on rotation. Two crews would be on site at any one time, one on day shift and one on night shift, with the other crews off-site on break. Hourly mining and maintenance personnel would work a two week on, two week off (2x2) rotation. Salaried supervisors and technical staff would work on the same 2x2 rotation.

Peak underground mining personnel requirements are summarized in Table 16.23.

Table 16.23: Underground Mine Operations Personnel - Peak Quarter and Average

Position	Peak Quantity	Avg. Quantity	Hourly/Salary
Mining Operations			
Mine Supervisor/Shift Boss	4	4	Salary
Coverage Miner	4	2	Hourly
Long-hole Drill Operator	4	0	Hourly
Jumbo Operator	8	5	Hourly
Ground Support/Bolter/Shotcrete	32	25	Hourly
Development Service	8	8	Hourly
Blaster	4	4	Hourly
LHD Operator	16	9	Hourly
Truck Driver	24	18	Hourly
Backfill/Construction	2	2	Hourly
Utility Vehicle Operator/Nipper	8	8	Hourly
Backfill FEL Operator	4	4	Hourly
Maintenance			
Heavy Duty Mechanic	14	13	Hourly
Electrician	6	6	Hourly
Site Technical Services			
Surveyor/Mine Technician	4	4	Salary
Grade Control Technician	2	2	Salary
Diamond Driller	4	1	Salary
Diamond Drill Geologist	2	0	Salary
Total Underground	**142**	**114**	

Source: JDS 2015

Hourly personnel were estimated based on development and production rates, operation productivities, and maintenance requirements.

16.3.10 Underground Development Schedule

Mine development is scheduled in two phases: pre-production development (prior to ore production) and ongoing development (during ore production).

The objective of pre-production development is to provide access to production areas and prepare enough ore to support the mine production rate and to also establish access to the lower levels. Pre-production development is scheduled to support the following:

- Provide access for trackless equipment;
- Establish primary ventilation circuits and emergency egress;
- Install mining services (power distribution, dewatering, explosives magazines, etc.); and
- Provide sufficient access to production areas to achieve and maintain the targeted mine production rate.

Development schedules were based on estimated cycle times for jumbo development and best practices for North American mining operations.

All waste development during pre-production is shown as capital development. During the production phase, the decline, ventilation drifts, and raises would be considered capital development but cross-cuts and drifting on ore were included in the operating costs.

Annual development metreage for the underground operation is summarized in Table 16.24.

Table 16.24: Development Schedule

Year	Total	1	2	3	4	5	6	7	8	9
Capital Lateral (m)	14,456	0	2,363	5,189	1,837	899	1,245	1,150	846	926
Capital Vertical (m)	1,485	0	354	903	200	0	0	0	0	28
Operating Lateral (m)	74,105	0	0	4,849	11,155	11,910	12,044	11,877	12,233	10,037

Source: JDS 2015

16.3.11 Underground Production Schedule

UG production is considered to have started as soon as first ore is mined. For the Umwelt underground operation, this is when the first production cut is taken.

The following strategies were used for production scheduling:

- In the early stages of mine life, improve Project economics by targeting mining blocks with higher grade mineralization; and
- Start production of the mining blocks from the bottom-up.

16.4 Project Production Schedule

16.4.1 Combined Open Pit & Underground Production Schedule

The open pit and underground mine production schedule for the Back River deposits incorporates the deposits at Goose (Umwelt, Llama, Goose Main). The mill-feed tonnage is planned to be simultaneously provided by a series of open pit mines and an underground mine. Due to the limited production capacity of the underground mine, it alone would not be able to produce the 3,000 tpd mill-feed requirement. Hence there is a need to combine the open pit mining to provide supplemental ore feed in conjunction with the underground mine.

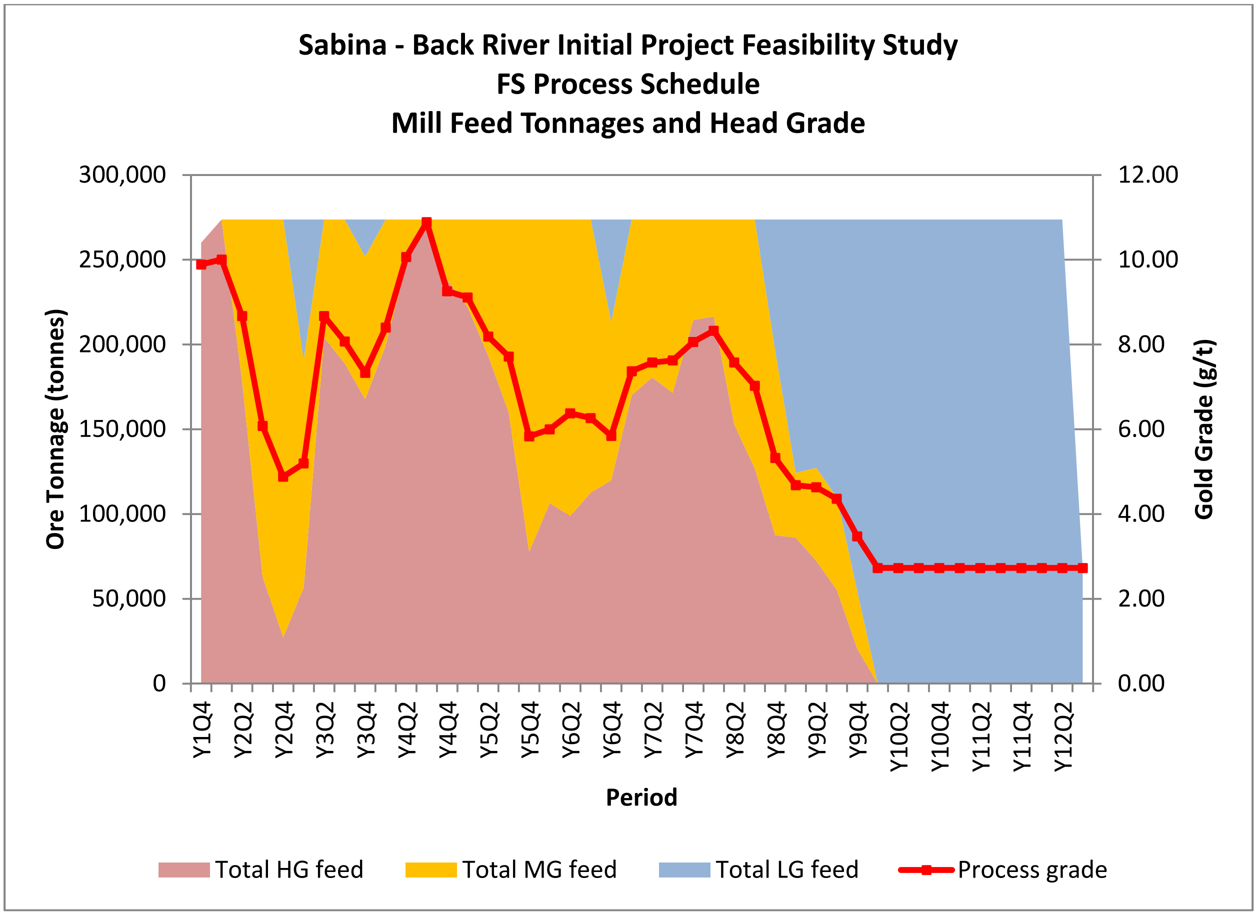
Table 16.25 summarizes the combined LOM production schedule for the Back River Project including the open pit and underground mines and the mill feed schedule and stockpile balances.

Table 16.25: Combined LOM Production Schedule

			YEAR SUMMARY												
Description	**Unit**	**Total**	**Y-1**	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**Y7**	**Y8**	**Y9**	**Y10**	**Y11**	**Y12**
OPEN PIT															
Umwelt															
Waste	t	15,424,574	6,864,912	8,061,230	498,432										
Total Ore	t	2,668,257	481,439	1,637,022	549,796										
Avg grade	g/t	6.49	5.57	6.15	8.32										
SR	w:o	5.78	14.3	4.9	0.9										
Total Material	t	18,092,831	7,346,350	9,698,252	1,048,228										
LLama															
Waste	t	30,389,273		3,956,404	12,505,337	10,599,096	3,235,010	93,427							
Total Ore	t	1,748,747		16,846	191,678	984,841	496,836	58,545							
Avg grade	g/t	7.15		8.43	5.77	6.79	7.96	10.52							
SR	w:o	17.38		234.9	65.2	10.8	6.5	1.6							
Total Material	t	32,138,020		3,973,250	12,697,015	11,583,936	3,731,846	151,972							
Goose Main															
Waste	t	47,285,489				1,384,523	8,728,302	12,573,270	13,210,307	9,477,134	1,911,952				
Total Ore	t	4,450,539				30,449	1,047,095	965,223	361,581	1,496,173	550,018				
Avg grade	g/t	5.00				5.75	4.98	5.22	4.26	4.74	5.80				
SR	w:o	10.62				45.5	8.3	13.0	36.5	6.3	3.5				
Total Material	t	51,736,028				1,414,973	9,775,397	13,538,493	13,571,888	10,973,307	2,461,970				
Open Pit Summary															
PAG	t	50,423,150	3,932,025	7,658,850	9,327,237	7,879,029	5,182,505	5,422,590	5,437,456	4,628,488	954,971				
NAG	t	42,676,183	2,932,886	4,358,786	3,676,531	4,104,589	6,780,805	7,244,111	7,772,851	4,848,644	956,981				
Total Waste		93,099,336	6,864,912	12,017,634	13,003,769	11,983,619	11,963,312	12,666,697	13,210,307	9,477,134	1,911,952				
Strip Ratio	w:o	10.50	14.26	7.27	17.54	11.80	7.75	12.37	36.53	6.33	3.48				
OP ore	t	8,867,542	481,439	1,653,868	741,475	1,015,290	1,543,931	1,023,769	361,581	1,496,173	550,018				
OP grade	g/t	5.87	5.57	6.17	7.66	6.76	5.94	5.52	4.26	4.74	5.80				
OP Au	ounces	1,674,721	86,227	328,292	182,685	220,535	294,898	181,751	49,566	228,145	102,623				
UNDERGROUND															
Umwelt UG															
Total Ore	t	3,491,504				224,954	533,420	565,522	564,516	560,320	569,335	473,437			
Avg grade	g/t	7.38				6.25	8.92	7.38	7.03	7.40	7.58	6.34			
Underground Summary															
UG ore	t	3,491,504				224,954	533,420	565,522	564,516	560,320	569,335	473,437			
UG grade	g/t	7.38				6.25	8.93	7.38	7.03	7.40	7.58	6.34			
UG Au	ounces	828,550				45,178	153,061	134,147	127,592	133,348	138,776	96,446			
Total OP/UG															
OP/UG ore	t	12,359,046	481,439	1,653,868	741,475	1,240,244	2,077,351	1,589,291	926,097	2,056,493	1,119,353	473,437			
Mine OP/UG grade	g/t	6.30	5.57	6.17	7.66	6.66	6.71	6.18	5.95	5.47	6.71	6.34			
OP/UG Au	ounces	2,503,271	86,227	328,292	182,685	265,713	447,959	315,898	177,159	361,493	241,399	96,446			
OP/UG Recovery	%	93.0%	92.0%	92.0%	91.8%	91.4%	93.3%	93.8%	93.2%	94.2%	93.5%	92.0%			
Mill-Feed Blender															
Total Mill feed from mine	t	7,300,623		685,525	501,383	963,758	1,093,139	942,482	723,760	1,095,000	822,139	473,437			
	g/t	8.24		9.56	9.74	7.75	9.65	8.20	6.87	7.66	8.03	6.34			
	ounces	1,933,842		210,779	157,039	240,239	339,127	248,605	159,776	269,590	212,240	96,446			
	%	92.7%		92.0%	91.8%	91.4%	92.8%	93.4%	92.9%	93.7%	93.1%	92.0%			
Total Mill feed from stockpile	t	5,058,424		108,350	593,617	131,242	1,861	152,518	371,240		272,861	621,563	1,095,000	1,095,000	615,171
	g/t	3.50		9.27	5.44	4.11	10.89	4.67	4.67		4.15	2.73	2.73	2.73	2.73
	ounces	569,429		32,284	103,758	17,353	651	22,889	55,713		36,387	54,487	95,989	95,989	53,927
	%	93.4%		92.0%	92.0%	91.9%	92.8%	93.8%	93.8%		94.1%	93.6%	93.6%	93.6%	93.6%
Total Mill Feed															
Process ore	t	12,359,046		793,875	1,095,000	1,095,000	1,095,000	1,095,000	1,095,000	1,095,000	1,095,000	1,095,000	1,095,000	1,095,000	615,171
Process grade	g/t	6.30		9.52	7.41	7.32	9.65	7.71	6.12	7.66	7.06	4.29	2.73	2.73	2.73
Process Au	ounces	2,503,271		243,062	260,797	257,593	339,778	271,494	215,490	269,590	248,628	150,933	95,989	95,989	53,927
Process Recovery	%	93.0%		92.0%	91.9%	91.4%	92.8%	93.4%	93.2%	93.7%	93.3%	92.9%	93.6%	93.6%	93.6%
Stockpile															
BALANCE															
HG ore stockpile	t		176,563												
MG ore stockpile	t		111,504		85,013		362,165	371,240		145,576					
LG ore stockpile	t		193,372	615,171	902,894	1,133,150	1,753,336	2,238,552	2,440,889	3,256,806	3,426,734	2,805,171	1,710,171	615,171	
Total in Stockpile	t		481,439	615,171	987,906	1,133,150	2,115,501	2,609,792	2,440,889	3,402,381	3,426,734	2,805,171	1,710,171	615,171	

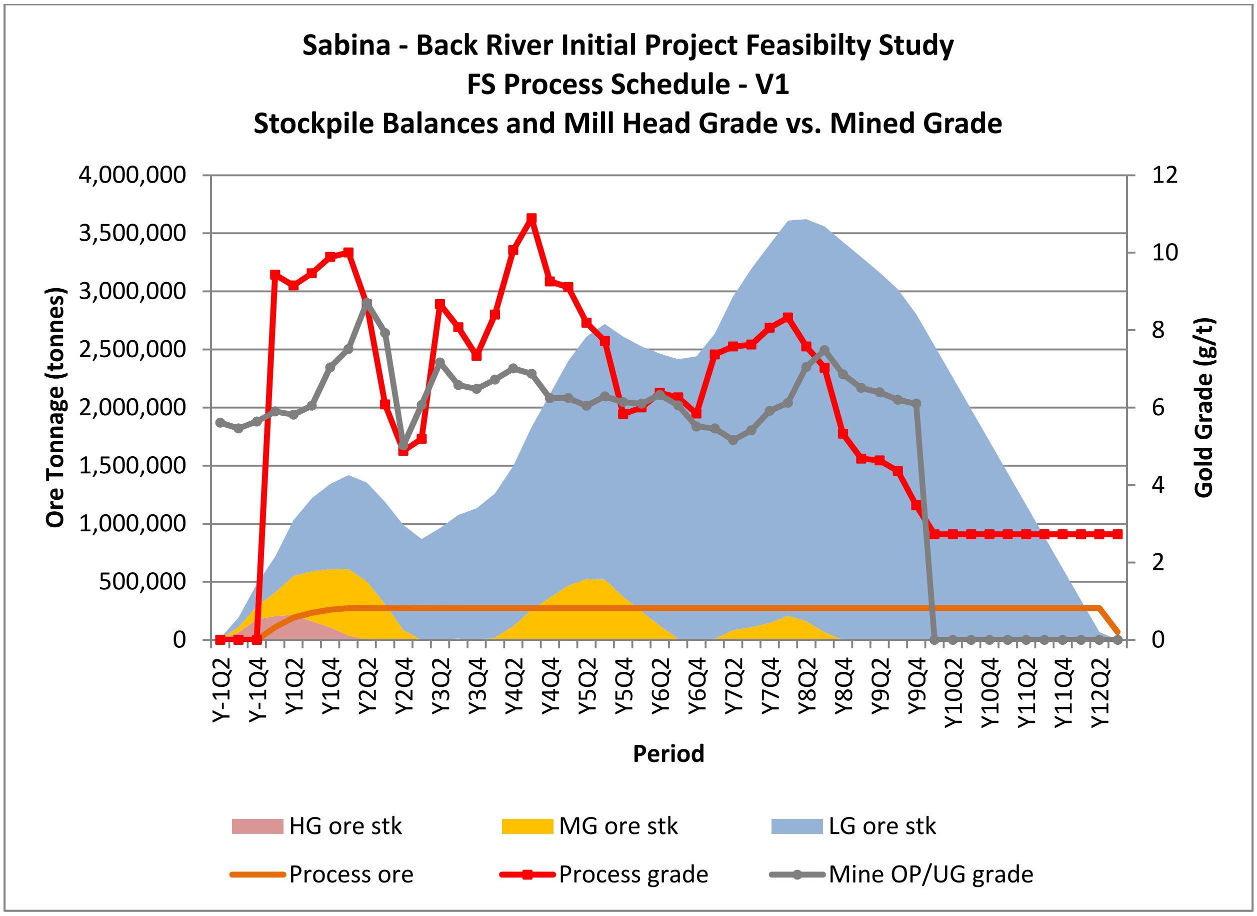
Figures 16.12 and 16.13 illustrate the combined LOM processing schedule for the Project including mill-feed grades and tonnages by grade bin, along with stockpile balances and mined versus mill grades. The graphs clearly indicate the benefits of a grade-bin stockpiling strategy and processing high grade ore early in the mine life. This in turn translates into economic benefits. This strategy, however, necessitates the extraction of ore and waste at a rate beyond what the mill can handle at start-up, but pays for itself by advancing considerable revenue in the early years enhancing the rate at which capital is paid back.

Figure 16.12: Process Plant Schedule and Head Grade



Source: JDS 2015

Figure 16.13: ROM Stockpile Balance



Source: JDS 2015

17 Process Description/Recovery Methods

17.1 Introduction

This section of the report describes recovery methods used in the Back River process facility.

The results of the metallurgical test work described in section 13 together with financial evaluation data were used to develop metallurgical design criteria, which in turn were used to design the process facility described in this report section. The majority of this section was previously written by Hatch for the NI 43-101 report "Technical Report and Feasibility Study for the Back River Gold Property, Nunavut" published in June 2015. Canenco has updated and edited that version for this technical report based on the design changes undertaken.

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This section of the report contains estimates, projections and conclusions that are forward-looking information within the meaning of applicable securities laws. Forward-looking statements are based upon the responsible QP's opinion at the time that they are made but in most cases involve significant risk and uncertainty. Although the QP has attempted to identify factors that could cause actual events or results to differ materially from those described in this report, there may be other factors that cause events or results to not be as anticipated, estimated or projected. There can be no assurance that forward-looking information in this section of the report will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements or information. Accordingly, readers should not place undue reliance on forward-looking information. Forward-looking information is made as of the effective date of this report, and the QP does not assume any obligation to update or revise it to reflect new events or circumstances, unless otherwise required under applicable laws.

17.2 Summary

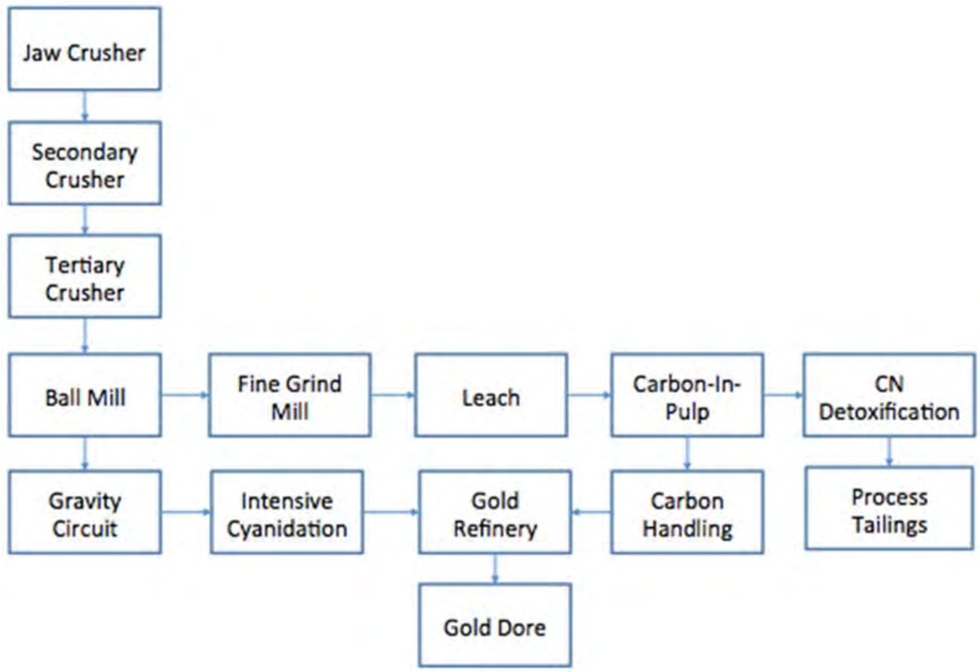
The process selected for the Back River Project is based on test work described in Section 13 and consists of a whole-ore cyanide leach and carbon adsorption process comprising crushing, grinding, gravity concentration, cyanide leaching, carbon adsorption, cyanide detoxification, carbon elution and regeneration, gold refining, and tailings disposal.

The mill is designed with a nominal capacity of 3,000 t/d at a planned average feed grade of 6.30 g/t gold (Au). The crushing circuit will operate at an availability of 70%. The milling and leaching circuits will operate 24 hours per day, 365 days per year at an availability of 90%.

The plant will consist of the following unit operations:

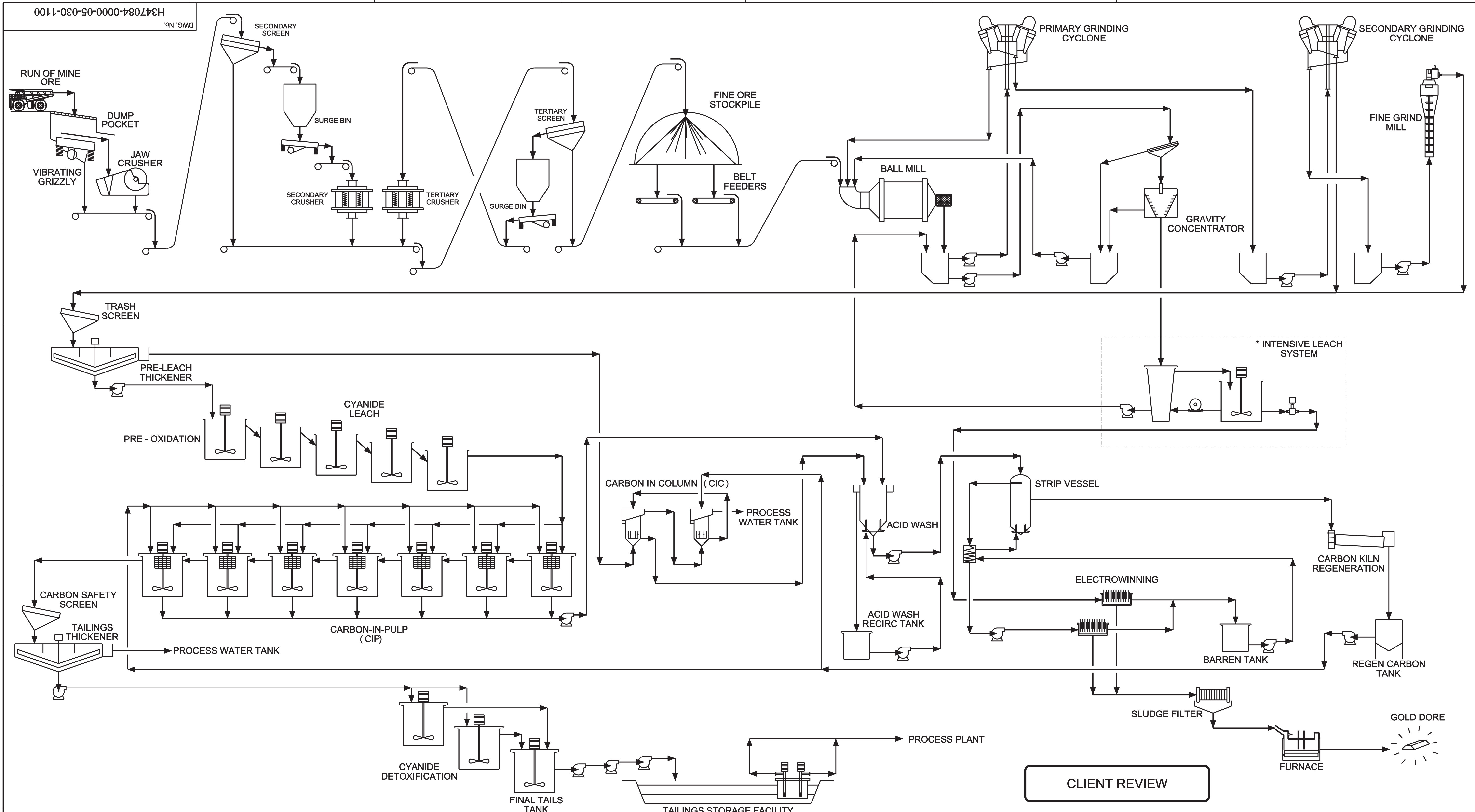
- Primary Crushing – A vibrating grizzly and jaw crusher in open circuit producing a final product P80 of 122 mm;
- Secondary Crushing – A vibrating double deck banana screen and cone crusher operating in closed circuit producing a final product P80 of 22 mm;
- Tertiary Crushing – A vibrating double deck banana screen and cone crusher operating in closed circuit producing a final product P80 of 8.5 mm;
- Fine Ore Stockpile and Reclaim – A 24 h live storage, covered, fine ore stockpile with two reclaim belt feeders feeding the ball mill feed conveyor;
- Primary Grinding – A ball mill in closed circuit with hydrocyclones producing a final product P80 of 180 µm;
- Secondary Grinding – A fine grind mill in open circuit with hydrocyclones producing a final product P80 of 50 µm;
- Gravity Concentration – Gravity concentration of cyclone underflow from the primary grinding circuit to produce a gold-rich concentrate for intensive leach;
- Intensive Cyanidation – Gravity gold dissolution within the intensive cyanidation reactor for gold recovery in electrowinning;
- Cyanide Leaching and Carbon Adsorption – Gold leaching by cyanidation, facilitated by oxygen, followed by adsorption of solution gold onto carbon particles;
- Cyanide Detoxification – Detoxification of cyanide slurry via sodium metabisulphite for SO_2, air and copper sulphate to <1 ppm CN_{WAD} (weak acid dissociable);
- Carbon Elution and Regeneration – Acid wash of carbon to remove inorganic foulants, elution of carbon to produce a gold-rich solution, and thermal regeneration of carbon to remove organic foulants; and
- Gold Refining – Gold electrowinning (sludge production), filtration, drying, and smelting to produce gold doré.

Figure 17.1: Block Summary Flow Diagram



Source: Canenco 2015

A summary of the process flowsheet appears as Figure 17.2. Models of the crushing and process facilities are given in Figure 17.3 and Figure 17.4, respectively.



HATCH

Sabina GOLD & SILVER CORP

ROLE	NAME	SIGNATURE	DATE
DRAFTSPERSON		NR	
DESIGNER		NR	
CHECKER			
DESIGN COORD.			
RESP. ENG.			
LEAD DISC. ENG.			
ENG. MANAGER			
PROJ. MANAGER			
CLIENT			

DRAWING APPROVAL STATUS: Client Review

No.	DESCRIPTION	BY	CHK'D	DATE
A	ISSUED FOR CLIENT REVIEW	–	–	2015/09/–

REVISIONS

DRAWING No.	DRAWING TITLE

REFERENCE DRAWINGS

REGISTERED PROFESSIONAL

BACK RIVER LOW CAPEX STUDY

GENERAL
OVERALL
PROCESS FLOW DIAGRAM

SCALE	DWG. No.	REV
NTS OR AS NOTED	H347084-0000-05-030-1100	A

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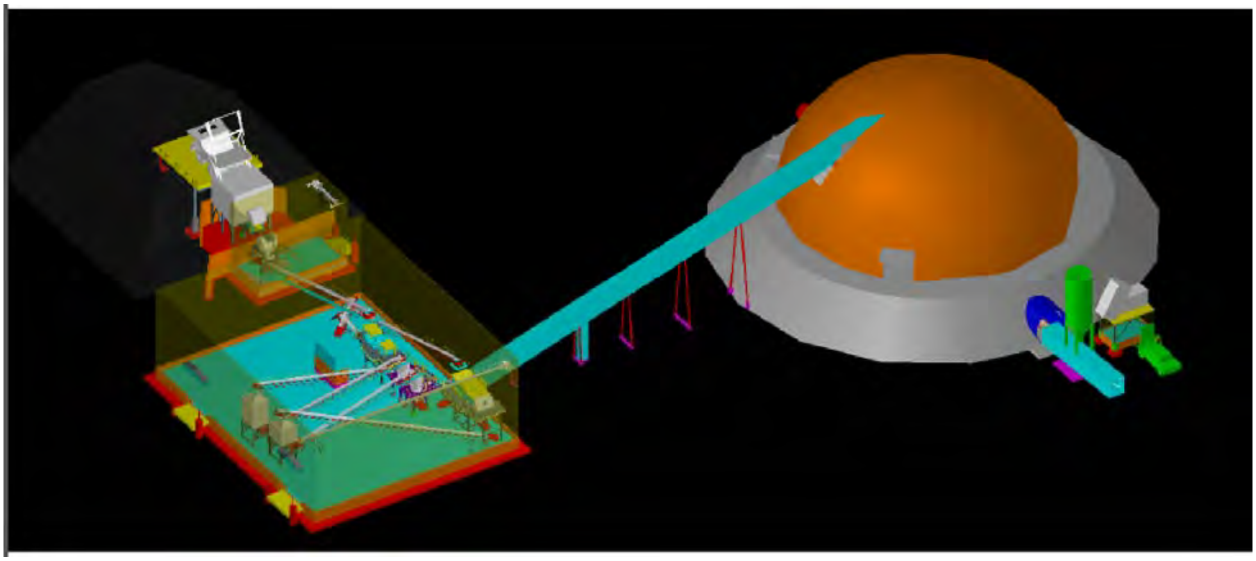
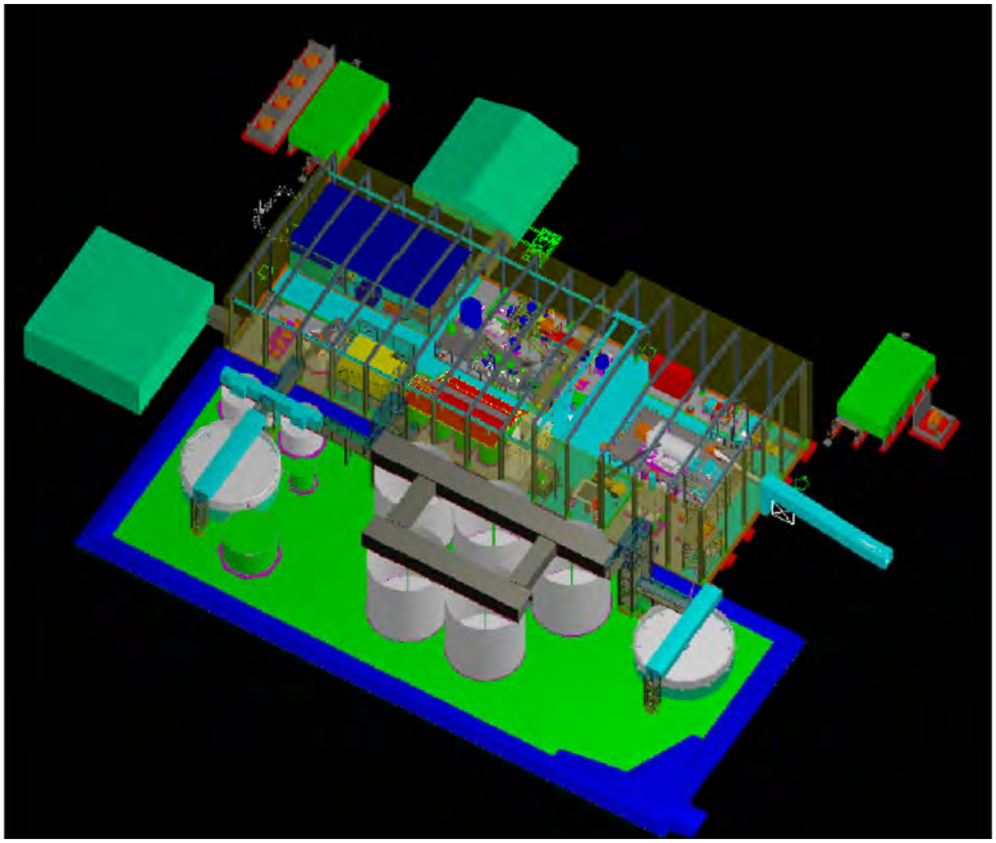
Figure 17.3: Crushing and Ore Storage 3D Model



Source: Canenco 2015

Figure 17.4: Process Plant 3D Model



Source: Canenco 2015

17.3 Process Design

17.3.1 Process Design Criteria

The Process Design Criteria and Mass Balance detail the annual ore and product capabilities, major mass flows and capacities, and plant availability. Consumption rates for major operating and maintenance consumables can be found in the operating cost estimate described in section 22. Key process design criteria are given in Table 17.1.

Table 17.1: Major Process Design Criteria

Area	Criteria	Unit	Nominal Value
General	Gold	g/t	6.28
	Silver[1]	g/t	2.3
	Daily throughput	t/d	3,000
	Process plant availability	%	90
	Overall Gold recovery	%	93.3
Crusher System	Availability/Utilities	%	70
	Crusher work index	kWh/t	14.6
	Number of crushing stages	-	3
	Crushing System product size (P_{80})	mm	8.5
Fine Ore Storage	Capacity (live)	t	3,400
	Capacity (total)	t	11,687
	Capacity (live)	h	27
Grinding	Bond Ball Mill work index (212µm)	kWh/t	15.8
	Bond Ball Mill work index (53µm)	kWh/t	17.8
	JKTech A×b		28.3
	Ball mill product size (P_{80})	µm	180
	Fine grind mill product size (P_{80})	µm	50
Gravity Recovery	Gold Recovery	%	39.9
	Feed percentage to gravity circuit	%	40
Screening and Thickening	Feed density	% w/w	35
	Thickener underflow density	% w/w	50
Leaching and Carbon-In-Pulp (CIP)	Leach retention	h	48.5
	CIP Carousel residence time	h	1.5
	Leach slurry feedrate	m^3/h	187
	CIP slurry feedrate	m^3/h	216
	CIP carbon retention time	d	24
	CIP carbon concentration	g/L	50
	Loaded carbon grade	g/t	7,779
Tailings Thickening	Thickener feed density	% w/w	45
	Thickener underflow density	% w/w	60
Cyanide Destruction	Discharge solution CN_{WAD}	mg/L	<1.0
	Residence time	h	1.6
Carbon Treatment	Acid Used	-	HCl
	No. of Acid Wash Vessels	-	1
	Acid Wash Batch Size	t	1.5
	Number of elution vessels	-	1
	Elution batch size	t	1.5
Electrowinning and Refining	EW recovery	%	99

Note: Silver is not included in the precious metal economics

Source: Canenco 2015

17.4 Process Plant Description

17.4.1 Primary Crushing

Ore from open pit and underground mining operations will feed a vibrating grizzly - primary jaw crusher system, which produces a product size P80 of 122 mm.

Feed material to the crusher system will be run of mine hauled by 64 t haul trucks from the run of mine stockpile or from the mines. Material will be stockpiled near the jaw crusher or direct dumped through a static grizzly into a dump pocket. Stockpiled material will be re-handled. Oversize material from the static grizzly will be removed, for later size reduction, using mobile machinery.

A vibrating grizzly feeder will draw material from the dump pocket. The spacing between the rails on the grizzly feeder will be 110 mm. The vibrating grizzly oversized material will discharge directly into the primary jaw crusher. The undersized material will bypass the crusher and feed directly onto the primary crusher discharge conveyor.

17.4.2 Secondary Crushing and Screening

The secondary screen feed conveyor will collect primary crusher product and feed onto a double deck vibrating banana screen. The top deck aperture will be 60 mm and a bottom deck aperture will be 30 mm. The oversize from the secondary screen will convey to the secondary crusher surge bin.

Ore from the secondary crusher feed bin will be reclaimed by a belt feeder controlled by a variable speed drive into a secondary cone crusher. The secondary crusher will reduce the ore to a nominal product size P80 of 22 mm with a closed side setting (CSS) of 22 mm.

17.4.3 Tertiary Crushing and Screening

The tertiary screen feed conveyors will collect secondary screen undersize, secondary crusher product, and tertiary crusher product and convey the combined feed onto a double deck vibrating banana screen. The top deck aperture will be 20 mm and a bottom deck aperture will be 10 mm, achieving a final undersize product P80 of 8.5 mm, which will be conveyed to the fine ore stockpile. The oversize from the tertiary screen will discharge onto the tertiary screen oversize conveyors into the tertiary crusher surge bin.

A belt feeder controlled by a variable speed drive will reclaim material from the tertiary crusher feed bin. The surge bin will regulate the feed into the tertiary crusher to maintain a choke feed operation. The tertiary crusher has a CSS of 13 mm.

17.4.4 Fine Ore Stockpile and Reclaim

The ore storage facility will consist of a dome-covered stockpile with two in-line belt feeders located within a corrugated pipe reclaim tunnel. The belt feeders will transfer ore to the conveyor feeding the ball mill.

The fine ore stockpile will have a 3,400 t live capacity that can support process plant operations for 27 h when the crushing plant is not operating. The total capacity of the stockpile is 11,687 t which corresponds to approximately 3 days storage. Each belt feeder is capable of providing the total throughput to the plant when required. Lime, for pH management, will be added to the reclaim conveyor from a lime silo via a screw feeder.

17.4.5 Grinding

The grinding circuit will consist of a ball mill operating in closed circuit with a hydrocyclone cluster and a fine grind mill operating in open circuit with a hydrocyclone cluster. Material from the fine ore stockpile will be fed to the ball mill via the ball mill feed conveyor. The grinding circuit will operate at a nominal throughput of 139 t/h (fresh feed), and produce a final particle size P80 of 50 µm. The ball mill will be 4.6 m in diameter by 5.6 m effective grinding length driven by a 1.9 MW motor.

Water will be added to the ball mill to maintain the ore charge in the mill at a constant slurry density. Slurry will overflow from the ball mill to a trommel screen, attached to the ball mill discharge end. The ball mill trommel screen oversize will overflow into a trash bin for removal from the system.

The ball mill hydrocyclone cluster will classify the feed slurry into coarse and fine fractions. The coarse underflow will feed the ball mill for additional grinding. The fine overflow with a nominal 80% passing 180 µm will flow by gravity to the fine grind mill cyclone feed pumpbox for classification prior to additional grinding. The ball mill hydrocyclones have been designed with a 300% recirculating load. Additional grinding will be performed by a 1.6 MW fine grinding mill.

A portion of the ball mill discharge will be pumped to a trash screen which in turn feeds a gravity concentrator circuit of 100 t/h capacity.

17.4.6 Gravity Concentration

The gravity recovery and intensive leach circuits will consist of a single gravity concentrator with a feed trash screen, gravity tails pumpbox and tails pump, feeding a concentrate hopper, and a skid-mounted intensive leach reactor. The overall target gold recovery is 39.9% or 1.7 t/d of concentrate at a grade of 4,900 g/t.

The scalping screen prior to the gravity concentrator removes coarse particles and/or metal pieces that would otherwise fill the concentrator with lower grade material, reducing the capacity, and/or damaging the concentrator. Scalping screen oversize will be directed by a launder to the gravity tailings box. Periodically, the centrifugal concentrator will be bypassed and switched to flushing mode to recover the collected concentrate.

Gravity concentrate will be leached to dissolve gold by contacting material with the leach solution in an intensive leach system. The leach solution will include sodium cyanide and caustic solution. The leach solution will be mixed with the gravity concentrate in the reactor feed tank.

At the completion of the batch leach cycle, the resulting gold-rich pregnant solution will be pumped to the gold refinery for gold recovery using electrowinning cells.

The leached concentrate (intensive leach reactor tails) along with decanted pre-wash concentrate fines will be pumped to the ball mill cyclone feed pump boxes for return to the grinding circuit.

17.4.7 Thickening and Carbon-in-Column (CIC)

The fine grind mill product and secondary cyclone overflow will flow by gravity to a vibrating trash screen for removal of trash material. The vibrating screen will have an area of 3.7 m^2 and have a screen opening of 600 µm.

The undersize from the screens will flow by gravity to the pre-leach thickener. Flocculant solution (anionic polyacrylamide) will be added to the thickener feed to promote the settling of fine solids. The pre-leach thickener will have a diameter of 18 m and produce a thickened product of 50% solids for the leach circuit. The CIP circuit will add process water to allow carbon adsorption to occur at 45% solids density.

The thickened slurry from the pre-leach thickener will be pumped to the leach circuit for continuous cyanide leaching. Thickener overflow solution will flow by gravity to a stand-pipe and pumped either to the process water tank or to the CIC tanks and then pumped again to the process water tank.

Because the process water, which will be recirculated within the process plant, contains cyanide from the tailings thickener, the grinding process will occur in a low-concentration cyanide solution. Grinding in cyanide will provide the slurry with a well-mixed and aerated environment.

This will cause some gold to leach out of the solids and into solution prior to the process slurry entering the CIP tanks. Two CIC tanks will be installed on the pre-leach thickener overflow to break the recirculation of gold in solution.

A controlled portion of the pre-leach thickener overflow passes through a linear trash screen and into two open, upflow CIC columns. Each column will contain 2.6 t of 6 x 12 mesh activated carbon and will operate as an expanded bed contactor. The two columns will be arranged for gravity flow from the first into the next column (stepped columns).

A flow rate of 86 $m^3/h/m^2$ will be maintained to sufficiently fluidize the bed of activated carbon without overflowing the carbon out of the column. To minimize carbon attrition, a recessed impeller pump will be used periodically to transfer carbon counter-current up the carbon train. As the carbon progresses up the circuit, the carbon from the first column, which has been fully loaded with gold, will be sent to the acid wash tank. Carbon from the sizing screen above the CIP tanks will discharge directly into the second CIC column.

The CIP circuit tailings slurry will be pumped to the tails thickener feed tank and will be washed with reclaim water to reduce the overall cyanide reporting to the cyanide detoxification circuit. Flocculant will be added to the tailings thickener to promote settling of fine solids.

17.4.8 Cyanide Leaching and Carbon Adsorption

The pre-leach thickener underflow will be pumped to a 14.5 m diameter by 15.5 m high pre-oxidation tank prior to being leached in four similar sized leach tanks. The pre-oxidation tank will oxidize some sulphide material to reduce cyanide consumption and improve gold recoveries. The leach circuit will increase gold concentration in the solution prior to contact with activated carbon in the CIP circuit. The leach circuit is designed to provide 48.5 h retention time, with an additional 1.5 h in the CIP tanks. All leach tanks will be located outside and adjacent to the main process building.

The circuit will be operated as a single train. The first tank will be utilized as a pre-oxidation tank, and lead nitrate will be dosed to assist with oxidizing sulphides in order to reduce cyanide consumption and improve recovery.

The solution returning to grinding also will contain cyanide, so there will be additional well agitated and aerated leaching occurring in the mills prior to the leach tanks. Lime slurry will be added to the first and second leach tanks to maintain protective alkalinity at a design pH of 11.0 to prevent evolution of hydrogen cyanide gas (HCN). Oxygen will be sparged from the bottom of the leach tanks at an aeration rate of 0.05 $Nm^3/h/m^{3.}$

Leached ore slurry from the final leach tank will flow by gravity to the CIP Carousel circuit for carbon adsorption. Dissolved gold and silver will be adsorbed onto activated carbon in the CIP tanks.

The CIP Carousel circuit is designed to provide a total slurry retention time of 1.5 h. The CIP Carousel circuit is a modification of the traditional CIP circuit. Leached slurry feeds a distribution launder. The distribution launder, using valving and piping, can feed any of the CIP tanks. There will be a carbon inventory in each of the CIP tanks, but they will not be pumped counter-current to the slurry flow. Instead, when CIP tank 1 carbon is loaded, the distribution launder will send fresh leach slurry to CIP tank 2. CIP tank 1 loaded carbon is then pumped to the carbon handling plant for gold refining, and newly regenerated carbon will fill CIP tank 1. CIP tank 1 will become the tails tank, and CIP tank 2 will be the head tank. This practice is continued through the tanks with CIP tank 1 eventually becoming the head tank once again. There will be an option to pump carbon slurry from a CIP tank to the first leach tank if preg-robbing ore is encountered.

Each CIP tank will have a single inter-stage screen/agitiator to retain carbon particles in the tank and to allow discharge of slurry to the next tank. All CIP tanks will be at the same elevation.

The average carbon concentration in the CIP Carousel circuit is expected to be 50 g/L. As the slurry proceeds through the circuit, metal values in the solids and solution will progressively decrease. Carbon will leave the first CIP tank once metal loading reaches about 7,779 g/t. In the CIP tank in the carousel that is acting as the head tank, a loaded carbon pump will pump slurry containing carbon to the loaded carbon screen. Loaded carbon will be collected and transferred to the acid wash tank on a daily basis. The tailings stream from the CIP Carousel circuit will flow by gravity onto a carbon safety screen to a stationary safety screen to capture any carbon particles that may have escaped from the final CIP tank.

Captured carbon particles will be collected in bins and disposed of periodically. Safety screen undersize will then be pumped to the tailings thickener for dewatering prior to CN detoxification. The tailings thickener has water added to it, which allows for some recovery of residual cyanide.

17.4.9 Carbon Acid Wash, Elution, and Regeneration

17.4.9.1 Carbon Acid Wash

Loaded carbon will be treated with hydrochloric acid solution in the acid wash tank to remove calcium deposits, magnesium, sodium salts, silica, and fine iron particles. Organic foulants such as oils and fats are unaffected by the acid and will be removed after the elution step by thermal reactivation utilizing a kiln.

The carbon will first be rinsed with fresh water. Acid will then be pumped from the acid wash circulation tank to the acid wash vessel. Acid will be pumped upward through the acid wash vessel and overflow back to the acid wash circulation tank. The carbon will then be rinsed with fresh water to remove the acid and any mineral impurities.

A recessed impeller pump will transfer acid washed carbon from the acid wash vessel into the elution vessel. Carbon slurry will discharge directly into the top of the elution vessel. Under normal operation, only one elution will take place each day.

17.4.9.2 Carbon Stripping (Elution)

The carbon stripping (elution) process will utilize barren solution to strip the carbon to create a pregnant solution, which will be pumped through electrowinning and back to the strip column.

The strip column will be a carbon steel tank which will hold approximately 1.5 t of carbon. During the strip cycle, solution containing approximately 1% sodium hydroxide and 0.2% sodium cyanide at a temperature of 140°C (284°F) and 450 kPa (65 psi) will be circulated through the strip vessel. Solution exiting the top of the elution vessel will be cooled below its boiling point by the heat recovery heat exchanger. Heat from the outgoing solution will be transferred to the incoming cold solution, prior to the cold solution passing through the solution heater. A diesel-powered boiler will be used as the primary solution heater.

17.4.9.3 Carbon Regeneration

A recessed impeller pump will transfer the stripped carbon from the elution vessel to the kiln feed dewatering screen. The kiln feed screen doubles as a dewatering screen and a carbon sizing screen, where fine carbon particles will be removed. Oversize carbon from the screen will discharge by gravity to the carbon regeneration kiln feed hopper. Screen undersize carbon, containing carbon fines and water, will drain by gravity into the carbon fines tank. Subsequently, the carbon fines will be collected into bags for disposal. A diesel-fired horizontal kiln with residual heat dryer will be utilized to treat 1.5 t of carbon per day, equivalent to 100% regeneration of carbon. The regeneration kiln discharge will be transferred to the carbon quench tank by gravity, cooled by fresh water and/or carbon fines water prior to being pumped back into the processing circuit. The carbon regeneration will use residual heat from the kiln to heat the pre-dryer.

To compensate for carbon losses by attrition, virgin carbon is added to the carbon attrition tank along with fresh water to mix and activate the carbon. The fresh carbon will then drain into the quench tank.

17.4.10 Gold Electrowinning and Refining

To facilitate metallurgical accounting, pregnant solution produced from the elution circuit and from the intensive leach reactor will not be mixed.

Pregnant solution from the strip vessel will be pumped to the refinery for electrowinning to produce a gold sludge. The electrowinning circuit will be a two-stage, single pass circuit. Resulting barren solution will be pumped back into the barren solution tank for reuse, with periodic bleeding to the CIP circuit.

Intensive leach reactor pregnant solution will be pumped from the intensive leach unit into a solution tank within the gold electrowinning room. The solution will then be pumped into a single electrowinning cell with solution overflowing back into the recirculating tank. Once the gold has been extracted from the intensive leach reactor pregnant solution, the resulting barren solution will be pumped to the CIP circuit.

Gold-rich sludge will then be washed off the steel cathodes in the electrowinning cells (using high pressure water) into the sludge holding tank. Periodically, the sludge will be drained, filtered, dried, mixed with fluxes and smelted in a diesel-powered, direct-fire furnace to produce gold doré. This process will take place within a secure and supervised area. The gold doré will be stored in a vault to await shipment.

17.4.11 Cyanide Detoxification

The cyanide detoxification circuit will consist of two mechanically agitated tanks, each with a capacity of 163 m^3. Cyanide will be destroyed using the SO_2/Air process. Treated slurry from the cyanide detoxification circuit will flow by gravity to a final tailings tank where the tailings will be pumped to the Tailings Storage Facility.

The cyanide detoxification circuit will treat thickened slurry from the tailings thickener, process spills from various contained areas and process bleed streams.

Process air will be sparged from near the bottom of the tanks, under the agitator impeller. Lime slurry will be added to maintain the optimum pH of 8.0 – 8.5 and copper sulphate ($CuSO_4$) will be added as a catalyst, maintaining 15ppm concentration in solution. Sodium metabisulphite (SMBS) will be dosed into the system as a solution as the source of SO_2. This system has been designed to reduce the total cyanide concentration to less than 1 mg/L CN_{WAD} prior to transfer to the TSF.

The cyanide detoxification product from the two tanks will flow by gravity to a mechanically agitated tailings tank with a 30-minute retention time. Three centrifugal pumps, two fixed speed and one variable, (with redundancy provided by an installed standby system) will pump the tailings through a 12 inch diameter HDPE pipeline for 2 km followed by the remaining length at 10 inch diameter to tailings storage.

17.4.12 Water Supply and Consumption

The following types of water will be used in the process plant.

- Process water: overflow water from the pre-leach thickener and tailings thickener will be used as process water. This water will have a low gold concentration and a CN-concentration of about 80 ppm. Process water will be used predominantly in the grinding circuit to dilute slurry to the required densities.
- Fresh water: fresh water for the process plant will be pumped from Goose Lake and will be used as reagent make-up water, gland water, and for cooling water services in the oxygen plant and strip circuit boiler. The estimated fresh water consumption in the process plant and for potable water is about 117 m^3/h.
- Reclaim water: water reclaimed from the TSF will be used as dilution water in the cyanide detoxification circuit and wash water in the tailings thickener. The estimated reclaim water consumption is about 75 m^3/h.

17.4.13 Air Supply

The air distribution system to supply instrument, plant, and process air will be centralised, except for the crushing area air system. The following compressed air supply centres are planned:

- An air compressor in the crushing area will provide air for that area;
- A 1380 Nm^3/hr @ 760 kPag air compressor will supply low pressure process air to the cyanide detoxification tanks; and
- An instrument and plant air system with compressors (one duty and one standby), dryers, filters, and receivers will be provided and located with the process air compressors in a compressor room inside the plant building.

17.5 Process Plant Operational Requirements

17.5.1 Manpower

Table 17.2 represents an estimate of the operating and maintenance personnel required to operate the process facility.

Table 17.2: Manpower Schedule for Process Plant

Position	Number of Personnel	Position	Number of Personnel
Mill Staff		**Laboratory**	
Mill Superintendent	1	Assayer*	0
Chief Metallurgist	1	**Total Laboratory**	**0**
Plant Metallurgist	2		
Operations General Foreman	2	**Shared Maintenance**	
Chief Assayer*	0	Maintenance Superintendent	1
Metallurgical Technician	2	**Total Shared Maintenance**	**1**
Operations Shift Foreman	4		
Total Mill Salaried	**12**	**Mill Maintenance**	
		Power Plant Operator	4
		Power Plant Maintenance	2
Mill Operations		Building Maintenance	2
Control Room Operator	4	Millwrights	9
Crusher Operator	4	Electricians	6
Grinding/ Gravity Operator	4	Process Control Technician	2
Leach/ CIP Operator	4	Welders	2
Strip/ Regeneration Operator	4	Instrument	2
Gold Refining Operator	2	Apprentice	8
Labourers	8	**Total Mill Maintenance**	**37**
Total Mill Operations	**30**	**Total Mill Labour**	**80**

*Laboratory supplied “Over-The-Fence” (OTF)

Source: Canenco 2015

17.5.2 Power Demand

The power demand for the processing plant from crushing to final tails and supporting ancillaries is shown in Table 17.3.

Table 17.3: Power Demand for Processing at Full Production

Work Breakdown Structure (WBS) - Description	Connected Load (kW)	Average Running Load (kW)	Annual Energy Usage (kWh)
3200 - Crushing	1001	716	4,335,122
3300 - Fine Ore Stockpilie	92	53	366,349
4200 - Grinding	3,960	3,386	26,763,370
4300 - Gravity Concentrating and Intensive Leaching	117	89	524,660
4400 - Cyanide Leaching and Carbon Adsorption	748	533	4,205,127
4500 - Acid Wash, Stripping and Regeneration	111	81	513,376
4600 - Gold Recovery	153	93	520436
4700 - Cyanide Destruction and Tailings	578	299	2,359,672
4800 - Reagents	949	924	7,195,722
4900/5200/5300/8100 - Process Utilities	1,322	982	6,762,097
Total	**9,031**	**7,156**	**53,545,931**

Source: Canenco 2015

18 Project Infrastructure and Services

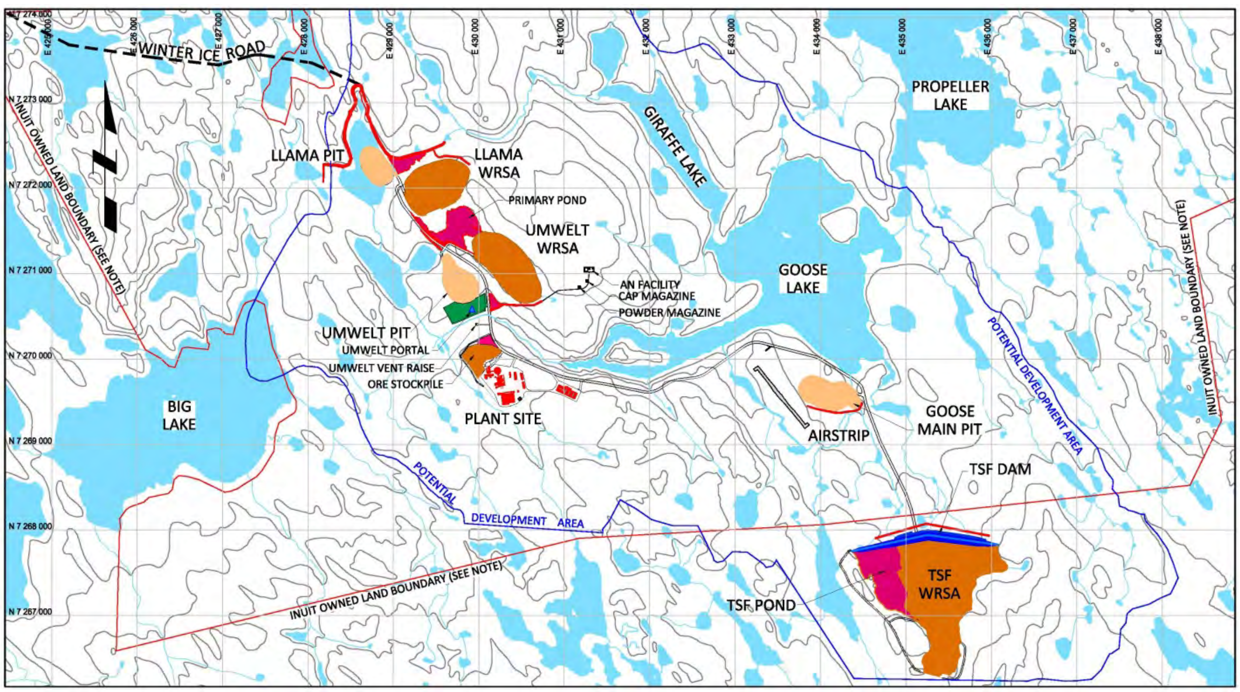
18.1 Overview and Design Criteria

The Back River Project consists of two main sites: the Goose Site, and the Marine Laydown Area (see Figure 4.1). The MLA is located on Bathurst Inlet, approximately 130 km north-northwest of the Goose Site.

The Goose Site is only accessible by air on a year-round basis and by a winter ice road from the MLA during the winter months. The MLA will be accessible by air on a seasonal basis (no land-based airstrip) and by sea during the summer months. Due to the remote nature of each site, significant infrastructure is required for access, power generation, consumable storage and accommodations.

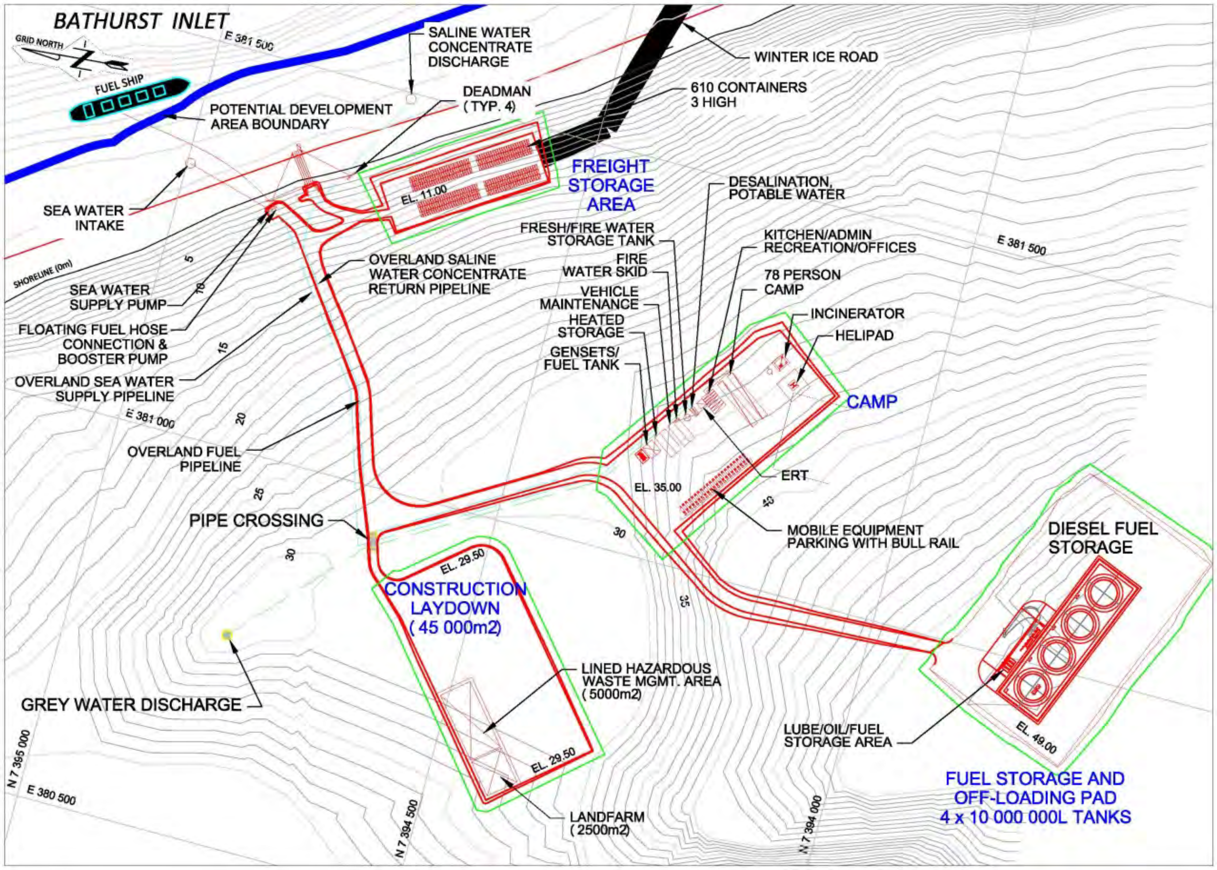
The site layouts for Goose, and the MLA are shown in Figure 18.1 and Figure 18.2, respectively.

Figure 18.1: Goose Site Layout



Source: JDS 2015

Figure 18.2: MLA Site Layout



Source: JDS 2015

18.1.1 General Infrastructure Design Criteria

The design of the facilities was dictated by the 12-year mine life. Therefore, broad design criteria were applied; facilities were fit-for-purpose to minimize cost, and of appropriate quality to ensure safe and reliable operations. Some design elements that were developed using this philosophy include the following:

- Paint structures only for protective or safety purposes (e.g., fuel tanks not painted);
- Use fabric buildings where practical;
- Use compacted-fill floors where appropriate;
- Compact building footprints to optimize heating and heat recovery;

- Minimize the difference in elevation and the horizontal distances between the open pits, underground portal, mill site, crushing plant, airstrip, and Tailings Storage Facility (TSF) to minimize the capital and operating costs for truck haulage, roads, earthworks, and pipelines;
- Minimize the Project footprint to keep ground disturbance and cost to a minimum;
- Locate key infrastructure in areas of shallow bedrock;
- Respect environmental design requirements, such as set-back from water;
- Consider site water management requirements;
- Consider traffic management and safety;
- Consider climatic conditions; and
- Consider local wind patterns with respect to noise, dust, and other atmospheric emissions.

18.1.2 Project Geotechnical Conditions

Over the past decade, multiple geotechnical characterization studies have been performed on the site involving drilling, sampling and materials testing of rock, soils and water. The results of the test work and subsequent analyses were used to characterize the geotechnical conditions for the Project and provide design recommendations for mining and infrastructure. A summary of the site conditions follows.

18.1.3 Soil Conditions

During the last Quaternary period, the region was subjected to multiple glaciations that have resulted in the striated landscape and overburden materials characteristic of a post-glacial environment with moraine sediments being the most dominant. Other sediments include glaciofluvial sediments, organic sediments, and marine sediments only in proximity to Bathurst Inlet, as well as exposed weathered bedrock. Overburden thickness varies from 1 m associated with outcropping weathered bedrock in the highlands, to greater than 37 m in topographic lows (Rescan 2014).

The two Project sites are located within a region of continuous permafrost. Permafrost temperatures below the point of zero amplitude range between -6 and -8˚C (Rescan 2014). Basal permafrost depths range from 490 to 570 m below ground surface (mbgs). The active layer depth ranges from approximately 1.3 to 4.2 mbgs with the greatest active layer depths occurring in areas with thin soil veneers. Due to the presence of salinity in some surficial groundwater, the active layer takes up to 60 days to refreeze in some areas (Rescan 2014).

Overburden soils at the Goose and onshore MLA sites generally consist of silty sands with some clay and gravel. Pockets of sandy, silty gravel till underlie these deposits at the Goose Property (SRK 2011a; Knight Piésold 2013b).

In general, ice content within the soils at the Goose and onshore MLA sites ranged from 10 to 30%.

18.1.3.1 Borrow Quarry Sources

18.1.3.1.1 Goose Property

Construction rock for infrastructure development at the Goose Property will be sourced from the existing quarry near the Goose airstrip (airstrip quarry) and from within the footprint of the Umwelt open pit (Umwelt quarry). This airstrip quarry is comprised mainly of greywacke and mudstone with minor amounts of BIF, large intrusive gabbro dykes, and smaller felsic to intermediate dykes. Acid rock drainage (ARD) results indicate that some of this quarry rock is potentially acid generating (PAG) and may have to be segregated and used appropriately or mixed to obtain acceptable geochemical conditions.

The Umwelt quarry area was selected to be entirely within the Upper Greywacke unit. The majority of these unit samples are classified as non-potentially acid generating (NPAG).

Based on analogous samples tested as part of the waste rock characterization program, contact water associated with quarry rock from both the airstrip quarry and the Umwelt quarry is expected to contain slightly elevated levels of arsenic but is relatively minor and no specific water management will be required.

18.1.3.1.2 MLA Site

Preliminary geochemical characterization was completed on a small number of surface outcrop samples representing quarry rock that will be excavated during construction of the MLA Fuel Storage Farm. These samples were described as weathered quartzite conglomerate and quartz arenite/quartzite (sandstone) and test results showed that these materials have a negligible potential for metal-leaching/acid rock drainage (ML/ARD).

18.1.4 Geotechnical Design Principles

18.1.4.1 Overburden Stripping

It is assumed that the overburden active layer, which ranges from 1 to 4 m, can be mined using conventional truck and shovel techniques. Some temporary access roads comprising competent material (quarried or run of mine rock (ROM)) may need to be constructed during the summer months when permafrost degradation is in full effect due to excavation. Alternatively, low–bearing pressure equipment may be used. Winter excavation of the overburden active layer or underlying permafrost in any season will require drilling and blasting. These soils will absorb a significant amount of the blast load and, as a result, closer drill hole spacing and higher blast load factors than for regular rock blasting will be required.

18.1.4.2 Overburden Stockpile Design

Overburden stockpiles are designed to be constructed at the quarry or mine areas, or directly deposited in WRSAs. Two overburden products will be produced in each area: (1) frozen overburden from permafrost areas, and (2) unfrozen overburden from summer stripping or talik zones. Geotechnical considerations have been heeded in the design of overburden stockpiles.

18.1.4.3 Tailings Storage Facility Dam Foundation

Characterization of the TSF dam foundation resulted in the identification of three distinct foundation zones, as described in Table 18.1.

Table 18.1: TSF Dam Foundation Zones and Material Properties

Zone Descriptor	Properties	Key Trench Treatment
Shallow bedrock	Bedrock is near surface, covered with less than 4 m of overburden.	Excavate all overburden and highly fractured bedrock. No other treatment required.
Deep ice-poor overburden	Deep overburden (greater than 4 m). Massive ice not present and low (less than 10%) interstitial ice content.	Excavate overburden to design extents of key trench.
Deep ice-rich overburden	Deep overburden (greater than 4 m). Massive ice and high (more than 10%) interstitial ice content.	Excavate overburden and massive ice zones contiguous with the key trench excavation.

Source: JDS 2015

18.1.4.4 Waste Rock Storage Area Foundations

Waste rock storage areas (WRSAs) constructed on permafrost soils (i.e. directly onto the tundra) should be designed to promote freeze back, thereby minimizing long-term environmental effects from possible metal-leaching and/or acid rock drainage (ML/ARD). This is provided the waste rock is proven to be geochemically compatible, meaning that it does not generate excessive heat and thereby degrade permafrost. Testing has indicated that excessive heat generation is not expected.

Permafrost soils are expected to provide suitable foundation conditions for WRSAs, provided the foundation remains frozen. To ensure the foundation remains frozen, it is recommended that the first lift of all new WRSAs be constructed during the winter season. In the event that the first lift of waste rock is constructed during the summer months, the WRSA will be subject to differential settlement due to consolidation settlement of the active layer. The amount of settlement will vary but will likely be between 10 and 30% of the active layer thickness which ranges from 1 to 4 m. This settlement will only occur during the first summer, expecting that appropriate freeze back is obtained during the subsequent winter.

The overall maximum height (i.e. total vertical thickness) of the WRSA should be limited to 100 m unless appropriate analysis to confirm otherwise is carried out.

In areas where the WRSA foundation is on exposed bedrock, no significant issues are expected. Therefore, placement on exposed bedrock is preferred and can proceed during any season provided adequate clearing of snow and ice has been completed.

18.1.4.5 Permafrost Foundations

Frozen overburden materials are expected to have sufficient bearing capacity, while thawed overburden soils are expected to have only a medium strength when drained. Since most material is likely to have limited clay and thawing is typically a slow process, silty soils observed at the two sites will likely be under drained conditions during seasonal thaw. However, some deformation can be expected during permafrost thaw and it is recommended that surface infrastructure be founded on frozen soils as often as practical. Care will be taken when designing infrastructure and pads to ensure that heat generated from buildings does not promote thaw of permafrost materials.

Structures that are particularly sensitive to differential settlement, such as the Goose process plant and fuel storage tanks, will be founded on exposed bedrock or with a compacted engineered-fill layer on top of bedrock.

18.1.4.6 Talik Foundations

A description of the presence and extent of taliks on the Property is presented in SRK (2015). If facilities are constructed in or near lakes, there may be foundation interactions with talik zones. Soils within talik zones may have lower bearing capacities and construction on these overburden soils and designs will take this into account.

18.1.4.7 Surface Water Management Facilities

Surface water management facilities such as water conveyance channels, storm water ditches, and sediment control ponds are planned for the Project. Excavation of channels and/or ditches into overburden soils will be avoided wherever possible. Ponded water on permafrost soils will be avoided.

18.1.5 Infrastructure Foundation Preparation Recommendations

Considering all of the conditions listed in the preceding sections, the specific foundation preparation recommendations for the Project are summarized in Table 18.2.

Table 18.2: Infrastructure Foundation Preparation Recommendations

Area	Recommendations
Goose Site	Bedrock foundation required for critical structures such as fuel storage tanks, heated buildings, and process equipment foundations.
	For bedrock foundations:
	Strip (doze) the upper 0.5 m of overburden and discard in overburden stockpile or place in non-critical pads. If winter construction is planned, drilling and blasting will be required.
	Drill and blast upper 3.5 m of fractured rock. Only 50% can be assumed to be useable as run of quarry (ROQ) construction fill. The remainder is likely to be unusable as it will contain predominantly oversize material. Discard material goes to the WRSAs or non-critical pads.
	Rock shatter not required. The exposed surface needs to be cleaned and roughly leveled.
	2.5 m compacted ROQ rock-fill pad (on top of undisturbed grade) required for unheated essential structures such as the airstrip.
	1.0 m compacted ROQ rock-fill pad (on top of undisturbed grade) required for unheated non-essential structures such as secondary roads.
	Rock-fill pads will ideally be done in lifts no greater than 1.5 m with the maximum rock size limited to 0.9 m.
	A 150 mm thick layer of 50 mm minus surfacing material is recommended as a topping layer for ROQ pads. No transition layer required provided the ROQ is well graded. There may be some holes that develop as a result of consolidation but minimal repair should be required. An allowance of 20% extra 50 mm minus material should be provisioned for.
	Mine haul roads should be 1.5 to 2.0 m thick to minimize deformation.
	Rock shatter required where roads cross over rock highs that impact road grade.
Marine Laydown Area	Bedrock foundation required for critical structures such as fuel storage tanks and heated buildings.
	For bedrock foundations:
	Strip (doze) the upper 0.5 m of overburden and use in non-critical pads. If winter construction is planned, drilling and blasting will be required.
	Drill and blast upper 1.5 m of bedrock. 100% can be assumed to be useable as ROQ construction fill.
	2.0 m compacted ROQ rock-fill pad (on top of undisturbed grade) required for unheated essential structures.
	1.0 m compacted ROQ rock-fill pad (on top of undisturbed grade) required for unheated non-essential structures such as secondary roads.

Source: SRK and JDS 2015

18.2 On-Site Infrastructure

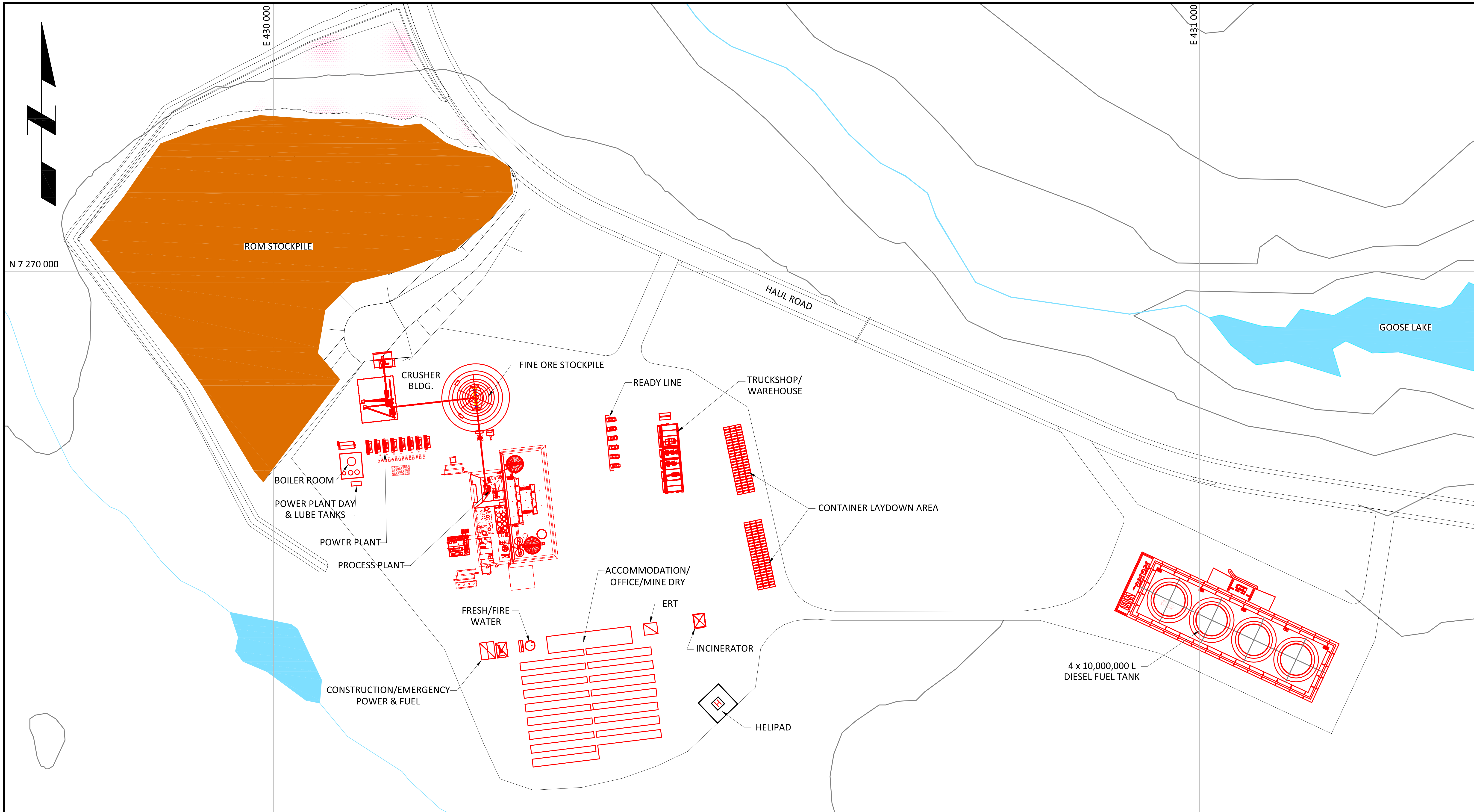
This section describes the on-site infrastructure at the Goose Site.

18.2.1 Goose Site

18.2.1.1 Site Infrastructure

The major infrastructure at the Goose Site includes the following (see Figure 18.3):

- Process-related facilities;
- Truck shop and warehouse building;
- Power plant;
- Fuel storage farm;
- Utilities;
- Permanent camp, administration office, and mine dry facility;
- Storage areas;
- Airstrip;
- Tailings management system, which includes a TSF and use of two mined-out pits to store tailings;
- WRSAs;
- Industrial waste management facilities, such as the incinerator; and
- Site water management facilities.


E 430 000
E 431 000
N 7 270 000
ROM STOCKPILE
HAUL ROAD
GOOSE LAKE
CRUSHER BLDG.
FINE ORE STOCKPILE
READY LINE
TRUCKSHOP/ WAREHOUSE
BOILER ROOM
POWER PLANT DAY & LUBE TANKS
POWER PLANT
PROCESS PLANT
CONTAINER LAYDOWN AREA
ACCOMMODATION/ OFFICE/MINE DRY
ERT
FRESH/FIRE WATER
INCINERATOR
CONSTRUCTION/EMERGENCY POWER & FUEL
HELIPAD
4 x 10,000,000 L DIESEL FUEL TANK

18.2.1.2 Plant: Site Integration

The process plant and fuel tanks are positioned at locations where the bedrock is very close to the surface. As much infrastructure as possible is located close to the process plant to make the operation of the site efficient, especially during the long and harsh arctic winter.

18.2.1.3 Truck Shop / Warehouse

The truck shop will consist of a 72 m long by 18 m wide by 12.5 m tall structural steel, pre-engineered building. It is designed to accommodate various facilities to repair and maintain mining equipment and light vehicles. The building will also provide storage space for spare parts and consumables.

The truck shop components are shown in Table 18.3.

Table 18.3: Truck Shop Components

Description	Area (m^2)	Comments
Service Bays	1080	four truck bays, + one wash bay each 12 m wide x 18 m deep
Warehouse	216	12m wide x 18m deep

Source: JDS 2015

The service bays are designated to service and repair the following major mining equipment: 64-tonne CAT 775 haul truck, CAT 988K FEL, CAT AD30 underground, and CAT 1600H underground LHD. The facilities will include automatic hose reels in one bay to dispense engine oil, transmission fluid, hydraulic oil, air, solvent, diluted coolant, and grease. The truck shop will be equipped with a 10-tonne overhead crane that will provide service to all service bays. Wash and tracked equipment bays are planned.

Tire repair will be done outside, weather permitting. In poor weather, tire repair will be done in the shop with the appropriate safety measures, such as adequate personnel access control and clearances.

18.2.1.4 Warehouse and Laydown Area

A container storage area is located to the east of the truck shop. Spare parts that do not require protection from the elements will be stored in the laydown area adjacent to the container storage yard. Break bulk freight can also be stored there.

A separate construction laydown area has not been designated, but the plant area pad size was developed to provide sufficient clearance around the infrastructure and allow for construction material laydown. Should additional storage for construction materials be required, it can be added to the south of the freight storage and fuel storage areas.

18.2.1.5 Crushing Building and Process Building

The three-stage crushing plant is located in a pre-engineered structural steel building. The crushing building will be heated (tempered) in the winter to -10ºC by glycol air handlers and unit heaters.

The process plant is located in a pre-engineered building. Overhead cranes are provided for equipment maintenance. The building is heated to 5°C by glycol air handlers and unit heaters.

18.2.1.6 Fuel Storage

Diesel fuel storage capacity at the Goose Site is designed for a 12-month operational period of the year with the maximum fuel usage. Table 18.4 provides the annual fuel usage for the Goose Site. Operational Year 5 requires the maximum fuel. A 12-month supply during this year results in a 40 ML total diesel fuel storage requirement. A total of four 10 ML field-erected fuel tanks will be constructed. One 10 ML fuel tank will be erected in Year -2 to support construction, two additional 10 ML fuel tanks will be constructed the following year, and the remaining one 10 ML fuel tank will be built in Year 2 to support the Umwelt underground mine.

Table 18.4: Goose Site Fuel Usage (thousand litres)

Area	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12
Infrastructure	17,661	18,351	20,143	20,566	20,541	20,574	20,548	20,557	20,213	17,661	17,661	13,537
UG Mining	0	242	3,985	5,676	5,623	5,562	5,517	5,564	4,552	0	0	0
OP Mining	8,320	8,519	8,851	9,742	10,295	9,991	9,035	2,960	664	615	615	406
Site Support	688	688	688	688	688	688	688	610	376	376	376	281
WIR* Constr.	290	308	320	322	322	322	319	303	295	279	141	0
WIR Freight	17	23	26	27	27	27	26	21	19	13	69	0
Earthworks	143	0	3	0	18	0	0	0	0	0	0	0
Total Fuel	**27,120**	**28,131**	**34,016**	**37,021**	**37,515**	**37,166**	**36,133**	**30,015**	**26,120**	**18,944**	**18,861**	**14,225**

*WIR = Winter Ice Road
Source: JDS 2015

The fuel tank farm bund will be lined with high-density polyethylene (HDPE) for spill containment. Fuel dispensing equipment for mining, plant services, and freight vehicles will be located adjacent to the fuel tank bund and the fueling area will drain into the bund. A fuel transfer module will provide fuel to the power plant day tank and diesel consumers in the process plant.

18.2.1.7 Explosives Storage and Preparation

Explosives storage at the Goose Site consists of the following three main components:

- Bulk ammonium nitrate (AN) storage;
- Ammonium nitrate – fuel oil (ANFO) manufacturing; and
- Explosive storage magazines.

Bulk ammonium nitrate (AN) prill will be shipped to site in one-tonne tote bags within 20-ft ISO containers. The AN storage area is sized to allow for a maximum of 3,900 t of AN or 207 ISO containers.

ANFO required for underground mining will be manufactured and bagged in one-tonne totes using bagging equipment located in the AN facility. The bagged ANFO will then be transported to the underground mine for use in blasting operations.

ANFO required for open pit blasting will be mixed on a bulk ANFO truck at the blast hole. The bulk ANFO truck will take on ammonium nitrate and fuel oil at the Goose AN facility. Bulk AN will be augered from the one-tonne tote bags into a 30-t silo and the bulk ANFO truck will drive underneath to load AN. Fuel oil will be loaded into the bulk ANFO truck from a 20,000-litre double-wall fuel tank also located on the AN facility pad.

Packaged explosives and explosive detonators will be stored in approved explosive magazines located on separate pads. The powder magazine will be a 40-ft container magazine capable of holding 32 t of explosives, and the cap magazine will be a 20-ft container magazine capable of holding approximately 600 cases of detonators.

The design of all storage facilities will meet government regulations and will be located according to required separation distances as regulated by the Explosives Regulatory Division (ERD) of Natural Resources Canada (NRC).

18.2.1.8 Camp / Administration Offices / Mine Dry Facilities

A camp, with a combination of single-occupancy and double-occupancy rooms, will be used during the construction phase at Goose Site to accommodate 303 workers. There will be 204 beds in double-occupancy rooms and 99 beds in single-occupancy rooms. All rooms will be a "Jack and Jill" bathroom arrangement. During operations, all rooms will be converted to single occupancy with 201 rooms available at the start of operations. An additional 23 rooms will be added in Year 1 and 38 rooms in Year 2 and Year 3 for the maximum number of 300 rooms.

The camp will include kitchen and dining facilities to seat 300 and a recreation area.

The camp complex will include an 838 m^2 office area for an integrated operations team and G&A staff. Additionally, a 900 m^2 mine dry/change facility will be included in this complex.

The camp complex will be constructed of modular units manufactured off-site in compliance with highway size restrictions for transportation. Camp modules will sit on wood cribbing. The camp will comply with all building and fire code requirements and be provided with sprinklers throughout.

Arctic corridors will be provided to connect the camp core facilities and dormitories with the process plant and truck shop, and all facilities, including the kitchen, will be sized and built for the highest design population.

18.2.1.9 Domestic Waste Management

Food waste from the kitchen facilities will be segregated and burned daily in the incinerator to limit wildlife attraction associated with the disposal of food waste.

All hazardous waste (such as batteries and biomedical waste) will be flown to processing facilities in Yellowknife. There will be sufficient backhaul space available on freight flights that bring regular cargo, such as food and emergency parts, to the site.

Recyclable waste is also expected to be backhauled by air to suitable off-site recycling facilities for proper disposal. Optionally, hazardous and recyclable waste can be backhauled on the return leg of the seasonal sealift.

Non-hazardous, non-leaching, inorganic garbage will be collected and disposed of within an on-site landfill, to be located in a WRSA.

A land farm for storing and treating hydrocarbon contaminated soil will also be located in a WRSA.

18.2.1.10 Ancillary Structures

A construction power plant will be installed when the camp is built to provide power during the construction period. This plant will be located inside a fabric building. When the process plant is operational, the construction power plant will function as a source of emergency power in case the main power plant malfunctions.

A batch plant will be installed to provide concrete for the construction of the Goose Site process plant and infrastructure. The batch plant will be located inside a large fabric building to allow year-round operation.

18.2.1.11 Utilities and Services

18.2.1.11.1 Sewage Treatment Plant (STP)

Sewage will be treated by a biological reactor housed in three 45-ft containers which are fully shop-assembled prior to shipment to site. A sludge drying system is also provided in a separate 40-ft container. The dewatered sludge is disposed of in the incinerator.

The treatment plant includes influent screening, equalization/bioreactor tank to handle the daily peaks in flow, membrane system, treated effluent storage, and UV disinfection. The treated effluent is discharged to the tundra during construction and to the tailings facilities during operations.

18.2.1.11.2 Fresh / Fire Water

Fresh water is drawn from Goose Lake by a pump station located on the shore at a point of sufficiently deep water. The fresh water is pumped via a heat-traced pipeline to the Fresh/Fire Water Tank. Fire water will be stored at the bottom of the tank, and the fresh water will be drawn off the upper portion of the tank.

18.2.1.11.3 Potable Water

Water will be pumped from the Fresh/Fire Water Tank to the Potable Water Treatment Plant. Treated water from the potable water plant will be stored in an insulated and heated Potable Water Storage Tank, which accommodates the potable water demand variances, and then distributed to the process plant, camp and mine dry facilities.

The plant is contained in one 20-ft shipping container which is fully shop-assembled prior to shipment to site. It contains the complete treatment system, including filtration, UV disinfection and chlorine disinfection.

18.2.1.11.4 Process Water

Process water is collected in the Process Water Tank and pumped to various points in the process plant.

The Process Water Tank is fed by tailings reclaim water via the tailings thickener as well as water from the CIC Tank tails and the pre-leach thickener overflow when the CIC tanks are bypassed.

The Process Water Tank is located inside the process building and is a bolted construction design with factory epoxy-coated panels.

18.2.1.11.5 Heating, Ventilation, Dust Control, and Fume Extraction

Continuous ventilation will be provided for all personnel-occupied spaces, as well as select unoccupied spaces. Ventilation rates will vary depending on the level of occupancy and the intended use of the space, in accordance with applicable codes and standards. Ventilation systems will include make-up air units for continuous supply of tempered air, exhaust fans to provide the required number of air changes per hour, and localized exhaust fans to remove fumes, where required. The process plant includes dust control and fume extraction systems.

18.2.1.11.6 Site Communications

Site-wide communications design will incorporate reliable communications systems to ensure that personnel at the mine site have adequate voice, data, and other communication channels available. A number of integrated systems will be provided for on- and off-site communication at Goose Site, including inside the process plant. On-site communications will be facilitated by satellite phones, VHF radio, optical fibre cable network, and a pit dispatch mesh on the surface, and a leaky feeder VHF for the underground mine.

A trunked radio system consisting of handheld, mobile, and base digital radios will provide wide area coverage for on-site communication by operations. The trunked radio system will be interfaced to the on-site Voice over Internet Protocol (VoIP) telephone system.

18.2.1.11.7 Fire Protection System

At a minimum, the Goose Site facilities will be protected from fire in accordance with applicable codes and standards. The fire alarm system will consist of manual pull stations at building exits and audible and visual notification devices throughout the work areas. Additionally, the fire alarm system will provide supervision of the proposed dry chemical system to cover lube oil storage and units to initiate an alarm on system discharge and inform the Control Room of this event.

A digital fire alarm system has been included for all buildings at the plant site.

All surface and underground mobile equipment will be fitted with fire extinguishers. The fleet of open pit mining equipment will also have fitted fire suppression systems.

The firewater main, hydrant, and stand-pipe system will service the Goose Site facilities by a fire water tank and modularized pump unit. A fire water truck will provide supplemental protection. All buildings and conveyors have fire extinguishers and some will have stand-pipe systems and fire truck connectivity. There are no sprinkler systems inside the Crushing Building or the Process Plant. The galleried conveyors represent safety risks for occupants due to the enclosure acting as a conduit for smoke and heat. Automatic sprinkler protection will be installed on these conveyors to mitigate the associated risks.

Fire suppression for the power generators will be provided by a Liquid Vehicle System (LVS) per unit because they are large diesel engines. The accommodations camp, truck shop, administration offices, mine dry, and warehouse will be fitted with sprinklers. The server rooms will have clean agent gas suppression and lube storage will have fixed dry chemical suppression.

18.2.1.11.8 Security

Security needs are served by a card access system and video cameras. Security is generally for safety reasons and theft mitigation to ensure access is restricted to qualified personnel.

18.2.1.11.9 Power Generation

A single captive power plant will be used to meet the electrical power demands needed to support the complete Goose Site operation, including the underground mine. The power plant will consist of diesel-fired reciprocating engine generator sets (gensets). To maximize the overall efficiency, this power plant will operate as a combined heat and power plant (CHP Plant) whereby the waste heat is recovered.

The estimated electrical loads are shown in Table 18.5.

Table 18.5: Project Electrical Load List

Description	Connected Load (MW)	Peak Demand Load (MW)	Average Running Load (MW)
Goose – Total	13.2	10.4	8.7
Goose Processing and Infrastructure	10.9	8.4	7.1
Goose Underground Loads	2.3	2	1.6
MLA – Total	0.9	0.7	0.6
Grand Total	14.1	11.1	9.3

Source: JDS 2015

The power plant design is based on eight diesel generator sets in an N+1 arrangement. All generators are rated for 1.45-MW and will run at 1,200 rpm generating 4,160 V. The peak gross power is 10.15 MW (7 operating at 100%). The peak electrical load is 10.4 MW which is slightly higher than the peak gross power, but it is close enough that this load can be satisfied by running the generators at 110% for short time periods.

18.2.1.11.10 Power Distribution

The power plant includes all switchgear and control equipment to accommodate the generators. This equipment includes 4,160-V switchgear for the generators and process plant feeders, load-sharing systems, neutral grounding equipment, surge suppression, local and master control systems, and all necessary low voltage distribution equipment for power plant ancillaries.

Power will be distributed throughout the plant site at 4,160 V. There will be five distribution feeders, including two spare positions.

The electrical loads at the Umwelt underground mine will be fed with 4,160 kV power lines that are laid alongside the roads and overland piping. The TSF and Tailings Facility (TF) pumps will be fed the same way. Seasonal open pit dewatering will be handled by diesel pumps to eliminate the need for long distance, high-voltage transmission lines to the Llama, Umwelt, and Goose Main open pits.

18.2.1.11.11 Waste Heat Recycling

Heating for buildings and facilities at the Goose Site will be provided primarily by heat recovery from the power plant. Waste heat from the power plant will be transferred with a glycol circulation system that extends throughout the plant site. A boiler system located between the camp and the mill building will be used as a backup heat source.

18.2.1.12 Site Roads

The road network for the Goose Site will consist of two types of roads: winter ice roads and all-season roads. The all-season roads will consist of haul roads and service roads.

Due to geotechnical conditions, all-season roads will be constructed with embankment fills only. The embankment material will be sourced from the airstrip quarry or from waste material from the open pits. Underfoot conditions will dictate the appropriate thicknesses of road bed material.

The Goose Site will have 9 km of haul roads and 4 km of service roads. Service roads are used for smaller vehicles (i.e., light trucks) to access ancillary infrastructure, such as water supply sources, the airstrip, the AN storage facility and the TSF.

Haul roads design criteria are described in the mining section. Service road design criteria are as follows:

- Design vehicle: light / medium truck;
- Minimum width of travelling surface: 8 m;
- Design speed: 50 km/h;
- Side slopes: 2H:1V;
- Maximum grade: 8%;
- Safety berms for fills greater than 3 m in height: 0.5 m; and
- Drainage: major culverts and bridges to be designed to a 1-in-100-year return period.

18.2.1.13 Airstrip

The Goose airstrip is a vital component of the site infrastructure because air transportation is the primary means of access for mine personnel and incidental freight to and from the Goose Site.

The existing Goose airstrip is designed to handle turboprop passenger aircraft similar in size to a de Havilland Dash 8. The airstrip is also sufficiently sized to handle cargo aircraft up to a de Havilland DHC-5A Buffalo. The airstrip will be designated as a “registered aerodrome” and the design will be in accordance with Transport Canada’s standards as set out in TP312E Aerodrome Standards and Recommended Practices.

The existing Goose airstrip is currently 915 m long and 30 m wide and includes apron areas and turn arounds on each end. This is deemed sufficient to accommodate the required aircraft.

In order to ensure year-round efficient operations, the Goose airstrip will be equipped with a Global Positioning System (GPS) Instrument Approach system; this allows instrument flight rule (IFR) approaches and departures under suitable weather conditions. The airstrip lighting package will include the following:

- Runway edge lighting;
- Taxi-way edge lighting;
- Precision approach path indicators (PAPI); and
- Omni-directional approach lighting system (ODALS).

A pre-fabricated modular operations centre will be located on the airstrip apron and will contain all electrical services and controls for the airstrip.

Passengers arriving at the Goose Site will be transported directly from the aircraft to the camp facilities using the site's crew buses. Departing passengers will then be shuttled from the camp to the aircraft.

18.3 Off-Site Infrastructure

18.3.1.1 Winter Ice Road

The winter ice road (WIR) will form a critical component of the off-site infrastructure that will be used to link the Goose Site with the MLA. The WIR will be constructed annually from the MLA at Bathurst Inlet to the Goose Site beginning in Year -3. The WIR will be the primary method of delivering equipment and materials to the Goose Site during construction and throughout operations.

Once the road is constructed and deemed suitable for hauling, the road will be monitored and maintained to ensure safe and continuous operation until all freight has been transferred from the MLA to the Goose Site. During the annual winter road season, a fleet of up to 25 trucks will transport an average of 11,800 t of freight and 29 million litres (ML) of fuel from the MLA to the Goose Site.

From Year -3 to Year 11, a total of 157 km of WIR will be constructed annually between the MLA and the Goose Site. The breakdown of distances constructed on ice and land for the WIR is summarized in Table 18.6.

Table 18.6: Winter Ice Road Segment Lengths

Description	Distance on Land (km)	Distance on Ice (km)	Total Distance (km)	# of Water Bodies
MLA to George Junction	39	63	102	22
George Junction to Goose	27	28	55	29
Total MLA to Goose Site	66	91	157	51

Source: JDS 2015

Distances on ice and land are important factors used to calculate both construction durations and haul cycle times. Construction progress rates vary significantly between ice and land; ice construction progresses faster. The number of water bodies along each road is also an important factor in determining haul cycle times, as speeds are significantly reduced for loaded trucks travelling onto and off of portages.

The following design criteria will be used for the construction and operation of the WIR:

Road Widths:

- On Ice – minimum 30 m width; and
- On Land – minimum 10 m width, with maximum 5% grade.

Operating Speeds:

Loaded Trucks (> 50% of maximum load limit):

- 25 km/h on ice;
- 30 km/h on land; and
- 10 km/h on and off portages.

Empty Trucks (< 50% of maximum load limit):

- 35 km/h on ice;
- 55 km/h on land; and
- Load Limits.

Table 18.7 provides minimum ice thickness and total allowable weight for various vehicle configurations.

Table 18.7: Load Limits at 100% of Highway Legal Gross Vehicle Weight

Vehicle Configuration	Minimum Ice Thickness (cm)	Total Allowable Weight (kg)
2-Axle Hotshot	66	14,600
3-Axle Hotshot	73	22,500
6-Axle Tractor Trailer	89	46,500
7-Axle B-Train	96	56,500

Source: JDS 2015

WIR construction is planned to begin in early December of each year starting in Year -3. At this time, it is expected that the tundra subgrade and water bodies will be frozen and able to support light tracked equipment. In order to construct the WIR to commence hauling in January, the construction will be advanced from two headings; one construction crew will start from the Goose Site and work north, while a second crew will start from the MLA and work south. Due to the seasonal nature of the WIR construction, it has been assumed that it will be performed by an experienced contractor. The contractor will provide all labour and operate the Owner's equipment.

The WIR will be constructed over frozen bodies of water and over land between the bodies of water (also known as portages). Ice construction consists of first determining if there is adequate ice thickness to support construction equipment using ground penetrating radar. Areas with thinner sections of ice might require manual flooding to increase ice thickness to accommodate legal loads. Once suitable ice thickness is achieved, snow is cleared from the right-of-way. Portage construction typically consists of layers of compacted snow and water placed on the frozen tundra to create a level driving surface suitable for highway legal loads.

The estimated construction rate-of-advance using two headings are as follows: on ice, it is 9 km/day total, and, on land, it is 2 km/day total. Rates of advance are based on consultations with contractors who are experienced in ice road construction in the Northwest Territories and Nunavut. Table 18.8 provides construction duration estimates.

Table 18.8: Winter Ice Road Construction Days

Description	Construction on Land			Construction on Ice			Total Construction Days
	Distance (km)	Rate (km/day)	Days	Distance (km)	Rate (km/day)	Days	
MLA to Goose Site	65.8	1.9	34.6	91.4	8.9	10.3	45

WIR construction will adhere to Fisheries and Oceans Canada requirements for ice bridges and snow fills as well as DFO Under-Ice Water Withdrawal Protocols for the withdrawal of water.

Emergency shelters will be temporarily placed along the WIR every 60 km. These shelters would be equipped with survival and communications equipment.

Freight and fuel quantities have been estimated for the life of the mine and are shown in Table 18.9.

Table 18.9: Winter Ice Road Freight and Fuel Quantities

	Freight (tonnes)	Fuel ('000 litres)
Year -2	10,568	0
Year -1	7,681	6,197
Year 1	9,393	27,120
Year 2	12,365	28,131
Year 3	14,389	34,016
Year 4	14,880	37,021
Year 5	14,910	37,515
Year 6	14,783	37,166
Year 7	14,016	36,133
Year 8	11,591	30,015
Year 9	10,258	26,120
Year 10	7,218	18,944
Year 11	11,383	33,086
Total LOM	**153,435**	**351,463**

Source: JDS 2015

Freight hauling is expected to average 25 t per load. Fuel hauling will use standard B-train tankers with a 50,000 L per load capacity.

The duration of a complete cycle between the MLA and the Goose Site exceeds the 12-hour maximum allowable shift. Therefore, it is anticipated that haul truck operators would work an 8-hour shift and each truck would take 16 hours to make a complete cycle. Table 18.10 provides cycle time details calculated for freight and fuel from the MLA to the Goose Site.

Table 18.10: Winter Ice Road Cycle Times

Description	Cycle Time (hours) MLA to Goose Site	
	Freight	Fuel
Unload / Reload Time @ MLA	1.5	1.0
Travel Time Loaded	7.3	7.3
Unload / Reload Time @ Goose	1.5	1.0
Travel Time Empty	5.4	5.4
Subtotal Cycle Time Before Delays	15.7	14.7
Shift Change Delays	0.3	1.3
Total Cycle Time (hrs)	16.0	16.0

Source: JDS 2015

The cycle times were used to determine the number of trucks and trailers required for each year.

Once the WIR is in full operation, the labour crews will be scaled back to perform road maintenance. WIR maintenance will take place on a 24-hour-a-day basis with two labour crews working 12-hour shifts: one crew based at the MLA and one crew based at Goose Site.

Maintenance crews will focus on the following tasks:

- Maintain road widths and repair damaged ice sheets, as required.
- Conduct focused flooding along the road in areas where icing is lagging.
- Sand portages, as and when required.
- Profile ice every second day until the road reaches 100% capacity, then weekly after that.
- Provide rescue and recovery work, as required.

Once hauling on the WIR is complete, perform the following tasks to decommission the road and prepare for demobilization:

- Remove all gravel on the ice surfaces that might have ended up on the ice as a result of sanding the portages;
- Gather all road signs and properly store them for future use;
- Remove any garbage that is found along the route;
- Remove any hydrocarbon spills that are found along the route;
- Conduct final maintenance of all ice road construction equipment; and
- Demobilize from site.

18.3.1.2 Marine Laydown Area

18.3.1.2.1 Introduction

The Project includes a marine receiving and staging facility located at Bathurst Inlet, approximately 130 km north-northwest of the Goose Site referred to as the MLA. During the construction phase and throughout the LOM, equipment, supplies, and fuel will be transported to the MLA by ocean-going, ice-class barges and ships from western and eastern ports in Canada. The MLA will be connected via WIRs to the Goose Site during construction and operations.

The MLA will be used to receive fuel, cargo, and consumables during construction and operation of the Project. Products will not be exported via the MLA as gold doré will be transported by air directly from the Goose Site.

Fuel and cargo will be received and staged at the MLA during the summer months when there is no sea ice, typically from August to September, and will be transported to the Goose Site by truck via the WIR from January to April. Outside of these periods, MLA activities will be limited to on-site storage and monitoring for loss prevention.

During periods of marine receiving and staging activities, crew transport between the MLA and Yellowknife, NWT will be facilitated by an ice airstrip during the winter months (January to April) and by float planes in the summer months (August to September). Between operational seasons, occasional personnel transport will be supported by helicopter service from the Goose Site.

18.3.1.2.2 MLA Functional and Design Criteria

The MLA has been designed to handle the off-loading and storage of 40 ML of diesel and 15,000 t of consumables per annum during operations, and 7 ML of diesel and 12,000 t of materials per annum during construction.

A laydown area capable of storing 610 ISO sea containers (20 ft long) will be constructed along with four 10 ML fuel tanks. Additional laydown area is provided to accommodate additional storage requirements during construction.

Fuel will be transported to the MLA using double-hulled ice-class tankers with the following specifications:

- Fuel capacity of 15,000 to 60,000 m^3;
- Dead weight tonnage (DWT) of 12,000 to 35,000 t; and
- Draft of 8 to 10 m.

The fuel tankers will use anchors to secure the ship offshore during off-loading. During the initial construction season, the MLA will have a Level 1 Oil Handling Facility (OHF) classification that permits a maximum oil transfer rate 150 m^3/hr. In Year 1 and for the remainder of the mine life, the MLA will have a Level 2 OHF classification which will permit transfer rates to a maximum of 750 m^3/hr. OHF standards are defined in TP-12402 – OHF Standards. The fuel supplier will supply and connect floating 200-mm discharge hoses to a shore installed connection that will allow the fuel to be pumped to the fuel storage facility. The fuel supplier will provide all spill response equipment required for the classification facility as detailed in TP-10783 – Arctic Waters Oil Transfer Guidelines.

Cargo will be transported to the MLA by either ocean-going barges or ships. The design specifications for ocean-going barges and ships are as follows:

Ocean-Going barges:

- DWT of 16,000 t; and
- Draft of 5.9 m.

Ships:

- DWT of 17,000 t; and
- Draft of 9.7 m.

The ships and barges will be self-sufficient for off-loading cargo. Lightering barges will be used to transfer cargo from the vessel to the lighter barge terminal at the MLA barge landing area. Freight will then be hauled to a laydown area where is will be stored until the annual WIR is open.

The Lightering Barge Terminal is designed to be removed at the end of each sealift season and re-installed prior to the arrival of the first sealift vessel the following year.

18.3.1.2.3 Navigation to Bathurst Inlet

Fuel and cargo will be shipped from the southern Canadian ports of Vancouver, BC and Bécancour, QC to the MLA based on the cost. The cargo will be transported to either port location. Since both routes would require the vessels to travel through Canadian Arctic waters, all vessels would comply with Transport Canada's Arctic Waters Pollution Prevention Act (AWPPA). The AWPPA deals with shipping in Canadian waters that lie north of 60° north latitude. The controlling regulation made under the AWPPA is called the Arctic Shipping Pollution Prevention Regulations (ASPPR). The ASPPR provides for construction and machinery standards for various classes of ice-strengthened vessels and establishes when and where in the Canadian Arctic such vessels can navigate based on their class and ice conditions in the area. Figure 18.4 shows the vessel routes from both Vancouver, BC and Bécancour, QC.

Figure 18.4: Shipping Routes

Source: Sabina 2015

The ASPPR governs the navigation through what is commonly known as the Zone/Date System. In the Zone/Date System, the Arctic waters are divided into 16 Shipping Safety Control Zones, with a schedule of earliest and latest entry dates for each zone corresponding to specific categories of vessels. Zone 1 has the most severe ice conditions and Zone 16 the least.

Type B vessels travelling the eastern route from Bécancour, QC will pass through Zone 6 and are only permitted to navigate in this zone from August 25 to September 30. A similar vessel travelling the western route from Vancouver, BC can pass through Zone 11 which allows navigation from July 15 to October 20.

All vessels transporting fuel and cargo will be equipped with appropriate navigational aids as per Transport Canada regulation; this includes the requirement under the ASPPR for an Ice Navigator to be on board all fuel tankers.

The MLA is located in a peninsula just north of the BIPR port site with a northing of 7394976 and easting of 381254 UTM NAD 83 Zone 13 N. This location allows the MLA to be constructed on relatively less steep terrain while providing sufficient water depth for ships at a reasonable distance from the shore.

18.3.1.2.4 MLA Layout and Marine Infrastructure

The MLA comprises the marine infrastructure, the laydown area, and the upland infrastructure. Upland infrastructure includes the following (see Figure 18.7):

- Diesel fuel storage tank farm;
- Container storage for 610 containers (20 ft, 1 TEU);
- Construction laydown area;
- Heated warehouse;
- Power plant;
- Maintenance shop;
- Desalination plant;
- Fresh/fire water storage and distribution;
- 94-person camp with offices; and
- Ancillary equipment for site operation.

The marine infrastructure comprises a single grounded terminal barge that will accept lighter barges. Lightering barges will shuttle freight from the ocean-going vessels that are moored at a water depth of approximately 12 m to the terminal barge. A foreshore ramp provides access from the terminal barge to the laydown area. The terminal barges will be secured in place by mooring them to onshore bollards. The terminal barge will be removed from the water at the end of each sealift season and re-installed for the next year’s sealift.

Other components of the MLA include an onshore fuel manifold for off-loading fuel from tankers.

18.3.1.3 Site Infrastructure

18.3.1.3.1 Maintenance Shop

A maintenance shop (fabric building with compacted-fill floor) will be provided to allow the maintenance of vehicles and other equipment at the MLA. The building will be heated with waste-oil/diesel-fired heaters and equipped with the necessary tools and equipment. This shop is not intended for the winter trucking fleet; maintenance on that fleet will be completed at the larger Goose truck shop.

18.3.1.3.2 Warehouse and Laydown Area

A heated warehouse (fabric building with compacted-fill floor) will be provided to store the antiscalant until it can be shipped from the MLA to the Goose Site. Material or equipment that requires protection from the elements at the MLA will be cold-stored in sea containers. Materials that can be exposed to the elements will be stored in the laydown areas.

Packaged explosives and detonators will not be shipped by sea so a dedicated explosives storage area or magazine is not required.

Because the operations freight area will be insufficient, a separate construction laydown area will be constructed to store construction material. The construction laydown pad will be built to a minimal thickness and can be expected to experience some freeze-thaw heaving.

18.3.1.3.3 Camp and Administration Offices

A camp with a single-occupancy room configuration will be used during the construction and operation phases at the MLA to accommodate up to 94 workers. There will be two dorms with a "gang" bathroom arrangement. The dorms will be connected to the kitchen/dining room/recreation complex with an arctic corridor.

A fabric building ERT with a compacted-fill floor is provided to house an F250 pick-up with an emergency response "camper". No fire engine is provided.

18.3.1.3.4 Site Communications, Fire Protection and Security

Communications systems, security and fire protection will be similar to those described for Goose.

18.3.1.3.5 Fuel Storage

Fuel storage at the MLA is designed with capacity for a one-year supply of diesel fuel required for operations of both sites. Table 18.11 provides the annual fuel storage requirements based on the usages from the Goose and MLA sites. Maximum fuel will be required in operational Year 5 (39 ML). A total of four 10 ML field-erected fuel tanks will be constructed at the MLA to handle the maximum requirement. Initially, one 10 ML fuel tank is erected in Year -3, two more 10ML fuel tanks are erected in Year -2, and the final 10 ML tank is installed in Year -1.

Table 18.11: MLA Site Fuel Storage Requirements (thousand litres)

	Goose Site	MLA Site	Total Fuel
Year 1	27,120	1,338	28,458
Year 2	28,131	1,401	29,532
Year 3	34,016	1,469	35,485
Year 4	37,021	1,491	38,512
Year 5	37,515	1,493	39,008
Year 6	37,166	1,491	38,656
Year 7	36,133	1,472	37,605
Year 8	30,015	1,394	31,409
Year 9	26,120	1,350	27,470
Year 10	18,944	1,260	20,204
Year 11	18,861	1,198	20,060
Year 12	14,225	-	14,225
Total Fuel	**345,266**	**15,356**	**360,622**

The fuel tank storage type and construction method is the same as at the Goose Site.

The fuel tanks are filled directly from the fuel supply ships through the shore manifold, and a booster pump module is also located at the shore. The fuel storage area is equipped with a tanker/light vehicle fueling module for filling the tanker trucks that transport the fuel to Goose on the WIR and for fueling local vehicles.

18.3.1.3.6 Blasting Agents Storage

Ammonium nitrate will be shipped in sea containers and stored at the MLA. Blasting agents are not mixed at the MLA Site.

18.3.1.3.7 Waste Management

Bermed and HDPE-lined areas will be constructed at the construction laydown area: one to function as a land farm, and a second to function as a hazardous materials storage area.

An incinerator will be provided to burn kitchen waste and other acceptable combustible materials.

18.3.1.3.8 Power Supply, Generation and Distribution

A small power plant will be installed at the MLA to meet the demands as summarized in Table 18.5. There will be two 500kW generator sets at the MLA. The sizing philosophy is N+1 for site power requirements.

18.3.1.3.9 Utilities and Services

Potable Water

Ocean water will be pumped from Bathurst Inlet (via a pump station located next to the inlet) to a treatment plant located near the camp. This plant will desalinate and treat the water for use as potable water and to feed the Fresh/Fire Water Tank. The plant is sized for 94 people based on a consumption of 275 L/person/day. The plant will be a Reverse Osmosis (RO)-type plant and will be contained in a 40-ft shipping container which is fully shop-assembled prior to shipment to site.

18.4 Tailings Management System Facility

18.4.1 Tailings Management System Design Requirements

The tailings management system at the Goose Site will entail deposition of 12.4 million tonnes (Mt) of tailings at two separate locations. The initial five years of tailings will be deposited in a purpose-built TSF located about 2 km south of the Goose Main open pit. Tailings deposition will then transition to in-pit deposition into the mined-out Llama open pit (referred to as Llama Tailings Facility or Llama TF) for the remaining seven years of the mine life. This is summarized in Table 18.12.

Table 18.12: Back River Project Tailings Management System Storage Requirements

Location	Period (Year and Quarter)	Tailings (Mt)	Tailings (Mm^3*)
TSF	Y1 Q1 to Y5 Q1	4.4	3.6
Llama TF	Y5 Q2 to Y12Q3	8	6.7
Total Project	Y1 Q1 to Y12 Q3	12.4	10.3

*The tailings density is 1.2 t/m^3. This value was selected after consideration of tailings test data and applying engineering judgement.
Source: JDS 2015

18.4.2 Tailings Geochemical Considerations

Based on two phases of geochemical testing, tailings will be considered PAG. Humidity cell tests indicate that sulphate and arsenic concentrations were elevated in all three tailings samples. Concentrations of cadmium, cobalt, nickel, and zinc increased in response to decreasing pH conditions that occurred in the master composite sample. These findings suggest that the tailings will need to be managed to prevent metal-leaching and acid rock drainage.

Llama TF will be flooded at closure which will prevent acidic conditions from developing. The TSF will be covered with waste rock as part of ongoing reclamation during the mine life.

18.4.3 TSF Containment Dam

18.4.3.1 Design Criteria

The complete TSF containment dam design criteria is listed in Table 18.13 and are consistent with best practices, including the Canadian Dam Association (CDA) (2013) guidelines.

Table 18.13: Summary of TSF Containment Dam Design Criteria

Component	Criteria
Dam Hazard Classification	High
Design Life Active use period as water retaining structure Design basis as active water retaining structure Total life until breach	 5 years 7 years 12 years
Dam staging	None
Tailings production rate	Ramp-up period, with a maximum rate of 3,000 t/d
Tailing slurry content	49% solids (by weight)
Tailings plasticity	Non-plastic
Tailings grind size	P80: 50 micron
Tailings solids specific gravity	2.9
Tailings settled density	1.2 t/m^3
Tailings storage requirement • By Mass • By Volume	 4.4 Mt 3.6 Mm^3
Ice entrainment allowance • Percentage of Tailings Capacity • By Volume	 20% 0.73 Mm^3
Contact water storage requirement	Average during operations 0.8 Mm^3 95^{th} percentile during operations 1.1 Mm^3 Max at TSF closure 174,000 m^3 (1:100 Yr)
Total TSF storage requirement (tailings, ice entrainment & contact water)	Average during operations 5.1 Mm^3 95^{th} percentile during operations 5.4 Mm^3
Tailings beach slope • Subaerial tailings • Sub-aqueous tailings	 1% 1%
Tailings deposition method	Single point spigot subaerial discharge (three locations over the LOM)
Maximum design earthquake (MDE)	1:2,475 year recurrence event; Peak Ground Acceleration (PGA) of 0.036 g
Inflow design flood (IDF)	1/3 between 1:1000 year event and probable maximum flood (PMF); 18-31 approx. 131 mm
Freeboard requirement: • Wind Set-up and Wave run-up allowance • Probable maximum flood storage allowance • Total freeboard (sum of above)	1.3 m 0.7 m 2.0 m
Stability Factors of Safety (Static)	1.3 during construction 1.5 during operation and closure
Stability Factors of Safety (Pseudo-Static)	1.0 following earthquake

18.4.3.2 Foundation Conditions

A geotechnical site investigation along the proposed centerline of the TSF was carried out during the winter of 2015 (SRK 2015x – SRK Consulting (Canada) Inc. (2015h), "Back River 2015 Overburden Geotechnical Drilling Program Report"; Draft Report prepared for Sabina Gold & Silver Corp., Project No. 1CS020.009, in progress). This investigation resulted in the adoption of the three foundation zone types as listed in Table 18.2.

18.4.3.3 Containment Dam Concept

A frozen foundation dam will allow for construction of a conventional dam on the deep permafrost foundation, while making use of the permafrost conditions to seal the water retaining feature of the dam with the foundation. There are no known deposits of low-permeability soils on the Property, and arctic conditions pose great challenges for the construction of low-permeability cores. Therefore, the decision was made to construct the TSF containment dam as a frozen foundation rock-fill dam built from ROM waste rock with a geosynthetic liner. For this concept to function, the liner will be frozen into the key trench permafrost, and over the life of the structure, the foundation should not thaw deep enough to compromise this seal.

18.4.3.4 Containment Dam Geometry

The TSF containment dam will be constructed as a relatively low and elongated structure, with a maximum height of about 14 m in the central section. The 1,744 m long dam centerline consists of two limbs articulated at an angle of 160° with the apex in the north direction. The two straight portions are similar in length, with the west side at 912 m and the east side at 832 m.

The crest of the dam was designed to have a 10 m width. This width was selected for ease of construction using the mining fleet of 64 t haul trucks. The upstream slope has a grade of 4H:1V while the downstream slope is 2.5H:1V.

The key trench will be aligned with the centerline of the dam such that the upstream toe of the key trench coincides with the dam centerline.

To ensure the TSF remains within the PDA, a small retaining dyke is required along the southern end of the facility. This South Dyke, constructed of waste rock, will be about 200 m long and 3 m high. As tailings will be deposited directly against the dyke, an impermeable liner will not be required.

18.4.4 Tailings Management Systems Operations

Tailings will be discharged from three points within the TSF, representing three periods in the deposition and tailings beach development. Deposition at the first and third discharge points are short in duration (2.5 and 4.3 months, respectively), with the bulk of the tailings deposited from the second discharge point at elevation 310 masl.

In the first period, tailings will be deposited from the crest of the South Dyke to fill in the southern end of the TSF and create a sloped surface that will direct the tailings and the water toward the north end of the TSF. During the second period of the deposition, tailings will be discharged from an elevation of 310 masl, which is 3 m higher than the crest of the South Dyke. However the first period tailings will limit the footprint of this cone. Beach development will be controlled in the beginning of the second period and local ponding will be prevented by trenching along the west embankment.

Tailings will be deposited in Llama TF using a single spigot discharge point. This discharge location will be changed over the life of deposition to ensure that a near struck (i.e. horizontal) tailings surface is created when production ceases.

18.4.4.1 Closure and Reclamation

Following completion of tailings deposition in the TSF, the facility will be drained and the water pumped to Llama TF. During this period, Goose Main open pit will be developed and the tailings surface of the TSF will be covered with 5 m of waste rock from this pit. This cover will ensure that the tailings surface will freeze. Once covered, only a portion of the west limb of the dam will be visible. That portion of the TSF will continue to allow storage of seasonal contact water until Year 12 and will remain functional until the dam is breached and the water is allowed to drain into Goose Main Pit when active water treatment ceases.

Llama TF will be closed once tailings deposition ceases in Year 12. Closure will entail a permanent water cover of 5 m. Water from the Llama TF will continue to be treated in accordance with the Project water management plan until such time as the discharge water quality meets Metal Mine Effluent Regulation (MMER) criteria.

18.5 Water Management Plan

18.5.1 Water Management Overview and Strategy

The water management plan includes lake dewatering, permanent and temporary diversions, pumping systems, and temporary and permanent holding ponds, from construction through operations and final closure. A water management plan is prepared for the Goose only, as the MLA does not require water management infrastructure beyond best management practices.

Water on the Project is categorized into the following types:

- Contact water – surface water runoff that comes in contact with disturbed areas. This includes runoff from WRSAs, ore stockpiles, open pits, the TSF, and infrastructure rock-fill pads.
- Saline connate water – groundwater inflows during underground development and open pit mining where a talik zone is present.
- Non-contact water – all other surface runoff that does not come in contact with disturbed areas.

Each type of site water is managed separately. Non-contact water diversions are constructed to minimize the volume of contact water on site. Saline connate water from groundwater inflows is pumped via a separate pipeline to the Umwelt open pit once mining is complete. Due to the presence of permafrost, excavation into overburden soils for diversion structures is minimized where possible.

The key objective of the water management plan is to provide infrastructure that can manage water inventories for a wide range of climatic conditions by applying best management practices to reduce effects to the environment. Primary objectives are listed as follows:

- Provide a reliable water supply to the process plant;
- Facilitate mining of the deposits by limiting inflows to the open pits and underground mine by timely removal of groundwater inflows and precipitation;
- Manage contact, non-contact, and saline connate water separately in order to minimize volume of contact water collected and requiring treatment on site; and
- Collect and treat contact water that would otherwise impair water quality of the receiving systems.

18.5.1.1 Water and Load Balance Model

A water and load balance model was developed for the Goose Site using a GoldSim model to optimize the water management strategy and evaluate water treatment needs during construction, operations, closure and post-closure in order to meet water quality guidelines. The water and load balance model is based on mass balance principles, available hydrology inputs, water management plans, the mine schedule, and the best available water chemistry inputs.

A water and load balance was not developed for the MLA, since the scope of activities are such that water management infrastructure beyond best management practices are not required.

18.5.1.2 Water Quality Predictions

Water quality predictions were evaluated in the TSF, Llama TF, and open pits at the Goose Site as well as downstream locations (prediction nodes). A total of ten prediction nodes were strategically chosen to assess the water quality effects from mine infrastructure and to optimize the required treatment to meet water quality objectives.

18.5.2 Water Management

18.5.2.1 Infrastructure

Contact water is stored in a series of event ponds and reservoirs, depending on water type. Event ponds are sized to contain a designated storm event and are dewatered on a regular basis. Long-term water storage facilities are denoted as reservoirs. The capacity requirements of ponds and reservoirs are shown in Tables 18.14.

Water on the site is conveyed either by gravity in open and purpose-designed drains or by pumping.

Prior to discharge to the environment, all contact water will be treated to reduce suspended solids, metals or other chemical constituents to meet regulatory water discharge criteria during construction, operations, and the closure phase.

Table 18.14: Goose Site Water Management Facilities

Description	Type of Water	Capacity (m³)
Llama WRSA Pond	Contact water	26,000
Umwelt WRSA Pond	Contact water	30,100
Ore Pond	Contact water	11,000
TSF WRSA Pond	Contact water	1,163,100
Primary Pond	Contact water	316,700
Umwelt Open Pit / Reservoir	Excess Contact Water / Saline Water	5,958,664
Goose Open Pit / Reservoir	Excess Contact Water	20,260,374

Source: JDS 2015

18.5.2.2 Water Treatment

Water treatment will be implemented during operations and closure to ensure the anticipated discharge limits are met. Water from Llama TF will need to be treated to remove high arsenic and copper concentrations. Water treatment will include a process of oxidizing residual cyanide using hydrogen peroxide. This process will liberate copper and other metals that may be complexed with cyanide, and oxidize available arsenite to arsenate. Ferric chloride will also be added to co-precipitate arsenic and potentially copper.

Water treatment discharge will be pumped back to Llama TF. Table 18.15 provides a summary of the treatment implemented during operations and closure. Llama TF water is treated until full and the water meets discharge criteria.

Table 18.15: Goose Site Water Treatment Summary

Phase	Feed Water	Discharge Location	Start	End	Flow Rate (m^3/d)	Primary Constituent	Comment
Operations	Llama TF	Pump back to Llama TF	Yr5,Q2	Yr12,Q3	8,500	TSS, Arsenic and Copper	Year-round operation until closure.
Closure	Llama TF	Pump back to Llama TF	Yr12,Q4	Yr18,Q2	8,500	TSS, Arsenic and Copper	Treat during open-water season only until Llama TF full and meets discharge criteria.

Source: SRK 2015

18.5.2.3 Underground Saline Water

As the Umwelt underground mine and the Llama open pit are developed, they are dewatered. The Umwelt underground development is below the basal permafrost and the Llama open pit intersects the talik zone beneath Llama Lake; as such, both of these mines are anticipated to be sources of groundwater inflows. This water has a high saline content (59,000 TDS) and is not suitable for discharge to the environment. This saline water is not treated but stored in the mined-out Umwelt open pit. Freshwater will be placed above the hypersaline water to create a meromictic lake at closure.

18.6 Mobile Equipment

Mobile site support equipment is used to provide support to operations at each of the two sites. A list of site support equipment by site is shown in 18.16.

Table 18.16: Mobile Equipment List

Equipment Description	Equipment Quantity	
	Goose Site	MLA Site
3/4 T Ambulance / Rescue - Ford F450	1	1
1 T Diesel Crew Cab Pick-up - Ford F350	7	1
2 T Diesel Pick-up (Blaster's Box) - Ford F550	1	0
2 T Diesel Pick-up c/w Heated Van - Ford F550	1	0
5 T Flat Deck Truck	1	1
10 T Fuel Truck - Western Star 4900 SA (Custom)	1	0
Tridem Water Tanker	1	0
Hydraulic Hammer for 320 Excavator	1	0
Fuel / Lube Truck - Western Star 6900 (Custom)	1	0
Welding / Service Truck - Ford F550 (Custom)	2	1
5 T Pumper - Fire Truck	1	0
100T Low-boy Trailer	1	0
Roll-Off Truck C/W Deck, Water Tank, Vacuum Tank, Garbage Bins (X2)	1	0
Dump Truck 10m3 Capacity- Western Star 4900	1	1
Winch Tractor with 60T Winch	1	0
Tri-Axle Single Drop, Scissor Neck Trailer	1	1
44 Passenger Bus - Freightliner	2	0
Excavator (~1.0 CU.M) CAT 320DL	1	0
Vibrating Packer - Cat CS56	1	0
Tool Carrier - Cat 930H (Old IT28)	2	0
Tool Carrier - Cat 966K (c/w Attachments)	1	1
Skid Steer Loader (1Cu.M)	2	1
5 T Fork Lift Zoom-Boom - Terex GTH-5519	1	0
Tire Manipulator Attachment for 966k	1	0
Container Handler - Taylor TXLC975	1	1
Track Dozer - CAT D6T	1	0
Mobile Crane - RT30 - Grove RT530E	1	0
Mobile Crushing/Screening Plant	1	0
165T Crawler Crane	1	0
Mobile Crane - RT90 - Grove RT890E	1	1
Pipe Fusing Machine (Able to Fuse 28" DR17)	1	0
Pipe Fusing Machine (Able to Fuse 12" DR11)	1	0
Portable Diesel Light Plants	5	2
Portable Diesel Heaters	5	4

18.7 Manpower

Each of the two sites will have a site support manpower crew that will provide support to the operations. The crews will be responsible for the following duties:

- Maintain and repair infrastructure facilities;
- Transfer freight from the storage areas to the warehouse and operation centres;
- Transport personnel between the camp and aircraft;
- Load and unload aircraft;
- Conduct airstrip operations and maintenance;
- Perform waste management duties (i.e., incineration, water treatment, hazardous waste handling);
- Provide plant-site snow removal;
- Manage site water;
- Oversee mobile crusher operations (Goose only); and
- Oversee tailings storage facilities operations (Goose only).

Table 18.17 provides the make-up of each site support manpower crew.

Table 18.17: Support Manpower Crew

Labour Position	On-Site Labour	
	Goose Site	MLA Site
Surface Foreman	1	1
Electrician	1	0
Facilities Maintenance - Tradesman	1	1
Mobile Equipment Operator	3	1
Labourer/Apprentice	2	1
Total Support Crew (Onsite)	**8**	**4**

Source: JDS 2015

19 Market Studies and Contracts

Detailed market studies on the potential sale of gold doré from the Back River Gold Project were not completed. However, JDS confirmed the refining and payable terms with a leading industry entity to determine indicative terms with respect to doré production. The terms were reviewed and found to be acceptable by QP Gordon Doerksen, P. Eng.

No contractual arrangements for shipping or refining exist at this time. Table 19.1 shows the terms used in the economic analysis.

Table 19.1: NSR Assumptions used in the Economic Analysis

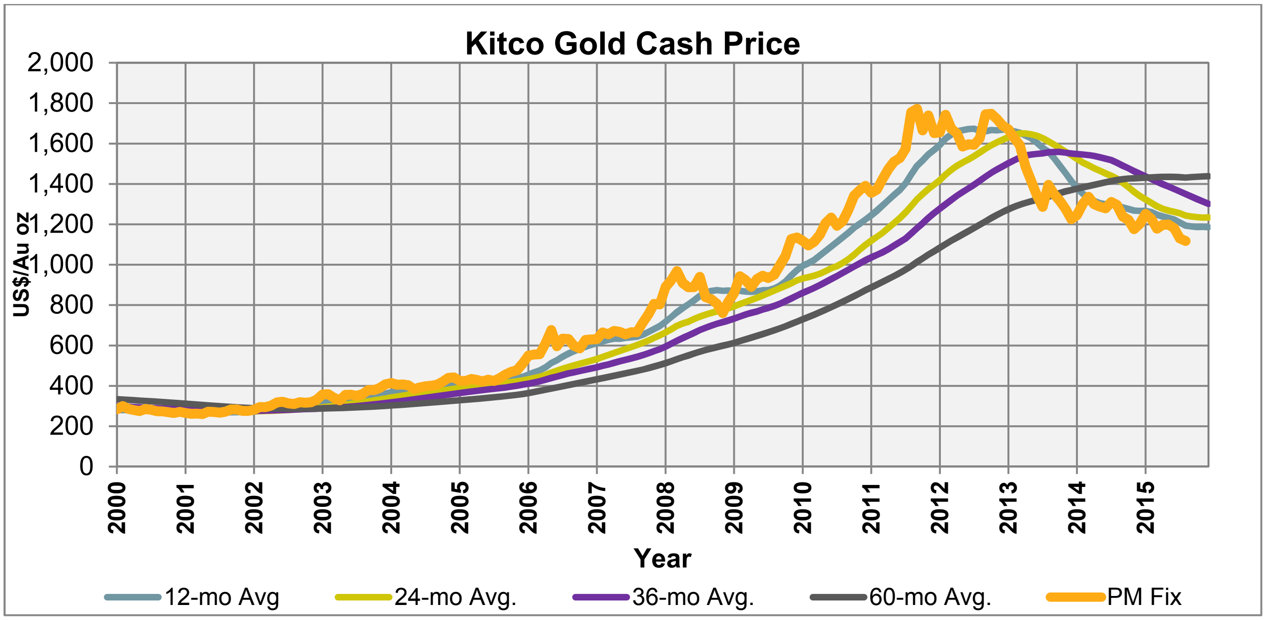
Assumptions	Unit	Value
Au Payable	%	99.8
Au Refining Charge	US$/oz	1
Insurance	% of payable value	0.15
Transport Cost	US$/oz	1

Source: JDS 2015

19.1 Metal Prices

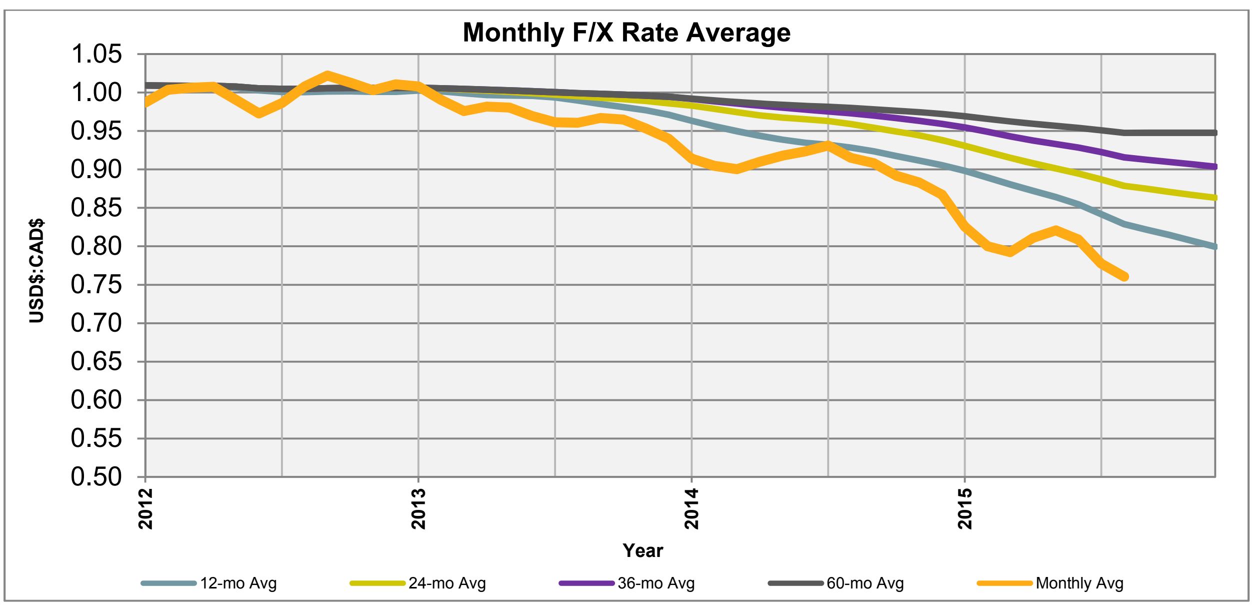
The precious metal markets are highly liquid and benefit from terminal markets around the world (e.g., London, New York, Tokyo, and Hong Kong). Historical gold prices are shown in Figure 19.1 and indicate the change in metal prices from 1998 to 2015. Historical, average US$:C$ exchange rates are shown in Figure 19.2.

Figure 19.1: Gold Price History



Source: JDS 2015

Figure 19.2: Monthly Average US$:C$: Foreign Exchange Rate (Bank of Canada)



Source: JDS 2015

The gold price used in the economic analysis is based on the August 2015, 3-month average price sourced from Kitco Metals Inc. The US$:C$ exchange rate used in the economic analysis is based on the August 2015, 6-month average price sourced from Bank of Canada. A sensitivity analysis was completed as part of the overall economic analysis. The results of this are discussed in section 23. Table 19.2 shows the metal price and exchange rate used in the economic analysis.

Table 19.2: Metal Price and Exchange Rate used in the Economic Analysis

Assumptions	Unit	Value
Au Price	US$/oz	1,150
F/X Rate	US$:C$	0.80

Source: JDS 2015

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Overview

One of Sabina's highest priorities is to manage and mitigate the potential effects of the Project to the surrounding environment. Sabina is committed to exploration and mining practices that are environmentally responsible and socially acceptable, and dedicated to creating and maintaining a safe environment for the land, its employees, and nearby communities. Sabina subscribes to the principles of sustainable development in mining. While mining cannot occur without some effect on the surrounding natural environment, Sabina's responsibility is to limit potentially negative environmental and social effects and to enhance potentially positive effects.

20.1.1 Sustainable Development Policy

Sabina Gold & Silver Corp. regards itself as a responsible explorer and mineral developer. We are committed to fostering sustainable development throughout all stages of our activities. We constantly strive to conduct our operations in a manner that balances the social, economic, cultural and environmental needs of the communities in which we operate. To build on this commitment Sabina will:

- Meet or strive to exceed all relevant legislated sustainable development requirements in the regions where we work;
- Ensure appropriate personnel, resources and training is made available to implement our sustainable development objectives;
- Establish clear lines of responsibility and accountability throughout the Company to meet these objectives;
- Implement proven management systems and procedures to facilitate our sustainable development objectives. A priority will be placed on developing and implementing management structures related to the environment, health and safety, emergency response and stakeholder engagement;
- Act as responsible stewards of the environment for both current and future generations. We will make use of appropriate assessment methodologies, technologies and controls to minimize environmental risks throughout all stages of mineral development;
- Work closely with local communities and project stakeholders to understand their needs, address their concerns and provide Project-related benefits to create win-win relationships. Our goal is to earn and maintain a social licence to operate at all our operations while building partnerships;
- Pursue economically feasible projects in order to generate shareholder profitability and support long-term positive socio-economic development in the regions where we work;

- Utilize a precautionary approach as it applies to potential effects from our activities. Work with employees, contractors and stakeholders to promote a culture of open and meaningful dialogue to ensure that any known or suspected departures from established protocols are reported to management in a timely manner;
- Regularly review this policy to ensure it is consistent with Sabina's current activities and the most recent legislation;
- Continually improve our performance and contributions to sustainable development including pollution prevention, waste minimization and resource consumption; and
- Implement programs at each of our operations to monitor and report compliance and proactively address potential deficiencies in our policies and procedures.

20.2 Environmental Assessment for Mining Projects

20.2.1 General

The Nunavut Lands Claim Agreement (NLCA) was the basis for creating Nunavut Territory in 1999. Through the NLCA, surface and subsurface rights for some parcels of land were entrusted to the Inuit. The designated Inuit organization under the NLCA is Nunavut Tunngavik Incorporated (NTI), who retains administration of the subsurface mineral rights for Inuit-Owned Land (IOL). Surface land rights (including water and wildlife) for IOL are vested from NTI to the Regional Inuit Associations (RIA). All other surface and subsurface rights in Nunavut are managed by the Crown through Aboriginal Affairs and Northern Development Canada (AANDC), except for the communities within the territory. Communities and municipalities are within the Commissioners' lands and they are managed by the Government of Nunavut.

The Project falls within the Kitikmeot Region of Nunavut and includes surface rights managed by the Kitikmeot Inuit Association (KIA), and AANDC. Subsurface mineral rights are administered by AANDC.

Five management boards were created within the NLCA, as listed in Table 20.1. These Institutes of Public Government include representatives of NTI, the Crown, and the Government of Nunavut (GN), and are responsible for resource management in Nunavut.

Table 20.1: Nunavut Boards (Institutes of Public Government) and Associated Responsibilities

Board	Responsibilities under NLCA
Nunavut Wildlife Management Board	Wildlife Management
Nunavut Planning Commission	Land Use Planning and Cumulative Effects
Nunavut Impact Review Board	Environmental Assessment
Nunavut Water Board	Water Resource Management
Nunavut Surface Rights Tribunal	Dispute Resolution for Land and Water Resource Management Issues

Source: Sabina Gold & Silver Corp. 2014

New and modified mining projects in Nunavut are subject to an environmental assessment (EA) and review prior to certification and issuance of permits to authorize construction and operations. The EA is a means of ensuring that potential adverse environmental, social, economic, health, and heritage effects, or potentially adverse effects on the local communities and Aboriginal peoples, are addressed before Project approval.

The two main stages in the EA process are the pre-application phase, when studies and consultations are undertaken, and the application review phase, when Project details and effects on environment and communities are reviewed. The scope, procedures, and methods of each assessment are generally flexible and tailored specifically to Project circumstances.

The primary environmental review and approval process that applies to the Project is the territorial EA administered by the Nunavut Impact Review Board (NIRB). Some federal regulatory requirements and processes that were applicable prior to the NLCA continue to apply in Nunavut. However, the *Canadian Environmental Assessment Act* is no longer applicable to the Nunavut Settlement Area due to the amendment of the NLCA on May 29, 2008.

In general, an EA contains the following four common main elements:

- Opportunities for all interested parties to identify issues and provide input, including Aboriginal and neighbouring jurisdictions;
- Technical studies regarding the relevant environmental, social, economic, heritage, and health effects of the proposed Project;
- Proposals to prevent or minimize undesirable effects and enhance desirable effects; and
- Consideration of the input of all interested parties in compiling the assessment findings and making recommendations about Project acceptability.

A Project Certificate, issued by the NIRB and approved by the Minister of AANDC, represents government approval in principle and allows the Proponent to pursue the necessary regulatory authorizations required to construct and operate the Project.

20.3 Environmental Studies

20.3.1 Overview

Regional environmental baseline programs have been conducted in the area over the past 10 years because of the various proposed mineral development, infrastructure projects (e.g., the Hackett River Project and the BIPR Project), and government monitoring programs. Data from these regional studies are also relevant to the Back River Property.

In 2008, baseline data collection was initiated on a limited scale at the Property. Hydrology and meteorology baseline data were collected in 2008, as well as fish habitat and community information. No baseline studies were conducted in 2009, with the exception of the continued operation of the meteorology station located near the Goose exploration camp.

Additional baseline studies were carried out each year between 2010 and 2015. Baseline work during this period included archaeology, metal-leaching / acid rock drainage (ML/ARD), hydrology, meteorology, air quality (including dustfall monitoring), noise, freshwater and marine water quality, sediment quality, aquatic biology, bathymetry (lake, marine, and pond), freshwater and marine fish and fish habitat, ecosystem mapping, vegetation/wetlands, soil and terrestrial landforms, rare plants, country foods, wildlife (terrestrial and marine), socioeconomics, traditional knowledge, and land use.

Six years of comprehensive environmental baseline investigations have been completed in support of the Project to date. The description of the existing environment in this Initial Project Feasibility Study is based on the results of the baseline studies. Further detail, including copies of the various baseline reports, is provided in the Draft Environmental Impact Statement (DEIS), which was submitted to the NIRB in January 2014.

The primary objectives of the baseline studies are to characterize the natural and human environment aspects of potentially impacted areas, as well as identification of reference locations for comparison throughout the Project life to assess impact.

Environmental baseline data describe the existing (pre-development) environment and assists with following:

- Helps to inform the Project design;
- Allows an assessment of likely environmental effects at the Project, including comparisons with established environmental guidelines, thresholds, and limits, where applicable; and
- Provides a reference for future environmental monitoring (i.e., it allows for the comparison of pre-development and post development conditions).

Standard field protocols and scientific methodologies have been used to accurately document the baseline studies. The studies also considered the information needs of regulatory agencies for approval of previous projects in Nunavut. The baseline studies included the collection of site-specific information, as well as documentation of applicable published material.

20.3.2 Fish Habitat Compensation

Construction and mining activities at the Goose Site will result in serious harm to fish habitat due to the development of the Goose Main deposit, and the loss of Llama Lake.

Freshwater offset options are currently being evaluated and are anticipated to incorporate a number of off-site habitat restoration features.

20.4 Community and Government Engagement and Consultation

Sabina is an active member of the Kitikmeot region community with a regional office in Cambridge Bay, staffed by a Community Liaison Officer (CLO). This office provides a local resource where residents can meet with Sabina personnel and obtain information about the Project. Sabina has also developed and advanced a community engagement program; for example, Sabina established a donations policy that focuses on supporting initiatives pertaining to "youth and education" and "community wellness and traditional lifestyles" in the Kitikmeot region. To date, Sabina has donated approximately $120,000 and a variety of in-kind support to Kitikmeot region initiatives.

Sabina has engaged local communities and the Kitikmeot Inuit Association in the Project planning activities. Through public and stakeholder meetings, site tours, and regular communications (such as newsletters and a Project website), Sabina strives to ensure engagement with all residents of the local communities. Public meetings have been held in various communities to discuss and help people understand the Project as it progresses.

Sabina's community engagement program commenced in early 2012 and has since made the following important advances:

- Four complete rounds of community visits to the Kitikmeot region were completed. In June 2012, April 2013, November 2013, and June 2015 each of the Kitikmeot communities were visited (i.e., Cambridge Bay, Kugluktuk, Gjoa Haven, Taloyoak, Kugaaruk), with the exception of Gjoa Haven in June 2015, and meetings were held with the public, Hunters and Trappers Organizations (HTOs), and Hamlet representatives. Public meetings were also held in November 2012 in Cambridge Bay, Kugluktuk, and Yellowknife (which included an informational meeting with the Yellowknife's Dene First Nation), and August 2013 in Cambridge Bay and Kugluktuk. Public meetings in Yellowknife and meetings with various NWT Aboriginal organizations were also held in November 2013, and June 2015. Public meetings in the Kitikmeot region have generally been well-attended, with anywhere from 30 to over 100 residents attending;
- Private meetings with Kingoak and Omingmaktok residents were arranged during November 2012, November 2013, October 2014, and June 2015 visits to Cambridge Bay. Representatives from these two communities also sit on the Cambridge Bay Community Advisory Group;
- Sabina participated in NIRB's scoping tour for the review of the Project proposal. Company representatives were in attendance for all of the public scoping meetings held in the Kitikmeot region and Yellowknife. Sabina representatives were available to the public throughout these meetings to answer community questions and share Project-related information. The details of these meetings are available in the NIRB report "Public Scoping Meetings Summary Report for the NIRB's Review of Sabina Gold & Silver Corp.'s "Back River" Project (NIRB file no. 12MN036)";
- Sabina participated in some of NIRB's March/April 2014 community information sessions for the review of Sabina's Back River Project DEIS. Company representatives were in attendance for the community information sessions held in Kugluktuk and Yellowknife, but were unable to attend the Cambridge Bay, Gjoa Haven, Taloyoak, and Kugaaruk sessions due to weather and flight cancellations. Sabina representatives were available to the public throughout the meetings they did attend to answer community questions and share Project-

related information. The details of these meetings have been captured in the NIRB report "Public Information Meetings Summary Report for the NIRB's Review of Sabina Gold & Silver Corp's Back River Project, March 24 - April 1, 2014";

- Sabina participated in the November 2014 Technical Meeting and Pre-hearing Conference hosted by NIRB on the Back River Project in Cambridge Bay. Company representatives were in attendance for the entire set of meetings to make presentations, answer questions, and share Project-related information. The details of these meetings have been captured in the NIRB report "Nunavut Impact Review Board Pre-hearing Conference Decision Concerning the Back River Project (NIRB File No. 12MN036) Proposed by Sabina Gold & Silver Corporation";
- Community Advisory Groups were established in Cambridge Bay and Kugluktuk that consist of elder and youth representatives, and representatives from local Hamlets and HTOs. Sabina has met with these groups approximately every 4 to 8 months; these groups also participated in a Back River Project site tour in September 2012 and July 2014; and
- Social media platforms were developed along with other means of distributing Project-related information to the Kitikmeot communities. For example, a Back River Project website (www.backriverproject.com) has been created that has Twitter and RSS feeds, in addition a regularly-produced community newsletter has also been developed. Sabina also participates in local radio shows and trade shows, and provides various informational materials (e.g., popular summaries, informational posters, handouts) to local communities.

Sabina will continue to advance its community engagement program during the EA and permitting process and throughout the development and operation of the mine. The results of the community engagement program (e.g., information obtained from communities and community stakeholders, and documented public comments and concerns) will also be integrated into Sabina's Final Environmental Impact Statement (FEIS) processes moving forward.

Sabina has also made the following significant efforts to provide government officials with Project information, plans, and timelines:

- Ongoing meetings and discussions have been conducted with the Northern Project Management Office of Canadian Northern Economic Development Agency (CanNor). The Project has been introduced through face to face meetings with the GN as well as with technical and senior staff in federal departments such as AANDC, Fisheries and Oceans Canada (DFO), Environment Canada, Transport Canada, and Natural Resources Canada (NRCan);
- A tour was conducted at the Back River Property in mid-September 2012, where nine officials representing five federal departments and the GN spent a day touring the Property to see the Project's location, areas of planned development, and to experience the current activities at the Project;
- In the fall of 2012, Sabina participated in an information session organized by the Northern Project Management Office. At this session, Sabina, in conjunction with MMG Canada Ltd. and Xstrata, presented the highlights of each of the projects' plans for the Kitikmeot region with an emphasis on coordination and coherence. There was significant participation across federal departments at this session;

- Through 2013 and 2014, numerous technical meetings were held with all relevant agencies to discuss aspects of the review related to their key areas of responsibility and mandate. Over 100 separate meetings were held and over 150 different officials were directly engaged;
- In 2013, both senior executive and technical officials toured the Project and over two days, were able to visit all of the potential Project areas;
- Various technical meetings were held to discuss key regulatory areas; and
- Property tours were completed in 2013 for both senior executive and technical level regulators.

The involvement of stakeholders will continue throughout the various Project stages. Key stakeholders include the following:

- Community governments of Kugluktuk, Cambridge Bay, Gjoa Haven, Taloyoak, and Kugaaruk;
- Aboriginal representative organizations such as the KIA and community HTOs;
- Various businesses, organizations, and non-government organizations;
- The general public;
- Federal government: AANDC, Environment Canada, DFO, NRCan, Transport Canada, CanNor;
- Nunavut Territorial Government: departments of Environment, Culture and Heritage, Labour, Economic Development and Transportation, and Health and Social Services; and
- Government of Northwest Territories, as related to potential cumulative and transboundary effects of the Project.

20.4.1 Aboriginal Communication and Engagement

The Project is located primarily on IOL under the NLCA, and Sabina is committed to working with the KIA as the Project advances. Under Article 26 of the NLCA, an Inuit Impact and Benefit Agreement (IIBA) at least 180 days prior to the start of major development negotiations should commence for the purpose of concluding an IIBA. Once negotiated, the IIBA will formally outline Sabina's social, economic, and environmental commitments to the people of the Kitikmeot region. IIBA terms and conditions have not yet been finalized.

It is important to Sabina to respect and properly document the Aboriginal traditional history of the Project area. For this reason, Sabina and the KIA signed a Traditional Knowledge Agreement in May 2012. Sabina has subsequently supported the preparation of Traditional Knowledge/Traditional Land Use studies that assess the use of the local area by Aboriginal peoples.

Under the direction of Sabina, the KIA prepared a report titled "Inuit Traditional Knowledge of Sabina Gold & Silver Corporation's Back River (Hannigayok) Project" (KIA 2012), which summarizes existing traditional knowledge in the vicinity of the Property. Traditional knowledge workshops in Kugluktuk and Cambridge Bay were conducted in August and September 2013 to help address traditional knowledge gaps. The data collection portion of these workshops focused on themes pertaining to Inuit heritage and land use, and the terrestrial and marine environments in the vicinity of the Property. The findings of these workshops have been presented in the report "Naonaiyaotit Traditional Knowledge Project – Hannigayok (Sabina Gold & Silver Corp. Proposed Back River Project), Results from Data Gaps Workshops, Final Report".

In an effort to develop a better understanding of the Arctic char fishery in the Nulahugyuk Creek - Hingittok Lake (Bernard Harbour) area (and related historic and contemporary environmental conditions), a traditional knowledge study was conducted in partnership with the Kugluktuk HTO. This is because Sabina (in partnership with the Kugluktuk HTO) is proposing stream restoration efforts in the Bernard Harbour area in order for Sabina to satisfy *Fisheries Act* offsetting requirements for the Back River Project.

The TK study is intended to complement the scientific baseline studies that have also been conducted and has involved one-on-one interviews with a number of Kugluktuk and Cambridge Bay residents. The TK study report was finalized in early 2015.

Finally, the document "Existing and Publically Available Traditional Knowledge from Selected Aboriginal Groups in the Northwest Territories" was prepared in 2014. This report identifies Traditional Knowledge and Traditional Use (TK/TU) of selected Aboriginal groups in the Northwest Territories in relation to the Project. One major focus of this review was on TK associated with caribou, as this is a topic where community concerns were deemed most likely to arise.

20.5 Regulatory Approval Process

20.5.1 Authorizations, Licences, and Permits

The major EA and regulatory requirements for mining projects in the Kitikmeot region are listed in Table 20.2. Additional ongoing permits will also be required for the life of the Project, such as land use permits and water licences. Article 20 of the NLCA requires compensation and an agreement for the alteration of quality or quantity of water on IOL. Several federal acts apply to the Project, including the *Fisheries Act* and the *Navigation Protection Act*.

20.5.1.1 Territorial Authorizations, Licences, and Permits

NTI is the organization that represents Inuit under the NLCA. NTI has established the Mining Policy for Nunavut, which states that the development of Mineral Resources in Nunavut will be supported and promoted by NTI if there are significant long-term social and economic benefits for the Inuit of Nunavut, and if the development is consistent with protecting the eco-systemic integrity of the Nunavut Settlement Area. The objectives of the NTI Mining Policy are as follows:

- Minimize negative impacts
  - Mining activities must take place in a way that is sensitive to wildlife, habitat, the environment, and traditional relationships with the land.
- Maximize the benefits of mining to Inuit
  - Benefits to Inuit must be maximized, while negative impacts to the land and Inuit culture must be minimized.
- Attract mining investment
  - NTI recognizes the value of mining to economic development in Nunavut, and the need for certainty with respect to mineral tenure and the right to mine.
- Resolve land use conflicts
  - NTI recognizes that objectives of Inuit and mining companies are not always the same. NTI will promote certainty and clarity for land access and resolve land use conflicts.
- Improve consultation and clarify decision-making
  - NTI's policy is to improve communications, consultation, and coordination among all stakeholders and clarify the decision-making process.

Sabina is committed to following these objectives during the planning, development, operational, and closure phases of the Project.

The settlement of the NLCA resulted in the transfer of Ownership and responsibility of certain surface and subsurface lands from the Crown (i.e., the federal government) to the Inuit. These lands are known as IOL. Surface activities on IOL are regulated through land use permits, quarry permits, and other licences that regulate specific activities managed through land leases. In the Kitikmeot region, these land use permits and land leases are issued and managed by the KIA. NTI manages subsurface IOL through mineral leases.

According to Article 20 of the *Nunavut Land Claims Agreement Act*, Inuit must be compensated for any substantial alteration of water quality or quantity on IOL. A water compensation agreement must be negotiated with the KIA. In addition a wildlife compensation agreement must be reached with the KIA.

20.5.1.2 Federal Authorizations, Licences, and Permits

Various pieces of federal legislation apply to mining projects in Nunavut. The agencies that would be involved in metal mining projects include AANDC, Environment Canada, DFO, Health Canada, Natural Resources Canada, and Transport Canada. AANDC regulates and manages the surface and subsurface Crown lands in Nunavut.

The *Fisheries Act*, administered by the DFO and Environment Canada, can play a substantial role in permitting mining projects in Nunavut. The requirements of DFO and Environment Canada (under the Metal Mining Effluent Regulations (MMER)) prohibit serious harm to fish or fish habitat in order to obtain their respective authorizations.

The *Navigation Protection Act*, administered by Transport Canada, can play a substantial role in permitting all-weather roads in Nunavut where stream crossings are required.

Table 20.2: Major EA and Regulatory Requirements for Mining Projects in Kitikmeot Region

Permit Process	Issuing/Lead Organization	Comments
Pre-environment Assessment	Nunavut Research Institute	Permits for allowing research activities (baseline studies) in Nunavut
		Wildlife Research Permits
	Department of Environment, Government of Nunavut	
	Department of Culture, Language, Elders, and Youth	Archaeological Research Permits
	DFO	Fisheries Research Permits
	Nunavut Planning Commission	Conformity with Approved Regional Land Use Plans if required
EA	NIRB	NIRB Screening, Part 5 Review, Part 6 Panel Review
Post-environmental Assessment	Environment Canada – DFO	MMER Schedule 2 Listing[1]
Permits and Licences	NTI	Subsurface IOL Leases
	KIA	Surface IOL Land Use Permits
		Surface IOL Quarry Permits
		Surface IOL Land Leases
		Inuit Impact and Benefits Agreements
		Wildlife Compensation
		Water Compensation
	Nunavut Water Board	Water Licence
	AANDC	Crown Land Leases
		Crown Land Mineral Leases
		Crown Land Use Permits
		Crown Quarry Permits
		Certificate of Occupation for Crown Land
		Licence of Occupation for Road Alignments on Crown Land
	DFO	Fisheries Authorizations
	Transport Canada	*Navigable Protection Act* authorizations; authorization for filling or dewatering a navigable waterway
		Canada Shipping Act
		Arctic Waters Pollution Prevention Act
	Natural Resources Canada	Blasting Permits
		Explosives Magazine Permits
		Radio Licensing
	Environment Canada	*Species at Risk Act*
		Migratory Birds Act
		Section 36 of the *Fisheries Act*
		Metal Mining Effluent Regulations

Source: Sabina Gold & Silver Corp. 2014

[1]: *This is not expected to be necessary.*

20.5.2 Environmental Considerations

The Project will entail significant on-site infrastructure development including the mines, Goose process plant, TSF, and ancillary facilities (e.g., on-site accommodations), as well as off-site infrastructure such as WIRs and the MLA. The complexity of the environmental review and permitting process will be dependent on the location of the facilities in relation to surface water at the potential development areas.

Through consultation and regulatory considerations, the following key environmental issues have been identified at the Project:

- Caribou and transboundary concerns;
- Waste rock and tailings management;
- Water quality;
- Geotechnical and permafrost;
- Shipping plan (sealifts); and
- Fisheries offset (Bernard Harbour Project).

Mitigation efforts and management plans will be an important component for addressing these issues and for addressing other issues that arise as the Project proceeds through the regulatory process.

20.5.3 Preliminary Prediction of Anticipated Environmental Effects

The design of the Project is ongoing, as is the development of appropriate mitigation measures. Table 20.3 provides a preliminary summary of the potential environmental effects associated with the construction, operation, and closure of the Project.

The following symbols and abbreviations are used in Table 20.3:

- "–" = negative;
- "D" = direct effect;
- "I" = indirect effect;
- "S" = short term; and
- "L" = long term.

Table 20.3: Preliminary Summary of Potential Environmental Effects

Component	Potential Effect
Mine Workings	Reduction in localized air quality due to the release of particulate from mining activities and heavy equipment diesel emissions (–DS). Increase localized sound emissions as a result of intermittent blasting activities, heavy equipment operation, and safety equipment (–DS). Alteration to the local terrain from excavation of the open pit and underground portals, forming a permanent surface depression in the landscape (–DL). Potential for loss of aquatic habitat due to the rerouting of streams and/or lakes to avoid the mine operation (–DL). Changes to surface water active zone due to the local landscape and mine dewatering activities (–DL). Potential effect on water quality from the release of treated effluent from the Project, including treated mine water (–IS). Reduction in terrestrial habitat caused by the mine footprint development, anticipated to be replaced by open pit lakes at closure (–DL).
Buildings and Storage	Reduction in localized air quality and increase in localized sound emissions during construction (–DS). Reduction in localized air quality due to the release of emissions from the process plant (–DS). Increase in localized sound emissions as a result of process plant and maintenance operations (–DS). Potential effect on water quality from the release of treated effluent from the Project, including treated process plant effluent and various wash water sources (–IL). Potential effect on the localized environment from accidents and malfunctions (–IS).
Waste Rock Storage Areas (WRSAs)	Reduction in localized air quality due to the release of particulate matter from stockpiling activities and heavy equipment emissions (–DS). Reduction in localized air quality due to dust release from the WRSAs (–DS). Increase in localized sound emissions as a result of heavy equipment operation, mineral waste deposition, and other noise producing equipment (–DS). Alteration to the local terrain through the formation of permanent WRSAs elevated above the existing landscape (–DL). Potential for loss of aquatic habitat by overprinting and/or rerouting of local creeks systems to accommodate stockpiling operations (–DL). Potential effect on water quality from the release of treated runoff and/or seepage from the WRSAs (–IL). Reduction in terrestrial habitat cause by the WRSA footprints (–DL).

Component	Potential Effect
TSF	Reduction in localized air quality due to the release of particulate matter from construction activities and heavy equipment operation (–DS). Increase in localized sound emissions as a result of heavy equipment operation during TSF dam construction (–DS). Alteration to the local terrain from the construction of a permanent facility raised above the surrounding landscape (–DL). Reduction in terrestrial habitat caused by the TSF footprint (–DL). Potential for loss of aquatic habitat by local creeks and wetlands (–DL). Potential alteration of groundwater infiltration rates (–DL). Potential effect on water quality from the release of effluent and seepage from the TSF (–DL).
On-site Infrastructure	Reduction in localized air quality and increase in localized sound emissions during construction (–DS). Reduction in localized air quality due to dust release from roads and vehicle emissions (–DS). Potential effect on the localized environment from accidents and malfunctions (–IS).

Source: Sabina Gold & Silver Corp. 2014

Sabina is confident that environmental issues associated with the development and operation of the Project could be effectively addressed and managed by:

- Applying sound engineering, environmental planning, and best management practices; and
- Complying with anticipated permits, licences, approvals, and existing federal and territorial environmental regulations and guidelines.

The DEIS concluded that all significant residual environmental impacts associated with the construction, operation and decommissioning of the Back River Project could be mitigated. The FEIS is scheduled to be submitted to the NIRB in Q4, 2015 with the same conclusion expected. Consequently, Sabina is confident that the Project can proceed, with the belief that all environmental and social issues can be addressed and will not materially impact Sabina's ability to extract the Mineral Resources or Mineral Reserves.

20.5.4 Environmental Management

An environmental management system (EMS) will be designed to ensure a consistent approach to responsible environmental management, and will be based on the following elements:

- Planning – define the scope of the EMS; establish an environmental policy for the Project; identify applicable legal and other (non-regulatory) requirements; set environmental performance objectives; develop Environmental Management Plans (EMPs).
- Implementation – resource allocation and the assignment of roles and responsibilities; environmental management training; internal and external communications; EMS documentation and records and document control; operating controls, including emergency response activities.
- Checking and corrective action – ongoing monitoring of environmental performance; inspection and evaluation of environmental management practices, including environmental compliance; EMS audits.
- Continual improvement – senior operational management review of the EMS; identification of improvements in environmental performance of the Project.

The EMS will emphasize key stakeholder engagement initiatives for environmental management, including educational and consultation programs with local communities and others.

20.5.5 Social and Cultural Considerations

The Project is located in the Kitikmeot Region of Nunavut with the closest settlements being the seasonal communities of Kingaok, located approximately 160 km to the north of Goose Site, and Omingmaktok, located approximately 250 km to the northeast. The communities of Kugluktuk (approximately 460 km northwest) and Cambridge Bay (approximately 400 km northeast) are the closest permanent regional settlements.

The communities of the eastern Kitikmeot region have been included as secondary study communities as they are within the boundaries of the KIA's administrative authority and are likely sources of workers and contractors. These communities include Gjoa Haven (575 km), Kugaaruk (795 km), and Taloyoak (705 km). Yellowknife, NT (525 km) is also an important community, as it will likely be a transport hub and a source for workers, goods, and services.

Sabina is committed to providing continuing employment and contracting opportunities for Nunavut residents. More specifically, Kitikmeot Inuit will be given hiring preference for Project-related jobs. Those communities located nearest to Sabina's operations (i.e., Kingaok, Omingmaktok, Cambridge Bay, and Kugluktuk) will be given particular preference for these opportunities, as a result of their traditional and contemporary ties to the Project area.

However, residents from other Kitikmeot communities (i.e., Gjoa Haven, Kugaaruk, and Taloyoak) will also be provided with preferential hiring opportunities wherever possible. Objectives for employment and contracting may be negotiated in consultation with the KIA in the IIBA.

Article 26 of the NLCA requires that all major projects negotiate an IIBA that would include, but not be limited to, agreements related to Inuit employment, preferred contracting, and local purchasing.

Sabina will undertake a socio-economic impact assessment that will be presented as a component of the overall FEIS being prepared for the Project. The details of this assessment will focus on matters pertaining to archaeology, employment, education and training, health and community well-being, economic development, business opportunities, subsistence economy and land use, non-traditional land and resource use, and country foods and human health. A Socio-Economic Monitoring Plan will also be prepared as a part of this process. Likewise, a Community Involvement Plan will be prepared that outlines Sabina's commitments to community engagement and consultation throughout the life of the Project.

20.5.6 Assessment Schedule

The overall regulatory process and permitting milestone schedule for the Project is presented in Table 20.4.

A key milestone in the schedule is the completion of the FEIS and its review by the NIRB. With an anticipated successful NIRB review, the EA decision is submitted to AANDC for approval and ultimate issuance of a Project Certificate by the NIRB. This process is anticipated to be completed in 2016.

Issuance of the Project Certificate is a key milestone in advancing the Project to a "go/no-go" decision. After this milestone, the permits and licences required for construction and operation can be submitted. Key permits and licences include a water licence for the mine and mill complex, and a fisheries authorization that includes approval of a Fisheries Offset Plan; these permits, licences, and authorizations are estimated to take approximately one to two years to complete after Project Certificate issuance. During this period, it is anticipated that any other permits and authorizations would also be obtained.

The implementation schedule for the regulatory phases and environmental studies are presented as consecutive tasks and actions; however, to facilitate these processes, regulatory bodies seek early engagement and discussions. Sabina continues to consult with the regulatory agencies as well as the Inuit communities.

20.5.7 Estimated Permitting Costs

The cumulative costs associated with the permitting process (including consulting fees) are estimated to be over $30 million for baseline studies, EIS preparation, and other permit applications for the development of the Project; the majority of these costs have already been incurred by the Company. These permitting costs will be incurred prior to a construction decision and therefore have been excluded from the initial capital cost estimate presented in this report.

Table 20.4: EA Timelines

	2013				2014				2015				2016				2017				2018				2019				2020			
	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Baseline Data Collection																																
Guidelines Issued by NIRB																																
Prefeasibility Study Issued																																
DEIS Issued																																
Technical Report and Feasibility Study Issued																																
FEIS Issued																																
Water Licence Engineering																																
Project Certificate Issued by NIRB																																
Construction																																
Type A Water Licence by NWB																																
First Gold Poured																																

Source: Sabina Gold & Silver Corp. 2015

20.6 Mine Closure and Reclamation Plan

20.6.1 Closure Objectives

The Property's mine closure and reclamation plan (MCRP) objectives are as follows:

- Design the mine for closure: identifying the processes that will act upon the mine components after mine closure so that they can be optimized into the mine design.
- Achieve physical stability: minimize risk to humans, wildlife, and the environment. Mine components that are to remain after mine closure will be constructed or modified at closure such that they are physically stable and do not erode, subside, or move from their final intended locations due to any forces.
- Achieve chemical stability: all mine components and wastes remaining after mine closure will be chemically stable. Chemical constituents released from the mine area will not endanger humans, wildlife, or the environment.
- Consider future use and aesthetics: compatibility with the surrounding lands after reclamation activities have been completed.

20.6.2 Closure Criteria

The MCRP for the Property is developed in accordance with current best practices which include relevant federal and territory closure guideline documents such as the Nunavut Mine Site Reclamation Policy for Nunavut (INAC, 2002) and the Mine Site Reclamation Guidelines for the Northwest Territories (INAC, 2007). With respect to the final water quality standards and site-specific thresholds, the MMER and the Canadian Council of Ministers for the Environment (CCME) water quality guidelines will be adopted and applied.

20.6.3 Closure Schedule

Progressive reclamation activities will begin as soon as mining at the Umwelt open pit has been completed in Year 2 (Phase 2, Stage 1) and will continue through the rest of the 12-year operating mine life. Active closure (Phase 3, Stage 1) will take approximately two years to complete and entails the bulk of the physical closure activities. Passive closure (Phase 3, Stage 2) is expected to last about four years and will consist primarily of water treatment followed by final decommissioning of the remaining elements of the Property. Finally, post-closure (Phase 4) will commence and last about five years during which confirmation monitoring occurs. This closure schedule is summarized in Table 20.5.

Table 20.5: Closure Schedule

Closure Stage	Project Year Start	Project Phase Start	Project Year End	Project Phase End
Progressive Reclamation	2	Phase 2 Stage 1	12	Phase 2 Stage 3
Active Closure	12	Phase 3 Stage 1	14	Phase 3 Stage 1
Passive Closure	14	Phase 3 Stage 2	18	Phase 3 Stage 2
Post-Closure	18	Phase 4	23	Phase 4

Source: SRK 2015

20.6.4 Reclamation Bond Requirements

Financial security is required under Type A Water Licences and is typically posted to AANDC for water-related closure costs and the land Owner (KIA) for land-based reclamation activities. The amount of security required will be agreed upon during the regulatory phase. The security will be deposited at agreed upon dates and milestones to ensure that the funds required for future reclamation will be available. Criteria will be established and will need to be met prior to release of any security held by the beneficiary. Funds may be released by the beneficiary back to Sabina, only if Sabina has satisfied its reclamation obligations. The Closure Plan incorporates provisions for progressive reclamation to take place, and this is captured in the financial model (section 23). To the extent funds are held in-trust to cover unforeseen future reclamation costs, or in the event that certain reclamation activities are not completed, the regulatory authorities will have the right to use the security funds to fulfill any necessary obligations.

20.6.5 Logistical Considerations

The remote setting of the Property presents unique challenges related to implementing a cost-effective closure plan. Specifically, these are the cost of shipping equipment, materials and supplies, as well as the construction and operation of winter roads between the MLA and Goose Site. The closure strategy, as far as practical, avoids backhauling and off-site shipping of any non-hazardous waste. Furthermore, materials and fuel needed for closure implementation will be brought to site by ship and winter roads during the final years of operations.

20.6.6 Temporary Closure Activities

Should mining cease temporarily before completion of the mine plan, a care and maintenance program will be necessary. Access to the mine area will be controlled and, as far as practical, be restricted to authorized personnel. All mine openings will be barricaded or guarded, and warning signs will be placed around all open pits and mine openings. Hazardous materials and explosives will be secured and stored safely. All machinery and mobile equipment will be locked out, and mobile equipment will be stored in safe locations.

All WRSAs and ore stockpiles will be maintained in a physically stable condition and annual geotechnical inspections will occur. Should temporary closure occur while the TSF is still in operation, dam safety inspections will continue to be carried out and pond levels will be monitored. All water management structures will be monitored and maintained accordingly. Surface water quality monitoring will continue at regular intervals. All infrastructure, including roads, airstrips, and camp areas will be maintained.

Temporary closure activities will continue until mining has resumed or until the decision is made to permanently close the mine. Should the mine close permanently, a final MCRP will be filed with the Nunavut Water Board (NWB) and final closure activities will begin.

20.6.7 Closure Activities

20.6.7.1 Open Pits (excluding Llama Tailings Facility (TF))

Boulder fences will be placed around each open pit as it is mined out. Boulders will be minimum 1 m in diameter, placed 3 m from the final pit crest, and will be spaced no more than 3 m apart. The intention is not to prevent access but to be a significant visual aid suggesting a change in landscape that will act as a warning sign to both humans and large mammals.

Pit sumps and associated pumps and pipelines are to be removed as each open pit is mined out. If the pumps are not to be re-purposed, hazardous material will be removed from the pumps and disposed of at an off-site licenced facility. The pumps will be landfilled on site. Associated pipelines, if not being reused, will be cleaned if necessary and landfilled.

Any mobile equipment used in the open pits that is past its service life will have all hazardous materials removed and disposed of at a licenced facility and the equipment will be landfilled on site.

The Umwelt, Llama, and Goose Main open pits will be flooded during the closure phase and will reach an overflow point in Year 21, Year 22, and Year 19, respectively. Once the outflow water quality requirements have been met, the pits will be allowed to overflow and discharge to the environment. Appropriate erosion protection measures will be constructed at the overflow locations to ensure management of suspended sediments. The Umwelt open pit will also contain about 1.3 Mm^3 of connate water at its base, making it a meromictic lake. The fresh water cover over the saline water will be about 58 m which is sufficient to prevent any turnover.

Table 20.6 summarizes the fill rate of all of the open pits under difference hydrological conditions.

Table 20.6: Estimated Project Open Pit Flood Rates under Different Hydrological Conditions

Open Pit/TF	Result	Fill Rate for Different Hydrological Conditions		
		5^{th} Percentile Conditions	50^{th} Percentile Conditions	95^{th} Percentile Conditions
Umwelt Reservoir	Fill Date	Year 19, Q2	Year 21, Q2	Year 27, Q4
	No. Days	2,446	3,900	5,549
Llama TF	Fill Date	Year 18, Q4	Year 22, Q1	Year 26, Q2
	No. Days	2,196	3,422	5,006
Goose Main Reservoir	Fill Date	Year 18, Q3	Year 19, Q3	Year 20, Q3
	No. Days	-118	244	497

Source: SRK 2015

20.6.7.2 Llama TF

The Llama open pit will be used as a TF. Water in Llama TF will be treated as soon as tailings deposition begins and treated water will be routed back to the TF to improve water quality within the facility. Water treatment in the facility will continue through the active and passive closure periods until water quality has met closure criteria. At this point a permanent water cover of 5 m will be maintained upon closure of the facility. Based on precedent, it is assumed the 5 m water cover is sufficient to prevent re-suspension of tailings solids due to wave action, surge, and ice scour.

Similar to the Umwelt and Goose Main open pits, Llama TF will be closed with permanent outflow measures once appropriate water quality criteria to the environment have been met. A boulder fence will also be placed around the pit as it is mined out.

20.6.7.3 Umwelt Underground

Pre-development for underground mining at Umwelt will being in Year 2 and mining will begin in Year 3. Mobile equipment used specifically in underground operations will have the hazardous material removed and disposed of at a licenced facility, and the equipment will be landfilled.

Mine dewatering pipelines, electrical transmission wires, substations, and pumping stations not suitable for reuse are to be cleaned, disposing of any hazardous waste off-site, and the remaining equipment either dismantled and landfilled, or left in place upon closure.

Underground void space will be backfilled with waste rock for structural stability. The mine portal will subsequently be sealed off with NPAG waste rock. The portal opening will be flush with the surrounding topography or, if required, at a slope angle of 3H:1V. All underground ventilation raises will be closed using engineered concrete caps, or alternately will be filled with NPAG waste rock flush with the surrounding ground surface.

20.6.7.4 Tailings Storage Facility

For approximately seven years following completion of tailings deposition in the TSF, the facility will continue to be used for seasonal contact water storage. During this period, Goose Main open pit will be developed and the tailings surface will be covered with waste rock from this pit. The entire tailings surface will be covered, including a zone 25 m downstream of the TSF containment dam. The entire covered surface, whether waste rock or tailings, will have at least 5 m of NPAG waste rock cover. This cover will ensure that the tailings surface will freeze.

Once covered, only a portion of the west limb of the dam will be visible. That portion of the TSF will continue to allow storage of seasonal contact water and will be known as the TSF WRSA Pond. It will be dewatered to Llama TF until mining at Goose Main Pit is complete in Year 8, at which point it will be dewatered to Goose Main Pit. At closure in Year 12, the TSF WRSA Pond will be fully dewatered and the west limb of the TSF containment dam will be breached to allow for any surface runoff to flow unimpeded towards Goose Lake within the pre-mining stream course.

Sediments remaining in the TSF WRSA Pond area will be tested and if not within industrial soil limit criteria, will be excavated and disposed of in the Goose Main open pit.

20.6.7.5 Waste Rock Storage Areas (WRSA)

WRSAs will be progressively capped with 5 m of NPAG rock and re-sloped such that the overall slopes are 3H:1V and no benches or terraces remain. All WRSAs are designed to freeze back within a period of 8 to 10 years. The active layer is expected to remain within the outer 5 m cap of NPAG, thus any risk of ARD will be mitigated.

20.6.7.6 Non-Hazardous Landfills

Non-hazardous landfills will be constructed within the confines of the WRSAs. All landfill areas will be capped with a minimum of 5 m of NPAG waste rock. Non-hazardous waste will also be disposed of within any of the open pits and will be covered with at least 5 m of water. Any available underground void space can also be used for disposal of non-hazardous waste.

20.6.7.7 Water Management Structures

All contact and non-contact water storage ponds will be pumped out and the containment berms breached. The Primary and Umwelt WRSA ponds will be breached to Umwelt open pit, and the Llama WRSA Pond will be breached to Llama TF. The Ore Stockpile Pond will be dewatered to Llama TF and the TSF WRSA Pond will be dewatered to Goose Open Pit.

Diversion structures will be breached and pre-mining flow channels will be re-established. Soil sampling will be carried out in all water storage facilities to determine whether the exposed sediments meet industrial standards. If not, the contaminated sediments will be excavated and disposed of in Llama TF. Any liners, both in containment structures and diversions, will be completely removed. The liners will be disposed of as non-hazardous waste. Pumps and pipelines will be removed, stripped of hazardous waste and landfilled if not being reused. Hazardous materials will be disposed of at a licenced facility.

20.6.7.8 Water Treatment Facilities

Once water treatment is complete at Llama TF in Year 18 and runoff quality at designated control points has met closure criteria, the treatment plant and all associated pipelines will be dismantled, cleaned, and disposed of in the landfill if not being reused.

20.6.7.9 Buildings and Equipment

At the end of mining, during the active closure period, the Goose Site's processing facilities, crusher, power plant, fuel storage facilities, shops and warehouses, and supplementary ancillary facilities will be dismantled. Hazardous materials will be removed, all reservoirs will be flushed out, and the remaining materials that are not deemed to have salvage value will be landfilled. Concrete foundations for any structures will be demolished and disposed of in the landfill.

At the MLA, surface infrastructure will be similarly handled. However, all elements designated for landfilling will be shipped off-site for disposal at a designated landfill near the port of destination or backhauled to Goose Site for landfilling. At both sites, once all buildings and equipment have been removed, the footprints (whether bedrock or thermal pads) will be re-contoured to allow for sheet flow drainage to the receiving environment.

A fully functional, modular 20 person camp (complete with associated support facilities) will be constructed at the Goose Site during this period to accommodate the closure and post-closure phases. Once this period, the camp will be dismantled and disposed of similar to the process followed during active closure.

20.6.7.10 Roads and Airstrip

Secondary access roads, mining haul roads, and service roads no longer required once operations are complete will be decommissioned. This will entail removing any culverts to maintain pre-construction surface drainage, and general grading of the road surface to promote runoff shedding. Primary access roads, the associated water management structures such as culverts, and the airstrip will be maintained throughout the active and passive closure phases. Culverts will be maintained and surfaces will be graded to minimize erosion and wear on equipment. Once passive closure is complete, the primary roads and airstrip will be reclaimed in a similar fashion to the secondary access roads.

WIRs are not expected to require any reclamation but the route will be inspected prior to completion of closure to identify any areas of potential physical instability (e.g., erosion). These areas will be remediated as required.

20.6.7.11 Contaminated Soils

A site investigation will be carried out to determine the volume of contaminated soil from hydrocarbon spills over the life of the mine. The investigation will be conducted using a direct-push drill rig and drilling will occur down to a base of the active layer (5 m). Drilling will be focused on all parking bays, fuel storage areas, wash bays, truck shops, maintenance areas, and generator areas, as well as along roads and in areas where spills have been known to occur.

If the volumes of contaminated soils are significant, the materials will be remediated with on-site land farms constructed specifically for this purpose. If the volumes are small, the material will be shipped off-site for disposal at a licenced facility.

As required, water contaminated with hydrocarbons will be treated using portable oil-separator units.

20.6.7.12 Hazardous Materials

All hazardous waste will be properly packaged and shipped off-site to a licenced facility for disposal. This will either be via return sealift or by backhaul flights once the MLA has been decommissioned.

20.6.7.13 Monitoring

Monitoring will be carried out during the closure and post-closure phases to ensure that closure activities are being undertaken and objectives are being met. The monitoring programs are outlined below.

20.6.7.13.1 Geotechnical Monitoring

Ground temperature cables (GTCs) will be installed within the WRSAs and the TSF during construction activities. The GTCs will be continuously monitored to confirm that freeze back is being achieved, as per design. As far as practical, the GTCs will be equipped with data loggers to allow for continuous data acquisition at a frequency determined by the Engineer of Record. Monitoring of the GTCs can cease once the data shows that freeze back has been achieved for a period of at least five years.

During operations and until complete decommissioning, an annual geotechnical inspection will be carried out by a qualified geotechnical engineer licenced to practice in Nunavut. These inspections should be carried out during the summer months when there is no snow cover. Areas to be included in the inspection are the TSF, TF, all WRSAs, open pit high walls, all contact and non-contact water storage ponds, diversion structures, and any other surface infrastructure elements possibly affecting permafrost.

20.6.7.13.2 Water Quality Monitoring

Water quality monitoring will start as soon as contact or process water is present at the start of construction. Monitoring at designated control points will be checked in accordance with the licence criteria, both with respect to frequency and the necessary testing parameters. Following the active and passive closure phases, water quality monitoring will systematically be scaled back, with ultimate cessation once there is at least five years of water quality monitoring that confirms that the final closure objectives have been met.

20.6.7.13.3 Terrestrial and Aquatic Effects Monitoring

Terrestrial and aquatic effects monitoring will be carried out in accordance with licence criteria. Cessation of this monitoring will be when a five year period has elapsed showing the system has achieved the stated closure objectives.

21 Capital Cost Estimate

21.1 Capital Costs

21.1.1 Introduction and Summary

Preparation of the capital cost estimate (CAPEX) is based on the JDS philosophy that emphasizes accuracy over contingency, and uses defined and proven Project execution strategies. The estimates were developed using first principles, applying directly-related Project experience, and the use of general industry factors. Almost all of the estimates used in this Project were obtained from engineers, estimators, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in this study.

The following cost estimates are described in this section:

- Initial Capital Cost – includes all costs incurred to develop the Property to a state of nameplate production (3,000 t/d);
- Sustaining Capital Cost – includes all costs incurred during production for initial and ongoing underground installations and development, LOM equipment acquisitions and replacements, and annual construction of the WIR; and
- Sunk costs and Owner's reserve are not considered in this section.

All cost estimates are based on the following key parameters:

- Owner-performed pre-production mining; and
- The specific scope and execution plans described in this study. Deviations from these plans will affect the capital costs.

Table 21.1 summarizes the capital cost estimate by area and activity. Table 21.2 shows the capital cost distribution.

A Work Breakdown Structure (WBS) was established for the initial capital cost estimate. Costs have been classified into the various WBS areas to ensure that the entire Project scope has been captured.

Table 21.1: Summary of Capital Costs by Category

Capital Cost	Initial $M	Sustaining $M	LOM $M
Mining	45.9	112.5	158.3
On-Site Development	15.3	1.3	16.6
Ore Crushing & Handling	15.6	0	15.6
Process Plant	55.5	0	55.5
On-Site Infrastructure (Goose)	68.1	14.9	83.0
Off-Site Infrastructure	25.0	39.6	64.7
MLA	26.3	2.0	28.3
Tailings	6.2	1.8	7.9
Indirects	65.5	0	65.5
EPCM	29.7	0	29.7
Owner's Costs	24.6	0	24.6
Reclamation	0	63.8	63.8
Subtotal	**377.7**	**235.8**	**613.5**
Contingency	37.2	13.2	50.5
Total Capital Costs	**414.9**	**249.1**	**664.0**

Source: JDS 2015

Table 21.2: Summary of Capital Cost Distribution

Capital Cost	Initial Capital Distribution (%)	Sustaining Capital Distribution (%)
Mining	11.1	45.1
On-Site Development	3.7	0.5
Ore Crushing & Handling	3.8	0
Process Plant	13.4	0
On-Site Infrastructure (Goose)	16.4	6.0
Off-Site Infrastructure	6.0	15.9
MLA	6.3	0.8
Tailings	1.5	0.7
Indirects	15.8	0
EPCM	7.2	0
Owner's Costs	5.9	0
Reclamation	0	25.6
Subtotal	**91.0**	**94.7**
Contingency	9.0	5.3
Total Initial Capital Cost Distribution	**100.0**	**100.0**

Source: JDS 2015

The accuracy of the capital cost estimate is in the range of +/-15%, which represents a JDS Feasibility Study Budget / AACE Class 2 Estimate.

The Project contingency was built up using factors applied to labour, equipment, materials and vendor packages for each capital cost category. The contingency factor is, however, only applied to the capital estimate as a rolled-up value and not to the individual components of the estimate.

This estimate was prepared with a base date of Q3-2015 and does not include any escalation beyond this date. The quotations used for this study were obtained between Q2-2015 and Q3-2015 and are valid for a period of 90 calendar days.

The capital cost estimate uses Canadian dollars as the base currency. When required, quotations received from vendors were converted to Canadian dollars using a currency exchange rate of US$:C$ 0.80. Duties and taxes are not included in the capital estimate.

21.1.2 Responsibility Matrix

This capital cost estimate was developed by a team of engineers, procurement specialists and cost estimators. JDS is responsible for the development and assembly of the overall capital cost estimate with input from companies shown in Table 21.3.

Table 21.3: Capital Cost Estimation Responsibility Matrix

Description	Responsibility	Scope
Open Pit Mining	JDS & Knight Piésold	• OP Mine Development & Production • OP Mining Equipment • OP Mine Services & Equipment
Underground Mining	JDS Hatch	• UG Mine Development & Production • UG Mining Equipment • UG Mine Services & Equipment • Electrical power feed from Goose power plant to Goose underground portals and portal emergency generator sets
On-Site Development	JDS Hatch	• Bulk Earthworks • Site Drainage • Water Management (Pipelines) • Airstrip • Roads • Water Management (Water Pumping System)
Ore Crushing & Handling	JDS & Canenco Hatch	• Buildings • Mechanical Equipment • Piping • Electrical Bulks • Instrumentation Equipment & Bulks • Detailed Civil Works • Concrete • Internal Steel • Electrical Supply/Distribution (MCC's & Switchgear) • Fine Ore Storage (reclaim tunnel, steel piles, stockpile cover, electrical, and instrumentation)
Process Plant	JDS & Canenco Hatch	• Buildings • Mechanical Equipment • Mechanical Platework (tanks, bins, chutes excluded from vendor packages) • Piping • Electrical Bulks • Instrumentation Equipment & Bulks • Detailed Civil Works • Concrete • Internal Steel • Electrical Supply/Distribution (MCC's & Switchgear) • Process control system (PCS), CCTV, Control Room
On-Site Infrastructure	JDS Sabina/Toric and Simplex Grinnel	• Detailed Civil Works • Buildings & Camp • Mechanical Equipment • Fuel Tanks • Piping • Electrical Supply (Power Plant) • Mobile Equipment • IT and Communications • Fire Protection and Security
On-Site Infrastructure	Hatch	• Concrete • Steel • Mechanical Platework (minor tanks, bins, chutes excluded from vendor packages) • Electrical Supply/Distribution (MCC's & Swtichgear) • E-Houses and Plant Lighting • Incinerator • First Aid/EMT • Fuel pumping and distribution
Off-Site Infrastructure	JDS	• Winter Roads
MLA Infrastructure	JDS Hatch Sabina/Toric and Simplex Grinnel	• Bulk Earthworks • Camp • Mechanical Equipment • Piping • Mobile Equipment • Concrete • Mechanical Equipment (fuel loading/unloading, dispensing module) • Mechanical Platework (tanks, bins, chutes excluded from vendor packages) • Pipeline (fuel) • Electrical • IT and Communications • Fire Protection and Security
Tailings	JDS SRK	• Mechanical Equipment • Piping (tailings and reclaim water) • Bulk Earthworks
Indirects	JDS	• Camp & Catering • Field Indirects • Freight • Vendor Reps • Spares • First Fills
EPCM	JDS	• Mining • Process Plant & Infrastructure • Tailings and Geotechnical
Owner's Costs	JDS	• G&A (labour, offices, freight, misc. items)
Contingency	JDS	• Mining, Process, Infrastructure, Tailings, Indirects, EPCM, and Owner's costs (labour, materials, equipment, sub-contract)

Source: JDS 2015

21.1.3 Mining

21.1.3.1 Open Pit Mining

The capital cost estimate for open pit mining operation is based on achieving the scheduled plant processing throughput rates as well as comparing the capital costs to similarly sized open pit gold operations. The open pit mining activities for the Back River Project open pits were assumed to be undertaken by an Owner-operated fleet with an estimated maximum capacity of 40,000 tpd total material.

The primary equipment types and costs required to achieve the target processing rate is estimated at $41.1 million (LOM) and is summarized in Table 21.3 (pre-production primary equipment capital is estimated at $31.0 million). The capital expenditures on equipment are in line with the timing of the costs outlined in Table 21.4. Budgetary quotes from regional equipment suppliers, mining cost service information, and factors based on experience were taken into consideration. Estimated delivery costs to the staging port, whether Vancouver, BC or Bécancour, QC, are included. Delivery costs from the port to Goose Site via the MLA are not included; these costs are included in the WIR haulage and freight costs (WBS 7230 and WBS 9700, respectively). No equipment was considered as leased.

Table 21.4: Open Pit Equipment Capital Cost Summary

Type	Unit	No. Required	LOM	Y-1	Y1	Y2	Y3	Y4	Y5	Y6-10
MD 6240 Drill (152–270 mm)	$M	2	2.1	2.1	-	-	-	-	-	-
MD 5125 Drill (89–152 mm)	$M	1	0.9	0.9	-	-	-	-	-	-
Cat 6015 Shovel (7 m^3)	$M	2	4.2	4.2	-	-	-	-	-	-
Cat 390 Excavators (4 m^3)	$M	2	2.0	1.0	-	1.0	-	-	-	-
Cat 988 Wheel Loader (7 m^3)	$M	1	1.8	1.8	-	-	-	-	-	-
Cat 775G Truck (64 t)	$M	16	20.3	13.6	1.4		2.7	0.9	1.8	-
Cat D8 Track Dozers	$M	5	4.4	2.1	1.0	-	-	-	-	1.3
Cat 824 Wheel Dozer (4.2 m blade)	$M	2	2.2	2.2	-	-	-	-	-	-
Cat 14M Grader	$M	3	3.1	3.1	-	-	-	-	-	-
Total Primary Equipment			**41.1**	**31.0**	**2.4**	**1.0**	**2.7**	**0.9**	**1.8**	**1.3**

Source: JDS 2015

Pre-stripping requirements were estimated using the pre-production total mined tonnage of 7.3 Mt and was considered part of the overall open pit initial capital cost. Using the estimated average mining cost for these periods, a pre-production capital cost of $19.4M is allocated to this pre-production period for open pit mining.

21.1.3.1.1 Equipment Replacement Criteria

Equipment suppliers provided estimates for equipment replacement life, and, where information was lacking, industry standards and related project experience were used. The estimated operating life for each piece of equipment is shown in Table 21.5.

Given the estimated life of the Project open pits (~ 8 years), no major equipment replacements are expected given the estimated equipment life cycles.

Table 21.5: Equipment Life Cycle

Equipment Unit	Operating Life (h)
Major Equipment (gross operating hours)	
Blasthole drill (250 mm)	50,000
Blasthole drill (115 mm)	45,000
Diesel hydraulic front shovel	70,000
Wheel loader	49,000
Haul truck	70,000
Support Equipment (gross operating hours)	
Track dozer	40,000
Rubber-tired dozer	56,000
Motor grader	56,000

Source: JDS 2015

Schedule of Initial Purchases

The schedule for initial equipment purchases is shown in Table 21.6; the "Year" indicates the first year of use for each unit.

Table 21.6: Planned Equipment Purchase Schedule

Type	Year											
	No. Required	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
MD 6240 Drill (152 – 270 mm)	2	2	-	-	-	-	-	-	-	-	-	-
MD 5125 Drill (89 – 152 mm)	1	1	-	-	-	-	-	-	-	-	-	-
Cat 775G Truck (64 t)	16	10	1	-	2	1	2	-	-	-	-	-
Cat 6015 Shovel (7 m^3)	2	2	-	-	-	-	-	-	-	-	-	-
Cat 390 Excavators(4 m^3)	2	1	-	1	-	-	-	-	-	-	-	-
Cat 988 Wheel Loader(7 m^3)	1	1	-	-	-	-	-	-	-	-	-	-
Cat D8 Track Dozers	5	2	1	-	-	-	-	2	-	-	-	-
Cat 824 Wheel Dozer (4.2 m blade)	2	2	-	-	-	-	-	-	-	-	-	-
Cat 14M Grader	3	3	-	-	-	-	-	-	-	-	-	-

Source: JDS 2015

21.1.3.2 Underground Mining

Underground mining capital costs for mobile and stationary equipment are based on the following:

- Budgetary quotes from equipment manufacturers for major mobile and stationary equipment;
- Budgetary quote from a contractor for raisebore development;
- In-house reference database; and
- Project construction timeline and mining schedule.

Underground mining capital costs comprise site development, mobile and stationary equipment, contractor raiseboring and Owner-operated mine development. Table 21.7 shows the total estimated annual capital spending at the Umwelt underground operation.

Table 21.7: Underground Mining Capital Costs (All Costs in M$)

Umwelt Underground	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
Site Development	**0.6**		0.6							
Mobile Equipment	**32.6**	8.8	9.5	1.6	0.2	4.1	4.6	3.3	0.5	
Stationary Equipment	**13.5**	10.1	2.2	1.1	0.1					
Development	**48.4**		8.0	18.1	6.2	2.8	4.1	3.6	2.6	3.0
Contractor Raiseboring	**6.7**		3.1	3.6						
Umwelt Total Mine Capital Cost	**101.9**	**18.9**	**23.4**	**24.3**	**6.6**	**6.9**	**8.7**	**7.0**	**3.0**	**3.0**

Source: JDS 2015

21.1.3.2.1 Site Development Costs

Site development costs are incurred in Year 2 when the pre-production development commences. Earthworks of $430,000 for construction of a small laydown area for underground equipment and consumables, and a roadway linking the laydown and portal area to the main haul road. Excavation costs of $170,000 for the Umwelt portal boxcut.

21.1.3.2.2 Mobile Equipment Capital Costs

No underground equipment purchases are planned in Year -2 and Year -1. It was assumed that the Owner would purchase and maintain a complete underground mining fleet for the Umwelt underground operation during Year 1 and Year 2. Because the mine life of the Umwelt underground mine is seven years, some equipment rebuilds and replacements are allowed for. The equipment will be rebuilt at 50% of its operating life at a cost of 60% of the initial purchase price. See table 21.8 for the underground mobile equipment fleet.

Table 21.8: Underground Mobile Equipment Fleet by Year

Underground Mobile Equipment	Max	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
Haulage Truck (30t)	**6**		1	4	6	5	5	5	5	4
LHD (10t)	**2**		1	2	2	2	2	2	2	2
LHD (6.7t)	**2**			1	1	2	2	2	2	1
Jumbo	**2**		1	2	2	2	2	2	2	2
Diamond Drill	**1**			1						
Production Drill	**1**			1						
Rockbolter	**4**		1	3	4	4	4	4	4	3
Shotcreting Machine	**1**		1	1	1	1	1	1	1	1
ANFO Loader	**2**		1	2	2	2	2	2	2	2
Boom Truck	**1**		1	1	1	1	1	1	1	1
Mechanics Truck	**1**		1	1	1	1	1	1	1	1
Fuel-Lube Truck	**1**		1	1	1	1	1	1	1	1
Supervisor Vehicle	**1**		1	1	1	1	1	1	1	1
Electrician Vehicle	**1**		1	1	1	1	1	1	1	1
Scissor Truck	**1**		1	1	1	1	1	1	1	1
Forklift/Telehandler	**1**		1	1	1	1	1	1	1	1
Utility Vehicle / Nipper	**1**		1	1	1	1	1	1	1	1
Portable Welder	**1**		1	1	1	1	1	1	1	1
Personnel Carrier	**1**		1	1	1	1	1	1	1	1
Grader	**1**		1	1	1	1	1	1	1	1
Front-End Loader	**1**		1	1	1	1	1	1	1	1

Source: JDS 2015

21.1.3.2.3 Stationary Equipment Capital Costs

No purchase of underground stationary equipment is planned for years -1 and -2. For the major pieces of stationary equipment the logistics time from point of purchase to required installation was taken into account by advancing 80% of the quoted price 12 months, with the remaining 20% of the quote being considered as the installation cost. See Table 21.9.

Table 21.9: Underground Mine Stationary Equipment CAPEX

Equipment	LOM Quantity	Unit Cost (K$)	Total LOM Cost (K$)
Elec. Surface Distr.	1	1,324	1,324
Emergency GenSet)	1	446	446
Elec. Underground Subs	3	256	768
Primary Ventilation	1	1,621	1,621
Primary Heating	1	4,895	4,895
Aux. Fans - 149 kW	3	62	186
Aux. Fans - 75 kW	6	22	132
Aux. Fans - 56 kW	1	21	21
Ventilation Bulkheads	19	10	190
Portable Refuge Stations	6	82	492
Mine Rescue Equipment	1	95	95
Leaky Feeder	1	150	150
Cap Lamps/Charger	100	0	30
Mine Engineering Equipment	1	225	225
Fuel Station (SF)	1	78	78
Cap Mag (UG)	1	20	20
ANFO Mag (UG)	1	25	25
Emergency Muster Station/Lunch Room	1	180	180
Mine Air Compressor - 85 kW (portable)	2	38	75
Level Sumps - 43 kW Submersible Pumps (UG)	16	38	608
Booster Pump - 56 kW (SF)	1	25	25
Brine System (short-term - decline dev.)	1	70	70
Mine Service Water System	1	73	73
Jackleg Drill	4	5	20
First Fills (Spares)	1	1,244	1,244
Escape-way Ladders (metres)	570	0.85	485
Total Stationary Equipment			**13,483**

Source: JDS 2015

21.1.3.2.4 Development Capital Costs

Underground development is scheduled to start in Year 2, with production start-up in Year 3. While the mine is in production, on-level waste development was considered an operating cost while the decline, vertical development, ventilation drifts and underground infrastructure were classified as sustaining capital costs.

It was assumed that all the lateral development and drop raises would be done by the Owner, and raisebored ventilation raises would be done by a contractor. Budgetary quotes for raiseboring were obtained from a contractor. Estimated Owner costs on a per metre of development basis, including direct labour, were applied for all lateral and drop raise development. Table 21.10 shows the capital development schedule by end type.

Table 21.10: Underground Capital Development by Year

Umwelt Underground	Unit	Total	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
Decline with Remucks	m	4,932	1,925	2,656	231	20	20	40	0	40
Level Access & Attack Ramps	m	6,689	86	896	1,050	843	1,165	1,060	804	784
Vent & Waste Pass Access	m	2,118	240	1,349	467	0	0	0	0	62
Other Infrastructure	m	717	112	288	89	36	60	50	42	40
Drop Raises - Vent	m	748	0	520	200	0	0	0	0	28
Raise Bore - Vent	m	737	354	383	0	0	0	0	0	0
Total CAPEX Development	**m**	**15,941**	**2,717**	**6,092**	**2,037**	**899**	**1,245**	**1,150**	**846**	**954**

Source: JDS 2015

21.1.4 Basis of Cost Estimate for the Ore Handling, Process Plant, Infrastructure and Tailings

The basis of cost estimate describes the methods, organization, assumptions and exclusions used to develop the Capital Cost Estimate the Project. The cost estimate includes the following elements:

Quantity Development

- CAPEX was developed largely from engineering quantities obtained from material takeoffs. In-house benchmarks were used where the engineering information were not sufficiently developed to prepare accurate quantities.

Direct Field Labour

- Direct field labour is the skilled and unskilled labour generally supplied by the contractor to install the permanent equipment and bulk materials at the Project site. Direct field installation man-hours were developed using estimated unit man-hours for each commodity multiplied by the quantity. Adjustments to standard man-hours were made to each commodity using a productivity factor (PF) to reflect the specific conditions at the Project site. These conditions include climate, physical extent of the site, working schedule, industrial environment, labour availability, etc.

Labour Rate

- A set of 'All-In Labour Rates' was developed for each commodity, each based on a specific crew mix and proposed work cycle, and applied against direct field man-hours to generate direct field labour costs. Rates are based on an agreement between Ledcor Industrial Alberta and CLAC Local 63 valid to July 2012, and have been adjusted for inflation at 2% annually to reflect 2015 costs. The rates have been 'built-up' to include all wages, benefits, government assessments, incentive pay, overtime costs, contractor indirects and contractor profit.

Productivity Factors for Labour

- A productivity factor has been applied to the standard base hours where the basis for the estimated work hours differs from the actual work environment. Factors for each commodity have been applied to reflect arctic work environments. The factors apply to productive labour only and do not affect non-productive labour (travel to the workface, tool box meetings, safety inductions etc.).Trades have been classified as either 'outdoor' or 'indoor'. Earthworks, concrete and architectural (buildings) are considered 'outdoor' while mechanical, piping, structural steel, electrical and instrumentation are considered 'indoor'.

Equipment Costs

- Estimates for major electrical and mechanical equipment are based on budget quotations. Major equipment is loosely defined as equipment costing greater than a million dollars and a delivery time greater than 10 months. For minor equipment, prices were obtained from budget quotations or from similar recent equipment quotes. Miscellaneous and or undefined equipment has been factored based on historical data where time and cost efficiencies can be achieved without significant impact on the estimate accuracy.

Bulk Material Costs

- Bulk material costs have been calculated as either part of the built-up rates applied to engineering MTOs or factored costs or allowances. Built-up unit rates are based on Project specific supply costs. Waste factors applied to bulk materials are shown in Table 21.11.

Table 21.11: **Waste Factors**

Commodity	Waste Factor
Civil & Earthworks	5%
Concrete	2%
Steel	1%
Piping	4%
Electrical Bulks	15%
Instrumentation Bulks	10%

Source: JDS 2015

Bulk material costs that were incorporated into the estimate include the following components:

- Site development and bulk earthworks;
- Concrete;
- Steel work;
- Mechanical bulks;
- Architectural;
- Piping; and
- Electrical and instrumentation bulks.

Facilities

- The costs developed for facilities are a combination of unit rates, allowances and budget quotes. These costs were assessed based on specifications and requirements outlined by engineering. The methodologies for costing of the major facilities are set out in Table 21.12.

Table 21.12: Facility Cost Basis

Facility	Cost Basis
Utilidors	Utilidors are not required on site with the exception of the camp due to the close proximity of the buildings.
Operation Camp	Budget quotes have been obtained based on the estimated camp sizes.
Ancillary Buildings	Costs have been carried from the 6,000tpd FS estimate or facilities have been sized based on design requirements and costs estimated based on similar installations.
Power Plant	Budget quotes have been obtained based on the estimated design electrical load.
Incinerators	Costs have been carried from the 6,000tpd FS estimate.
Truck Shop & Wash Bay	Building sizes have been determined by project requirements. The 'Pre-Engineered' truck shop has been quoted and the internal finishing costs have been included based on unit costs from JDS internal data.
Fresh, Fire, Process and Potable Water	Major holding tanks and pipelines have been quantified by engineering and priced based on project commodity costs. The potable water treatment plant has been based on budget quotes carried from the 6,000tpd FS estimate.
Sewage Treatment	A budget quote has been obtained for the Sewage Treatment plant as being part of the camp.

Source: JDS 2015

21.1.5 On-Site Development

The on-site development is described in section 18 of the report and contains detailed descriptions of the site earthworks, drainage, airstrip and miscellaneous infrastructure required for the Goose Site. A summary of the estimated costs for on-site development is shown in Table 21.13.

Table 21.13: On-Site Development Cost Estimate (WBS 2000)

Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
On-Site Development – Goose (WBS 2000)			
Bulk Earthworks	6.1	0	6.1
Site Drainage	5.4	1.3	6.6
Airstrip	1.3	0	1.3
Infrastructure	2.6	0	2.6
Total	**15.3**	**1.3**	**16.6**

Source: JDS 2015

21.1.6 Ore Crushing and Handling and Process Plant

The ore crushing and handling facilities and Goose process plant are described in section 17 of the report. Section 17 also contains images from the 3D model that are used to derive quantities. A summary of the estimated costs for ore handling and process plant are shown in Table 21.14.

Table 21.14: Ore Crushing & Handling and Process Plant Cost Estimate (WBS 3000 & 4000)

Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
Ore Crushing & Handling – Goose	**15.6**	**0**	**15.6**
Primary Crushing Building	1.1	0	1.1
Crushing & Screening	9.9	0	9.9
Fine Ore Storage & Reclaim	4.5	0	4.5
Process Plant – Goose	**55.5**	**0**	**55.5**
Process Plant Building	10.3	0	10.3
Grinding	10.8	0	10.8
Gravity Concentrating and Intensive Leaching	1.4	0	1.4
Cyanide Leaching and Carbon Adsorption	16.8	0	16.8
Acid Wash, Stripping, and Regeneration	0.5	0	0.5
Gold Recovery	4.2	0	4.2
Cyanide Destruction and Tailings	3.9	0	3.9
Reagents	3.1	0	3.1
Process Utilities	4.4	0	4.4
Total	**71.1**	**0**	**71.1**

Source: JDS 2015

The ore handling and process plant sections of the estimate include the following scope:

- Detailed earthworks;
- Concrete;
- Internal steel (equipment supports and access platforms);
- Mechanical equipment;
- Platework;
- Piping;
- Electrical;
- Instrumentation and process control; and
- Buildings (as a separate WBS section, including process plant piping).

No changes or additions to the equipment or structures of the crushing circuit and process plant are envisaged over the Life of Mine in this Initial Project Feasibility Study.

21.1.7 Infrastructure

21.1.7.1 On-Site Infrastructure (Goose)

The on-site infrastructure is described in section 18 of the report. A summary of the on-site infrastructure costs are shown in Table 21.15.

Table 21.15: On-Site Infrastructure Capital Cost Estimate (WBS 5000)

Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
On-Site Infrastructure - Goose	**59.5**	**13.0**	**72.5**
Electrical Supply & Distribution	19.5	7.6	27.1
Water Supply & Distribution	1.0	0	1.0
Assay Laboratory	0.4	0	0.4
Construction Camp / Permanent Camp	19.4	4.8	24.2
Waste Management & Removal	0.8	0	0.8
Bulk Fuel Storage & Distribution	7.7	0	7.7
IT & Communications	5.8	0.5	6.3
Plant Mobile Fleet	4.9	0	4.9
Ancillary Facilities	**8.6**	**1.9**	**10.5**
Administration Offices & Mine Dry (part of Truck Shop)	0	0	0
Truck Shop & Truck Wash	4.2	0	4.2
On-Site Services & Utilities & Infrastructure Pre-Production OPEX	2.5	0	2.5
First Aid & Mine Rescue	0.2	0	0.2
Water Treatment Plant	0	1.9	1.9
Sewage Treatment Plant / Effluent Treatment Plant (part of Camp)	0	0	0
ANFO & Blasting Facilities	1.7	0	1.7
Total	**68.1**	**14.9**	**83.0**

Source: JDS 2015

21.1.7.2 Off-Site Infrastructure

Off-site infrastructure comprises the WIR including associated mobile equipment and labour requirements. The off-site infrastructure is described in section 18 of the report. A summary of the WIR costs are shown in Table 21.16.

Table 21.16: Off-Site Infrastructure Capital Cost Estimate (WBS 6000)

	Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
7200	**Winter Ice Roads**			
7210	Winter Ice Road Construction	16.0	35.6	51.6
7220	Winter Ice Road Maintenance	2.0	0.0	2.0
7230	Winter Ice Road Haulage	7.1	4.0	11.1
	Total	**25.0**	**39.6**	**64.7**

Source: JDS 2015

21.1.7.3 Marine Laydown Area (MLA)

The MLA comprises is described in section 18 of the report. A summary of the bulk earthworks, fuel storage, mobile equipment, infrastructure and port facility costs are shown in Table 21.17.

Table 21.17: MLA Capital Cost Estimate (WBS 7000)

	Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
7100	**MLA**			
7110	MLA - Site Preparation & Quarry	5.2	0.0	5.2
7120	MLA - Fuel Storage and Distribution	9.3	2.0	11.3
7130	MLA - Airstrip	0.0	0.0	0.0
7140	MLA - Mobile Equipment	1.9	0.0	1.9
7150	MLA - Port Facilities & HSE	1.6	0.0	1.6
7160	MLA – Camp	4.2	0.0	4.2
7170	MLA - On-Site Services / Utilities & Pre-Production OPEX	2.1	0.0	2.1
7180	MLA – Electrical Supply & Distribution	1.3	0.0	1.3
7190	MLA - On-Site Communications	0.8	0.0	0.8
	Total	**26.3**	**2.0**	**28.3**

Source: JDS 2015

21.1.8 Tailings Management Facility, Reclaim Water, and Pipelines

Tailings management is described in section 18 of the report. A summary of the estimated costs for tailings management, reclaim water, and pipelines for the Goose Site are shown in Table 21.18.

Table 21.18: Tailings Management Facility and Pipelines Cost Estimate (WBS 8000)

Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
Tailings Management - Goose			
Tailings Pipeline & Distribution	1.0	0	1.0
Tailings Storage Facility	3.5	1.8	5.3
Reclaim Water	1.7	0	1.7
Total	**6.2**	**1.8**	**7.9**

Source: JDS 2015

21.1.9 Indirect Costs

Indirect costs include items that are necessary for the completion of the Project but are not part of the direct costs. They are considered Project indirects and are in addition to contractor indirects. The indirect costs are shown in Table 21.19.

Table 21.19: Indirect Capital Cost Estimate (WBS 9000)

Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
Camp & Catering	8.4	0	8.4
Construction Field Indirects	14.3	0	14.3
Freight & Logistics	36.5	0	36.5
Vendors Reps.	0.5	0	0.5
Start-up & Commissioning	1.3	0	1.3
Spares	2.4	0	2.4
First Fills	2.0	0	2.0
Total	**65.5**	**0**	**65.5**

Source: JDS 2015

21.1.9.1 Camp & Catering

Camp and catering costs have been estimated based on the project man-hours and construction schedule. Preliminary quotes have been obtained for the camp and catering services based on the estimated construction camp size.

21.1.9.2 Construction Field Indirects

Construction field indirect costs are split into the following items:

- Shared services labour (pre-production G&A);
- Equipment rentals and purchases;
- Water management equipment;
- Temporary construction facilities;
- First aid & medical;
- Waste management;
- Mobilization/demobilization; and
- Contractor supervision.

21.1.9.3 Freight / Logistics

During construction, the annual material, equipment and supplies requirements are planned to be shipped from the facilities of various suppliers to the marshalling areas at ports on either the west or east coast of Canada. The supplies will then be shipped by ocean-going vessels to the MLA. Section 25 of this report provides additional details of the Project freight requirements.

Certain supplies that have limited storage onsite, such as packaged explosives, will be air freighted to site using fixed-wing aircraft.

The freight cost estimate includes the following items:

- Freight to staging port costs;
- Sealift costs (staging port to MLA);
- Air freight costs;
- Backhaul costs;
- Sea-container rental costs; and
- Sealift support costs.

21.1.9.4 Vendor Representatives

Vendor representatives will be required at the project site during construction to verify that the installation of the main equipment has been performed in compliance with technical specifications. Representatives will also be required during the pre-commissioning stage. Vendor representative costs were not quoted but factored as a percentage of equipment costs.

21.1.9.5 Commissioning and Start-up

Commissioning and start-up costs were based on supervision required for the plant and major equipment. Commissioning costs were factored as a percentage of the equipment costs.

21.1.9.6 Spare Parts

Spare parts have been considered for start-up, one year of operations and capital. Spare parts for equipment were factored as a percentage of the equipment costs.

21.1.9.7 First Fills

First fills are required for start-up and include the following:

- Mill balls and grinding media;
- Lime and reagents;
- Lubricants;
- Glycol for district heating; and
- Other fills for initial set-up.

First fills were not quoted but factored as a percentage of the equipment costs.

21.1.10 Engineering, Procurement, and Construction Management (EPCM)

The EPCM estimate uses a first principles approach based on man-hours and consultant rates. The EPCM costs are summarized in Table 21.20.

Table 21.20: EPCM Capital Cost Estimate (WBS 10000)

Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
Engineering & Procurement – EP	11.5	0	11.5
Construction Management – CM	18.2	0	18.2
Total	**29.7**	**0**	**29.7**

Source: JDS 2015

Associated services include the following:

- Detailed engineering;
- Procurement;
- Contract management;
- Construction management and supervision;
- Administration and document control;
- Field engineering;
- Quality assurance / quality control (QA/QC);
- Health and safety;
- Surveying; and
- Commissioning.

Further discussion on Project EPCM can be found in section 25 of this report.

21.1.11 Owner's Costs

Owner's costs that are included in the cost estimate are based on the following:

- Owner's team and consultants during the implementation phase. This includes Owner's labour, offices, Owner's consultants, and head office overhead and costs during detailed engineering and construction period;
- Third-party costs,;
- Insurances and fees;
- Owner's start-up and commissioning crew;
- Recruitment and training of operation and maintenance staff;
- Community associated costs;
- Administration; and

A summary of the Owner's costs are shown in Table 21.21.

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Table 21.21: Owner's Cost Estimate (WBS 11000)

Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
G&A Labour	20.2	0	20.2
Health, Safety, Medical & First Aid	0.5	0	0.5
Environmental	1.0	0	1.0
Human Resources	0.2	0	0.2
Insurance & Legal	2.3	0	2.3
Office & Miscellaneous Costs	0.2	0	0.2
Satellite Offices	0.2	0	0.2
Total Owner's Cost	**24.6**	**0**	**24.6**

Source: JDS 2015

21.1.12 Contingency

Contingency is a provision of funds for unforeseen or inestimable costs within the defined Project scope relating to the level of engineering effort undertaken and estimate/engineering accuracy. The contingency is meant to cover events or incidents that occur during the course of the project, which cannot be quantified during the estimate preparation and do not include any allowance for Project risk. No provision is made, or contingency allowed, for design changes or changes to the scope of work.

It is important to note that contingency does not cover force majeure, adverse weather conditions, government policy changes, currency fluctuations, escalation and other Project risks. As well, the contingency will be based solely on the capital estimate and no other Project risks, such as schedule delays or HAZOP assessments.

The contingency factors take into account that most mobile and equipment cost estimates were based on quotations and, therefore, attracted a lower quantum of contingency. Similarly, usage was generally built up from first principles. Sub-Contract or “Other” costs generally included vendor packages with a mix of equipment and labour costs and, consequently, a slightly higher contingency factor was applied. Labour built up from first principles (e.g., mining, Owner’s costs, EPCM and site services) was assigned a lower factor than the labour used for construction. Table 21.22 shows the contingency factors used in the “Initial” and Sustaining Capital “S/C” estimate.

Table 21.22: Contingency Factors

Capital Cost Category	Labour (%)	Materials (%)	Equip (%)	Usage (%)	Other (%)	Initial CAPEX (%)	Sust. CAPEX (%)	LOM Total (%)
Mining	10.0	10.0	0.0	10.0	10.0	3.1	1.2	
On-Site Development	13.0	13.0	10.0	13.0	13.0	12.7	13.0	
Ore Crushing & Handling	11.7	11.0	10.3	11.7	11.7	11.0	0.0	
Process Plant	11.7	11.0	10.3	11.7	11.7	11.0	0.0	
On-Site Infrastructure	10.2	10.2	10.2	10.2	10.2	10.2	6.9	
Off-Site Infrastructure	8.0	8.0	0.0	8.0	8.0	6.1	10.0	
MLA	10.7	10.7	10.7	10.7	10.7	10.7	10.6	
Tailings	12.3	12.3	12.3	12.3	12.3	12.3	0.0	
Indirects	12.0	12.0	10.0	12.0	12.0	11.7	0.0	
EPCM	11.5	11.5	11.5	11.5	11.5	11.5	0.0	
Owner's Costs	11.6	11.6	11.6	11.6	11.6	11.6	0.0	
Reclamation	0.0	0.0	0.0	0.0	0.0	0.0	10.3	
Total Contingency %						**9.9**	**7.7**	**9.2**
Total Contingency ($M)						**37.2**	**13.2**	**50.4**

Source: JDS 2015

21.1.13 Sustaining Capital

The main sustaining capital cost comprises underground mine development occurring during the operations phase. Sustaining capital for underground mining represents the permanent infrastructure and includes the main access ramps, ventilation raise accesses, level accesses, sumps, ore pass accesses, and permanent explosive storage cut-outs, main ventilation raises, and mining equipment.

The following sustaining capital items will be required for the site:

- Open pit sustaining capital is used for the replacement of equipment over the mine life.
- On-site development sustaining capital is used for running additional pipelines for the water management and drainage of the site.
- On-site infrastructure sustaining capital is used for additional gensets, camp beds, a water treatment plant, and IT & Communications for the site.
- The annual cost of constructing the WIR (but excluding road maintenance) is considered a sustaining capital cost.
- MLA infrastructure sustaining capital is used for an additional 10ML tank at the MLA Site.
- Tailings sustaining capital is used for running additional pipelines for the tailings management system.

21.1.14 Closure Cost Estimate

The closure cost estimate assumes closure activities will commence once mining operations stop. Closure will consist of two phases: Closure and Post-closure. During the 2-year active closure stage of Closure, the majority of earthworks and facility decommissioning will take place. Approximately 27,000 man-days will be required for active closure, with the majority expected to be completed in Year 14. Subsequent to the active closure stage, the passive closure stage (part of the Closure phase) will commence and reclamation activities will continue as water is collected and treated and post-closure monitoring continues through the Post-closure phase.

Mine closure and reclamation activities include the following:

- Constructing an on-site demolition landfill;
- Managing hazardous waste;
- Demolishing and disposing of all structures and equipment;
- Landfilling all inert waste, including equipment drained of all oils and hazardous materials;
- Transporting all hazardous waste from the Project sites;
- Disposing all liners and pipelines;
- Re-sloping and crowning all WRSAs and landfills;
- Decommissioning the airstrips and all site roads;
- Sealing all underground mine portals and vent raises;
- Draining and treating all water from the TSF;
- Collecting and treating all contaminated soils;
- Re-contouring the site areas to be consistent with the surrounding geography; and
- Scarifying disturbed surfaces.

The following assumptions were used to build up the closure cost estimate and are summarized in Table 21.23:

- Mobile equipment required for closure was assumed to be provided by a contractor;
- Closure cost estimates used a blended rate of $95/hr for contractor labour;
- Unit cost estimates were based on the contractor equipment fleet; and
- No salvage costs were included in the closure cost estimate due to the impracticality and cost of transporting equipment to market, as well as resulting in the need to construct a dedicated WIR and the provision of shipping.

Table 21.23: Basis of Closure and Reclamation Estimate Summary

Category	Estimate Basis
Open Pit	Open pit closure costs were estimated by applying unit costs from first principles and previous projects to estimated quantities based on current designs.
Underground	Underground closure costs were estimated by applying unit costs from first principles and previous projects to estimated quantities based on current designs. Vent plugs were estimated using first principles.
Waste Rock Stockpiles and Landfills	Capping and sloping of waste rock stockpiles and landfills were estimated by applying unit costs to estimated volumes based on proposed footprints and tonnage.
Water Management Structures and TSF	Water management structure and TSF closure costs were estimated by applying unit costs to material quantities estimated on current designs.
Buildings and Equipment	Buildings and equipment closure costs were estimated using previous project-closure production data which was scaled by area and material quantities.
Roads and Airstrips	Roads and airstrip closure costs were estimated using first principles consistent with the designs.
Water Treatment	Decommissioning costs used unit cost data from previous projects. Pumping and treatment costs were based on operating costs for similarly sized equipment.
Contaminated Soil	Soil investigations were based on unit costs per metre drilled at the required intervals over the testing footprint.
Post-Closure / Closure Monitoring	Cost allowances were based on similar projects.
Indirects	Indirect costs were based on the required man-days to complete active and post-closure activities and associated accommodation, fuel, tool, transportation and supply costs.

Source: JDS 2015

The estimated closure costs by period are shown in Table 21.24.

Table 21.24: Closure Costs by Period

Cost Category	Total Cost (M$)	%
Closure	47.2	74
Post-Closure	16.6	26
Total	63.8	100

Source: JDS 2015

21.1.15 Capital Cost Exclusions

The following items have been excluded from this capital cost estimate:

- Working or deferred capital;
- Financing costs;
- Refundable duties (with the exception of excise and other refundable fuel taxes);
- Currency fluctuations;
- Lost time due to severe weather conditions;
- Lost time due to force majeure;
- Additional costs for accelerated or decelerated deliveries of equipment, materials or services resulting from a change in Project schedule;
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares;
- Any Project sunk costs (studies, exploration programs, etc.);
- Escalation cost;
- Depreciation and depletion allowances;
- Environmental permits;
- Performance bond;
- Builders risk insurance;
- Surface land rights, including water and wildlife compensation; and
- Hiring and relocation.

22 Operating Cost Estimate

22.1 Introduction & Summary

The operating cost estimate was developed using first principles and applying direct applicable Project experience, thus avoiding the use of general industry factors. The operating cost was based on the Owner purchasing and operating the mining and services fleet. Minimal use of permanent contractors was assumed except where value was provided through expertise and / or the provision of seasonal services. Most estimates were derived from engineers, contractors, and suppliers who have provided similar services to existing operations (particularly in northern Canada) and have demonstrated success in executing the plans set forth in this study.

The target accuracy of the operating cost is -15 / +15%, which represents a typical Feasibility Study Budget Class 2 Estimate (AACE).

The operating cost estimate is broken into six major sections:

- Open Pit Mining;
- Underground Mining;
- Processing;
- Site Services;
- Freight Transportation; and
- General & Administrative.

Certain operating costs are associated with activities that commence during the construction phase and continue through the LOM. Some of the costs incurred during the pre-production period relate to the purchase of items such as consumables required for the following year of production. The timing of these costs has been accounted for in the economic analysis. Refer to section 23.2.1 for further details on working capital calculations and breakdown.

Underground lateral and vertical waste development after the pre-production period has been capitalized and will not appear as an operating cost (refer to section 21.1.13 – Sustaining Capital Cost). As is standard industry practice, capital waste development represents the mine's permanent infrastructure and includes the main access ramp, ventilation raise accesses, level accesses, sumps, ore pass accesses and permanent explosive storage cut-outs, as well as main ventilation raises.

The total operating unit cost is $114.58/t processed. LOM operating costs are summarized in Table 22.1.

Note that some totals within the tables in this section might not add exactly due to rounding.

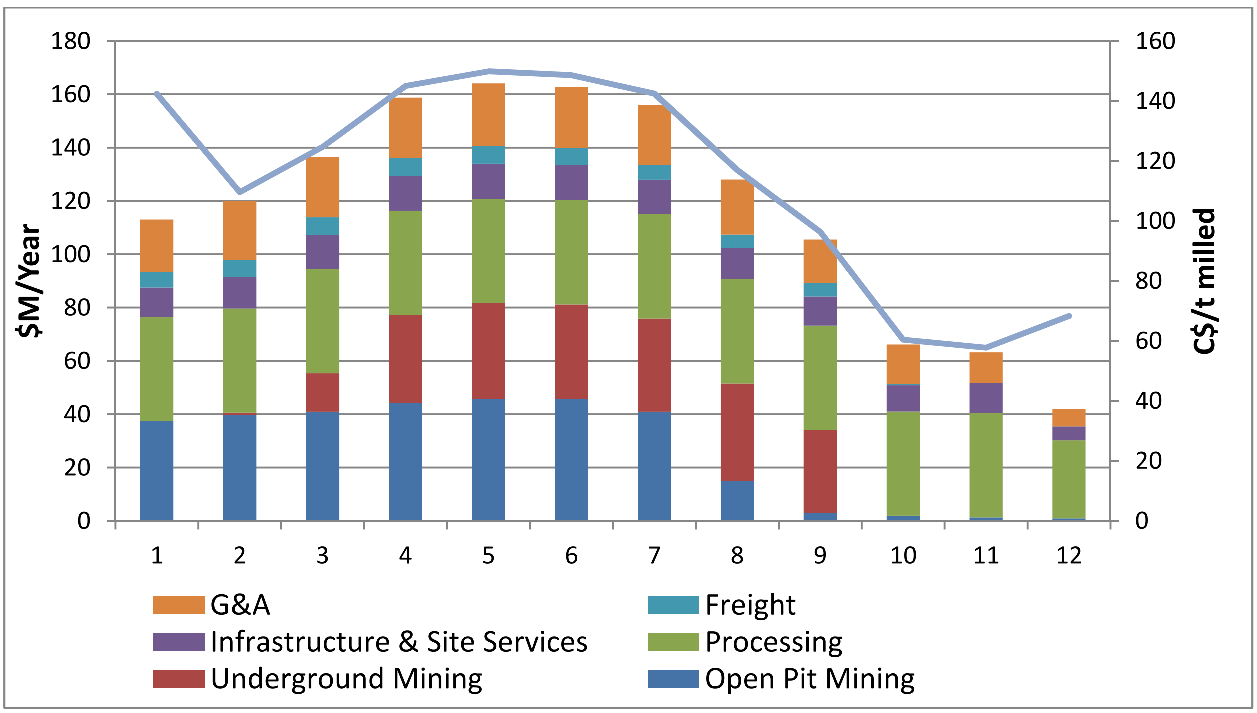
Table 22.1: Estimated Average Operating Cost by Activity

Operating Cost†	Average $M/yr	LOM $M	$/t processed
Mining*	46	539	43.64
Processing	39	459	37.16
Site Services	12	137	11.08
Freight Transportation	5	55	4.42
General & Administrative	19	226	18.28
Total Operating Costs	121	1,416	114.58

(†): Operating Costs include $47.3M of working capital claimed in the pre-production period and excludes pre-stripping costs
(*): Average LOM Open Pit Mining cost amounts to $3.35/t mined at a 10.5:1 strip ratio; average LOM Underground Mining cost amounts to $63.61/t mined

Source: JDS 2015

Figure 22.1: Annual Operating Cost by Area



Source: JDS 2015

Operating costs by category are shown in Table 22.2.

Table 22.2: Estimated Average Operating Cost by Category

Operating Cost	$/t processed	Description
Labour	44.54	All labour except contractors
Fuel	25.84	All fuel usage including power generation, equipment operation
Operating and Maintenance Consumables	26.29	Maintenance parts, reagents, explosives, lubricants, ground engaging tools (GET), PPE, tools
Services	17.91	Contractors, insurance, travel, software, consultants,
Total Operating Cost	**114.58**	

Source: JDS 2015

22.1.1 Operations Labour

Table 22.3 summarizes the total planned workforce during Project operations.

Table 22.3: Summary of Average Employment by Activity

Department	Total Persons Employed (Average)*
Open Pit Mining	131
Underground Mining	114
Processing	95
Site Services	35
Freight	84
G&A	92
Shared Services	35
Total	**586‡**

(*) The average is based on the quantities of total persons employed over the production period. Seasonal averages exclude periods where there is no labour required.
(‡) Total will not sum as the averages are based on average of each discipline over the production period. Peaks will not occur at the same year. The total is the average manpower over the production period inclusive of all disciplines.

Source: JDS 2015

Labour base rates and burdens were determined by reference to other northern Canadian operations. Labour burdens were assembled using first principles and include overtime (scheduled and unscheduled), travel pay, production bonus for underground miners, CPP, EI and WCB, statutory holiday, pension and vacation pay allowances of 6% of scheduled hours, and insurance allowance of 8% of base pay.

22.1.2 Fuel

Based on an assumed long term fuel indicator price and delivery costs, it is expected that fuel deliveries will originate on the west coast of North America utilizing a west coast shipping contractor. Non-mobile equipment (primarily power generation) is exempt from the federal excise tax of $0.04/L and mining related projects receive a rebate on the $0.09/L Nunavut petroleum tax. Fuel delivery costs are based on a quote from a vendor with experience in shipping fuel in the Canadian Arctic and include transportation and off-loading. Fuel costs used for mobile and non-mobile equipment used in the estimate are summarized in Table 22.4. Fuel prices were calculated based on delivered to the MLA only, costs associated with hauling the fuel from the MLA to the Goose and George sites are included in the freight cost.

A long-term diesel price of $2.40 / gallon has been applied to the appropriate project operating costs. This price is closer to the long-term historical diesel price and considered more realistic for the duration of the operation. The indicator diesel price as of September 2015 was $1.42 / gallon.

Table 22.4: Fuel Price Estimation

Fuel Assumptions	Mobile Equipment	Non-mobile Equipment
Per Gallon (US$/US gal)	2.40	2.40
F/X Rate (US$:C$)	0.80	0.80
Price per Gallon (C$/US gal)	3.000	3.000
L to US Gallon Factor	3.785	3.785
Diesel Rack Rate (C$/L)	0.793	0.793
Transport to Site (C$/L)	0.12	0.12
Federal Excise Tax (C$/L)	0.04	0
Nunavut Petroleum tax (C$/L)	0	0
Total Fuel Price (C$/L)	**0.953**	**0.913**

Source: JDS 2015

22.2 Mining

22.2.1 Open Pit Mine Operating Costs

Open pit mining activities are assumed to be undertaken by the Owner. The open pit mining costs include pit and waste operations, road maintenance, mine supervision, and technical services. The average open pit operating costs for the LOM plan are presented in Table 22.5 and Table 22.6, both by mining facility and category.

Table 22.5: Open Pit Operating Cost Estimate – by Activity

Activity	$/tonne mined
Drill & Blast	0.74
Load & Haul	1.96
Mine General	0.05
Mine Maintenance	0.28
Technical Services	0.05
Shared Services	0.27
Total Open Pit Operating Cost	**3.35**

Source: JDS 2015

Table 22.6: Open Pit Operating Cost Estimate – by Category

Open Pit Operating Costs	$/t mined	Description
Labour*	1.58	All labour except contractors
Fuel	0.69	All fuel usage including explosive mixture and mobile equipment operation
Operating and Maintenance Consumables	0.99	Parts, Consumables, oil & lube, GET, Tires, Explosives
Services	0.10	Contractors
Total Operating Costs	**3.35**	

(*) Includes shared services

Source: JDS 2015

22.2.2 Underground Mine Operating Costs

Development, production, mine maintenance, mine services and labour are components that are in the underground mine operating cost build-up. Equipment operating hours, productivities, labour and consumables were estimated for the underground operation. Consumable usage was based on vendor quotes and consumption rates and included rock bolts, explosives, drill bits, wire mesh, piping, and electrical power cables. Electrical power costs were estimated at $0.264/kWh.

Equipment consumables such as parts, tires, electrical power, diesel fuel, and ground engaging tools were included in the equipment operating costs.

Stoping and drifting productivities and operating costs were estimated based on calculated cycle times for each operation, assuming standard drift and stope dimensions.

The average underground operating cost over the LOM is $63.61/t mined. This cost is based on the LOM schedule presented in this report and accounts for the material tonnages mined and their associated costs. Tables 22.7 and 22.8 summarize the total LOM underground mining costs by activity and category, respectively.

Drilling, mucking and hauling operating costs were developed from first principles from the mine plan. Haulage profiles were developed for ore and waste rock to determine required haulage hours.

Table 22.7: Average LOM Underground Operating Cost Estimate – by Activity

Cost Activity	$/t mined	Description
Production	44.54	Labour, fuel, equip maintenance, lubes, tires, consumables, explosives
Mine General	10.02	Power, rock monitoring tools
Maintenance	4.87	Labour, consumables
Technical Services	4.19	Labour including shared services
Total Underground Mining Cost	**63.61**	

Source: JDS 2015

Table 22.8: Average LOM Underground Operating Cost Estimate – by Category

Cost Category	$/t mined	Description
Labour	29.37	All labour including shared services
Fuel	12.67	Electrical power, heat, etc.
Operating and Maintenance Consumables	21.57	Lubes, tires, explosives, pipe, vent, bolts, etc.
Total Underground Mining Cost	**63.61**	

Source: JDS 2015

22.2.3 Shared Mining Services

The labour costs associated with shared mining services includes operations management, technical services, and integrated operations. These costs were proportioned between the open pit and underground mining on the basis of ROM ore tonnes. A total of 72% of the shared services costs were allocated to open pit mining and 28% to underground mining.

22.3 Process Operating Costs

The processing plant operating cost estimate includes:

- Process plant labour and consumables for operations;
- Process plant maintenance for crushing, grinding, leaching, carbon handling, gold refining, and tailings disposal;
- Power plant labour, consumables, and maintenance; and
- Over-the-fence (OTF) contracts for oxygen and laboratory.

Although the power plant provides electricity to the infrastructure facilities, a large component of the power consumed will be for the process plant (particularly crushing and grinding). Consequently, this power cost was included in the process operating cost. Underground power costs were excluded from this component of the power cost and reported separately in the underground operating cost section.

A summary of the process plant operating cost is presented in Table 22.9.

Table 22.9: Processing Operating Costs by Category

Cost Category	$/t processed	Description
Labour	14.35	Includes all process plant and laboratory operations
Power	13.45	Power plant and process equipment such as kiln, furnace, dryer
Operating and Maintenance Consumables	9.35	Spare parts, consumables, reagents for process plant and power plant
Total Process Operating Costs	**37.16**	

Source: Canenco 2015

22.4 Infrastructure and Site Services Operating Costs

Infrastructure and site services operating costs account for the costs such as infrastructure operations and maintenance, winter road haul, winter road maintenance, and support equipment. The infrastructure and site services costs that occur during the pre-production period are included in Pre-production CAPEX. Details of this can be found in section 21. Table 22.10 summarizes the infrastructure and site services operating costs.

Table 22.10: Site Services and Infrastructure Operating Costs

	$/t processed
Infrastructure - Goose	3.70
Infrastructure - MLA	0.38
Winter Road Haul	2.81
Winter Road Maintenance	1.17
Support Equipment	3.03
Total	**11.08**

Source: JDS 2015

22.5 Freight Transportation Operating Costs

During operations, the annual operating supplies are planned to be shipped from the facilities of various suppliers to the marshalling areas at ports on either the west or east coast of Canada. The supplies will then be shipped by ocean-going vessels to the MLA. These supplies include process plant reagents and grinding media, underground mining supplies, ammonium nitrate, maintenance parts, and lubes and oils.

Certain supplies that have limited storage onsite, such as packaged explosives, will be air freighted to site using fixed-wing aircraft.

The freight operating cost estimate includes the following items:

- Freight to port costs;
- Sealift costs (staging port to MLA);
- Air freight costs;
- Backhaul costs;
- Sea-container rental costs; and
- Sealift support costs.

The estimated LOM operating cost associated with freight transportation is $54.7M or $4.42/t processed (Table 22.11).

Table 22.11: Freight Transportation Operating Costs

	$/t processed	Comments/Description
Fuel	0.02	Equipment
Maintenance & Operating Consumables	0.05	Parts, lubricants, consumables
Services	4.35	Contractor labour, freight providers
Total	**4.42**	

Source: JDS 2015

22.6 General and Administrative Operating Costs

General and administrative costs are grouped into the following categories:

- Labour;
- On-site items;
- Satellite office and off-site warehousing; and
- Employee travel (to and from site).

The total G&A unit operating cost is estimated at $18.28 per tonne processed as summarized in Table 22.12.

Table 22.12: Summary of G&A Costs

	$/t processed	Comments/Description
Labour	7.40	All G&A Labour
Services	10.88	HR, IR, insurance, consultants, environment, community, flights, camp catering, etc.
Total	**18.28**	

Source: JDS 2015

23 Economic Analysis

An engineering economic model was developed to estimate annual cash flows and sensitivities. Pre-tax estimates of Project values were prepared for comparative purposes, and after-tax estimates were developed to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Sensitivity analyses were performed for variations in metal price, foreign exchange rate, head grades, operating costs, capital costs, and discount rates to determine their relative importance as Project value drivers.

This technical report contains forward-looking information regarding projected mine production rates, construction schedules, and forecasts of resulting cash flows as part of this study. The mill head grades are based on sufficient sampling that is expected to be reasonably representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits, construct and operate a mine, obtain major equipment or skilled labour on a timely basis, achieve the assumed mine production rates at the assumed grades can all cause actual results to differ materially from those presented in this economic analysis.

The capital and operating cost estimates have been developed specifically for this Project and are summarized in Sections 21 and 22 of this report. They are presented in 2015 Canadian dollars (C$). The economic analysis has been run with no inflation (constant dollar basis).

23.1 Assumptions

All costs and economic results are reported in Canadian dollars (C$), unless otherwise noted. Metal prices are reported in US dollars (US$). Table 23.1 outlines the planned LOM tonnage and grade estimates.

Table 23.1: Life of Mine Plan Summary

Parameter	Unit	Value
Au Price	US $/oz	1,150
F/X Rate	US$:C$	0.80
Mine Life	Years	11.8
Total Ore	Mt	12.4
Processing Rate	t/d	2,882
Average Au Head Grade	g/t	6.3
Au Payable	LOM koz	2,319
	koz/a	197

Source: JDS 2015

Other economic factors used in the economic analysis include the following:

- Discount rate of 5% (sensitivities using other discount rates have been calculated for each scenario);
- Costs based on nominal 2015 Canadian dollar values;
- No application of inflation;
- No PST, GST or duties;
- Numbers are presented on 100% Ownership and do not include management fees or financing costs;
- Exclusion of all pre-development and sunk costs such as exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, and others. Note: Pre-development and sunk costs are used in the tax calculations; and
- Reclamation costs of $63.8M (excluding contingency).

23.2 Timing of Revenue and Working Capital

23.2.1 Working Capital

Working capital for the first year of production has been accounted for in the economic analysis due to the timing difference between cash outflows and cash inflows with respect to the operating costs. This is particularly important in light of the timing related to procurement, delivery to site, and usage, which is estimated to be 12 months.

The following describes how the operating costs were scheduled to occur in the economic analysis:

Mining Operating Costs

- 100% of consumables, material and fuel are assumed to be purchased one year prior to the actual consumption. This models the incurrence of the costs for the consumables, materials, and fuel before the actual use due to the seasonality of the freight transport;
- Labour costs are assumed to be incurred as they are paid; and
- A total of $19.4M of mine operating costs is assumed to occur in the pre-production period of cash flows (Year -1) to account for working capital.

Processing Operating Costs

- 100% of consumables and fuel are assumed to be incurred one year prior to the actual use based on the proposed processing schedule;
- Labour costs are assumed to be incurred as they are paid; and
- A total of $23.0M of processing operating costs is calculated to occur in the pre-production period of cash flows (Year -1) as working capital.

Site Services Operating Costs

- 100% of consumables and fuel are assumed to be incurred one year prior to the actual occurrence/requirement; Labour costs are assumed to be incurred as they are paid; and
- A total of $4.9M of site services operating costs is calculated to occur in the pre-production period of cash flows (Year -1).

A total of $47.3M has been considered as working capital in the pre-production period.

23.2.2 Revenue & Net Smelter Revenue Parameters

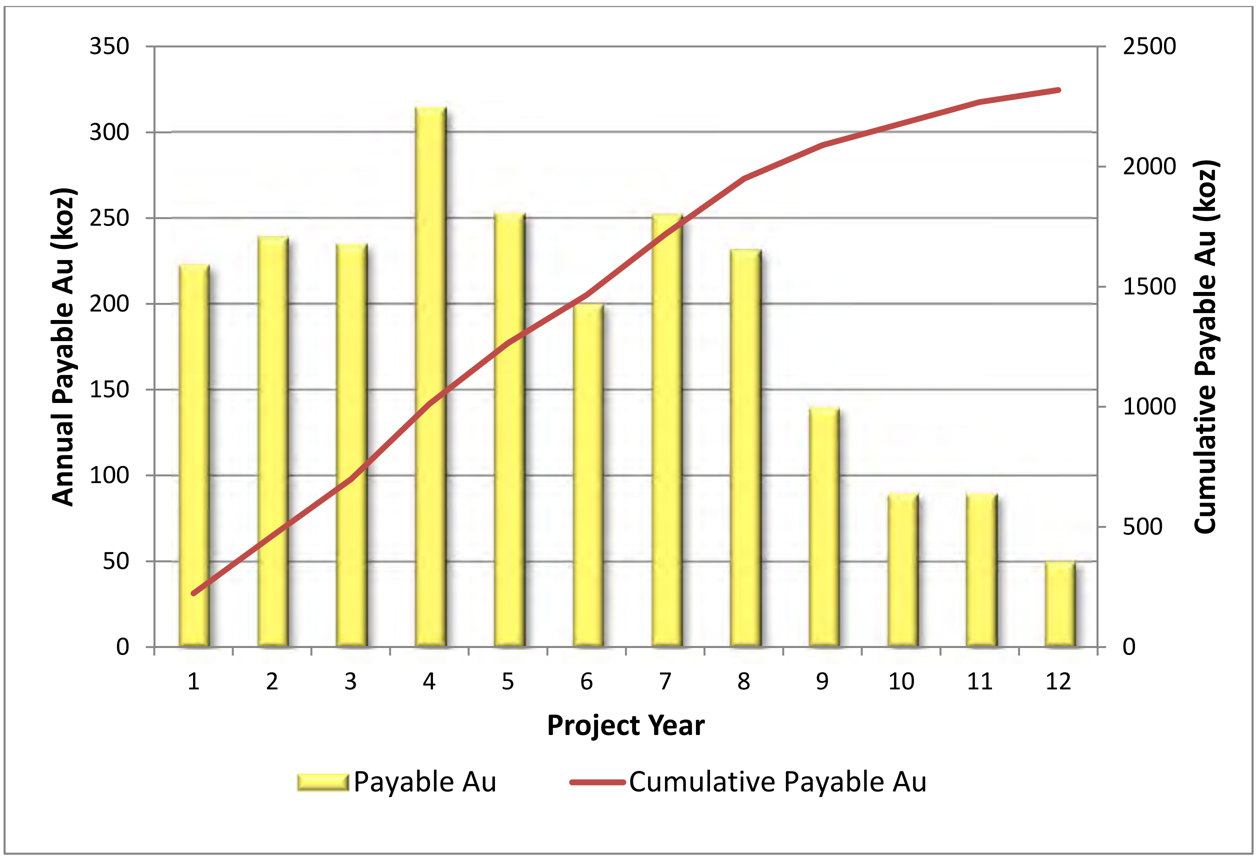
Mine revenue is derived from the sale of gold doré. No contractual arrangements for refining exist at this time. However, the parameters used in the economic analysis were confirmed by a leading industry entity. Gold production and sales are assumed to begin in Year 1 and continue for 11 years. Table 23.2 outlines the market terms used in the economic analysis. Figure 23.1 shows the annual payable gold and cumulative payable gold by Project year.

Table 23.2: NSR Assumptions used in the Economic Analysis

Assumptions	Unit	Value
Au Payable	%	99.8
Au Refining Charge	US$/oz	1
Insurance	% of payable value	0.15
Transport Cost	US$/oz	1

Source: JDS 2015

Figure 23.1: Annual and Cumulative Payable Gold Production



Source: JDS 2015

23.3 Taxes

The Project has been evaluated on an after-tax basis to reflect an indicative, but still approximate, value of the Project. NWT and Nunavut Mineral Royalties, Federal Income Taxes and Nunavut Income Taxes were applied to the Project. A detailed tax analysis was completed by PwC in Vancouver, BC, for the purpose of the after-tax valuation of the Project. Commodity taxes have been excluded from the economic analysis.

The following assumptions and methodologies were used in the analysis:

NWT and Nunavut Mineral Royalties (NTNMR)

- NTNMR have been evaluated as part of the after-tax analysis. The federal government, under the NTNMR requires a royalty be paid to the federal government on defined mining profits. The Crown royalty is levied on a mine-by-mine basis and is equal to the lessor of 13% of the net value of mine output during a fiscal year, and an escalating rate from 0% to 14% on incremental levels of net value of the mine output during a fiscal year.
- Generally, the formula to calculate the output of a mine for a fiscal year is based on the profits from both mining and processing operations, minus a processing allowance that removes from taxable profits a given return on the investment in processing assets. Profits are net of mine site operating costs, exploration costs, depreciation on depreciable mine assets and a development allowance on pre-production costs. The royalties payable under the NTNMR are not subject to the rules in the Income Tax Act (Canada); however, any royalties paid are deductible for income tax purposes under the Income Tax Act (Canada).
- Mineral claims or leases established prior to the Nunavut Land Claims Agreement are grandfathered properties; these have the option of paying royalties based on NTNMR or negotiating a royalty agreement with the designated Inuit organization. Nunavut Tunngavic Incorporated (NTI) is the designated Inuit organization that has vested title to IOL.
- The Back River Mineral Resources considered in this study occur on grandfathered properties subject to royalties under the NTNMR.

Federal and Territorial Corporate Income Tax

- Federal income taxes have been calculated using the current enacted corporate rate of 15% to all estimated pre-tax cash flow generated by the Project. The component of pre-tax cash flow related to the Project will generally be determined by the net operating profits, including deductions for any territorial royalty or mining taxes paid and discretionary deductions for capital cost allowance (CCA), Canadian exploration expenses (CEE), Canadian development expenses (CDE) and reclamation costs paid up to three years after the cessation of pre-tax income. Opening balances of tax pools were incorporated based on Sabina's existing balances, as provided by Management.
- Nunavut income taxes have been calculated using the current enacted corporate rate of 12% to all estimated pre-tax cash flow generated by the Project. Nunavut income tax is levied on taxable income as determined for federal purposes.

- Federal income taxes will be reduced by any available investment tax credit (ITC) balance on qualifying pre-production mining expenditures. 2015 is the final year that Sabina can earn ITC, which is 4% of the qualifying pre-production mining expenditures incurred in 2015. Qualifying pre-production mining expenditures do not include expenditures that have been renounced to flow-through share investors.

Mineral Property Tax Pools

- CEE consists of Canadian exploration expenses incurred for the purpose of determining the existence, location, extent or quality of a Mineral Resource in Canada. CEE does not include depreciable Property of another prescribed class. CEE accumulates in a cumulative CEE pool and a deduction up to the lesser of the CEE pool balance or taxable income may be claimed each year. The CEE pool balance does not expire and can be carried forward indefinitely. CEE renounced to flow-through share investors will reduce the CEE pool.
- CDE consists of resource Property acquisition costs, many pre-production expenses of bringing a new mine into production, including constructing shafts and haulage ways or similar underground work and any exploration drilling costs incurred after the mine comes into commercial production. CDE does not include depreciable Property of another prescribed class. CDE is accumulated in a cumulative CDE pool and is eligible for a deduction of up to 30% of the unclaimed CDE balance each year calculated on a declining basis. The CDE pool balance does not expire and can be carried forward indefinitely.

Capital Cost Allowance (CCA)

- Specific capital cost class CCA rates were applied and used to calculate the appropriate CCA the Company can claim during the entire life of the Project.
- Capital assets acquired for the purpose of producing income from a mine have been included in Class 41. A CCA deduction is permitted in computing taxable income of up to a maximum of 25% on a declining basis and is subject to a 50% reduction on assets acquired during the year. An accelerated rate, over the 25%, is permitted in certain circumstances.

23.4 Third Party Royalties

The Project is subject to NSR royalties payable to various third-party royalties which have been considered in the economic analysis. A total of $96.7M, or effective rate of 3.64%, of third-party royalties are payable over the LOM based on the Project's mine schedule and base case assumptions.

In 2011, Sabina completed the purchase of certain royalties on the Back River and Wishbone Project. The remaining NSR royalties that would apply to the Goose claim area deposit (and other mineral claim areas on the Project) are:

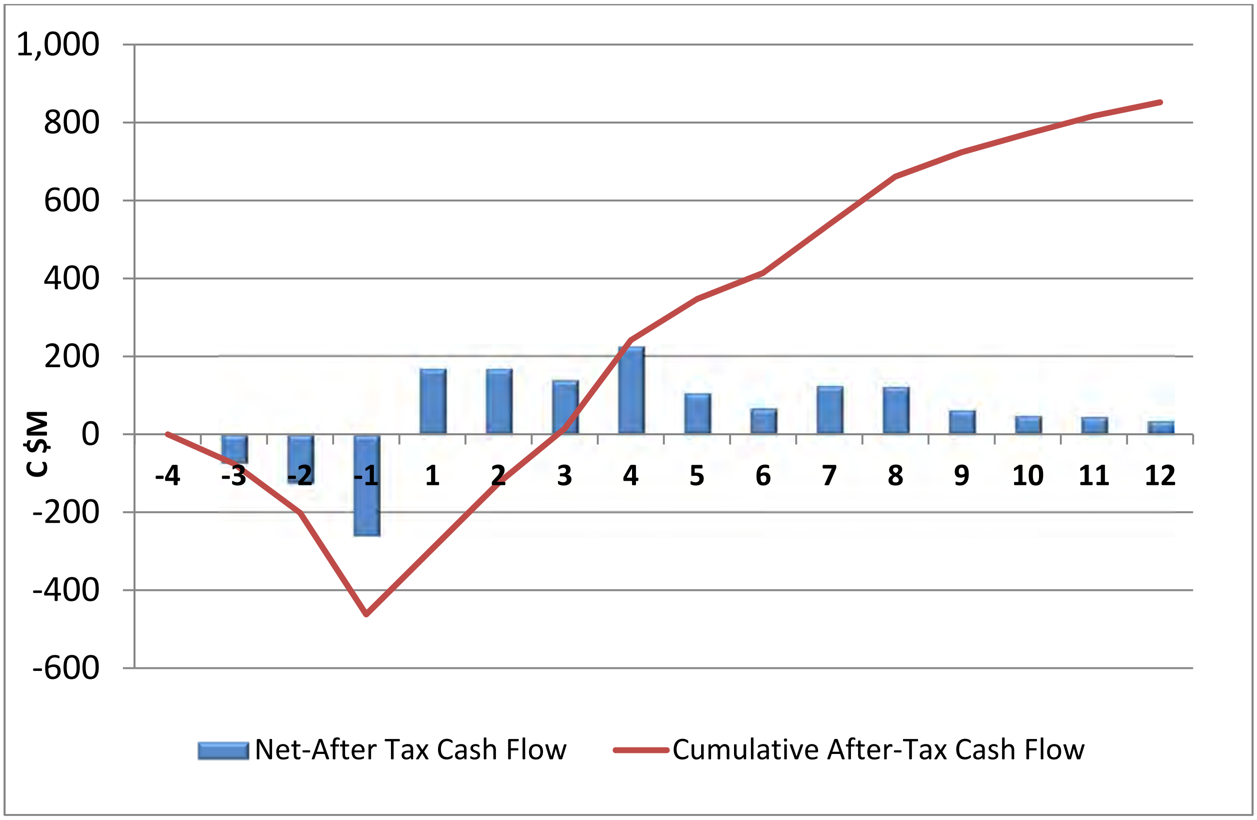
- Goose properties: 0.7% NSR payable on the first 400,000 oz of gold production, increasing to 4.25% on gold production over 400,000 oz.

23.5 Economic Analysis

Based on the findings of the FS, it can be concluded that the Project would be economically viable with an after-tax internal rate of return (IRR) of 24.2% and a net present value ($NPV_{5\%}$) of $480M. Figure 23.2 shows the projected cash flows used in the economic analysis. Table 23.3 shows the detailed results of the evaluated scenario.

Payback is calculated on annual cash flows without considering discount rates or inflation.

Figure 23.2: Annual and Cumulative After-Tax Cash Flows



Source: JDS 2015

Table 23.3: Summary of Economic Results

Category	Unit	Value
Net Revenues	$M	3,201
Operating Costs†	$M	1,416
Cash Flow from Operations	$M	1,833
Capital Costs*	$M	664
Cash Cost‡	US$/oz	534
All-In Sustaining Cash Costs (°)	US$/oz	598
Net Pre-Tax Cash Flow	$M	1,122
Pre-Tax $NPV_{5\%}$	$M	699
Pre-Tax IRR	%	28.2%
Pre-Tax Payback	Years	2.9
Break-Even Pre-Tax Gold Price ($NPV_{5\%}$ = 0)	US$/oz	794
Total Taxes	$M	340
Net After-Tax $NPV_{5\%}$	$M	480
After-Tax IRR	%	24.2%
After-Tax Payback	Years	2.9
Break-Even After-Tax Gold Price ($NPV_{5\%}$ = 0)	US$/oz	795

(†): Operating Costs include $47.3M of working capital claimed in the pre-production
(*): Includes pre-production, sustaining, closure and reclamation capital costs, and contingency
(‡): (Refining Costs + Insurance + Transport Costs + Third Party Royalties + Operating Costs) / Payable Au oz
(°): (Refining Costs + Insurance + Transport Costs + Third Party Royalties + Operating Costs + Sustaining Capital Costs) / Payable Au oz
Source: JDS 2015

23.6 Sensitivity

To test Project value drivers, a sensitivity analysis was performed on the Project's NPV and IRR using a 5% discount rate. The results of this analysis are shown in Table 23.4, Table 23.5, and Figure 23.3. The Project proved to be most sensitive to changes in the foreign exchange rate followed by metal prices, head grades and the least sensitive to capital and operating costs.

A sensitivity analysis of the pre-tax and after-tax results was performed using various discount rates. The results of this analysis are shown in Table 23.6.

Table 23.4: Pre-Tax NPV$_{5\%}$ Sensitivity Results

Factor	Pre-Tax NPV$_{5\%}$ ($M)						
	-15%	-10%	-5%	100%	5%	10%	15%
Metal Price	347	465	582	699	816	933	1,051
F/X Rate	1,098	950	818	699	591	494	404
Head Grade	363	475	587	699	811	923	1,035
OPEX	843	795	747	699	651	603	555
CAPEX	783	755	727	699	671	643	615

Source: JDS 2015

Table 23.5: After-Tax NPV$_{5\%}$ Sensitivity Results

Factor	After-Tax NPV$_{5\%}$ ($M)						
	-15%	-10%	-5%	100%	5%	10%	15%
Metal Price	250	328	404	480	555	631	706
F/X Rate	736	641	556	480	410	347	288
Head Grade	261	336	408	480	552	623	695
OPEX	572	542	511	480	449	418	387
CAPEX	564	536	508	480	452	425	397

Source: JDS 2015

Table 23.6: Pre-Tax IRR Sensitivity Table

Factor	Pre-Tax IRR						
	-15%	-10%	-5%	100%	5%	10%	15%
Metal Price	18.20%	21.80%	25.10%	28.20%	31.20%	34.00%	36.70%
F/X Rate	37.90%	34.50%	31.30%	28.20%	25.40%	22.60%	20.00%
Head Grade	18.70%	22.10%	25.30%	28.20%	31.10%	33.80%	36.40%
OPEX	31.80%	30.60%	29.40%	28.20%	27.00%	25.80%	24.50%
CAPEX	33.60%	31.70%	29.90%	28.20%	26.70%	25.20%	23.90%

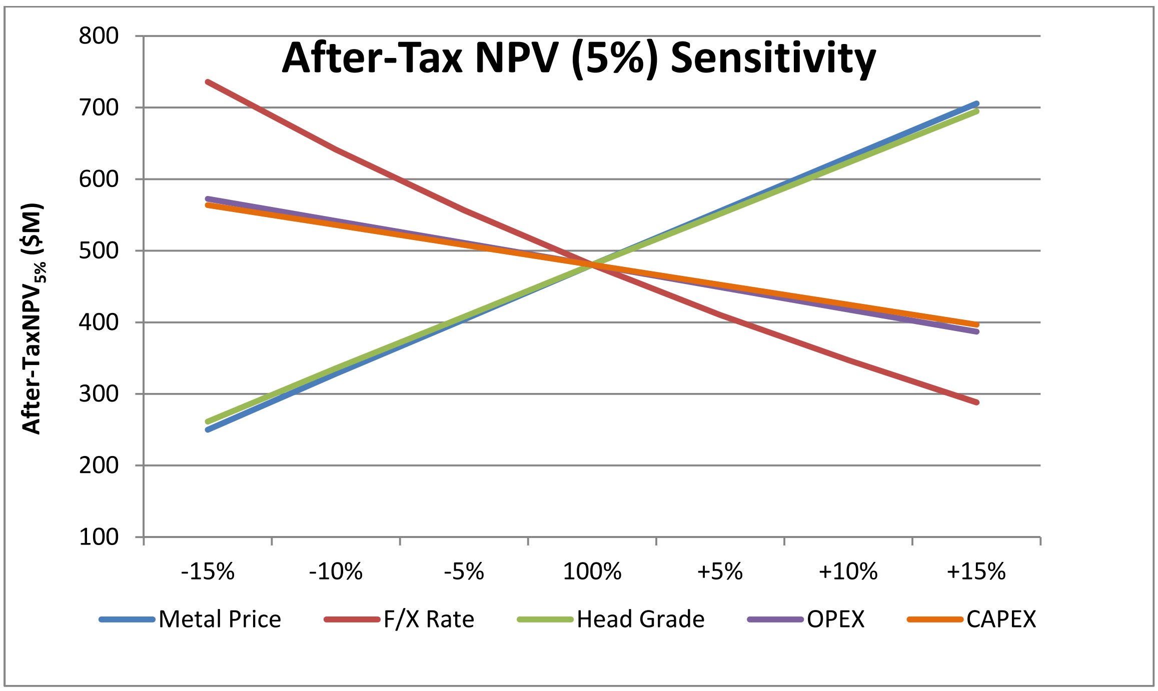
Source: JDS 2015

Table 23.7: After-Tax IRR Sensitivity Table

Factor	After-Tax IRR						
	-15%	-10%	-5%	100%	5%	10%	15%
Metal Price	15.90%	18.90%	21.60%	24.20%	26.70%	29.10%	31.40%
F/X Rate	32.30%	29.40%	26.70%	24.20%	21.80%	19.60%	17.30%
Head Grade	16.30%	19.10%	21.70%	24.20%	26.60%	28.90%	31.10%
OPEX	27.10%	26.20%	25.20%	24.20%	23.20%	22.10%	21.10%
CAPEX	29.80%	27.80%	25.90%	24.20%	22.60%	21.10%	19.70%

Source: JDS 2015

Figure 23.3: After-Tax NPV5% Sensitivity Graph



Source: JDS 2015

Table 23.8: Discount Rate Sensitivity Test Results on NPV

Discount Rate	Pre-Tax NPV ($M)	After-Tax NPV ($M)
0%	1,122	782
5%	699	480
7%	576	391
8%	522	351
10%	426	281

Source: JDS 2015

Back River IPFS Annual Cash Flow Model	Unit	Pre-Production	Production	Total LOM	Year -4	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15 to Year 25
Total Open Pit + Underground																							
Total Ore Mined	M tonnes	0.5	11.9	**12.4**	0.0	0.0	0.0	0.5	1.7	0.7	1.2	2.1	1.6	0.9	2.1	1.1	0.5	0.0	0.0	0.0	0.0	0.0	0.0
Total Waste Mined	M tonnes	5.4	86.2	**91.6**	0.0	0.0	0.0	5.4	12.0	13.0	12.0	12.0	12.7	13.2	9.5	1.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total Mined	**M tonnes**	5.9	98.1	**104.0**	**0.0**	**0.0**	**0.0**	**5.9**	**13.7**	**13.7**	**13.2**	**14.0**	**14.3**	**14.1**	**11.5**	**3.0**	**0.5**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Combined Au Grade	g/t	5.57	6.33	**6.30**	0.00	0.00	0.00	5.57	6.17	7.66	6.66	6.71	6.18	5.95	5.47	6.71	6.34	0.00	0.00	0.00	0.00	0.00	0.00
Re-handled Material	M tonnes	0.0	5.1	5.1				0.0	0.1	0.6	0.1	0.0	0.2	0.4	0.0	0.3	0.6	1.1	1.1	0.6	0.0	0.0	0.0
Ore Haulage Quantities - Goose	M tonnes	0.0	12.4	12.4				0.0	0.8	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	0.6	0.0	0.0	0.0
RECOVERED METAL																							
Total Recovered Au																							
Total Ore Processed	M tonnes	0.0	12.4	**12.4**				0.0	0.8	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	0.6	0.0	0.0	0.0
Au Grade	g/t	0.00	6.30	**6.30**				0.00	9.52	7.41	7.32	9.65	7.71	6.12	7.66	7.06	4.29	2.73	2.73	2.73	0.00	0.00	0.00
Total Contained Au	k oz	0	2,503	**2,503**				0	243	261	258	340	271	215	270	249	151	96	96	54	0	0	0
Overall Recovery	%	0.00	93.0%	**93.0%**				0.0%	92.0%	91.9%	91.4%	92.8%	93.4%	93.2%	93.7%	93.3%	92.9%	93.6%	93.6%	93.6%	0.0%	0.0%	0.0%
Total Recovered Au	**k oz**	0	2,324	**2,324**	**0**	**0**	**0**	**0**	**224**	**240**	**236**	**315**	**254**	**201**	**253**	**232**	**140**	**90**	**90**	**50**	**0**	**0**	**0**
PAYABLE METAL																							
Payable Au	%	99.8%	99.8%	**99.8%**	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%
	k oz	0	2,319	**2,319**	0	0	0	0	223	239	235	315	253	200	252	232	140	90	90	50	0	0	0
	C$M	0	3,333	**3,333**	0	0	0	0	321	344	338	452	364	288	362	333	201	129	129	72	0	0	0
	US$M	0	2,667	**2,667**	0	0	0	0	257	275	270	362	291	230	290	266	161	103	103	58	0	0	0
Cumulative Payable Au	k oz				0	0	0	0	223	462	697	1,012	1,265	1,466	1,718	1,949	2,089	2,179	2,269	2,319	2,319	2,319	2,319
Refining Charge	US$/oz	$1.00	$1.00	**$1.00**	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00
	US$M	0	2	**2**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Insurance	%	0.15%	0.15%	**0.15%**	0.15%	0.15%	0.15%	0.15%	0.15%	0.15%	0.15%	0.15%	0.15%	0.15%	0.15%	0.15%	0.15%	0.15%	0.15%	0.15%	0.15%	0.15%	0.15%
	US$M	0	4	**4**	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	0	0
Transport Cost	US$/oz	$1.00	$1.00	**$1.00**	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00	$1.00
	US$M	0	2	**2**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
NSR Pre-Royalty	**US$M**	0	2,658	**2,658**	**0**	**0**	**0**	**0**	**256**	**274**	**269**	**361**	**290**	**230**	**289**	**265**	**160**	**103**	**103**	**58**	**0**	**0**	**0**
Total Royalty Payments	**US$M**	0.0	96.7	**96.7**	0.0	0.0	0.0	0.0	1.8	4.4	11.5	15.3	12.3	9.8	12.3	11.3	6.8	4.4	4.4	2.5	0.0	0.0	0.0
NSR (After Royalty Payments)	**US$M**	0	2,561	**2,561**	**0**	**0**	**0**	**0**	**254**	**270**	**258**	**345**	**278**	**220**	**277**	**254**	**154**	**98**	**98**	**55**	**0**	**0**	**0**
	C$M	0	3,202	**3,202**	**0**	**0**	**0**	**0**	**317**	**337**	**323**	**432**	**347**	**275**	**346**	**318**	**192**	**123**	**123**	**69**	**0**	**0**	**0**
Operating Costs																							
Open Pit Mining	C$/tonne mined			**$3.35**	$0.00	$0.00	$0.00	$0.00	$2.90	$2.93	$3.30	$3.33	$3.34	$3.18	$2.25	$4.25	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00
	C$M	0.0	297.8	**297.8**	0.0	0.0	0.0	0.0	39.6	40.3	42.9	45.0	45.7	43.1	24.7	10.5	3.0	1.9	1.0	0.2	0.0	0.0	0.0
Underground Mining	C$/tonne mined			**$63.61**	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$112.91	$65.07	$62.56	$62.56	$63.76	$57.09	$31.91	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00
	C$M	0.0	222.1	**222.1**	0.0	0.0	0.0	0.0	0.0	8.0	25.4	34.7	35.4	35.3	35.7	32.5	15.1	0.0	0.0	0.0	0.0	0.0	0.0
Processing	C$/tonne processed			**$37.16**	$0.00	$0.00	$0.00	$0.00	$49.23	$35.69	$35.69	$35.69	$35.69	$35.69	$35.69	$35.69	$35.69	$35.69	$30.43	$19.56	$0.00	$0.00	$0.00
	C$M	0.0	436.2	**436.2**	0.0	0.0	0.0	0.0	39.1	39.1	39.1	39.1	39.1	39.1	39.1	39.1	39.1	39.1	33.3	12.0	0.0	0.0	0.0
Site Services	C$/tonne processed			**$11.08**	$0.00	$0.00	$0.00	$0.00	$14.02	$10.93	$11.65	$12.01	$12.06	$12.03	$11.65	$10.51	$9.83	$9.01	$9.00	$3.10	$0.00	$0.00	$0.00
	C$M	0.0	132.0	**132.0**	0.0	0.0	0.0	0.0	11.1	12.0	12.8	13.2	13.2	13.2	12.8	11.5	10.8	9.9	9.9	1.9	0.0	0.0	0.0
Freight	C$/tonne processed			**$4.42**	$0.00	$0.00	$0.00	$0.00	$7.38	$5.79	$6.08	$6.17	$6.11	$5.82	$4.99	$4.54	$4.64	$0.40	$0.00	$0.00	$0.00	$0.00	$0.00
	C$M	0.0	54.6	**54.6**	0.0	0.0	0.0	0.0	5.9	6.3	6.7	6.8	6.7	6.4	5.5	5.0	5.1	0.4	0.0	0.0	0.0	0.0	0.0
Ore Haulage - Goose	C$/tonne processed			**$0.00**	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00
	C$M	0.0	0	**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
G&A	C$/tonne processed			**$18.28**	$0.00	$0.00	$0.00	$0.00	$24.74	$20.17	$20.71	$20.76	$21.45	$20.84	$20.62	$18.86	$14.85	$13.56	$10.57	$10.71	$0.00	$0.00	$0.00
	C$M	0.0	226.0	**226.0**	0.0	0.0	0.0	0.0	19.6	22.1	22.7	22.7	23.5	22.8	22.6	20.6	16.3	14.8	11.6	6.6	0.0	0.0	0.0
Total Operating Costs	**C$/tonne processed**	**$0.00**	**$114.58**	**$114.58**	**$0.00**	**$0.00**	**$0.00**	**$0.00**	**$145.26**	**$116.61**	**$136.46**	**$147.43**	**$149.38**	**$146.01**	**$128.15**	**$108.84**	**$81.53**	**$60.44**	**$50.90**	**$33.69**	**$0.00**	**$0.00**	**$0.00**
	C$M	**0.0**	**1,368.7**	**1,368.7**	**0.0**	**0.0**	**0.0**	**0.0**	**115.3**	**127.7**	**149.4**	**161.4**	**163.6**	**159.9**	**140.3**	**119.2**	**89.3**	**66.2**	**55.7**	**20.7**	**0.0**	**0.0**	**0.0**
Operating Income	**C$M**	**0.0**	**1,833.0**	**1,833.0**	**0.0**	**0.0**	**0.0**	**0.0**	**202.2**	**209.5**	**173.1**	**270.2**	**183.7**	**115.0**	**205.6**	**198.5**	**102.7**	**56.8**	**67.3**	**48.4**	**0.0**	**0.0**	**0.0**
	C$/tonne processed	**$0.00**	**$148.32**	**$148.32**	**$0.00**	**$0.00**	**$0.00**	**$0.00**	**$254.67**	**$191.31**	**$158.08**	**$246.79**	**$167.75**	**$105.03**	**$187.75**	**$181.29**	**$93.82**	**$51.91**	**$61.45**	**$78.66**	**$0.00**	**$0.00**	**$0.00**
Au Cash Cost	**C$/oz**	**$0**	**$647**	**$647**	**$0**	**$0**	**$0**	**$0**	**$531**	**$562**	**$701**	**$579**	**$712**	**$863**	**$622**	**$580**	**$703**	**$804**	**$687**	**$477**	**$0**	**$0**	**$0**
	US$/oz	**$0**	**$518**	**$518**	**$0**	**$0**	**$0**	**$0**	**$425**	**$449**	**$561**	**$463**	**$569**	**$691**	**$498**	**$464**	**$563**	**$643**	**$550**	**$382**	**$0**	**$0**	**$0**
Capital Costs																							
Mining	C$M	45.9	112.5	**158.3**	0.0	0.0	0.5	45.4	21.5	24.5	27.1	7.6	8.7	10.1	7.0	3.0	3.0	0.0	0.0	0.0	0.0	0.0	0.0
On-Site Development	C$M	15.3	1.3	**16.6**	0.0	1.0	10.3	4.1	0.1	0.0	0.0	0.0	1.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Ore Crushing and Handling	C$M	15.6	0.0	**15.6**	0.0	0.0	3.9	11.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Process Plant	C$M	55.5	0.0	**55.5**	0.0	0.0	6.9	48.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
On-Site Infrastructure (Goose)	C$M	68.1	14.9	**83.0**	0.0	0.8	39.7	27.6	1.2	9.4	1.8	0.0	2.3	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Off-Site Infrastructure	C$M	25.0	39.6	**64.7**	0.0	9.1	8.4	7.5	7.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	1.1	0.0	0.0	0.0	0.0
MLA	C$M	26.3	2.0	**28.3**	0.0	26.3	0.0	0.0	0.0	1.9	0.0	0.0	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Tailings	C$M	6.2	1.8	**7.9**	0.0	0.0	0.0	6.2	1.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Indirects	C$M	65.5	0.0	**65.5**	0.0	16.8	27.1	21.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
EPCM	C$M	29.7	0.0	**29.7**	0.0	13.6	6.7	9.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Owner Costs	C$M	24.6	0.0	**24.6**	0.0	1.3	10.9	12.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Reclamation	C$M	0.0	63.8	**63.8**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	44.6	2.5	16.6
Subtotal	C$M	377.7	235.8	**613.5**	**0.0**	**68.9**	**114.3**	**194.5**	**32.0**	**39.3**	**32.4**	**11.1**	**15.7**	**13.6**	**10.5**	**6.5**	**6.5**	**3.5**	**1.1**	**0.0**	**44.6**	**2.5**	**16.6**
Contingency	C$M	37.2	13.2	50.5	0.0	6.8	11.3	19.2	1.3	1.5	0.8	0.4	0.8	0.5	0.4	0.4	0.4	0.3	0.1	0.0	4.5	0.3	1.7
Total Capital Costs (incl. Contingency)	**C$M**	414.9	249.1	**664.0**	**0.0**	**75.7**	**125.6**	**213.6**	**33.3**	**40.8**	**33.2**	**11.5**	**16.4**	**14.1**	**10.9**	**6.9**	**6.8**	**3.8**	**1.2**	**0.0**	**49.1**	**2.8**	**18.3**
Working Capital	**C$M**	47.3	0.0	**47.3**	0.0	0.0	0.0	47.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Net Pre-Tax Cash Flow	**C$M**	-462.3	1,584.0	**1,121.7**	**0.0**	**-75.7**	**-125.6**	**-261.0**	**168.9**	**168.7**	**139.9**	**258.7**	**167.3**	**100.9**	**194.7**	**191.6**	**95.9**	**53.0**	**66.1**	**48.4**	**-49.1**	**-2.8**	**-18.3**
Cumulative Pre-Tax Cash Flow	C$M				0.0	-75.7	-201.3	-462.3	-293.4	-124.7	15.2	273.9	441.2	542.1	736.8	928.4	1,024.3	1,077.3	1,143.4	1,191.8	1,142.7	1,139.9	1,121.7
Taxes	**C$M**	0.0	339.9	**339.9**		0.0	0.0	0.0	0.0	0.1	0.1	32.1	61.7	34.1	69.9	69.1	33.4	5.2	20.6	13.7	0.0	0.0	0.0
Net After-Tax Cash Flow	**C$M**	-462.3	1,244.1	**781.8**	**0.0**	**-75.7**	**-125.6**	**-261.0**	**168.9**	**168.6**	**139.8**	**226.7**	**105.6**	**66.8**	**124.8**	**122.5**	**62.5**	**47.9**	**45.6**	**34.7**	**-49.1**	**-2.8**	**-18.3**
Cumulative After-Tax Cash Flow	C$M				0.0	-75.7	-201.3	-462.3	-293.4	-124.8	15.0	241.6	347.2	414.0	538.8	661.3	723.8	771.7	817.3	851.9	802.8	800.0	781.8

24 Adjacent Properties

In 2015, Sabina was the only exploration company actively exploring in the vicinity of the Back River area.

The Wishbone Property, located approximately 50 km west-southwest of Goose camp, consists of the Wishbone claims, Malley claims and the Needle lease, as shown in Figure 4.2. This claim and lease package is owned by Sabina, but no exploration drilling was conducted in 2015.

The Hackett River Project is located 50 km west-northwest of the George camp, as shown in Figure 4.1. The Hackett River Project is a large, undeveloped, volcanogenic massive sulphide deposit that Sabina sold to Xstrata in 2011. It is currently owned by Glencore Canada Corp. following its 2013 takeover of Xstrata.

The following text is taken from Sabina's news release, dated June 2, 2011:

Under the terms of the Agreement, Xstrata has agreed to pay cash consideration of $50 million. Sabina will reserve a silver production royalty equal to 22.5% of the first 190 million ounces of payable silver from the current resource at Hackett River and other properties and 12.5% of all payable silver from the Properties thereafter at no future cost to Sabina. Additionally, Xstrata has agreed to incur not less than $50 million on exploration and other expenditures on the Properties over a four year period in order to advance the Properties and complete a National Instrument 43-101 compliant Feasibility Study.

25 Other Relevant Data and Information

25.1 Operational Logistics

25.1.1 Introduction

The annual transportation of equipment, fuel, and consumables to the Goose Site is a critical aspect of Project operations. The majority of freight will be delivered to site first by annual sealift to the MLA during the open-water season, then by WIR from the MLA to Goose Site. Fixed-wing aircraft will support operational activities with delivery of select materials, perishables (food), and passengers throughout the life of the mine.

The methodology to determine the distribution of freight between the available delivery methods, as well as the available source data and methods used to determine the feasible capacity of each delivery method, is described in this section.

25.1.2 Load List

An estimated 140 kilotonnes (kt) of freight is required to support operational activities over the life of the Project. A comprehensive load list was prepared to analyse the weights and volumes of all supplies, equipment, and materials required during operations for each site. Of the 140 kt of freight, there are 0.9 kt of operational freight and 2.3 kt of catering freight that will be transported to the sites by air. The remaining 136 kt of freight will be transported by the annual sealift. The inbound freight classification is shown in Table 25.1.

Table 25.1: Production Freight Requirements (tonnes)

	Sea Freight	Air Freight - Operations	Air Freight - Catering	Total Freight
Year 1	10,020	10	200	10,230
Year 2	12,420	50	240	12,710
Year 3	14,450	130	250	14,830
Year 4	14,940	120	250	15,310
Year 5	14,970	150	260	15,380
Year 6	14,850	120	260	15,230
Year 7	14,080	120	250	14,450
Year 8	11,650	110	210	11,970
Year 9	10,310	90	130	10,530
Year 10	7,240	0	120	7,360
Year 11	7,240	0	100	7,340
Year 12	4,160	0	50	4,210
Total	136,330	900	2,320	139,550

Source: JDS 2015

Freight quantities shown in Table 25.1 are summarized for the year they will be consumed on site. Freight delivered by the annual sealift will need to be transported to the MLA one year prior to being required on site.

In addition to the freight requirements, a total of 361 million litres (ML) of fuel is required over the 12-year operations period. All fuel required during the production period will be transported by ocean-going vessels during the annual sealift. The inbound fuel breakdown by site is shown in Table 25.2. Quantities are shown in the year they are consumed; fuel quantities need to be transported to the MLA one year before they are required on site.

Table 25.2: Production Fuel Requirements (thousand litres)

	MLA	Goose	Total Fuel
Year 1	27,120	1,338	28,458
Year 2	28,131	1,401	29,532
Year 3	34,016	1,469	35,485
Year 4	37,021	1,491	38,512
Year 5	37,515	1,493	39,008
Year 6	37,166	1,491	38,657
Year 7	36,133	1,472	37,605
Year 8	30,015	1,394	31,409
Year 9	26,120	1,350	27,470
Year 10	18,944	1,260	20,204
Year 11	18,861	1,198	20,059
Year 12	14,225	0	14,225
Total	345,267	15,357	360,624

Source: JDS 2015

25.1.3 Air Freight

Fixed-wing aircraft will be used to transport certain operating supplies (primarily packaged explosives), catering supplies, and passengers to the Goose and MLA sites throughout the production period.

The primary aircraft to be used at each site, and their associated capacities, are shown in Table 25.3; these numbers are based on the runway capabilities described in the section 18.

Table 25.3: Primary Aircraft and Capacities

Site	Aircraft	Primary Usage	Capacity
Goose	De Havilland DHC-5D Buffalo	Supplies	8,164 kg
Goose	De Havilland Dash 7	Passengers	40 person
MLA	De Havilland DHC-6 Twin Otter	Supplies	1,090 kg
MLA	De Havilland DHC-6 Twin Otter	Passenger	10 person

Source: JDS 2015

Deliveries for both sites are anticipated to originate in Yellowknife, NWT. Passengers from southern Canada will fly commercially from Edmonton, AB to Yellowknife and then be transported via charter aircraft to the Goose and MLA sites.

Approximate straight-line distances for typical routes are as follows:

- Yellowknife to Goose Site 520 km; and
- Yellowknife to MLA 570 km.

The following sections further describe the requirements for each type of cargo that will be delivered by fixed-wing aircraft.

25.1.3.1 Freight

Freight requirements for each site are summarized by year in Table 25.4.

Table 25.4: Fixed-Wing Freight Requirements

Site	Goose Site		MLA Site		Total All Sites	
Aircraft	Buffalo		Twin Otter		All Aircraft	
	Tonnes	Flights	Tonnes	Flights	Tonnes	Flights
Year 1	174	22	33	31	207	53
Year 2	258	32	31	29	289	61
Year 3	350	43	32	30	382	73
Year 4	343	43	32	30	375	73
Year 5	376	47	32	30	408	77
Year 6	346	43	32	30	378	73
Year 7	334	41	32	30	366	71
Year 8	283	35	32	30	315	65
Year 9	187	23	32	30	219	53
Year 10	86	11	33	31	119	42
Year 11	80	10	20	19	100	29
Year 12	54	7	0	0	54	7
Total	2,871	357	341	320	3,212	677

Source: JDS 2015

25.1.3.2 Fixed-Wing Passenger Transportation

Passenger flights to the Goose Site will originate in Yellowknife and use a De Havilland Dash 7 aircraft. Passenger flights to the MLA will originate in Yellowknife and use a Twin Otter. An average capacity factor of 90% has been assumed for load planning. Annual passenger delivery requirements are summarized in Table 25.5.

Table 25.5: Fixed-Wing Passenger Requirements

Site	Goose Site		MLA Site		Total All Sites	
Aircraft	Dash 7		Twin Otter		All Aircraft	
	Passenger	Flights	Passenger	Flights	Passenger	Flights
Year 1	3,071	86	592	66	3,663	152
Year 2	3,718	104	567	63	4,285	167
Year 3	3,875	108	573	64	4,448	172
Year 4	3,939	110	579	65	4,518	175
Year 5	4,048	113	579	65	4,627	178
Year 6	3,983	111	579	65	4,562	176
Year 7	3,866	108	579	65	4,445	173
Year 8	3,187	89	580	65	3,767	154
Year 9	1,927	54	579	65	2,506	119
Year 10	1,707	48	586	66	2,293	114
Year 11	1,610	45	391	44	2,001	89
Year 12	1,087	31	0	0	1,087	31
Total	36,018	1,007	6,184	693	42,202	1,700

Source: JDS 2015

25.1.4 Sealift Freight

The majority of fuel, production supplies, equipment, and materials will be transported to the MLA by ocean-going vessels during the annual sealift period. Dry freight will be shipped by either barges or ships. Fuel will be shipped by tankers.

Dry freight will be consolidated at one of two port locations depending on the point of origin. Supplies, equipment, and material originating in western North America or China will be consolidated at Vancouver, BC. Supplies, equipment, and material originating in eastern North America or Europe will be consolidated at Bécancour, QC.

Dry freight was categorized by the following shipping method:

- Roll-on/roll-off (RORO) – typically mobile mining and auxiliary equipment;
- Modular – large modular components that cannot be containerized (e.g., camp modules);
- Rack – typically steel and piping; and
- Containerized – materials containerized in standard 20-ft ISO containers.

The weight and volume of all RORO, modular, rack freight, and the total number of containers required were determined from engineering equipment lists. These freight parameters were provided to qualified shipping companies on the west and east coasts of Canada to determine sealift costs for dry freight. West coast freight quantities are shown in Table 25.6.

East coast freight quantities are shown in Table 25.7. Freight quantities are provided in the year they are transported to the MLA; however, they will not be consumed at each site until the following year.

Table 25.6: West Coast Dry Freight Quantities

Year	RORO		Modular		Rack		Containers
	Tonnes	Volume (m^3)	Tonnes	Volume (m^3)	Tonnes	Volume (m^3)	#
-1	361	3,606	0	0	0	0	526
1	100	475	110	419	0	0	599
2	102	445	0	0	0	0	618
3	51	223	0	0	0	0	635
4	218	838	0	0	0	0	639
5	0	0	0	0	0	0	639
6	0	0	0	0	0	0	600
7	0	0	0	0	0	0	472
8	0	0	0	0	0	0	425
9	0	0	0	0	0	0	388
10	0	0	0	0	0	0	612
Total	832	5,587	110	419	0	0	6,153

Source: JDS 2015

Table 25.7: East Coast Dry Freight Quantities

Year	RORO		Modular		Rack		Containers
	Tonnes	Volume (m3)	Tonnes	Volume (m3)	Tonnes	Volume (m3)	#
-1	26	124	0	0	0	0	26
1	189	503	0	0	0	0	197
2	230	540	0	0	0	0	232
3	102	236	0	0	0	0	102
4	0	0	0	0	0	0	0
5	0	0	0	0	0	0	0
6	0	0	0	0	0	0	0
7	0	0	0	0	0	0	0
8	0	0	0	0	0	0	0
9	0	0	0	0	0	0	0
10	0	0	0	0	0	0	0
11	0	0	0	0	0	0	0
Total	**547**	**1,403**	**0**	**0**	**0**	**0**	**557**

Source: JDS 2015

The dry freight requirements for Years 11 and 12 will be significantly reduced. Therefore, all sea freight required for operations in Years 11 and 12 will be transported to the MLA during the final sealift in Year 10.

Cargo will be transported to the MLA by ice-class ocean-going barges or ships. The design specifications for ocean-going barges and ships are as follows:

Ocean-going barges:

- Dead weight tonnage of 16,000 t; and
- Draft of 5.9 m.

Ships:

- DWT of 17,000 t; and
- Draft of 9.7 m.

The barges and ships will be self-sufficient for off-loading cargo. Lightering barges will be used to transfer cargo from the vessel to the lighter barge terminal at the MLA barge landing area.

25.1.5 Fuel Freight

Fuel freight for production purposes will be sourced by the fuel shipping contractor using brokerage companies that can supply large volumes of fuel at competitive prices. Based on fuel indicator pricing and delivery costs, it is expected that fuel deliveries will originate on the west coast of North America and use a west coast shipping contractor. Ice-class fuel tankers will use anchors to secure the ship offshore during off-loading. During the production period, the MLA will have a Level 2 Oil Handling Facility (OHF) classification that permits a maximum oil transfer rate of up to 750 m^3/hr.

25.1.6 Manpower

Sabina's management team will be responsible for managing the Project logistics. The Owner's team will be responsible for the following:

- Coordinating delivery of equipment and material to the appropriate ports;
- Ensuring timely delivery;
- Tracking movement of equipment and material to the port sites;
- Verifying deliveries to the port sites;
- Working with the shipping contractor to set shipping priorities;
- Validating shipping manifests;
- Coordinating off-loading activities at the MLA;
- Developing and implementing MLA storage plans; and
- The primary Sabina staff responsible for project logistics will be the Procurement/Contracts/Logistics Manager and the Logistics Coordinator.

During the sealift period, additional personnel will be mobilized to the MLA to assist with dry freight off-loading and storage. Due to the temporary nature of the sealift period, it is anticipated that the sealift support crew will be provided by a contractor. Table 25.8 provides a list of the contractor personnel that will be on site to assist during the sealift period.

Table 25.8: Sealift Support Crew

Contractor Position	On-Site Personnel
Superintendent	1
Supervisor	1
Administrator / Dispatch	1
HEO – Skilled	4
HEO – Semi-Skilled	4
Labourer	2
Mechanic	1
Total On-Site Labour	**14**

Source: JDS 2015

25.2 Project Execution and Development Plan

25.2.1 Introduction & Philosophy

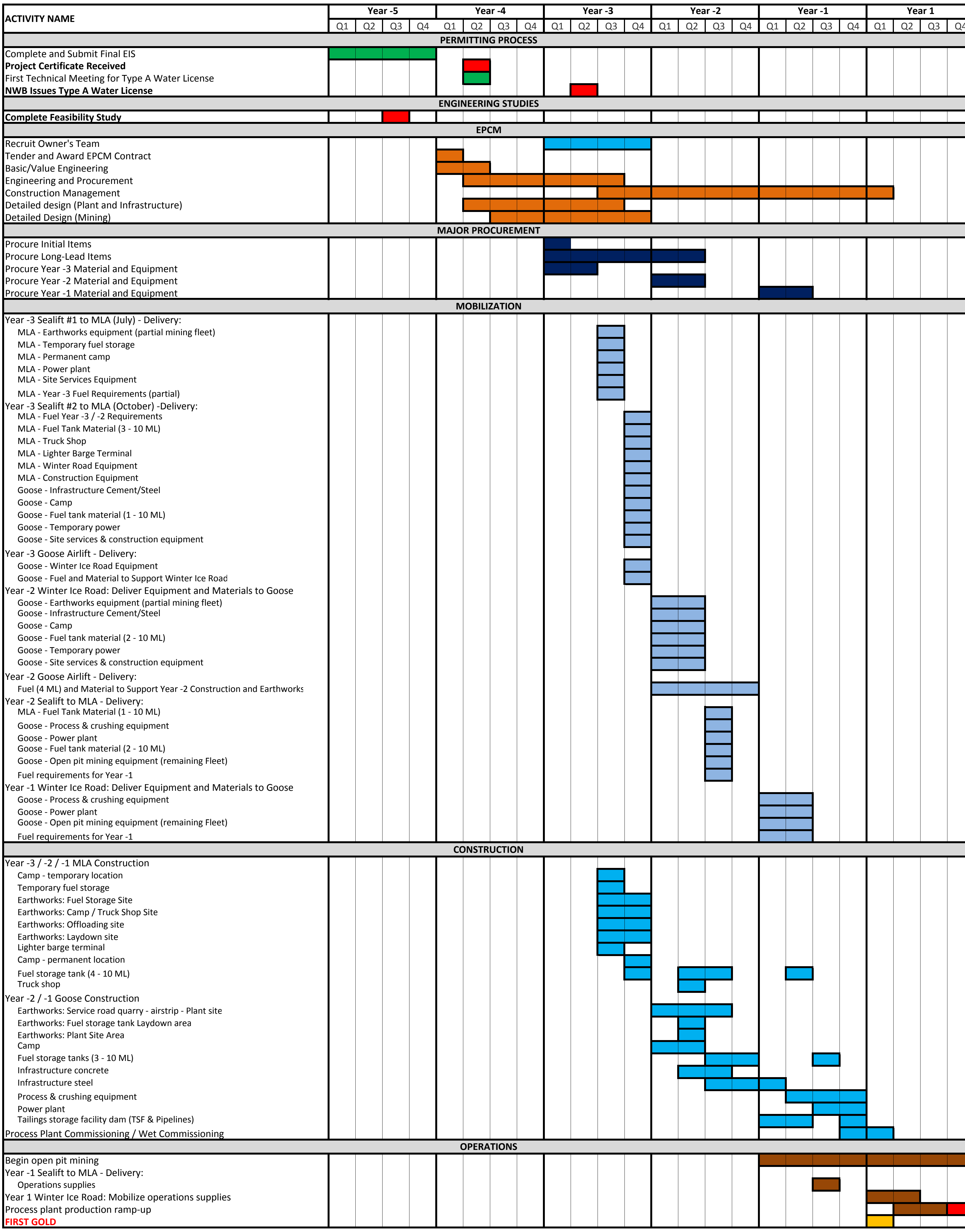
The project execution plan for the Back River FS is based on principles tested and proven in the development of remote, logistically-challenging projects in northern Canada. These principles include the following:

- Promote safety in design, construction, and operations to succeed;
- Minimize disturbance footprint using simple, passive environmental solutions;
- Use fit-for-purpose designs, constructions, and operations;
- Consolidate construction and operational needs to the extent practical due to the high cost of transportation to and from site (i.e., “bring it in: it stays”);
- Purchase common equipment fleet at the outset (by Owner) and use for construction needs;
- Minimize on-site labour requirements for an efficient operation;
- Negotiate contracts with suppliers, contractors, and engineers with proven track records in northern Canadian mine developments;
- Complete Project construction components early and turn over to operations;
- Eliminate surplus management organizations; and
- Accommodate all site personnel at the same camp (i.e., no management quarters).

Similar to the operational plan, the majority of construction freight will be delivered using an annual sealift and WIR. Fixed-wing aircraft will deliver an amount of fuel and equipment in the first year of construction and select bulk freight and passengers throughout the duration of construction.

25.2.2 Project Scheduling

The Project schedule is driven by the annual sealift and WIR windows in Year -2 and Year -1. A detailed, resource-loaded schedule has been developed for the MLA and Goose Site construction activities, using the Initial Project Feasibility Study cost estimate as the basis for the required man-hours. This scheduling exercise indicates that mechanical completion and wet commissioning can be accomplished within the 2-year construction period. The key scheduling milestones are shown in Figure 25.1.


ACTIVITY NAME
Year -5
Year -4
Year -3
Year -2
Year -1
Year 1
Q1
Q2
Q3
Q4
PERMITTING PROCESS
Complete and Submit Final EIS
Project Certificate Received
First Technical Meeting for Type A Water License
NWB Issues Type A Water License
ENGINEERING STUDIES
Complete Feasibility Study
EPCM
Recruit Owner's Team
Tender and Award EPCM Contract
Basic/Value Engineering
Engineering and Procurement
Construction Management
Detailed design (Plant and Infrastructure)
Detailed Design (Mining)
MAJOR PROCUREMENT
Procure Initial Items
Procure Long-Lead Items
Procure Year -3 Material and Equipment
Procure Year -2 Material and Equipment
Procure Year -1 Material and Equipment
MOBILIZATION
Year -3 Sealift #1 to MLA (July) - Delivery:
MLA - Earthworks equipment (partial mining fleet)
MLA - Temporary fuel storage
MLA - Permanent camp
MLA - Power plant
MLA - Site Services Equipment
MLA - Year -3 Fuel Requirements (partial)
Year -3 Sealift #2 to MLA (October) -Delivery:
MLA - Fuel Year -3 / -2 Requirements
MLA - Fuel Tank Material (3 - 10 ML)
MLA - Truck Shop
MLA - Lighter Barge Terminal
MLA - Winter Road Equipment
MLA - Construction Equipment
Goose - Infrastructure Cement/Steel
Goose - Camp
Goose - Fuel tank material (1 - 10 ML)
Goose - Temporary power
Goose - Site services & construction equipment
Year -3 Goose Airlift - Delivery:
Goose - Winter Ice Road Equipment
Goose - Fuel and Material to Support Winter Ice Road
Year -2 Winter Ice Road: Deliver Equipment and Materials to Goose
Goose - Earthworks equipment (partial mining fleet)
Goose - Infrastructure Cement/Steel
Goose - Camp
Goose - Fuel tank material (2 - 10 ML)
Goose - Temporary power
Goose - Site services & construction equipment
Year -2 Goose Airlift - Delivery:
Fuel (4 ML) and Material to Support Year -2 Construction and Earthworks
Year -2 Sealift to MLA - Delivery:
MLA - Fuel Tank Material (1 - 10 ML)
Goose - Process & crushing equipment
Goose - Power plant
Goose - Fuel tank material (2 - 10 ML)
Goose - Open pit mining equipment (remaining Fleet)
Fuel requirements for Year -1
Year -1 Winter Ice Road: Deliver Equipment and Materials to Goose
Goose - Process & crushing equipment
Goose - Power plant
Goose - Open pit mining equipment (remaining Fleet)
Fuel requirements for Year -1
CONSTRUCTION
Year -3 / -2 / -1 MLA Construction
Camp - temporary location
Temporary fuel storage
Earthworks: Fuel Storage Site
Earthworks: Camp / Truck Shop Site
Earthworks: Offloading site
Earthworks: Laydown site
Lighter barge terminal
Camp - permanent location
Fuel storage tank (4 - 10 ML)
Truck shop
Year -2 / -1 Goose Construction
Earthworks: Service road quarry - airstrip - Plant site
Earthworks: Fuel storage tank Laydown area
Earthworks: Plant Site Area
Camp
Fuel storage tanks (3 - 10 ML)
Infrastructure concrete
Infrastructure steel
Process & crushing equipment
Power plant
Tailings storage facility dam (TSF & Pipelines)
Process Plant Commissioning / Wet Commissioning
OPERATIONS
Begin open pit mining
Year -1 Sealift to MLA - Delivery:
Operations supplies
Year 1 Winter Ice Road: Mobilize operations supplies
Process plant production ramp-up
FIRST GOLD

25.3 Temporary Facilities for Construction

25.3.1 Goose Site

The Goose Site currently has the following facilities:

- Exploration camp with approximately 140-person capacity and ancillary structures;
- 915-m gravel airstrip;
- Fuel storage for approximately 975,000 L of diesel fuel;
- Diesel generators (2 x ~400 kW; 1 x~175 kW);
- Workshop and warehouse facilities;
- Mobile crushing and screening plant; and
- Various pieces of heavy equipment to support exploration activities.

To carry out construction at the Goose Site in preparation for operations, a significant increase in site facilities is required, and includes the following:

- Expansion of site camp capacity to 303 people during construction and approximately 300 people during operations. The existing exploration camp will be used for initial earthworks activities and during the construction of the main camp facilities. The exploration camp will be used during construction as an overflow facility during periods of peak activity. Construction of the main camp will be the first priority in Year -2.
- Construction of additional diesel storage at the plant site area. A total of four 10-ML field-erected tanks will be constructed to store diesel required for operations. One tank will be constructed during Year -2 to support construction and pre-stripping activities. Two additional tanks will be constructed in Year -1 in preparation for mining and processing, and the final tank will be constructed in Year 2, ahead of underground mining activities.
- Addition of three 1-MW diesel generators in Year -2 to support the expanded camp and to provide construction power. These will be deployed as emergency backup generators during operations.
- Construction of additional ancillary structures: waste disposal, STP, laydown areas.

25.3.2 Marine Laydown Area

Currently, there are no facilities at the MLA Site. Construction and operations of the MLA will require the following infrastructure:

- Initial diesel storage of eight 90,000-L double-wall fuel tanks with portable containment berms to support construction activities. This will be increased to four 10-ML field-erected tanks to store annual diesel requirements until the bulk is transported on the WIR to the Goose Site. One tank will be constructed in Year -3, two additional tanks in Year -2, and the remaining tank in Year -1.
- A construction and operations camp with capacity for 94 people. The camp will be transported to the MLA in Year -3 and set up for construction. Once earthworks are complete, it will be relocated to its final position for operations.
- Two 500-kW diesel generators for use during construction and operations.

25.4 Construction Materials

All construction materials and equipment required during the construction of the MLA and Goose sites will be transported by aircraft or by sealift. Materials delivered by sealift to the MLA and destined for the Goose Site will be trucked over WIRs. Rock-fill and concrete aggregates will be produced on site from the Umwelt open pit.

Annual material and equipment quantities required during the pre-production period, excluding catering freight, are shown in Table 25.9.

Table 25.9: Pre-production Annual Material and Equipment Quantities (tonnes)

Description	Year -3	Year -2	Year -1	Total
Operations Equipment	1,180	1,140	1,370	3,690
Construction Equipment & Material	2,770	10,130	4,510	17,410
Packaged Explosives	350	760	50	1,160
Oil & Lubes	20	70	100	190
Maintenance Parts	30	100	160	290
Tires	10	40	110	160
Total Equipment & Material	4,360	12,240	7,530	24,130

Source: JDS 2015

Of the total equipment and material required during pre-production, 1.3 kt and 0.4 kt will be airfreighted to Goose and MLA sites, respectively. The remaining 22.4 kt will be sealifted to the MLA.

25.5 Site Access and Mobilization

Currently, both sites are only accessible by air: the Goose Site has a 915-m gravel airstrip capable of handling medium-sized turboprop aircraft, and the MLA is only accessible by float plane (during summer months) and helicopters dispatched from the Goose Site.

25.5.1 Goose Site

Prior to infrastructure construction at the Goose Site, an ice airstrip on Goose Lake will be used to accommodate large turboprop and jet aircraft while the existing 915-m airstrip will be used for smaller turboprop aircraft.

The construction of the Goose Site will occur over a 24-month period. To meet this timeline, the following will be required:

- Air delivery of equipment and supplies necessary to construct the first winter ice road between Goose Site and the MLA beginning late in Year -3;
- Construction of the MLA facilities in Year -3;
- Summer sealift campaigns from Vancouver, BC and/or Bécancour, QC to the MLA in Years -3 and -2;
- Storage of freight at the MLA from the end of the sealift period until the WIR is constructed and freight can be hauled to site;
- Construction of the WIR from the MLA to Goose Site starting in December of Years -3 and -2; and
- Haulage of freight by the WIR from the MLA to Goose Site during the first quarter of Years -2 and -1.

25.5.2 Marine Laydown Area

The construction of the Goose Site is dependent on the successful construction and operations at MLA because most of the construction equipment and material will be shipped through this site.

The construction of the majority of MLA will occur over a 1-year period (mid-Year -3 to mid-Year -2). To meet this timeline, the following will be required:

Two sealifts in Year -3 to deliver the following:

Sealift #1 – July

- Earthworks equipment (partial mining fleet);
- Temporary fuel storage (8 x 90,000 L);
- Permanent camp;
- Power plant;

- Site Services Equipment; and
- Year -3 Fuel Requirements (partial).

Sealift #2 – October

- MLA – Fuel Year -3 / -2 Requirements
- MLA - Fuel Tank Material (3 x 10 ML)
- MLA - Truck Shop
- MLA - Lighter Barge Terminal
- MLA - Winter Road Equipment
- MLA - Construction Equipment
- Goose - Infrastructure Cement/Steel
- Goose - Camp
- Goose - Fuel Tank Material (1 x 10 ML)
- Goose - Temporary power
- Goose - Site services & construction equipment
- A major sealift in Year -2 to deliver the following:
- MLA - Fuel Tank Material (1 x 10 ML)
- Goose - Process & crushing equipment
- Goose - Power Plant
- Goose - Fuel Tank Material (2 x 10 ML)
- Goose - Open pit mining equipment (remaining fleet)

Construction of the MLA facilities will commence in Year -3, including:

- Lighter barge terminal;
- Fuel Tank Material (4 x 10 ML);
- Truck shop;
- Camp and associated infrastructure;
- Diesel generators; and
- Site laydown areas.

25.6 Fuel Requirements

Fuel requirements for the Goose Site during construction are shown in Table 25.10.

Table 25.10: Goose Pre-production Fuel Requirements (thousand litres)

Activity	Year -3	Year -2	Year -1	Total Pre-production Fuel
Infrastructure & Construction Power	50	1,203	1,203	2,456
Open Pit Mining - Operations	0	0	3,191	3,191
Site Support - Goose	0	295	455	750
Winter Road Construction	162	235	235	632
Winter Road Maintenance	0	62	46	108
Winter Road Freight Haul	0	20	14	34
Construction Equipment	0	748	684	1,432
Earthworks	0	1,258	369	1,627
Total Fuel Requirements	**212**	**3,821**	**6,197**	**10,230**

Source: JDS 2015

Fuel required for Year -3 and Year -2 activities will be flown to site using a Boeing 737 jet during winter months and an ice strip on Goose Lake or a De Havilland DHC-5D Buffalo and the existing Goose airstrip. Fuel required for Year -1 will be transported to the MLA using an ocean tanker and trucked to the Goose Site on the WIR.

Fuel requirements for the MLA Site construction are shown in Table 25.11.

Table 25.11: MLA Pre-production Fuel Requirements (thousand litres)

Activity	Year -3	Year -2	Year -1	Total Pre-production Fuel
Infrastructure	336	740	740	1,816
Site Support - MLA	73	47	43	163
Sealift Support	17	21	21	59
Winter Road Construction	162	235	235	632
Winter Road Maintenance	0	62	46	108
Winter Road Freight Haul	0	125	121	246
Construction Equipment	237	116	0	353
Earthworks	534	0	0	534
Total Fuel Requirements	**1,359**	**1,346**	**1,206**	**3,911**

Source: JDS 2015

Fuel will be transported to the MLA Site via ice-class ocean tankers. In the first year it will be stored in the temporary fuel tanks, and in subsequent years it will be stored in the permanent fuel storage tanks.

25.7 Other Temporary Equipment and Facilities

Other temporary equipment and facilities that will be used to construct both sites are shown in Table 25.12.

Table 25.12: Other Temporary Equipment and Facilities

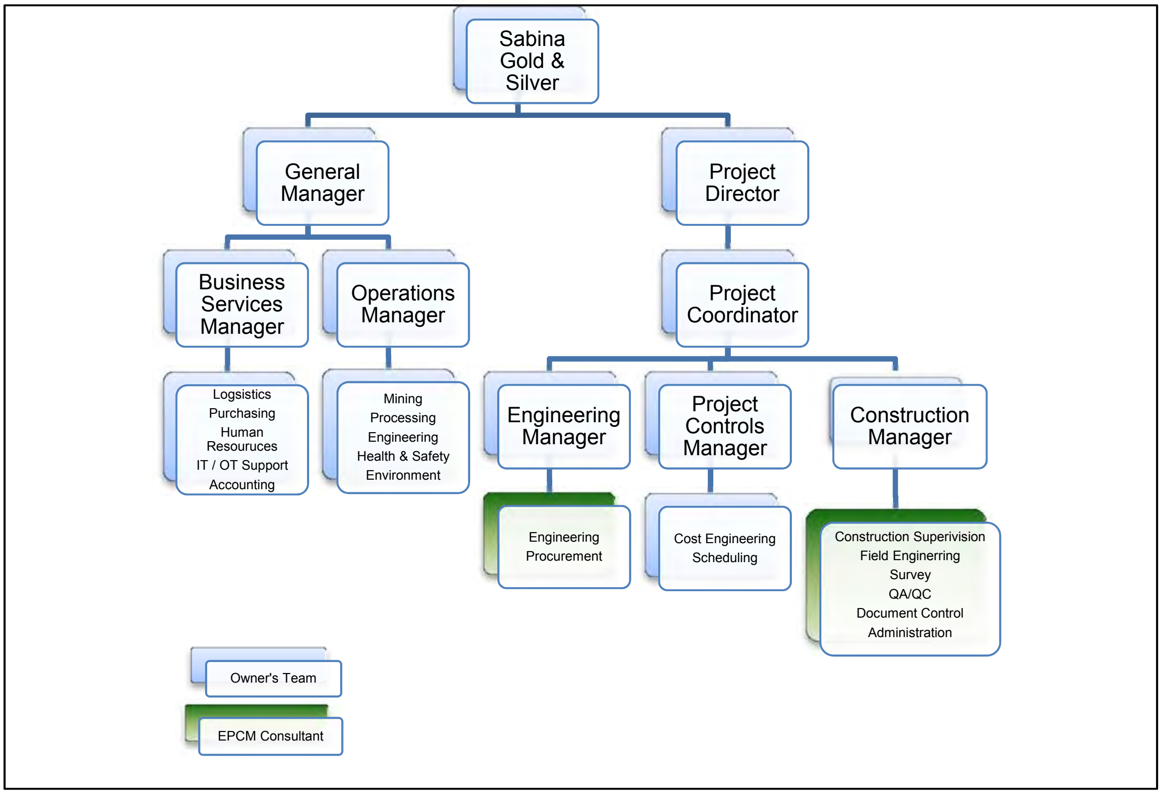
Equipment Description	Quantity
5 T Fork Lift Zoom-Boom - Terex GTH-5519	2
65 FT Man-Lift - Genie S-65	3
85 FT Man-Lift - Genie S-65	3
125 FT Man-Lift Straight Boom - Genie S-125	2
125 FT Man-Lift Articulating Boom - Genie ZX-135/70	2
Mobile Crane - RT150 - Grove RT9150E	1
Mobile Crane - RT90 - Grove RT890E	2
Crawler Crane - 165T - Manitowoc MLC 165	1
Mobile Concrete Batch Plant	1
Panel Lifter	2
Concrete Mixer Truck	2
Concrete Pump Truck	1
Tool Carrier - Cat 966K (c/w Attachments)	1
1 T Diesel Crew Cab Pick-up - Ford F350	4
Winch Tractor with 60T Winch	1
Tri-Axle Single Drop, Scissor Neck Trailer	1
Mobile Concrete Batch Plant	1
Portable Diesel Light Plants	6
Portable Diesel Heaters	8
Scissor Lift - Genie GS4069RT	1
Portable Office / Wash Module	2

Source: JDS 2015

25.8 Engineering, Procurement, and Construction Management

Project management will be an integrated team comprising the Owner's project management personnel and the project management consultant. The project management organizational chart is shown in Figure 25.2.

Figure 25.2: Project Management Organization Chart



Source: JDS 2015

The project management team will oversee the detailed EPCM activities for the project. The project management team will also coordinate the work of the engineering consultants and other specialized consultants as required.

The project management team will be responsible for all project activities from detailed design through to commissioning and turnover to operations. The project management team will be available to backstop the operations teams with key supervision and management assistance when the operations personnel assume control of project components as they are completed.

25.8.1 Engineering Team

The engineering team will design each facility and component of the plant and infrastructure using estimated costs, procurement lead time, and installation requirements. Each design package will then be turned over to the procurement team following a review by the project management team. The engineering team will also monitor construction and commissioning to ensure design compliance. This may include representation at the vendors' factories, if required. The engineering team's tasks will include the following:

- Create detailed designs;
- Estimate costs;
- Produce drawings;
- Review tenders for technical and design compliance;
- Conduct field engineering; and
- Conduct QA/QC.

25.8.2 Procurement Team

In general, the procurement team will oversee the selection and tendering of all tagged equipment, bulk materials, and commodities as a function of managing the engineering consultant. Tagged equipment is defined as uniquely designed and engineered equipment and assemblies required for the project as documented in the project equipment lists. Generally, bulk materials are not specifically engineered items and are not identified on the project equipment list. All bulk materials for the project will be purchased, tracked, and referenced to applicable specifications and standards.

The procurement team will use the design packages provided by engineering to obtain competitive tenders. The procurement team’s tasks will include the following:

- Prepare tender and contract documents;
- Pre-qualify vendors;
- Review tender submissions for compliance and commercial terms;
- Award and execute contracts;
- Coordinate logistics with Owner’s team;
- Pay contractors and vendors;
- Resolve any contractor claims and disputes; and
- Complete and close-out contract.

The purchasing schedules for the majority of the procurement packages will be driven by the sealift schedules in Year -3 and Year -2.

25.8.3 Owner's Team

The Owner's team will be responsible for the following areas:

- Project controls;
- Schedule development, review, and tracking;
- Logistics;
- Warehousing;
- Contract administration;
- Purchasing;
- Cost tracking and forecasting;
- Scheduling and cost compliance; and
- Environmental procedures and compliance.

25.8.4 Construction Management Team

The construction management team's tasks will include the following:

- Review construction designs and contract documents;
- Ensure construction progresses on schedule and on budget;
- Assist with and review schedule development;
- Coordinate and manage site operations;
- Oversee site health and safety;
- Manage and maintain camp;
- Oversee site-wide labour relations;
- Oversee site services;
- Manage document control;
- Review contractor invoices and forward to procurement;
- Certify contract completion and close-out contract;
- Provide input to contractor claims and dispute resolutions;
- Oversee survey control and as-builts;
- Conduct QA/QC; and
- Oversee commissioning.

25.9 Construction/Pre-Production Logistics

The pre-production logistics plan relies primarily on two sealift and two WIR campaigns for delivery of equipment and materials. A fixed-wing aircraft campaign will support construction activities with delivery of fuel, select bulk materials, and passengers through the duration of construction.

The pre-production logistics plan addresses the requirements for sealift freight, airfreight, and freight to port costs. Personnel transport and catering freight to support the project schedule is also provided. The available source data and methods used to determine the feasible capacity of each delivery method are discussed within the logistics plan.

25.9.1 Load List and Methods of Delivery

There are an estimated 24.6 kt of freight required to support construction and pre-production activities for the Project. A comprehensive load list was prepared to analyse the weights and volumes of all equipment and materials required on site. Of the 24.6 kt of freight, there are 1.6 kt of material and equipment along with 0.5 kt of catering freight that will be transported to the sites by air. The remaining 22.5 kt of freight will be transported via the annual sealift. The inbound freight breakdown by area is shown Table 25.13.

Table 25.13: Pre-Production Freight Requirements (tonnes)

Freight Classification	Year -3		Year -2		Year -1		Total
	MLA	Goose	MLA	Goose	MLA	Goose	
Sealift Freight	3,840	0	300	10,880	150	7,330	22,500
Airfreight – Construction	350	170	0	1,060	0	50	1,630
Airfreight – Catering	40	10	30	120	30	220	450
Total	**4,230**	**180**	**330**	**12,060**	**180**	**7,600**	**24,580**

Source: JDS 2015

In addition to the pre-production freight requirements listed above, a total of 14.1 ML of fuel is required for the pre-production period. A comprehensive fuel demand list was prepared to analyse the volume of fuel required by area. Of the 14.1 ML of fuel, 4.0 ML will be airfreighted to Goose Site. The remaining 10.1 ML of fuel will be transported by ocean-going vessels during the annual sealift. The inbound fuel breakdown by area is shown in Table 25.14.

Table 25.14: Pre-Production Fuel Requirements (thousand litres)

Mode	Year -3		Year -2		Year -1		Total
	MLA	Goose	MLA	Goose	MLA	Goose	
Sealift Fuel	1,359	0	1,346	0	1,206	6,197	10,108
Airfreight Fuel	0	212	0	3,821	0	0	4,033
Total	**1,359**	**212**	**1,346**	**3,821**	**1,206**	**6,197**	**14,141**

Source: JDS 2015

25.9.2 Air Freight

A fixed-wing aircraft campaign will support construction activities with delivery of fuel, select bulk materials, and passengers through the duration of construction.

25.9.2.1 Fuel

An estimated 4 ML of fuel are required to be delivered by air in Year -3 and Year -2 to the Goose Site before sufficient infrastructure and conditions will permit bulk fuel transport by ocean-going vessels. During the winter months, a Boeing 737 jet will be used to transport fuel using an ice strip constructed on Goose Lake. For the remainder of the year a De Havilland DHC-5D Buffalo and the existing Goose airstrip will be utilized. A total of 58 flights for the Boeing 737 and 161 flights with the Buffalo are anticipated.

25.9.2.2 Freight

An estimated 1,300 t of dry freight are required to be delivered by air during construction to the Goose Site. During the winter months of Year -3, Hercules aircraft will be used to transport freight utilizing an ice strip constructed on Goose Lake. For the remainder of the construction period a De Havilland DHC-5D Buffalo and the existing Goose airstrip will be utilized. Goose freight requirements for each type of aircraft are summarized by year in Table 25.15.

Table 25.15: Goose Fixed-Wing Freight Requirements

	Hercules		Buffalo		Total	
	Tonnes	**Flights**	**Tonnes**	**Flights**	**Tonnes**	**Flights**
Year -3	159	10	8	1	167	11
Year -2	0	0	877	108	877	108
Year -1	0	0	264	33	264	33
Total	**159**	**10**	**1,149**	**142**	**1,308**	**152**

Source: JDS 2015

Annual MLA freight requirements for each type of aircraft are summarized in Table 25.16.

Table 25.16: MLA Fixed-Wing Freight Requirements

	Hercules		Twin Otter		Total	
	Tonnes	**Flights**	**Tonnes**	**Flights**	**Tonnes**	**Flights**
Year -3	**353**	**21**	**37**	**34**	**390**	**55**
Year -2	**0**	**0**	**32**	**30**	**32**	**30**
Year -1	**0**	**0**	**28**	**26**	**28**	**26**
Total	**353**	**21**	**97**	**90**	**450**	**111**

Source: JDS 2015

25.9.2.3 Fixed-Wing Passenger Transportation

Passenger flights to the Goose Site will originate in Yellowknife and use a De Havilland Dash 7. Passenger flights to the MLA will originate in Yellowknife and use a Twin Otter. An average capacity factor of 90% has been assumed for load planning. Annual passenger delivery requirements are summarized in Table 25.17.

Table 25.17: Fixed-Wing Passenger Requirements

Site	Goose Site		MLA Site		Total All Sites	
Aircraft	Dash 7		Twin Otter		All Aircraft	
	Passengers	Flights	Passengers	Flights	Passengers	Flights
Year -3	248	7	568	64	816	71
Year -2	2,219	62	586	66	2,805	128
Year -1	4,022	112	514	58	4,536	170
Total	**6,489**	**181**	**1,668**	**188**	**8,157**	**369**

Source: JDS 2015

25.9.2.3 Sealift Freight

The majority of pre-production equipment and materials will be transported to the Goose and MLA sites via ocean-going vessels during the annual sealift period. Dry freight such as construction equipment and materials will be shipped by either ocean-going barges or ocean vessels. Fuel will be shipped by ocean tankers. The modes of sealift transport are as described previously for operations supply. Estimated quantities of pre-production sealift freight can be found in Table 25.13 above.

26 Interpretations and Conclusions

The economic results of this Initial Project Feasibility Study (3KFS) demonstrate that the Project has positive economics and warrants development.

Standard industry practices, equipment and processes were used in this study. The authors of this report are not aware of any unusual or significant risks, or uncertainties that could affect the reliability or confidence in the Project based on the data and information made available.

26.1 Risk and Opportunity Management

Most mining projects are exposed to risks that might impact the economics of the Project to varying degrees. Most risks are external and largely beyond the control of the Project proponents. They can be difficult to anticipate and mitigate; although, in many instances, some reduction in risk might be achieved by regular reviews and interventions over the life of the Project. Certain opportunities that can enhance the Project economics might also be identified during further optimization in the detailed engineering phase of Project development.

External risks are, to a certain extent, beyond the control of the Project proponents and are much more difficult to anticipate and mitigate, although, in many instances, some risk reduction can be achieved. External risks are things such as the political situation in the Project region, metal prices, exchange rates and government legislation. These external risks are generally applicable to all mining projects.

Table 26.1 and Table 26.2 summarize the significant Project opportunities and risks, respectively, for the Back River Gold Project, including the potential impacts, and possible mitigations.

The Project risk assessment was conducted in a series of workshops involving Sabina personnel and consultant leads representing their respective Project disciplines. Emphasis was placed on the identification of currently significant Project risks and possible high-level mitigation efforts. A formal review of the likelihood and consequence ratings and pre- and post-mitigation rankings was not conducted: this will be performed during the detailed engineering phase of development.

The remote location of the Back River mine site imposes an additional level of risk to the Project due to the large volume of equipment, fuel and materials that need to be transported from Canadian western and eastern ports to the mine site via Bathurst Inlet. Superimposed on methods of transportation are seasonal constraints that require ice-free sea routes for a short window in summer and an ice road in winter. These transportation methodologies, routes and logistics have been carefully considered in the 3KFS. However, seasonal weather variations are rarely predictable and appropriate mitigations will be required in the event of cooler summers or milder winters. Procurement timelines aligned to logistical requirements have been developed in the 3KFS.

Permitting processes are known to be long and outcomes are sometimes unpredictable. The uncertainty of permitting timelines could have an impact on the Project development schedule.

However, the permitting process for the Project is well underway and major impacts on the Project schedule are not expected.

The WIR will be used for the first time in Year -2. Although the volume of material to be transported on this road during the first year is lower than subsequent years, the uncertainties of a new, unproven ice road will exist.

Extreme winter temperatures, between -25°C to -40°C, will impact personnel and equipment productivities during construction and operations.

The typical risks associated with open pit and underground mining related to geotechnical conditions, equipment availability and productivity, and personnel productivity are generally similar to those expected at similar operations.

Although measures to mitigate many of these issues have been identified and applied in the FS, risk identification and review of mitigations will continue to be a priority during Project development, construction and operations.

Table 26.1: Significant Project Opportunities

Area	Description	Aspect	Impact		Opportunity Steps	Ref. No.
			Description	Opportunity		
PROJECT CONCEPT	Alternate Scale of Project (See review of 6KFS in section 26.2)	Optimization of resource utilization and project value	A higher production rate and Mineral Reserve base provides higher project valuation.	An improvement in capital markets may result in a preference to finance a higher CAPEX project with a higher NPV.	Revert to the 6KFS.	1
GEOLOGY AND EXPLORATION	Geological Model	Accuracy of geological model	Model does not represent the deposit location and grade distribution accurately.	Mineral Resources and Reserves are understated.	Infill drilling and in-pit geological mapping as mining advances. Build flexibility into plan.	2
MINERAL PROCESSING AND METALLURGICAL TEST WORK	Metallurgy	Gold recovery	Gold recovery process is not fully optimized for the mineralogy and variations within it.	Increased recoveries improve Project economics.	Additional sampling and metallurgical test work; further review of tailings mineralogy and test work done to date.	3
MINERAL RESOURCE ESTIMATE	Mineral Resources	Resource expansion	Significant Inferred material is still available for possible conversion, all deposits continue at depth and some extend laterally, and there are indications of increasing grade and thickness at depth.	Additional Mineral Resources that can be converted to Mineral Reserves improve Project economics. Significant exploration potential within large land package with multiple greenfield targets.	Infill and exploration drilling.	4
MINERAL RESERVE ESTIMATE	Mineral Reserves	George resources	Historical Reserves identified in 6KFS at George Site were excluded from this FS in order to simplify the mine plan but may still be economic.	Additional Mineral Reserves may improve Project economics including extending the Project mine life.	Continue investigating opportunities to reduce operating costs especially related to hauling or processing the ore on site at George.	5
UNDERGROUND MINE	Geotechnical Considerations	Ground conditions	Distribution of Middle Mudstone, faults and variation in rock mass quality different than modelled with impact on planned stope dimensions.	Reduced ground support requirements and pillar sizes; increased ore recovery and production rate; reduced mining costs.	Ongoing geotechnical monitoring and mapping; ongoing stope design and rock support review.	8
	Mining Method	Optimized production	Further evaluation of mining methods available for the Umwelt deposit.	Alternate mining methods in certain (or potential all) areas of Umwelt underground; increased ore recovery and production rate; reduced mining costs.	Initiate further underground mine engineering investigations.	9
RECOVERY METHOD	Ore Segregation	Coarse beneficiation	Competent ore (relatively coarse after crushing, for example) tends to have less contained gold.	Screening and hence, segregating harder ore can result in increased plant throughput and reduced processing energy consumption. This harder and lower grade ore can be stockpiled for later processing.	Conduct testwork to evaluate the relationship between size, hardness and contained gold. Apply the results to cut-off grades, equipment selections, plant throughput, operating cost estimates, operating approach to George, etc.	14
	Grinding	Mill throughput	Coarser grind size (greater than 50 microns) is adequate.	Increase grind size for reduced operating costs (power).	Ongoing sampling and metallurgical test work; further review of test work done to date.	15
	Flotation	Mill Throughput	Considerably coarser grind size is adequate with minimal recovery losses.	Increased grind size for reduced operating costs (power).	Redo the feasibility metallurgical variability test program with flotation instead of whole-ore-leach.	16
WASTE ROCK MANAGEMENT	Geochemical Considerations	NPAG/PAG ratio	Improper segregation or identification of NPAG and PAG waste.	More NPAG than estimated allows greater scheduling flexibility.	Ongoing monitoring and review of WRSA designs, especially with regards to regulatory requirement.	21
FINANCIAL	Payable Metal	Silver	Insufficient sampling and assaying to create an indicated silver resource on the deposits.	Additional payable metal at no additional capital or operating cost.	Although a silver resource could be created, the immediate cost benefit doesn't warrant this. Silver will be accounted for with the production of payable metal.	26
	Capital Costs	Used equipment	Estimate is based on all new equipment for the process, main components of the mobile fleet, and accommodations camp. Capital costs would be less if good used equipment was sourced.	Used mobile equipment including CAT 777 trucks, cranes, etc. are widely available. Similarly, processing equipment of suitable specifications may be available on the used market.	Source and make commitments to secure used equipment matching the design basis.	27
	Capital Costs	Negotiated cost reductions	Quoted capital costs do not account for cost reductions expected through negotiations with Vendors when bundled or through volume discounts.	Opportunity to reduce OPEX, CAPEX and/or establish improved commercial terms and payment schedules.	Negotiations with vendors during procurement.	28
	Income Tax	Reduce taxes paid	Additional tax pools and deductions may be available.	In particular, for the NTNMR, Sabina may be able to deduct certain historical qualifying expenditures on the Project.	Analysis of historical expenditures. Development of a tax planning strategy.	30
	Royalties	Third-party NSR royalties	Reduce third-party Royalties.	Opportunity to buy out third-party NSR royalty holders.	Analysis of third-party royalties and agreement terms and discussions with holders.	31

Table 26.2: Significant Project Risks

Area	Description	Aspect	Impact		Risk Mitigation	Ref. No.
			Description	Risk		
GEOLOGY AND EXPLORATION	Geological Model	Accuracy of geological model	Model does not represent the deposit location and grade distribution accurately.	Faults and mineralized zones might not be accurately identified; significant change in mine design required. Mineral Resources and Reserves are overstated. Project economics reduced.	Infill drilling and in-pit geological mapping as mining advances.	2
					Build flexibility into plan.	
MINERAL PROCESSING AND METALLURGICAL TEST WORK	Metallurgy	Gold recovery	Gold recovery process is not optimized for the mineralogy and variations within it.	Reduced recoveries reduce Project economics. Redesign of the gold recovery process plant might lead to additional capital expenditure.	Additional sampling and metallurgical test work; further review of test work done to date.	3
OPEN PIT MINE	Geotechnical Considerations	Pit wall stability	Wall failure and overtopping of safety berm is possible.	Loss of life, lost production, increased ground support requirements, equipment loss.	Ongoing geotechnical monitoring and mapping to optimize pit wall design. Reliance on ROM stockpiles as buffer to production losses; use of backup equipment (from construction).	6
	Mobile Equipment	Key production equipment	Loss of key production equipment due to fire, major mechanical failure or accidents.	Reduced productivity; increased cost to repair/replace.	Install fire detection and suppression equipment; equipment health monitoring and preventative maintenance; adequate spares inventory.	7
UNDERGROUND MINE	Geotechnical Considerations	Ground conditions	Distribution of Middle Mudstone, faults and variation in rock mass quality different than modelled with impact on planned stope dimensions.	Loss of life in the event of failure; increased ground support requirements and pillar sizes; reduced ore recoveries and production rate; increased mining costs; cut-off of ventilation.	Ongoing geotechnical monitoring and mapping; ongoing stope design and rock support review.	8
		Ground failure	Catastrophic rock failure due to inadequate support for actual ground conditions.	Loss of production and life.	Ongoing geotechnical monitoring and mapping; ongoing stope design and rock support review.	10
		Ingress of tailings	Failure in crown pillar results in tailings stored in open pit to flow into underground workings.	Loss of life, equipment, and recoverable ore (if underground mining lags open pit).	Ongoing geotechnical monitoring and mapping; ongoing review of open pit and underground scheduling.	11
	Mobile Equipment	Key production equipment	Loss of key production equipment due to fire, major mechanical failure or accidents.	Reduced productivity; increased cost to repair/replace.	Install fire detection and suppression equipment; equipment health monitoring and preventative maintenance; adequate spares inventory; operator training.	12
MINING EXECUTION	Dilution	Lowering of head grade	A reduction of head grade as a result of unanticipated dilution would have a direct negative impact on project economics.	Loss of revenue and increase operating cost per ounce.	An appropriate grade control plan with practical execution.	13
RECOVERY METHODS	Grinding	Mill throughput	Final ore size of 50 microns not achieved.	Reduction in gold recovery.	Ongoing sampling and metallurgical test work; further review of test work done to date.	15
	Gold Recovery	Gold recovery	Gold recovery lower than test work indicates.	Reduced revenue and negative impact on Project economics.	Ongoing sampling and metallurgical test work; further review of test work done to date.	17

Area	Description	Aspect	Impact		Risk Mitigation	Ref. No.
			Description	Risk		
ON-SITE INFRASTRUCTURE	Power Supply	Loss of major electrical equipment	Fire, explosion, impact, lightning strike results in significant loss in power production.	Loss of production; cost to repair or replace.	Emergency standby generators, separate circuits, lightning protection, blast walls separating transformers, bollards and other barriers, monitoring and SOP enforcement.	18
	Health, Safety, Environment	Extreme weather	Temporarily halt in productions or access to the site due to weather.	Risk to personnel, restricted evacuation and medical response ability, satellite communications affected. Loss of production.	On-site medical expertise, remote diagnostics via off-site doctor. Use ROM stockpile capacity as a buffer to reduced mine production.	19
OFF-SITE INFRASTRUCTURE	Winter Ice Road	Ice road construction	Lack of water and / or snow to construct the road due to weather conditions.	Materials and equipment cannot be transported to site as planned or required. Results in loss of production and / or increase in cost of air freighting.	Air transport (ice strip at MLA), prioritize loads, CAT train, licence for additional water demand.	20
WASTE ROCK MANAGEMENT	Geochemical Considerations	NPAG/PAG ratio	Improper segregation or identification of NPAG and PAG waste.	Less NPAG than estimated results in a reduction in waste dumping flexibility and possibly increased re-handling costs.	Ongoing monitoring and review of WRSA designs, especially with regards to regulatory requirement.	21
ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT	Permitting	Regulatory process	Delay or failure to obtain necessary permits.		Robust permitting process that addresses all Project requirements within the regulatory timelines.	22
				Delay in commencement of mining operations reduces Project economics and might impact Project financing.	Maximize synergy between engineering, environmental assessment, and permitting processes to reduce costs of mitigation programs, improve efficiency, and improve timing for Project management.	
MANPOWER	Manpower	Workforce complement and skills	Inability to attract or retain personnel with appropriate skills.	Loss in production, increased cost to recruitment and salary bill.	Improve attractiveness of employment by maintaining an attractive and positive working environment.	23
FREIGHT/	Freight and Logistics	Access by marine transport during construction	Reduced or no open-water season.	Delayed construction with increase in construction costs, reduced Project valuation.	Air freight materials, ice road from south and / or CAT train, ice-breaking to access MLA.	24
LOGISTICS	Freight and Logistics	Access by marine transport during operations	Reduced or no open-water season.	Loss of production. Increased operating costs.	Air freight materials, ice road from south and / or CAT train, ice-breaking to access MLA.	25
FINANCIAL	Financing	Securing capital	Failure to secure funding could slow or stop Project development.	Stop Project development.	Develop financing plan and analysis of reduced CAPEX option to reduce financing risk.	29
	Income Tax	Taxation	Taxation regulations may change or taxation details may be different than estimated.	Increase tax costs.	Analysis of historical capital expenditure pool and the development of a tax planning strategy.	30
	Capital and Operating Costs	Lack of Fuel tax rebate	Nunavut Petroleum Tax rebates of $0.091 for motive fuel and $0.031 for non-motive have been accounted for in estimates.	Although no issues are expected in securing the rebates, negotiations have not been held yet.	Complete the negotiations for the rebates.	32

Source: JDS 2015

26.2 6,000 tpd CAPEX Opportunity

26.2.1 Introduction

The Initial Project Feasibility Study (3KFS) summarized in this technical report offers a Project with a lower execution risk than the 6,000 tpd Project (6KFS). The latter was detailed in the "Technical Report and Feasibility Study for the Back River Gold Property, Nunavut" dated June 22, 2015, with an effective date of May 20, 2015. The risk has been reduced by simplifying the mine plan and significantly reducing the CAPEX. The larger Project is still attractive for its strong economic results. The 3,000 tpd and 6,000 tpd Project scenarios provide Sabina management with viable alternatives for project development options. The details of the 6,000 tpd Feasibility Study are summarized below.

26.2.2 Mine Planning

The Mineral Reserves for the 6KFS were developed by examining each deposit at both the Goose and George sites to determine the optimum practical mining method. Cut-off grades were then determined based on appropriate mine design criteria and the adopted mining method. The 6KFS used lower cut-off grades than the 3KFS due to lower unit operating costs associated with higher production rates. The use of the lower cut-off grades resulted in more deposits being economic. A total of eight open pits and seven underground mines were viable in the 6KFS compared to three open pits and one underground mine in the 3KFS.

The estimated Proven and Probable Mineral Reserves in the 6KFS total 19.8 Mt at 5.70 g/t Au, containing 3.63 Moz Au. These Reserves are now deemed historical and are superseded by the reserves in this report.

Four mining methods were chosen: shovel-and-truck open pit mining, underground mining using post pillar cut-and-fill (PPCF), drift and fill (DF), and longitudinal open stoping (LOS). Conventional shovel-and-truck open pits combined with underground mines are projected to provide the process plant feed at a nominal rate of 6,000 t/d or 2.2 (Mt/a) for a period of 10 years. Annual mine production of ore and waste is profiled to peak at 13.6 Mt/a from the open pits, with a LOM waste to ore stripping ratio of 7.2:1. Ore production from underground mining will peak at 1.7 Mt/a and will supplement the feed from the open pits. In order to optimize the Project cash flow, the run of mine ore is planned to be segregated into high, medium, and low-grade stockpiles located adjacent to the processing plant.

The tonnes, grade, and ounces of the 6KFS compared to the 3KFS are shown in Table 26.3. The total mineralized material that is above the cut-off grade in the 6KFS is about 60% more than that in the 3KFS, the combined grade is about 0.6 g/t lower and the mine life is about two years shorter. Recovered gold is about 1.1 Moz more than in the 3KFS.

Table 26.3: Mine Plan

Description	Unit	2015 6KFS	2015 3KFS
Total Mineralized Material Mined	Mt	19.8	12.4
Total Waste Mined	Mt	77.2	93.1
Total Mined	Mt	96.9	105.5
Combined Au Grade	g/t	5.7	6.3
Production life	years	9.6	11.8
Total Contained Au	Moz	3.4	2.3

Source: JDS 2015

26.2.3 Mineral Processing

The basic mineral processing flowsheet is essentially the same between the 6KFS and the 3KFS but with a mill throughput rate of 6,000 t/d instead of 3,000 t/d. The most significant differences are the use of 2-stage crushing and a 10-hour live fine ore stockpile compared to the 3KFS which has 3-stage crushing and a 24-hour live fine ore stockpile. The process plant for 6KFS is intended to be largely “stick-built” using conventional manufacturing and supply methods whereas a more modular design approach has been taken in the 3KFS.

26.2.4 Infrastructure and Logistics

With mining of George deposits being economically feasible in the 6KFS, the Project has three discrete infrastructure locations. Proportional for the larger Project, the 6KFS infrastructure, including camp, water and waste management facilities, and ore stockpiles, is appropriately larger. Generally, the Goose and MLA sites have larger footprints than for the 3KFS case.

The logistics concepts for the transportation of freight, consumables, and personnel are essentially the same for the 6KFS as the 3KFS. Transportation includes the annual summer sealift of freight to the MLA and the use of a WIR. However, the WIR network is extended to connect George to both MLA and Goose sites. The estimated cost for all materials transport was greater for the 6KFS in total annual dollars and the cost was also slightly more on a unit rate (per tonne processed) basis.

26.2.5 CAPEX and OPEX Estimates

CAPEX and OPEX estimates were built up from supplier quotes, engineering first principles, applying directly-related Project experience, and the use of general industry factors.

The CAPEX estimate, shown in Table 26.4, indicates that a 6,000 t/d mine and mill could be built for about $695M. This compares with $415M for the 3KFS.

Table 26.4: CAPEX Estimates for 6KFS

Capital Cost	Initial M$	Sustaining M$	LOM M$
Mining	105.9	292.4	398.3
Processing	111.7	0	111.7
On-Site Infrastructure	172.3	54.7	227.0
Off-Site Infrastructure	73.4	35.3	108.7
Water & Waste Management	19.7	31.9	51.6
Owner's Costs	30.3	0	30.3
Indirect Costs	125.6	3.9	129.5
Reclamation	0.0	89.0	89.0
Subtotal	638.9	507.2	1,146.2
Contingency	55.8	22.1	77.9
Total Capital Costs	694.7	529.4	1,224.1

Source: JDS 2015

The OPEX estimates for the 6KFS are shown in Table 26.5 and are approximately $18/t milled lower than in the 3KFS.

Table 26.5: OPEX Estimates for 6KFS

Operating Cost†	Average $M/yr	LOM $M	$/t processed
Open Pit & Underground Mining*	74.6	714.0	36.07
Processing	53.8	515.3	26.04
Infrastructure and Site Services	27.0	258.9	13.08
Freight Transportation	9.3	88.7	4.48
George Ore Haul °	6.2	59.3	3.00
General & Administrative	28.2	269.5	13.62
Total Operating Costs	199.1	1,905.6	96.28

(†): Operating Costs include the working capital during the pre-production period and excludes pre-stripping costs.
(*): Average LOM Open Pit Mining cost amounts to $3.95/t mined at a 7.2 strip ratio; average LOM underground Mining cost amounts to $48.84/t mined
(°): George Ore Haul cost is for 3.06 Mt over the LOM
Source: JDS 2015

26.2.6 Economic Results

A standalone engineering economic model was developed to estimate the value and investment return of the 6KFS, as for the 3KFS. Pre-tax estimates of Project values were prepared for comparative purposes and after-tax estimates were developed to better indicate the true investment value. Sensitivity analyses reflecting variations in metal prices, grades, operating costs, and capital costs were performed to determine their relative importance as Project value drivers.

The main economic assumptions used in the 6KFS (metal price, exchange rate, payables, etc.) are different than those used in the 3KFS to reflect the prevailing economic environment (Q1 2015 versus Q3 2015, respectively). The significant changes are shown in Table 26.6.

Comparing the post-tax economics to the 3KFS, the 6KFS had a nearly 3% lower IRR, a payback period shorter by 0.7 years, and an approximately 10% increase in NPV.

Table 26.6: Comparison of Financial Results for 6KFS and 3KFS

	Unit	2015 6KFS	2015 3KFS
Gold Production	Moz	3.37	2.32
Mine Life	years	9.6	11.8
Initial Capital Cost	$M	695	415
Sustaining Capital Cost	$M	437	185
Closure Cost	$M	89	64
Operating Cost	C$/t	96.28	114.58
Cash Costs	US$/oz	583	534
All-In Cash Costs	US$/oz	899	763
Pre-Tax IRR	%	26.4	28.2
Pre-Tax NPV 5%	$M	826	699
Post-Tax IRR	%	21.3	24.2
Post-Tax NPV 5%	$M	526	480
Payback	years	2.2	2.9

Source: JDS 2015

If financing the larger Project is not deemed to be an impediment to developing Back River, the 6KFS represents a viable option that provides good economics and the best use of the available Mineral Resources at the Goose and George sites.

27 Recommendations

The Initial Project Feasibility Study (3KFS) offers strong economic results and has good potential to be financed under current market conditions. It is recommended that the project be advanced to construction through the normal process of permit acquisition, financing detailed engineering and construction. Costs for engineering and construction are included in the capital cost of this study.

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29 Units of Measure, Abbreviations and Acronyms

actual cubic feet per minute	Acfm
ampere	A
annum (year)	a
bed volumes per hour	BV/h
billion	B
billion tonnes	Bt
billion years ago	bya
billions of years	Ga
British thermal unit	BTU
centimetre	cm
centipoise	cP
cubic centimetre	cm^3
cubic feet per minute	cfm
cubic feet per second	ft^3/s
cubic foot	ft^3
cubic inch	in^3
cubic metre	m^3
cubic metres per hour	m^3/h
cubic metres per second	m^3/s
day	d
days per week	d/wk
days per year (annum)	d/a
dead weight tonnes	DWT
decibel adjusted	dBa
decibel	dB
degree	°
degrees Celsius	°C
diameter	ø
dollar (American)	US$
dollar (Canadian)	C$
dry metric ton	dmt
foot	ft
gallon (US)	gal
gallons per minute (US)	gpm
Gigajoule	GJ
Gigapascal	GPa
Gigawatt	GW
gram	g
grams per litre	g/L
grams per tonne	g/t
hectare (10,000 m^2)	ha
hertz	Hz
horsepower	hp
hour	h
hours per day	h/d

hours per week	h/wk
hours per year	h/a
hydraulic conductivity	K
inch	in
kilo (thousand)	k
kilogram	kg
kilograms per cubic metre	kg/m^3
kilograms per hour	kg/h
kilograms per square metre	kg/m^2
kilometre	km
kilometres per hour	km/h
kilopascal	kPa
kilotonne	kt
kilovolt	kV
kilovolt-ampere	kVA
kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne	kWh/t
kilowatt hours per year	kWh/a
litre	L
litres per minute	L/min
litres per second	L/s
megabytes per second	Mb/s
megapascal	MPa
megavolt-ampere	MVA
megawatt	MW
metre	m
metres above mean sea level	mamsl
metres below ground surface	mbgs
metres below sea level	mbsl
metres per minute	m/min
metres per second	m/s
microns	µm
milligram	mg
milligrams per litre	mg/L
millilitre m	L
millimetre	mm
million	M
million bank cubic metres	Mbm^3
million bank cubic metres per annum	Mbm^3/a
million tonnes	Mt
minute (plane angle)	'
minute (time)	min
month	mo
Normal cubic metres per hour	Nm^3/h
parts per billion	ppb
parts per million	ppm
pascal	Pa
pounds per square inch	psi

revolutions per minute	rpm
second (plane angle)	"
second (time)	s
specific gravity	SG
square centimetre	cm^2
square foot	ft^2
square inch	in^2
square kilometre	km^2
square metre	m^2
standard cubic feet per minute	Scfm
tonne (1,000 kg) (metric ton)	t
tonnes per day	t/d
tonnes per hour	t/h
tonnes per year	t/a
tonnes seconds per hour metre cubed	ts/hm^3
Troy ounce	oz
volt	V
week	wk
weight/weight	w/w
wet metric ton	wmt

29.1 Abbreviations and Acronyms

Aboriginal Affairs and Northern Development Canada	AANDC
abrasion index	Ai
acid rock drainage	ARD
air terminal building	ATB
Aircraft Radio Control of Aerodrome Lighting	ARCAL
All-weather road	AWR
American Society of Testing and Materials	ASTM
AMC Mining Consultants (Canada) Ltd.	AMC
ammonium nitrate fuel oil	ANFO
Arctic Shipping Pollution Prevention Regulations	ASPPR
Arctic Waters Pollution Prevention Act	AWPPA
Area Navigation	RNAV
atomic absorption spectroscopy	AAS
Automated Weather Observation Station	AWOS
back scattered electron	BSE
Bench Face Angle	BFA
Bond Ball Mill work index	BMWi
Back River Joint Venture	BRJV
Canadian Dam Association	CDA
Canadian development expenses	CDE
Canadian exploration expenses	CEE
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Canadian Northern Economic Development Agency	CanNor
Canadian Pension Plan	CPP
capital cost allowance	CCA

capital expenditure	CAPEX
carbon dioxide	CO_2
carbon-in-leach	CIL
carbon-in-pulp	CIP
carbon monoxide	CO
Certified Reference Material	CRM
Coefficient of Variation	CV
Community Liaison Officer	CLO
copper sulphate	$CuSO_4$
crushing work index	CWi
cumulative net cash flow	CNCF
cut-off grade	COG
dead weight tonnage	DWT
diamond drill hole	DDH
Digital Elevation Model	DEM
direct shear	DS
discounted cash flow	DCF
discounted operational surplus	DOS
Distributed Control System	DCS
disturbance factor	D
Draft Environmental Impact Statement	DEIS
drift and fill	DF
Designated Inuit Organization	DIO
Dundee Precious Metals Inc.	DPM
exhaust air raise	EAR
earthquake design ground motion	EDGM
electrowinning	EW
Employment Insurance	EI
engineering, procurement, and construction management	EPCM
Environmental Management Plan	EMP
Environmental Management System	EMS
extended gravity recoverable gold	E-GRG
factor of safety	FoS
fresh air raise	FAR
field electrical centre	FEC
Final Environmental Impact Statement	FEIS
Footwall	FW
Geological Strength Index	GSI
Global Positioning System	GPS
gold	Au
Government of the Northwest Territories	GNWT
Government of Nunavut	GN
ground penetrating radar	PR
hanging wall	HW
harmful alteration, disruption or destruction	HADD
Hatch Ltd.	Hatch
high-density polyethylene	HDPE
high pressure grinding rolls	HPGR
high pressure index	BHPi

horizontal-loop electromagnetic	HLEM
Human Machine Interface	HMI
hydrated lime	$Ca(OH)_2$
hydrochloric acid	HCl
Indian and Northern Affairs Canada	INAC
induced polarization	IP
inductively coupled plasma	ICP
inductively coupled plasma mass spectrometry	ICP-MS
inflow design flood	IDF
input/output	I/O
Instrument Approach Procedures	IAP
Instrument Flight Rules	IFR
integrated communications cap lamp	ICCL
internal rate of return	IRR
International Electrotechnical Commission	IEC
International Standards Organization	ISO
internet protocol	IP
inter-ramp angle	IRA
Inuit Impact and Benefit Agreement	IIBA
Inuit-Owned Lands	IOL
inverse distance cubed	ID^3
inverse distance squared	ID^2
investment tax credit	ITC
JDS Energy & Mining Inc.	JDS
Kitikmeot Inuit Association	KIA
Knight Piésold Ltd.	KP
Lerchs-Grossman	LG
letter of credit	LOC
life of mine	LOM
Light Detection and Ranging	LiDAR
limit-equilibrium	L-E
load-haul-dump	LHD
local area network	LAN
longitudinal open stoping	LOS
Marine Laydown Area	MLA
maximum credible earthquake	MCE
maximum design earthquake	MDE
metal leaching	ML
Metal Mining Effluent Regulations	MMER
methyl isobutyl carbinol	MIBC
Mine Closure and Reclamation Plan	MCRP
Mineable Shape Optimizer	MSO
motor control centre	MCC
National Topographic System	NTS
Natural Resources Canada	NRCan
Nearest Neighbour	NN
net cash flow	NCF
net present value	NPV

net smelter return	NSR
neutralization potential/acid production	NP/AP
non-potentially acid generating	NPAG
Northwest Territories and Nunavut Mining Regulations	NTNMR
Nunavut Impact Review Board	NIRB
Nunavut Land Claim Agreement	NLCA
Nunavut Tunngavik Inc.	NTI
Nunavut Water Board	NWB
Obstacle Limitation Surfaces	OLS
Omni-Directional Approach Lighting System	ODALS
operating basis earthquake	OBE
operating expenditure	OPEX
ordinary kriging	OK
Overall Slope Angle	OSA
overburden	OVB
oversize	O/S
particle size analyser	PSA
peak ground acceleration	PGA
personal computer	PC
post pillar cut-and-fill	PPCF
potassium amyl xanthate	PAX
potential development area	PDA
potentially acid generating	PAG
Precision Approach Path Indicator	PAPI
Prefeasibility Study	PFS
preferred orientation	PO
Preliminary Economic Assessment	PEA
PricewaterhouseCoopers	PwC
programmable logic controller	PLC
Qualified Person	QP
quality assurance/quality control	QA/QC
radio-frequency identification	RFID
Rapid Mineral Scan	RMS
Regional Inuit Associations	RIA
Registered Retirement Savings Plans	RRSPs
relative paired difference	RPD
Rock Mass Rating (1989 version)	RMR89
Rock Quality Designation	RQD
SAG power index	SPI
scanning electron microscope	SEM
secondary	Sec
semi-autogenous grinding	SAG
short take-off and landing	STOL
Smee & Associates Consulting Ltd.	Smee
sodium cyanide	NaCN
sodium hydroxide	NaOH
sodium metabisulphite	SMBS
specific gravity	SG
SRK Consulting (Canada) Inc.	SRK

sulphur dioxide	SO_2
Tailings Storage Facility	TSF
Technical Decision Memorandum	TDM
three-dimensional	3D
Tibbit-Contwoyto Winter Road	TCWR
time-domain electromagnetic	TDEM
total dissolved solids	TDS
total suspended solids	TSS
two dimensional	2D
ultra-high frequency	UHF
undersize	U/S
unconfined compressive strength	UCS
uninterruptible power supply	UPS
variable frequency drive	VFD
visual flight rules	VFR
Voice over Internet Protocol	VoIP
Volcanic-turbidite series	VTS
waste rock storage area	WRSA
wide area network	WAN
weak acid dissoluble	WAD
weak acid dissoluble cyanide	CN_{WAD}
work breakdown structure	WBS
Workers Compensation Board	WCB

29.1.1 Geological Abbreviations

banded iron formation	BIF
deep iron formation	DIF
Echo	EC
gabbro	GAB
George Property (geologically)	GRL
Goose Main	GM
Goose Property (geologically)	GSE
greywacke	GWK
Indicated (resource classification)	IND
Inferred (resource classification)	INF
Llama	LL
Locale 1	Loc1
Locale 2	Loc2
Lone Cow Pond North	LCPn
Lone Cow Pond South	LCPs
lower greywacke	LGWK
lower iron formation	LIF
Measured (resource classification)	MEAS
middle mudstone	MM
mineralization	MIN
pelite	PEL
phyllite	PHYL
quartz porphyry dyke	QFP
Umwelt	UM
upper greywacke	UGWK
upper iron formation	UIF
volcanic-turbidite series	VTS

30 Appendices

Appendices shown in Table 30.1 are available upon Request.

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