1.0 SUMMARY

1.1 INTRODUCTION

The Seabridge Gold Inc. (Seabridge) Courageous Lake Project (the Project) involves the development of a major gold deposit located approximately 240 km northeast of Yellowknife, Northwest Territories (NWT), Canada.

This National Instrument 43-101 (NI 43-101)-compliant Prefeasibility Study (PFS) on the Courageous Lake property has been prepared by Tetra Tech-Wardrop (Tetra Tech) for Seabridge, and has been based on work produced by Tetra Tech and the following independent consultants:

- Resource Modeling Inc. (RMI)
- Moose Mountain Technical Services (MMTS)
- EBA Engineering Consultants Ltd., a Tetra Tech company (EBA)
- WN Brazier Associates Inc. (Brazier)
- Rescan Environmental Services Ltd. (Rescan)
- Golder Associates Ltd. (Golder)
- SRK Consulting (Canada) Inc. (SRK).

Dr. Jianhui (John) Huang (P.Eng.) of Tetra Tech visited the property on August 24, 2010, and is the Qualified Person (QP) for matters relating to mineral processing and metallurgical testing, infrastructure (excluding power supply and airstrip), operating costs (excluding mining operating costs), capital cost estimate, project development plan, and overall report preparation.

Dr. Sabry Abdel Hafez (P.Eng.) of Tetra Tech is the QP for matters relating to the financial analysis.

Mr. Michael J. Lechner (P.Geo., RPG, CPG) of RMI visited the property most recently between August 3 and August 5, 2010, and is the QP for all matters relating to the mineral resource estimate.

Mr. Jim Gray (P.Eng.) of MMTS visited the property on July 13 and July 14, 2010. He is the QP for matters relating to mining, mining capital, and mine operating costs.

Mr. Nigel Goldup (P.Eng.) of EBA visited the property on July 13 and 14, 2010, and is QP for matters relating to the tailing, surface water management, mine rock storage facilities, and surficial geology.

Mr. Kevin Jones (P.Eng.) of EBA visited the property on September 21 and 22, 2010, and on July 29, 2011, and is QP for matters relating to the airstrip upgrade.

Mr. Neil Brazier (P.Eng.) of Brazier visited the property on August 24, 2010, and from August 21 to 28 (inclusive), 2012, and is the QP for matters relating to power generation.

Mr. Pierre Pelletier (P.Eng.) of Rescan visited the property on August 24, 2010, and is the QP for the environmental setting and baseline studies.

Mr. Cameron Clayton (M.Eng., P.Geo.) of Golder visited the property from July 13 to 16, 2010, and from July 20 to 23, 2010. He is the QP for matters relating to the open pit slope stability.

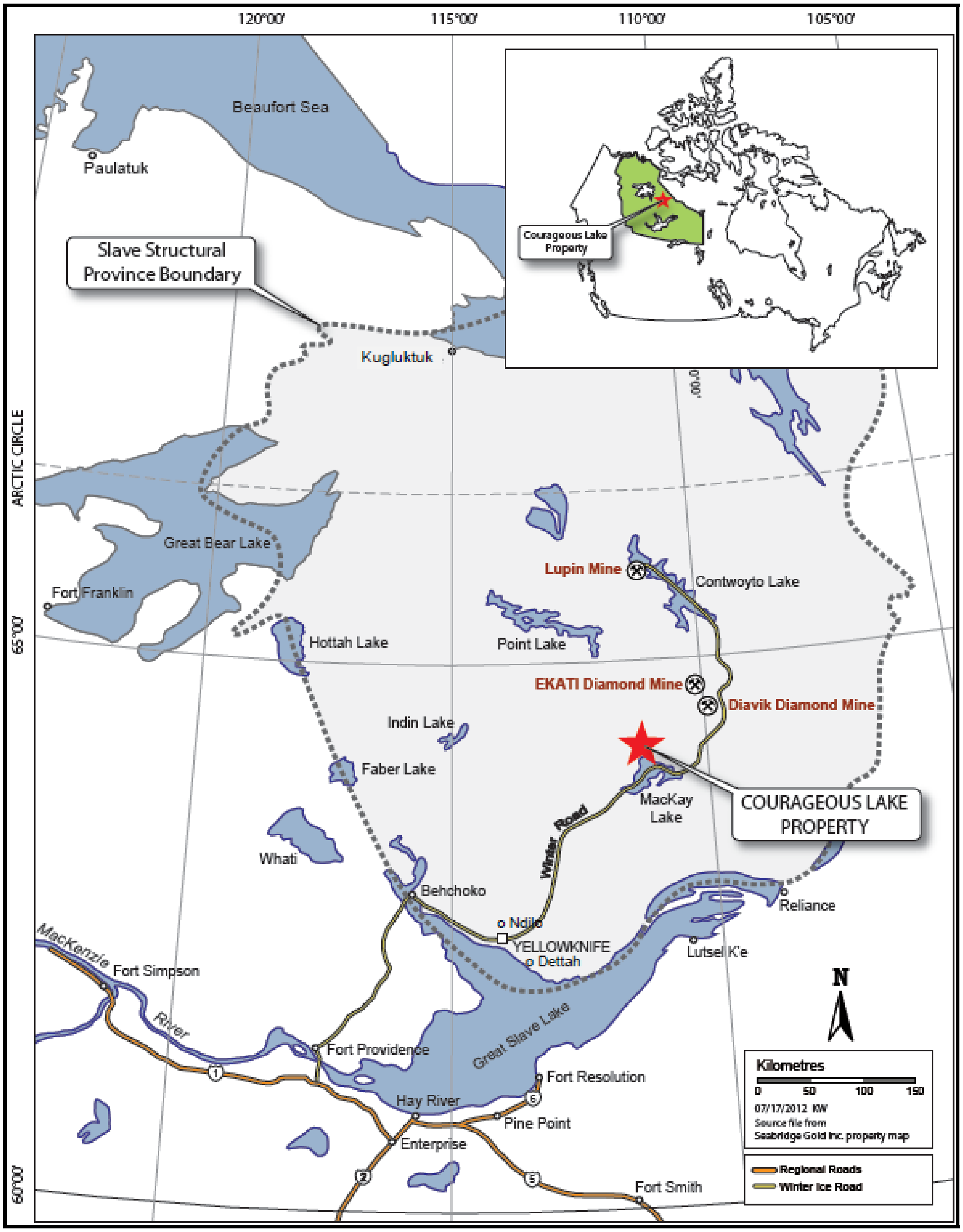
Mr. Stephen Day (M.Sc., P.Geo.) of SRK visited the property on September 29 and 30, 2010, and is the QP for matters relating to metal leaching and acid rock drainage.

This PFS has been completed to a +25/-15% level of accuracy. All dollar figures presented in this report are stated in US dollars (US$) unless otherwise specified.

1.2 Property Location

The property is located 240 km northeast of Yellowknife in the NWT (Figure 1.1). The centre of the deposit is located at approximately NAD83 Universal Transverse Mercator (UTM) coordinates 486,700 East and 7,109,600 North.

Figure 1.1 Property Location Map



1.3 PROPERTY OWNERSHIP

The property, owned by Seabridge, is a collection of mineral leases and mining claims which trend north-south along the approximate 54 km length of the Courageous Lake Greenstone Belt (CLGB).

As of August 2012, the Courageous Lake property is comprised of 62 Federal Mining Leases and 26 Federal Mining Claims, having a combined area of 124,189.86 acres. Seventeen of the Mining Leases were acquired by Seabridge through a "Purchase and Sale Agreement" with Newmont Canada Ltd. (51%) and Total Resources (Canada) Ltd. (Total) (49%), dated July 16, 2002. The Mining Leases are encumbered by two Royalty Agreements (G21883 and G21885) and two Debentures (G21884 and G21886) registered in favour of Newmont Canada Ltd. and Total, respectively. The property is subject to a 2 km Area of Interest from, and parallel to, all exterior boundaries of the Mining Leases.

1.4 HISTORY

The Project is an advanced gold exploration project located in the NWT, Canada. Gold was initially discovered in this region in the early 1940s. Regional geologic mapping studies were carried out intermittently by the Geologic Society of Canada during the mid-1940s through 1980.

In 1980, Noranda Exploration Ltd. (Noranda) began a surface diamond drilling program that resulted in the discovery of the Courageous Lake or FAT deposit. Around 1982, Noranda entered into a joint venture agreement (Tundra Joint Venture) with Getty Canadian Metals Ltd. (Getty) with Noranda as the operator. In 1987, Total Energold Corp. (now Total Resources Canada Ltd.) purchased Getty and assumed the Getty interest in the joint venture. In 1988, Noranda sank a 475 m shaft in order to carry out underground diamond drilling programs to assess the viability of high-grade mineralization that could be exploited by underground mining methods. The results of that program did not meet Noranda's criteria. In 1992, Noranda consolidated its Hemlo Gold Mines Inc. unit with Battle Mountain Gold Company (Battle Mountain Gold) and operated the Tundra Joint Venture as Battle Mountain Gold.

In 1997, Placer Dome optioned the property from Battle Mountain Gold with the concept of developing a bulk tonnage open pit deposit. To test that concept, Placer Dome carried out surface diamond drilling programs during the fall of 1997 and summer/fall of 1998. In 2001, Battle Mountain Gold merged with Newmont Gold Corp. (Newmont) and ownership of the Tundra Joint Venture was transferred to Newmont.

In 2002, Seabridge Gold Corp. (now Seabridge Gold Inc.) purchased the property from the Newmont-Total Tundra Joint Venture, consolidating the property for the first time. Seabridge has carried out a number of drilling campaigns and has conducted various metallurgical and mining studies.

1.5 GEOLOGY

The Project is located within the CLGB, which is a steeply east dipping homocline sequence of metavolcanic and metasedimentary rocks of the Yellowknife Supergroup. The CLGB is bounded to the west by a sodic granite pluton, referred to as the Courageous Lake Batholith, and to the east by conformably overlying turbidite sequences. Dynamothermal regional metamorphism within the CLGB has created mineral assemblages indicative of mid-greenschist facies metamorphic grade. Lower amphibolite facies grade metamorphism has been identified north and south of the CLGB.

The volcanic material within the CLGB represents a tholeiitic to calc-alkaline suite of volcanic rocks common to many Archean greenstone belts. U-Pb and Rb-Sr age determinations reveal an age of 2.66 Ga (Dillon-Leitch, 1981).

Felsic volcanic rocks and their intrusive equivalents in the CLGB were derived from peraluminous, sub-alkaline magmas of calc-alkaline affinity (Wells, 1998). These felsic volcanic lithologies are the predominant host of the FAT deposit.

Within the felsic volcanic rocks are abundant lense-shaped epiclastic intercalations that are thought to be derived from a tuffaceous source. The lithologies are tuffaceous greywacke, thinly laminated siltstone, and fine-grained arkosic sandstone.

The mineral domains or zones of the FAT deposit are defined by a discrete suite of hydrothermal alteration assemblages. The lateral continuity and stratigraphic thickness of the hydrothermal system indicates that the FAT deposit is robust in volume and durations. The predominant hydrothermal alteration minerals in the FAT deposit are illite group sheet silicates, referred to as “sericite”. Silicic alteration of varying intensity is ubiquitous throughout the defined mineralized zones and is represented by silica flooding of groundmass material in volcanic rock. Generally the most intense zones of silica alteration are not indicative of higher gold concentrations. Carbonate alteration is also quite ubiquitous and occurs as calcite, ankerite, and siderite.

1.6 MINERALIZATION

Sulfide mineralogy in the FAT deposit is relatively simple and consists of pyrite, pyrrhotite, arsenopyrite, sphalerite, and chalcopyrite in decreasing order of abundance. While all of these minerals can be found in the mineralized zones, only arsenopyrite has a consistent correlative relationship to gold concentrations. Arsenopyrite occurs in three distinct habits: acicular disseminated crystals, anhedral disseminated clots, and euhedral crystals in fractures. The acicular variety tends to have the clearest association with higher-grade gold mineralization.

1.7 METALLURGY

There were five major testing programs performed on the mineral samples from the property since 2003.

From June 2003 to July 2004, SGS-Lakefield Research Ltd. (Lakefield) conducted a metallurgical testing program which included a comprehensive investigation into flotation and gravity concentration, flotation concentrate pre-treatments by bio-oxidation (BIOX) and pressure oxidation (POX), cyanide leaching, and POX slurry neutralization.

The second test work program carried out by G&T Metallurgical Services Ltd. (G&T) in 2007 focused on optimizing flotation performance. The program included two stages of metallurgical test work – one in early 2007, and the other in mid-2007.

Further metallurgical test work was conducted by Lakefield during 2010 and 2011 on the samples collected from the geologically identified mineralization zones in the FAT deposit. The test work investigated the metallurgical responses of the various mineral samples to flotation, POX, and cyanidation.

Based on the 2007 work program, G&T conducted the 2012 test program to further investigate flotation optimization, and prepared concentrates for POX tests that were carried out in 2012 by Sherritt Technologies, a division of Sherritt International Corp. (Sherritt). Sherritt also conducted cyanide amenability (CNA) tests on the POX residues and cyanide destruction tests on the cyanide leach residues.

The Lakefield study revealed that the dominant sulfides in the mineralization were arsenopyrite (<5-350 µm), pyrite (~5-350 µm), marcasite (20-350 µm), and pyrrhotite (~5-350 µm). The gold occurred as liberated gold, or associated with sulfides and silicates. Gold grain sizes ranged from microscopic invisible to 70 µm. The degree of the sulfide oxidation appeared to be very low. The G&T investigation indicated that between 43 and 54% of sulfides were liberated when the sample was ground to a particle size of 80% passing 165 µm.

These testing programs also determined mineral sample resistance to various comminution processes. The test work determined the Bond ball mill work index and hardness parameters related to semi-autogenous mill grinding (SAG) and high pressure grinding rolls (HPGR) crushing.

The grindability test results indicated that the sample is moderately hard for grinding by ball mills but is very hard for milling by SAG mills. The HPGR locked cycle tests showed that the gross specific energy requirement for particle size reduction by HPGR was 2.20 kWh/t with a specific throughput of 257 ts/hm^3.

The mineralization responded well to flotation concentration. The various test programs produced very similar metallurgical performances. Gold recovery by flotation was high, ranging from 85 to 95%. The pressure oxidation and cyanidation tests by Lakefield and Sherritt indicated a significant improvement in gold extraction when the flotation concentrate underwent a high degree of pressure oxidation. The three separate testing programs by the two laboratories showed that over 98% of the sulfide sulfur can be oxidized with the standard conditions practiced in the POX industry. The test work indicated that gold extraction improved substantially with increasing sulfur oxidation. The Lakefield and Sherritt test results showed that the gold extractions from the POX residues varied from 94 to 99%.

The test results also showed that the flotation concentrate was amenable to the BIOX process; however, it appears that the mineralization did not respond well to gold recovery by gravity concentration.

Table 1.1 projects the average annual metallurgical performances according to the test results and the proposed mining plan.

Table 1.1 Metallurgical Performance Projection

Year	Tonnage (kt)	Feed Grade (Au g/t)	Recovery (Au %)	Annual Gold Production	
				kg	oz
1	4,482	2.171	89.4	8,701	279,729
2	5,760	2.160	89.4	11,121	357,544
3	6,397	2.264	89.4	12,947	416,266
4	6,397	2.195	89.4	12,552	403,545
5	6,397	2.061	89.4	11,784	378,868
6	6,397	2.003	89.4	11,453	368,217
7	6,397	1.870	89.0	10,649	342,358
8	6,397	2.245	89.4	12,838	412,752
9	6,397	2.202	89.4	12,595	404,948
10	6,397	2.161	89.4	12,358	397,306
11	6,397	2.136	89.4	12,218	392,821
12	6,397	2.182	89.4	12,476	401,128
13	6,397	2.274	89.4	13,002	418,031
14	6,397	2.581	89.8	14,829	476,777
15	4,120	2.744	89.8	10,154	326,445
Total	**91,126**	**2.205**	**89.4**	**179,677**	**5,776,735**

1.8 MINERAL RESOURCES

RMI has completed four historical estimates of Mineral Resources for the Project. In late 2011 and early 2012, RMI constructed a new resource model incorporating 2011 drilling results and an updated geologic interpretation that was completed by

Seabridge's geologic staff. Similar to RMI's 2010 model, block gold grades were estimated using a series of nested inverse distance cubed interpolation runs within mineral zone wireframe boundaries. Additional constraints were implemented for the updated model using indicator probabilities and a more selective search strategy referred to as "dynamic anisotropy". The estimated block grades were classified into Measured, Indicated, and Inferred categories using a combination of distance to drilling data, the number of drill holes used to estimate block grades, and a wireframe shape reflecting mineralized continuity. Table 1.2 summarizes the undiluted Measured, Indicated, and Inferred Mineral Resources of the Courageous Lake deposit at a 0.83 g/t gold cut-off grade.

Table 1.2 Summary of Undiluted Gold Resources

Tonnes (000)	Au (g/t)	Au (oz 000)
Measured		
13,401	2.53	1,090
Indicated		
93,914	2.28	6,884
Measured + Indicated		
107,315	2.31	7,974
Inferred		
48,963	2.18	3,432

Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

1.9 MINING

1.9.1 MINE PLANNING – LERCHS-GROSSMAN PIT LIMITS

MMTS has produced a series of Lerchs-Grossman (LG) pit shell optimizations for the Courageous Lake deposit using a resource model provided by RMI. The pit optimizations use mining, processing, tailing management, general and administrative (G&A) costs, and process metal recoveries. The processing cost includes a gold plant to produce doré on site. Measured and Indicated Resource classes are used in the pit optimization.

Cut-off Grade (COG) is determined using an estimated Net Smelter Return (NSR) in Cdn$/t, which is calculated using Net Smelter Prices (NSP). The NSR (net of offsite refining charges and onsite mill recovery) is used as a cut-off item for break-even economic material selection. The NSP includes metal prices, US$ exchange rate, off-site transportation, and refining charges. The metal price and resultant NSP is shown in Table 1.3.

Table 1.3 Metal Price and NSP

Metal	Metal Price (US$/oz)	NSP (Cdn$/g)
Au	1,244	41.98

MMTS notes that the economic pit limits are based on Measured and Indicated resource classes as well as estimated mining unit costs at a PFS level of study. These costs are derived to meet the local conditions for the Project and the specific project arrangements for mine rock management and water management, as well as certain input parameters such as pit slope angles, process recoveries, allowances within the mine operating costs for environmental considerations, and reclamation requirements. All of these components affect the mining quantities and activities to release the specified mineralization and, as such, affect the economic pit limits. Changes to these design elements and parameters will not only affect the cost estimates within the plan but will also impact the economic pit limits in future studies.

The ultimate economic pit limit for this study is selected using the Base Case price (Table 1.3). Typically a time discounted value analysis would be used on a project with a 15- to 20-year mine life to maximize the NPV and IRR. However, when this is done, deeper ore grade material is discounted more heavily and often the pit size is decreased. The previous study ("Courageous Lake Updated PEA 2011"; Wardrop, 2011), which (as a PEA) was allowed to include Inferred material in the pit limit economics, reported a pit delineated Resource similar to this PFS Base Case ultimate pit, where Inferred material is counted as waste. Even though a discounted value analysis could possibly improve the financial results of this prefeasibility-level study by limiting the mining to shallower ore, the discounted method was not chosen. Instead the larger, less economic, pit limit has been selected as a basis for this study to maximize the mineable resource in anticipation that future exploration will upgrade the Inferred material internal in the pit, to a Measured or Indicated Resource. Future studies will consider a time discounted economic pit analysis after the Inferred material has been drilled off.

The in-situ LG pit delineated resource summarized in Table 1.4 uses a NSR COG of Cdn$20.1/t but does not include any mining dilution or mining loss.

Table 1.4 Measured and Indicated LG Pit Resources

In situ Pit Resource (Mt)	Au (g/t)	Mine Rock (Mt)	Strip Ratio
86	2.35	935	10.8

Note: NSR cut-off used is Cdn$20.1/t.

The total in-situ metal contained in the chosen LG ultimate pit is 6.5 Moz of gold.

FUTURE OPEN PIT OPPORTUNITY

The Inferred Resources within the selected LG pit limits used for this study are included in the waste tonnages in Table 1.4. These Resources, included within the ultimate pit limits, have the potential to be upgraded to Reserves with future exploration drilling or when the pit grade control and blast hole assaying systems is in place. Additionally, Inferred material may also have the potential to expand the future economic pit limit if they are upgraded by future exploration drilling.

To analyze this potential, an additional set of pit optimization runs have been completed allowing Inferred material to be given a value as well. Table 1.5 summarizes the results of these Inferred LG pit limits incremental to the Measured and Indicated pit resources shown in Table 1.4.

Table 1.5 Incremental Inferred LG Pit Resources

In situ Pit Resource (Mt)	Au (g/t)	Mine Rock (Mt)	Strip Ratio
29	2.74	395	13.5

The incremental in-situ metal contained in the inferred LG pit shell is 2.6 Moz of gold.

1.9.2 MINE ROCK MANAGEMENT FACILITY

The mine rock management facility for the Project is situated east of the pit area, and is constructed using a combination of bottom-up and top-down methods. Foundation preparation will be completed, as required. The proposed schedule of mine rock placement enables flotation tailing to be contained within the footprint of this facility. Allowances are made to address reclamation and post-closure requirements by configuring the constructed slopes at the overall reclamation slope angle. Leach residue tailing will be stored between the ultimate pit and immediately west of the mine rock management facility.

Overburden inside the ultimate pit limit is stripped and placed in the overburden stockpile to the west of the pit. This stockpile is used for reclamation material.

1.9.3 MINING OPERATIONS

Detailed pit phases are developed from the results of the LG sensitivity analysis, which integrates the detailed pit slope criteria and high wall roads. The ultimate pit is divided into smaller mining phases, or pushbacks, to enable a low strip ratio starter pit and to allow for more even waste stripping during the optimized scheduling stage of the project design.

Dilution and mining loss estimates consider the selective mining method required to efficiently extract the narrow near vertical lenses that characterize the Courageous

Lake mineralized zones. Proven and Probable Reserves are estimated in Table 1.7 using diluted whole block grades with additional mining dilution and loss varying by the number of block model resource contact edges with waste blocks. COG, mining dilution, loss, and dilution grades are estimated in Table 1.6. The grade of dilution material is derived from blocks in the model that are just below the specified cut-off grade. Internal dilution contained in the block model accounts for the rest of the expected mining dilution.

Table 1.6 Cut-off Grade, Mining Dilution, Loss, and Dilution Grades

Contact Edges	Cut-off Grade NSR ($/t)	Dilution (%)	Loss (%)	Dilution Grade (Au g/t)
0	20.5	0	0	0.404
1	20.5	5	5	0.404
2	20.5	5	5	0.404
3	20.5	5	5	0.404
4	20.5	5	5	0.404

Table 1.7 Summary – Reserves

Pit	Ore (Mt)	Au (g/t)	Mine Rock (Mt)	Strip Ratio (t:t)
P651	34.3	2.19	198.6	5.8
P652i	25.8	2.04	463.8	18.0
P653i	31.1	2.36	479.9	15.5
Total	**91.1**	**2.20**	**1,142.3**	**12.5**

Note: NSR cut-off used is Cdn $20.5/t.

Proven and probable reserves are summarized in Table 1.8.

Table 1.8 Proven and Probable Reserves

Class	Ore (Mt)	Au (g/t)	Contained Metal (M oz)
Proven	12.3	2.41	0.96
Probable	78.8	2.17	5.50

The production schedule has been developed using Mintec Inc.'s (Mintec) MineSight® schedule software. A summary of the production schedule is provided in Table 1.9.

Table 1.9 Summarized Production Schedule

	Unit	Year																LOM
		-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	
Pit to Mill																		
Ore Tonnes	kt	-	4,165	5,760	6,397	6,397	6,397	6,397	6,397	6,397	6,397	6,397	6,397	6,397	6,397	6,397	4,079	90,767
Grade – Gold	g/t		2.160	2.160	2.264	2.195	2.061	2.003	1.870	2.245	2.202	2.161	2.136	2.182	2.274	2.581	2.764	2.205
Pit to Stockpile																		
Ore Tonnes	kt	359	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	359
Grade – Gold	g/t	2.148	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2.148
Stockpile Reclaim																		
Ore Tonnes	kt	-	317	-	-	-	-	-	-	-	-	-	-	-	-	-	42	359
Grade – Gold	g/t	-	2.324	-	-	-	-	-	-	-	-	-	-	-	-	-	0.819	2.149
Stockpile Inventory	kt	359	42	42	42	42	42	42	42	42	42	42	42	42	42	42	0	
Mill Feed																		
Ore Tonnes	kt	-	4,482	5,760	6,397	6,397	6,397	6,397	6,397	6,397	6,397	6,397	6,397	6,397	6,397	6,397	4,120	91,126
Grade – Gold	g/t	-	2.171	2.160	2.264	2.195	2.061	2.003	1.870	2.245	2.202	2.161	2.136	2.182	2.274	2.581	2.744	2.205
Metal to the Mill - Gold	M oz	-	0.3	0.4	0.5	0.5	0.4	0.4	0.4	0.5	0.5	0.4	0.4	0.4	0.5	0.5	0.4	6.5
Waste Mined																		
Total Waste Tonnes Mined	kt	7,641	51,816	49,997	49,998	49,998	89,998	120,935	159,997	159,998	159,998	99,999	64,624	30,184	22,861	17,243	6,970	1,142,256
Strip Ratio (waste mined/Plant Feed)	t/t	-	11.6	8.7	7.8	7.8	14.1	18.9	25.0	25.0	25.0	15.6	10.1	4.7	3.6	2.7	1.7	12.5

Mining operations, methods, and equipment will be typical of open-pit mining in northern Canada. The Project will be a large-capacity operation that utilizes large-scale equipment for the major operating areas in order to generate high productivities, and reduce unit and overall mining costs. The maximum size of the large mining equipment will be constrained by the maximum loads, which can be delivered along the winter road.

The mine plan and production schedule will undergo further refinement during higher levels of study for the Project. Additional geotechnical information on high wall capabilities should confirm the pit slopes and determine if the ultimate pit can be designed to a deeper depth. Further details on rock storage management, water management, and final land use will be developed for the Environmental Assessment application, the result of which will impact the mine plan. These elements, along with other optimization details, will be integrated into feasibility-stage mine planning.

1.9.4 OPEN PIT MINING – GEOTECHNICAL CONSIDERATIONS

Pit slope design criteria are required as a basis for input to the MineSight® pit optimization program to establish the economic pit shells. Golder completed a study to develop prefeasibility slope design criteria for the Project, and presented these in a report titled “Pre-Feasibility Level Pit Slope Design Criteria for the FAT Deposit, Courageous Lake Project” (Golder, 2011a), dated May 18, 2011. The report presents the results of the geotechnical site investigation carried out by Golder personnel from July 2010 to September 2010, which included the following activities:

- geotechnical logging of specific intervals of five exploration boreholes
- point load strength testing
- geotechnical logging of three inclined, oriented boreholes
- collection of 125 core samples
- geotechnical mapping of Peggy’s pit
- six hydrogeological conductivity tests in three boreholes
- installation of two thermistors in two boreholes
- installation of one vibrating wire piezometer
- installation of two electrical conductivity probes in two boreholes
- a site visit from senior Golder personnel.

Additional oriented geotechnical data collection was carried out by Golder in 2011 as part of hydrogeological studies that also involved borehole hydraulic conductivity testing and the installation of two Westbay monitoring and sampling well systems (installed by others). The installation of the wells for water sampling purposes provided an opportunity to collect additional oriented geotechnical data from areas deeper in the deposit than previously drilled. The orientation data were compared with previous core orientation data and confirmed that the structural orientations used to develop the prefeasibility slope design criteria are consistent with depth in the deposit.

Golder completed a study in 2012 to evaluate the east pit wall stability with a revised mine plan that places the leach residue tailing storage facility adjacent to the east pit wall crest. The results were presented in a draft report titled “East Pit Wall Slope Stability Assessment for the FAT Deposit, Courageous Lake”, dated May 3, 2012. The results indicated the east pit wall to be stable with the surcharge from the leach residue tailing storage facility at the proposed location, and for the model parameters considered.

Based on the results of the 2010 and 2011 investigations, and the information gathered from previous studies, design criteria were developed for prefeasibility-level final pit slope angles.

OVERBURDEN SLOPES

The final pit slope walls will expose variable thicknesses of overburden overlying rocks of the CLGB volcanic sequence and the overlying Yellowknife Group Sedimentary rocks. Stability analyses were carried out for the overburden materials in the "Courageous Lake Updated PEA 2011". The preliminary slope design criteria shown in Table 1.10 are suggested in order to establish the approximate footprint area of the proposed pit.

Table 1.10 Proposed Overburden Slope Design

Slope Design Element	Value
Bench Height (m)	5
Catch Bench Width (m)	5
Bench Face Angle (degrees)	30
Overall Slope Angle (degrees)	22

Once geotechnical characterization of the soils has been completed, these slope angles may need to be revised and may be shallower. Adequate dewatering of the soil materials will be required in order to achieve stable slopes within the overburden. A minimum 10 m-wide catchment berm at the interface between soil and rock should be incorporated into the pit slope design to:

- accommodate sloughing of material
- allow the installation of a drainage ditch to manage water at the pit crest
- allow equipment to access any areas that require maintenance.

ROCK SLOPES

The pit slope design criteria are based on:

- geotechnical data collected by Golder at the Courageous Lake Project site during the 2010 and 2011 geotechnical investigations
- surface geotechnical mapping investigations by Golder
- previous geotechnical drilling investigations by others.

The FAT deposit is defined by one structural domain, represented by the persistent steeply west dipping foliation. The proposed pit has been sub-divided into five design sectors on the basis of wall orientations. The bench face angles (BFA) and inter-ramp angles (IRA) within each of the design sectors were formulated to minimize structurally-controlled failures based on limit equilibrium analyses. The east and west wall stability will be controlled predominantly by the continuous foliation. The north and south walls will be controlled predominantly by the orientation of the more widely spaced, discontinuous subordinate joint sets. The presence of elevated pressure heads in the toe region of the high rock slopes will have an impact on the stability of the overall pit slopes. Design sectors are named by the wall orientation that each sector represents. The results of the stability analyses indicate that slope depressurization will be required once the pit extends deeper than the base of permafrost.

Table 1.11 summarizes the proposed bench design criteria for the FAT deposit.

Table 1.11 Summary of Prefeasibility Pit Slope Design Criteria – FAT Deposit

Slope Design Element	Wall Sector Azimuth and Description				
	166° to 226° South to Southwest Wall	**226° to 312° West Wall and Tundra Fault**	**312° to 012° Northwest to North Wall**	**012° to 066° North to Northeast Wall**	**066° to 166° East Wall**
BFA	64°	80°	64°	67°	70°
Final Bench Height (m)	30 m	30 m	30 m	30 m	30 m
Catch Bench Width (m)	12 m	16 m	12 m	12 m	12 m
IRA	48.4°	54.6°	48.4°	50.5°	52.6°
Design Basis and Limiting Factors	Design controlled by inter-ramp slope instability relating to J3 and J0 interaction; limit Inter-ramp to maximum 48°.	Bench scale toppling along in-dipping foliation, additional ravelling expected beneath/adjacent to Tundra Fault; additional catchment may be required; develop ramp in West Wall where possible.	Design controlled by inter-ramp slope instability relating to J1 and J0 interaction; limit Inter-ramp to maximum 48°.	BFA controlled by J4; inter-ramp slope instability on J4/J0; limit inter-ramp to maximum 51° for overall wall stability.	BFA controlled by foliation orientation, foliation is continuous feature, design to 15 to 20% PoF*; Limit inter-ramp to maximum 53° for overall wall stability by adjusting catch bench width.

* PoF = Probability of Failure.

ALTERNATE BENCH DESIGN AND OVERALL/INTER-RAMP SLOPE ANGLES

Table 1.12 presents an alternate slope design for the west wall of the pit, assuming that carefully controlled blasting practices are used, including the use of angled pre-shear blast holes. Typically, uncontrolled blasting will result in significant undercutting of structure and ravelling of material; therefore, it requires wider catchment benches in order to accommodate this material and maintain a safe catchment bench width. Alternatively, by using controlled blasting methods, the angled pre-shear drilling and blasting would reduce the degree that material ravelling would occur due to the undercutting of the foliation. Consequently, a narrower catch-bench width would be achievable.

Table 1.12 Alternate Prefeasibility Pit Slope Design Criteria – FAT Deposit

	Wall Sector Azimuth and Description
Slope Design Element	**226° to 312° West Wall and Tundra Fault**
BFA	70°
Final Bench Height (m)	30 m
Catch Bench Width (m)	10 m
IRA	55.1°
Design Basis and Limiting Factors	Bench scale toppling along in-dipping foliation, additional ravelling expected beneath/adjacent to Tundra Fault; additional catchment may be required; develop ramp in West Wall where possible

In all cases, the use of controlled blasting techniques will be required in order to achieve the recommended bench configurations. During feasibility studies, additional geotechnical boreholes drilled to confirm the structural orientations used for this level of study should be undertaken.

PIT WATER INFLOWS, SLOPE DEPRESSURIZATION, AND GROUNDWATER QUALITY

A baseline hydrogeological investigation was undertaken by Golder in 2010, and the results were summarized in a Technical Memorandum titled “Preliminary Estimate of Water Inflows to the Proposed Courageous Lake Project Open Pit” (Golder, 2010a), dated December 23, 2010. Additional hydraulic conductivity testing was carried out in 2011 in two boreholes drilled for the installation of Westbay water quality monitoring and sampling wells, as previously described. The data collected were compared with the results of the previous hydraulic conductivity testing and confirmed that the 2011 results are consistent with the results of the 2010 study.

A three-dimensional hydrogeological model was developed to estimate the potential daily pit inflows to the proposed pit. Table 1.13 presents the estimates of water inflows to the pit on an annual basis assuming depressurization of the pit slopes is required for stability reasons. Depressurization could be achieved by installing active pumping wells from a dedicated bench approximately one year prior to the pit reaching the base of the permafrost.

Table 1.13 Estimated Water Inflows to Open Pit

Mining Year	Pit Depth (m Below Ground Surface)	Estimated Water Inflow for Depressurized Case (m^3/d)
-1	0	Inflows from surface runoff not estimated.
1	40	Inflows from surface runoff not estimated.
2	90	Inflows from surface runoff not estimated.
3	150	Inflows from surface runoff not estimated.
4	210	Inflows from surface runoff not estimated.
5	270	7,900
6	330	10,100
7	390	10,100
8	430	10,100
9	480	9,200
10	550	9,200
11	560	9,200
12	580	8,600
13	590	8,600
14	600	8,600

The inflows to the pit determined by the modelling are assumed to come from the sub-permafrost aquifer; therefore, there are no predicted groundwater inflows when the pit is entirely within permafrost.

As the pit is mined down through the permafrost, it will approach the bottom of the permafrost in Year 4 to Year 5. A three-dimensional model was developed to predict inflow quality and total dissolved solids (TDS) concentrations to the open pit. Table 1.13 presents the model predicted values.

Table 1.13 Predicted Pit Inflows Quantity and Quality

Year	Average Inflow (m^3/d)	Average TDS Concentration (mg/L)	Total TDS Mass (kg)
5	7,900	5,000	1.29E+07
6 to 8	10,100	7,000	7.77E+07
9 to 11	9,200	11,000	1.09E+08
12 to 14	8,600	14,000	1.20E+08
Total	**33,400,000 m^3**		**3.06E+08**

RECOMMENDATIONS FOR FUTURE STUDIES

Additional hydraulic conductivity testing in exploration boreholes is suggested in order to refine the hydrogeological model, and to confirm previous test results and water inflow quantity estimates. As the Project is advanced to a feasibility-level of study, additional oriented coring will need to be completed to collect oriented geotechnical data from other areas of the deposit in order to verify the assumptions used in the development of the prefeasibility slope design criteria.

1.10 MINERAL PROCESSING

Tetra Tech updated the process flowsheet for the Project, according to the results from the testing programs. The proposed process plant will process 17,500 t/d of mineralization. The plant will be operated 365 d/a at an availability of 92%.

The flowsheet proposed for the Project includes HPGR/grinding comminution, conventional flotation, flotation concentrate pressure oxidation, cyanidation, and gold recovery/refining circuits. The comminution will consist of primary crushing by gyratory crusher, secondary crushing by cone crusher, and tertiary crushing by HPGR followed by ball mill grinding. The conventional flotation will include rougher flotation, scavenger flotation, and cleaner flotation on scavenger flotation concentrate. The rougher flotation concentrate together with the scavenger cleaner concentrate will be oxidized under pressure after being reground and acid pre-leached. A portion of the slurry and acid-bearing solution from the POX circuit will be recycled back to the POX pre-leaching. The slurry and the acid-bearing solution of the POX discharge will be separately neutralized. The POX residue or oxidized concentrate will be leached in a carbon-in-leach (CIL) circuit; the cleaner flotation tailing will be reground and cyanide leached together with the POX residue. Gold recovery will be completed by stripping the loaded carbon from the CIL circuit, followed by electrowinning to produce gold doré as a final product. The flotation tailing and the cyanide residue will be thickened and disposed of separately. The residual cyanide in the leach residue will be destroyed by a sulphur dioxide (SO_2)/air oxidation procedure prior to disposal to the lined leach residue storage facility.

The HPGR circuit is recommended, instead of a SAG mill grinding arrangement, to reduce energy consumption.

The description of the process flowsheet and the design criteria for the proposed process plant are detailed in the Section 17.0 of this report.

1.11 GEOTECHNICAL, TAILING, AND WATER MANAGEMENT

1.11.1 GEOTECHNICAL INVESTIGATION

EBA was retained by Seabridge to conduct three geotechnical investigations to support the proposed development, which were conducted during the following periods:

- September 3 to September 17, 2010, with the primary focus on the mine rock and tailing disposal areas, plant site, and proposed diversion channel Option 1A, and comprised 10 boreholes and the installation of two ground temperature cables.
- March 21 to April 12, 2011, with the purpose of completing the remaining boreholes from the summer 2010 investigation and comprised 14 boreholes and the installation of three ground temperature cables.
- March 9 and March 30, 2012, with the purpose of investigating the alternate Matthews Creek diversion channel alignments (Options 2, 3, and 4), the alternate process plant site (Option 1), and at the southeast end of the proposed airstrip extension. The investigation consisted of 14 boreholes and the installation of three ground temperature cables. In addition, a geophysical survey was undertaken of the Matthews Creek diversion channel alignments (Options 2, 3, and 4), the North Dam, and the airstrip extension using ground penetrating radar and capacitively-coupled resistivity.

A total of 38 infrastructure geotechnical boreholes have been drilled at the site to depths ranging between 5 m and 24.5 m together with the installation of 8 ground temperature cables. Twenty-one of the boreholes targeted the general arrangements of the supporting mine infrastructure (i.e. mine rock storage facility, tailing storage facilities, water retention structures, and water diversion dams). The remaining 17 boreholes specifically targeted the diversion channel routing options for the proposed Matthews Creek diversion channel, an alternative plant site location, and the airstrip extension.

1.11.2 SURFICIAL GEOLOGY AND PERMAFROST

In the project area, surficial unconsolidated material interspersed with patches of the exposed bedrock has been deposited during the Late Pleistocene to the Holocene time in glacial, periglacial, and postglacial environments. It consists of glacial (till), glaciofluvial (esker), glaciolacustrine, fluvial (alluvium), and organic deposits.

Till dominates the project area terrain. It consists predominantly of unsorted sand and silt with variable proportions of gravel and trace of clay, with cobbles and boulders disseminated throughout. The thickness of the till ranges from less than 1 m (a till veneer) to 21.4 m (ground moraine). The till is ice-rich, locally with ground ice bodies up to 1.3 m thick encountered in the upper 3 m.

The glaciofluvial deposits in the project area form well-defined east-west trending esker ridges, mounds, and flanking aprons. The deposits consist of sand and silt, some to trace gravel in planar, cross-stratified and massive beds with cobbles and boulders disseminated throughout.

Minor glaciolacustrine deposits comprise mainly silt and sand, with lenses of organic material, cross-stratified to planar bedded. These deposits usually occur locally as beaches along old lake shores.

Fluvial deposits occur as a floodplain of the Matthews Creek valley. They consist of silt to gravel size stratified and moderately sorted sediment with lenses and layers of organic material.

Organic deposits, peat, and muck up to 3.0 m thick occur as patterned peatlands in depressions and along creek valley and drainage channel bottoms. They are ice-rich and contain ground ice wedges, lenses, and layers of segregated ice that are manifested in peatland topography as ice-wedge polygons, thermokarst collapse structures, and polygonal peat plateaus.

The underlying bedrock consists of volcanic (felsic ash tuff of high to extremely high strength and excellent quality) metasedimentary rock with rock quality ranging from very poor (in the uppermost portion with localized zones of frost-jacked blocks of rock protruding above the exposed bedrock surface) to excellent, with rock strengths varying from medium (in the uppermost portion) to very high, and Precambrian granitic rocks. The granitic rock is found in the western portion of the project area, which has not been investigated during the current geotechnical program.

The project area is underlain by continuous, locally ice-rich permafrost more than 320 m thick. The permafrost temperatures were measured in a range between -0.8°C to -4.3°C below the depth of zero annual amplitude. Permafrost features include frost-crack polygons, frost-jacked rock blocks and boulders, thermokarst features, non-sorted circles, patterned peatlands, and small frost mounds in peatlands.

1.11.3 MINE ROCK, TAILING, AND DESALINATION SOLIDS MANAGEMENT FACILITIES

Mine rock and run-of-mine (ROM) waste products from the mining operations will comprise approximately:

- 1,142 Mt of mine rock
- 18 Mt of neutralized leach residue tailing, including the flotation tailing that is used for neutralizing the POX residue
- 73 Mt of flotation tailing
- 0.38 Mt of dry salt product from the pit depressurization desalination plant.

The tailing, residue, and mine rock will be stored in a tailing/mine rock management facility on a flat open area, east of the open pit and south of Courageous Lake. In addition, the dry salt product (associated with the pit depressurization) will be temporarily stored in encapsulated cells within a landfill located immediately west of pit.

NEUTRALIZED LEACH RESIDUE TAILING MANAGEMENT FACILITY

The neutralized leach residue tailing comprise approximately 20% of the full tailing stream leaving the process plant. The tailing is expected to be somewhat finer than the flotation tailing stream and will be deposited into a containment structure east of the open pit. The neutralized leach residue tailing management facility will provide a capacity of 18 Mm^3, which is greater than the 15.4 Mm^3 required for a 15-year mine life. The tailing will be pumped as slurry to the storage facility and will be deposited using multiple spigot locations.

The neutralized leach residue containment structure will primarily be constructed from mine rock and will be lined. A starter dam will be constructed during the pre-mining construction, to provide sufficient tailing storage for five years. Based on the tailing production schedule, the perimeter dam will be progressively raised over the operating life, to reach an ultimate capacity of 18 Mm^3 and a final crest elevation of 456 m, which equates to a dam height of approximately 30 m.

The neutralized leach residue tailing management facility is designed to accommodate a maximum of 2 Mm^3 of supernatant water. The excess supernatant water (approximately 60,000 m^3/month based on the water balance assessment) will be reclaimed and pumped to the process plant as make-up water.

FLOTATION TAILING MANAGEMENT FACILITY

The flotation tailing comprise approximately 80% of the full tailing stream leaving the process plant. The flotation tailing will be stored within the mine rock storage facility. The mine rock will form the primary containment structure and will be internally lined with crushed bedding and transition materials. A starter containment berm will be constructed to a crest elevation of 456 m during pre-mining operations. The flotation tailing will be pumped to the flotation tailing storage facility and will be discharged using multiple discharge points along the containment berm. Approximately 53.1 Mm^3 of flotation tailing will be disposed of in the tailing storage facility during the life of the mining operations.

The supernatant water accumulated within the flotation tailing management facility will be either be reclaimed to the process plant as the make-up water or pumped to the water treatment plant for treatment and discharge.

MINE ROCK MANAGEMENT FACILITY

Mine rock represents the largest waste stream from the mining operations with an estimated volume of 519 Mm^3. The mine rock management facility will be located to the east of the pit and will occupy an area of 660 ha. The mine rock will be directly hauled from the pit and end-dumped in 30 m lifts and the mine rock management facility will attain a final elevation of 570 m, which is equivalent to a height between 130-150 m. Seepage and runoff from this facility will be collected and managed within the water storage pond.

DESALINATION SOLIDS STORAGE FACILITY

From Year 5 to the end of mining, the water collected by the pit depressurization/ dewatering system will be treated to meet regulatory requirements and discharge in to Courageous Lake. Multiple options were investigated for potential saline water handling processes. It is currently proposed to treat the water using reverse osmosis and thereafter evaporate the concentrated brine to produce a dry salt product. The estimated volume of salt that will be produced by the desalination and evaporation plant during the life of mine is approximately 378,500 t (472,500 m^3). It is proposed to temporally store this dry product in encapsulated cells within a landfill located immediately west of the pit. At the end of mining operations, this saline product will be moved to the base of the pit prior to pit flooding.

1.11.4 *MINE AREA WATER MANAGEMENT*

Currently, Matthews Creek flows through the proposed open pit area so it will be necessary to divert the creek away from the open pit mining operations. In terms of the proposed mine infrastructure layout, a diversion channel options assessment determined that the most suitable routing for the diversion channel is approximately 1.5 km west of the existing Matthews Creek. The proposed diversion will require a cut channel from the western edge of Matthews Lake through to Lake UnL-10, then an additional cut channel from Lake UnL-10 northward to Lake UnL-12. Thereafter the proposed diversion will make use of an existing creek system (some widening will be required) to drain naturally towards and into Courageous Lake.

Prior to the commencement of open pit mining operations, it will be necessary to first construct the Matthews Creek diversion channel (without the final breakthrough into Matthews Lake). The bulk of this construction should be undertaken during the winter with some summer refinements and dressing. After the following spring freshet peak flow event, the upper reaches of the diversion channel should be broken through into Matthews Lake. Only once flow is confirmed along the diversion channel can the existing Matthews Creek be permanently blocked by the Matthews Creek upstream diversion dam.

In terms of surface water management, it will be necessary to construct three water retention dams (North Dam 1, North Dam 2, and East Dam) prior to the commencement of mining operations. These structures will constitute a water storage pond at the north end of the mine rock management facility to collect surface runoff from catchment areas and seepage water from the flotation tailing. The water retention dams will comprise internally lined structures with passive cooled key trenches founded within the overburden or bedrock material. During the latter stages of mine rock placement (after Year 10), it will be necessary to manage surface water to the south of the mine rock pile as the mine rock placement crosses the southern catchment boundary. Southward draining surface water will be collected and managed within two water collection ponds. Overall, the water balance assessment indicates that there will be a net water surplus to the Project. This surplus will be managed by a combination of storage, discharge during freshet, or treatment and discharge.

During mining in Years 1 to 4, pit water seepage will be pumped to the water storage pond while the pit base elevation is above the lower permafrost boundary. From Year 5, the dewatering well inflow and pit seepage inflow will be collected and pumped to a separate saline water treatment facility for treatment and discharge. The total inflow volumes into the pit between Years 5 and 15 range from 7,900 m^3/d to 10,100 m^3/d.

1.11.5 *Mine Area Closure Plan*

At the cessation of mining operations, all supernatant pond water from the tailing storage facilities will be drained or pumped into the pit and the tailing storage areas will be progressively capped. Seepage reporting to the various seepage collection ponds will also be directed into the pit until this water seepage is proven to meet accepted discharge criteria. Thereafter North Dam 1, North Dam 2, and the East Dam will be breached to allow surface water to flow naturally into Courageous Lake.

Pit filling will be augmented by drawing water from Courageous Lake. Some upper pit benches will be modified to provide shallow aquatic habitat features. Once the pit is full, a channel will be constructed to connect the pit to Courageous Lake. Salt from the desalinization process that was moved to the base of the pit prior to pit filling will produce a dense brine, which is anticipated to form a deep stable water layer within the pit lake. Further study is planned at the feasibility level to optimize the closure options.

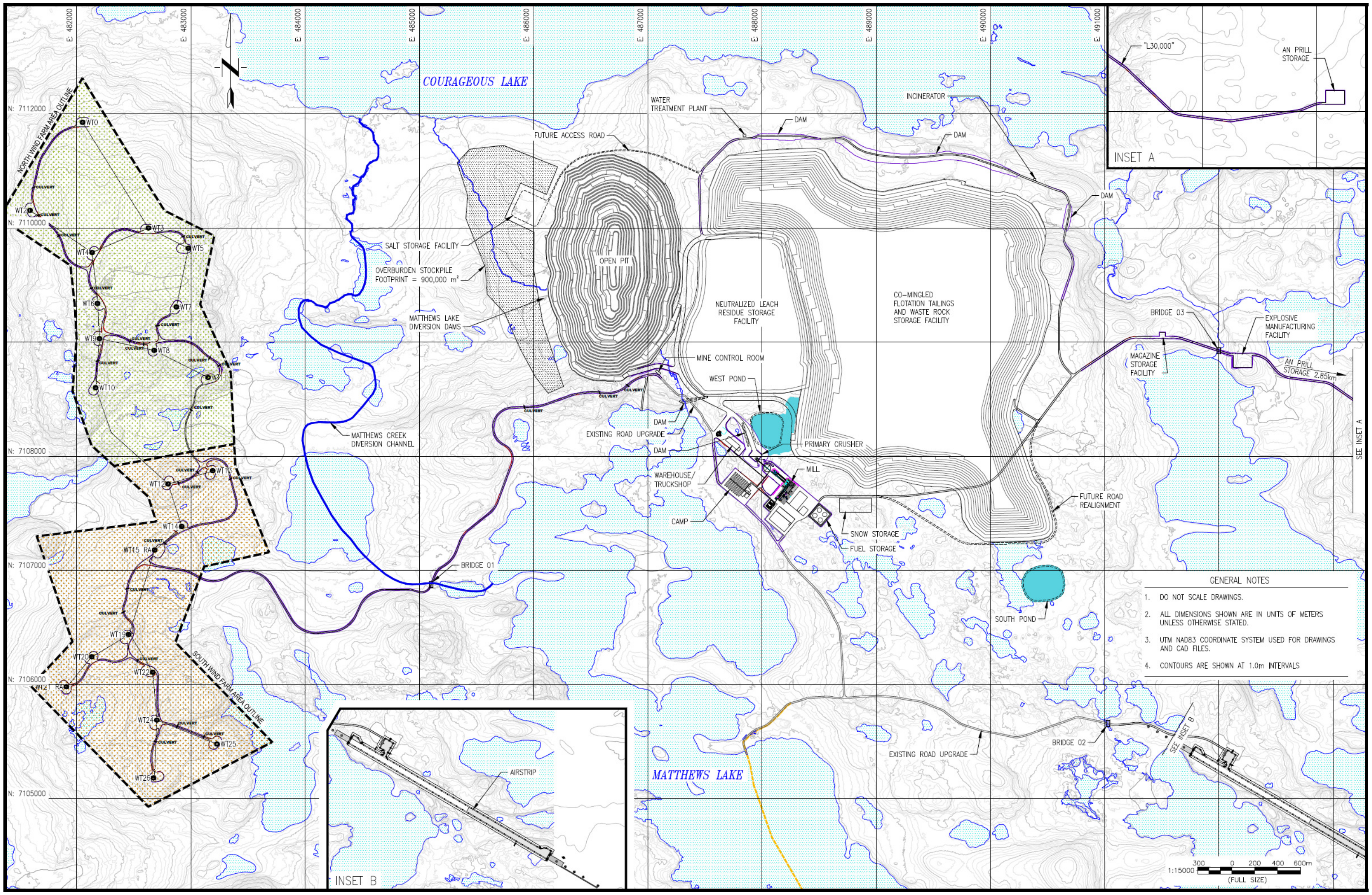
1.12 INFRASTRUCTURE

1.12.1 INTRODUCTION

The Project site is in an isolated location, 240 km northeast of Yellowknife in the NWT. The Project site will be accessible by air or by the Tibbitt to Contwoyto winter road that is normally open from late January/early February until early April of each year. Therefore, the Project will be required to generate its own power, maintain a permanent camp, provide access by air and by winter road, and maximize warehousing and storage.

The Project site will have open pit mining-related facilities, process-related facilities, a power supply plant (consisting of diesel power facility and wind power facility), and a permanent camp. The general site layout is shown in Figure 1.2.

Figure 1.2 General Site Layout



1.12.2 ACCESS

AIRSTRIP

Currently there is an existing airstrip southeast of the mine. To meet the Project's requirements, the existing airstrip will be upgraded to accommodate more frequent flights with larger aircraft. The Phase 1 expansion will accommodate larger turbo-prop aircraft, such as the Hercules and Dash 8 Q400. Phase 2 of the expansion will extend the runway to accommodate B737-200 or equivalent aircraft on a continuous basis. Phase 3 will include widening of the shoulders to achieve certifiable standards. Associated infrastructure and systems include: airfield lighting, wind indicators, an automatic weather observation station, and Wide Area Augmentation System with localizer performance with vertical guidance (WAAS/LPV) instrument approaches.

WINTER ICE ROAD

Currently, there is the Tibbitt to Contwoyto winter ice road connecting Yellowknife, NWT, with the Diavik and EKATI diamond mines. For the purposes of this study, it is assumed that the Project will use the existing winter road.

1.12.3 ON-SITE TRANSPORT ROAD

On the site, the road connecting the existing airstrip and the Project site will be upgraded. On-site service roads will be constructed connecting to the wind power generation towers, ammonium nitrate (AN) prill storage and explosive manufacturing facilities, tailing/residue storage facility, and open pit.

1.12.4 MAIN FACILITIES

The main facilities for the Project will include:

- primary crushing facility
- secondary crushing (cone crushers) and tertiary crushing (HPGR) facility
- main process plant, including primary grinding, flotation, regrinding, pressure oxidation, cyanide leaching, loaded carbon striping and gold recovery, cyanide destruction and water treatment
- oxygen generation plant
- mine rock/tailing/leach residue storage facility, including separate flotation tailing storage pond and leach residue storage pond within the mine rock storage facility
- power generation plant, including diesel power generation and wind power generation

- truck shop, including warehouse
- permanent camp, including general offices
- cold storage, including lime storage facility
- explosive manufacturing facility and related raw material storage facilities
- airstrip, including aviation control system
- sewage treatment facility.

Arctic corridors will be constructed for workers' access to main buildings, and for housing power lines, heating system piping, and other pipe lines. The arctic corridors will connect the camp, main process plant, oxygen plant, diesel power plant, and truck shop. The recovered heat from the diesel power plant and oxygen plant will be used for heating buildings during winter.

1.12.5 POWER SUPPLY AND DISTRIBUTION

DIESEL POWER SUPPLY EQUIPMENT

The selected power supply option for the Project includes a combination of thermal power, based on a diesel power plant, and wind power generation. The proposed diesel power plant makes use of medium speed diesel engines burning light diesel fuel suitable for winter transport to the mine site, all similar to the existing diamond mines in the area. Detailed fuel supply and operational studies have been completed to establish the operating costs for the diesel power installation.

The diesel engines will be equipped with complete waste heat recovery systems including water jacket heat and exhaust gas hot water boilers to provide hot water/glycol for all process buildings and accommodation centre heating. The recovered heat will also be used for concentrated brine dewatering.

The diesel generator sets will be controlled by programmable logic controllers (PLCs) and designed for automatic unattended operation, as has been common in the industry for the past 30 years. However, due to the extensive heat recovery system and the proposed combined operation in conjunction with a wind farm, allowance has been made in the operational and maintenance (O&M) costs for fully staffing the plant with combined trades people operators.

Diesel generator sets have been included to match the plant projected normal running load plus two spare sets (n + 2 criteria) to allow for one set to be down for normal preventative maintenance and service and another set to be out of service due to forced outage or for a major overhaul. This redundant capacity is especially critical for a remote plant where heavy equipment can only be delivered to site for a short period of time in the winter. The generator sets will have a continuous rated capacity in the range of 4.4 MW each. Larger sets cannot be reliably shipped to site over the ice road without complete disassembly of the engines, which is not practical.

To meet the estimated process plant and ancillary average annual load of 24.4 MW, a normal running load of 26.0 MW, and a peak load of 29.4 MW, a total of 9 generator sets are required, each with a nominal continuous rating of around 4.4 MW and a “prime” (short time overload) rating of 4.8 MW. With 7 sets operating, the total continuous capacity is 30.8 MW and the short time “prime” capacity is 33.6 MW. This arrangement provides two redundant generator sets (one permitted to be down for service or maintenance, and one on hot standby to allow for a forced outage) as previously described.

Based on the project delivered fuel price for power generation (fuel cost varies depending on end use and thus tax rate), including O&M and fuel, the total cost of diesel generator power, with no wind power generation contribution is projected to be Cdn$0.300 (US$0.288) per kilowatt hour. This is exclusive of amortization of the plant capital cost, which is included in the total project budget. However, the amortization cost for a diesel power plant is a small component of the total cost of power that is dominated by fuel cost.

SUPPLEMENTAL WIND GENERATION

The wind resource at the Courageous Lake mine site has been studied in detail based on data accumulated from a 60 m tall wind-monitoring tower installed in early July 2009. The resultant data, uploaded via satellite, has been evaluated using industry-leading software to establish the wind resource and generation potential.

Based on the analysis of the Courageous Lake wind speed data, a mean wind power density at 50 m is indicated as being 385 W/m^2, which is defined as Wind Class 3 – “Fair”. As “Fair” covers the range of 300 to 400 W/m^2, the Courageous Lake site is at the upper end of the industry “Fair” classification.

A detailed review of the Project indicates that, although the wind power class is only rated as “Fair” by industry guidelines, it nonetheless represents an attractive supplemental source of energy for the Project. This is due to the fact that the alternative is very expensive diesel generated power, which costs Cdn$0.30/kWh, representing a price point considerably higher than that realized by a typical wind farm selling power to an electric utility.

Calculations based on a selection of actual wind turbines, with equipment maintenance and other appropriate factors included, results in a conservative loss factor of 17.7%. Based on representative turbines, the capacity factor (that is the actual annual generation divided by the generation if the equipment operated at 100% output) is 33%. Turbines providing somewhat higher projected capacity factors are available but this is an economic trade-off to be made at the project implementation stage, when firm equipment price quotations have been received.

A wind farm of 31.5 MW installed capacity has been selected for this study. Based on detailed data analysis, the wind farm will produce 43% of the process plant and mine annual kilowatt-hour energy requirements using a wind turbine of good efficiency optimized for lower wind velocities. It should be noted that a wind farm actually only operates at its rated capacity a small percentage of the time and thus a larger wind farm would further reduce the combined electricity per kilowatt-hour cost, but would also significantly increase the overall capital cost. Further studies are recommended in order to arrive at the optimal wind farm size.

The proposed wind farm has been laid out such that the turbine towers are arranged on rock to minimize foundation requirements. The towers and access roads have also been located so as to minimize environmental impacts.

Turbines in the range of 1.5 MW each have been selected. The current trend is to larger machines as this reduces the per megawatt of capacity-installed unit cost but, in this case, turbines ranging from 1.5 MW to perhaps 1.8 MW each are the largest size that can be reliably shipped to site over the winter road. The supply and installation costs include the cost of a large crawler crane and small hydraulic helper crane, as required for site installation.

COMBINED WIND AND DIESEL GENERATION

The projected total annual energy consumption for the process plant is 213,829 MWh. Based on the proposed wind farm, there would be 91,900 MWh produced by wind and 121,931 MWh by diesel generation. Thus, with a 31.5 MW wind farm consisting of 21 turbines, approximately 43% of the required energy would be provided by wind in an average year.

The total operating costs of the combined power generating facility, including fuel and O&M, will be Cdn$0.184/kWh. The total fuel consumption will be 28,115,860 L/a, which represents a savings of 21,190,592 L of diesel fuel over using solely diesel generation. The required fuel for power generation will represent approximately 525 truckloads, based on the average ice road load.

The diesel power plant output will be automatically coordinated with the wind generation to meet the instantaneous power demand. For combined wind and diesel generation at a remote site, there are issues when the wind output approaches the total plant load, as the governors on the diesel engines are no longer controlling sufficient output (unless spare sets are kept idling on-line) to adequately smooth the wind farm output, which always exhibits some instantaneous fluctuation due to gusting. To address this issue, a module of commercially available high-speed flywheel energy storage equipment has been included.

1.13 Environmental Considerations

1.13.1 *Environmental*

The formal environmental assessment of the Project will commence with preliminary screening of an application to the Mackenzie Valley Land and Water Board (MVLWB) for a Class A Water License, issued under the *Mackenzie Valley Resource Management Act* (*MVRMA* 1998, c. 25). After preliminary screening, the Project will be referred to the Mackenzie Valley Environmental Impact Review Board (MVEIRB), an independent body set up under the *MVRMA* to conduct environmental assessments of projects in the NWT, either by the MVLWB, or any other regulatory agency involved. The environmental assessment is conducted in a number of phases and documentation is submitted to the Minister of Aboriginal Affairs and Northern Development (AAND) for decision making.

Environmental baseline work was initiated at the site by EBA in 2004; Rescan restarted the environmental baseline in the spring of 2010, and a second year of baseline was completed in 2011. In 2012, baseline work is continuing and will address information required to further advance the Project. Results of this baseline work were integrated into mine planning for this PFS.

Seabridge and its team are engaging with local communities and their respective leaders, regulatory agencies, regional, municipal and aboriginal governments, Treaty Nations, and First Nations as part of their efforts to advance the proposed project through the review process.

1.13.2 *Metal Leaching and Acid Rock Drainage*

Waste rock, low grade ore, flotation tailing, and POX residues expected to be produced from mining the Courageous Lake gold deposit were characterized for metal leaching and acid rock drainage (ML/ARD) potential as part of waste and water management planning for the Project. Geochemical characterization tests were initiated in May 2010, with testing ongoing.

Based on results obtained to date, waste is characterized as follows:

- **Waste Rock:** non-potentially ARD generating (non-PAG) but arsenic leaching may require management
- **Low Grade Ore:** uncertain PAG but arsenic leaching will likely require management
- **Flotation Tailing:** non-PAG but arsenic leaching may require management
- **POX Residue:** ARD generating and arsenic, sulphate, and cyanide leaching will require management.

1.14 LOGISTICS

Tetra Tech conducted a high-level logistics study to determine the cost for hauling freight via truck from Edmonton, Alberta, to the property. The property can be accessed on ground via the Tibbitt to Contowyto winter ice road that connects Yellowknife, NWT, with the Diavik and EKATI diamond mines.

Equipment and materials will be procured for delivery to Edmonton, stored, and then transported to the site. Edmonton and Yellowknife are chosen as the staging areas for equipment and materials delivery as they are central trading hubs in their respective regions with established transportation routes that include major highways and rail lines. Trucking services to the site will be provided by transport companies that currently provide services to the operating mines in the area.

Transport services along the ice roads are available on average for a period of nine weeks per year, generally starting from the last week in January until the first week in April. The operating period is heavily dependent on weather and ice conditions. In order to transport goods for the Project, it will be mandatory to sign a Third Party User Agreement with the Tibbitt to Contwoyto Winter Road Joint Venture in advance before any cargo can be dispatched from Yellowknife.

1.15 CAPITAL AND OPERATING COST ESTIMATES

1.15.1 CAPITAL COSTS

The initial capital estimate for the Project is US$1.52 B, based on capital cost estimates developed by the following consultants:

- Tetra Tech – process plant and associated infrastructure costs including general site and plant site preparation, logistics, and tailing and residue storages (based on the material take-offs provided by EBA)
- MMTS – mine capital costs, mine rock storage, and pit area pioneering works
- Brazier – power supply, including diesel power generation and wind power generation, and saline water treatment
- EBA – material take-offs for water management and tailing/leach residue starter dam earthworks, water diversion channel, and airstrip
- Golder – pit depressurization
- Seabridge – Owner’s costs.

All currencies in this section are expressed in US dollars. Costs in this estimate have been converted using a fixed currency exchange rate based on the Bank of Canada three-year average of Cdn$1.00 to US$0.98 (base case). The expected accuracy range of the capital cost estimate is +25%/-15%. This capital cost estimate includes only initial capital, which is defined as all capital expenditures that are required up until the start of gold doré production. A summary of the major capital costs is shown in Table 1.14.

This PFS estimate is prepared with a base date of Q2 2012 and does not include any escalation past this date. Budget quotations were obtained for major equipment. The vendors provided equipment prices, delivery lead times, freight costs to a designated marshalling yard, and spares allowances. The quotations used in this estimate were obtained in Q1/Q2 2012, and are budgetary and non-binding. For non-major equipment (i.e. equipment less than $100,000), costing is based on in-house data or quotes from recent similar projects. All equipment and material costs include Free Carrier (FCA) manufacturer plant Inco terms 2010. Other costs such as spares and freight are covered separately in the Indirects section of the estimate.

Table 1.14 Capital Cost Summary

Area	Description	Capital Cost (US$000)
Direct Costs		
10	Overall Site	59,745
20	Open Pit Mining	96,701
30	Crushing and Stockpiles	83,238
35	Grinding and Flotation	135,039
40	Pressure Oxidation	88,660
45	Thickening, Neutralization, and Cyanide Leaching	38,940
48	Gold ADR Circuit, Cyanide Handling and Electrowinning	14,833
50	Reagents and Consumables	23,536
60	Tailing Management Facility	53,422
65	Water Treatment Plant	8,774
68	Site Services and Utilities	34,352
70	Ancillary Buildings	66,839
75	Airstrip and Loading/Unloading Facilities	12,203
77	Plant Mobile Equipment	3,058
78	Temporary Services	49,085
80	Electrical Power Supply	179,838
88	Yellowknife and Edmonton Facilities	17,227
Sub-total Direct Costs		**965,490**
Indirect Costs		
90	Indirects	315,187
98	Owner's Costs	55,059
99	Contingency	186,703
Total Capital Cost		**1,522,439**

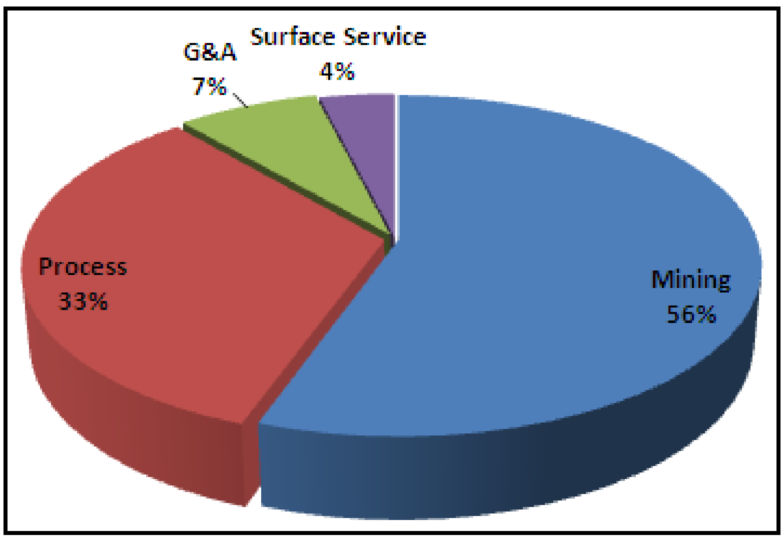
1.15.2 OPERATING COSTS

The operating costs for the Project, as shown Table 1.15, were estimated at US$47.35/t of mineralization processed. The estimate was based on an average annual process rate of 6,387,500 t mineralization milled at a gold grade of 2.20 g/t including dilution. The cost estimates in this section are based on budget prices in Q1/Q2 2012 or based on information from the databases of the consulting firms involved in the cost estimates. When required, costs in this report have been converted using a three-year average currency exchange rate of Cdn$1.00 to US$0.98. All costs are reflected in 2012 US dollars. The expected accuracy range of the operating cost estimate is +25%/-15%.

Table 1.15 Operating Cost Summary

	US$/a (000)	US$/t Milled
Mine	167,620	26.24
Mill	100,420	15.72
G&A	22,300	3.49
Surface Services	12,100	1.90
Tailing Handling	included in sustaining cost	
Total	**302,440**	**47.35**

Figure 1.3 Operating Cost Distribution



The operating costs are defined as the direct operating costs including mining, processing, surface service, and G&A. The power is estimated to be US$0.18/kWh. The power cost is based on the on-site power generation by a combination of diesel power generation and wind power generation. The tailing storage costs are included in the sustaining costs.

1.16 ECONOMIC EVALUATION

An economic evaluation of the Project was carried out by Tetra Tech incorporating all the relevant capital, operating, working, sustaining costs, and royalties (2% of NSR). The evaluation was based on a pre-tax financial model. For the 15-year mine life and 91 Mt inventory, the following pre-tax financial parameters were calculated using the base case gold price:

- 7.3% internal rate of return (IRR)
- 11.2-year payback on US$1.52 B capital
- US$303 M net present value (NPV) at 5% discount value.

The gold price used for the base case is US$1,384.00/oz using the three-year trailing average (as of July 3, 2012).

The revenues projected in the cash flow model were based on the average metal values indicated in Table 1.16.

Table 1.16 Metal Production from Courageous Lake Project

	Years 1 to 5	LOM
Total Tonnes to Mill (000s)	29,433	91,126
Annual Tonnes to Mill (000s)	5,887	6,075
Average Grade		
Gold (g/t)	2.170	2.205
Total Production		
Gold (000s oz)	1,836	5,777
Average Annual Production		
Gold (000s oz)	367	385

Two additional metal price scenarios were also developed using the spot metal price on July 3, 2012 (including the closing exchange rate of that day), and using an alternate gold price of US$1,925/oz (Table 1.17).

Table 1.17 Summary of the Economic Evaluations

	Unit	Base Case	Spot Price Case	Alternate Case
Metal Price				
Gold	US$/oz	1,384.00	1,617.50	1,925.00
Exchange Rate	US$:Cdn$	0.9803	0.9877	0.9877
Economic Results				
NPV (at 0%) *	US$ M	1,507	2,785	4,519
NPV (at 5%)	US$ M	303	1,054	2,080
IRR	%	7.3	12.5	18.7
Payback	years	11.2	7.4	4.0
Cash Cost/oz Au	US$/oz	780	789	796
Total Cost/oz Au	US$/oz	1,123	1,134	1,141

* undiscounted cash flow.

1.16.1 Sensitivity Analysis

Sensitivity analyses were carried out on the following parameters:

- gold price
- exchange rate
- initial capital expenditure
- on-site operating costs.

The analyses are presented graphically as financial outcomes in terms of NPV and IRR. Both the Project NPV and IRR are most sensitive to gold price and exchange rate followed by operating costs, with initial capital having the least impact. The NPV and IRR sensitivities can be seen in Figure 1.4 and Figure 1.5.

Figure 1.4 Base Case Sensitivity to NPV at 5% Discount Rate

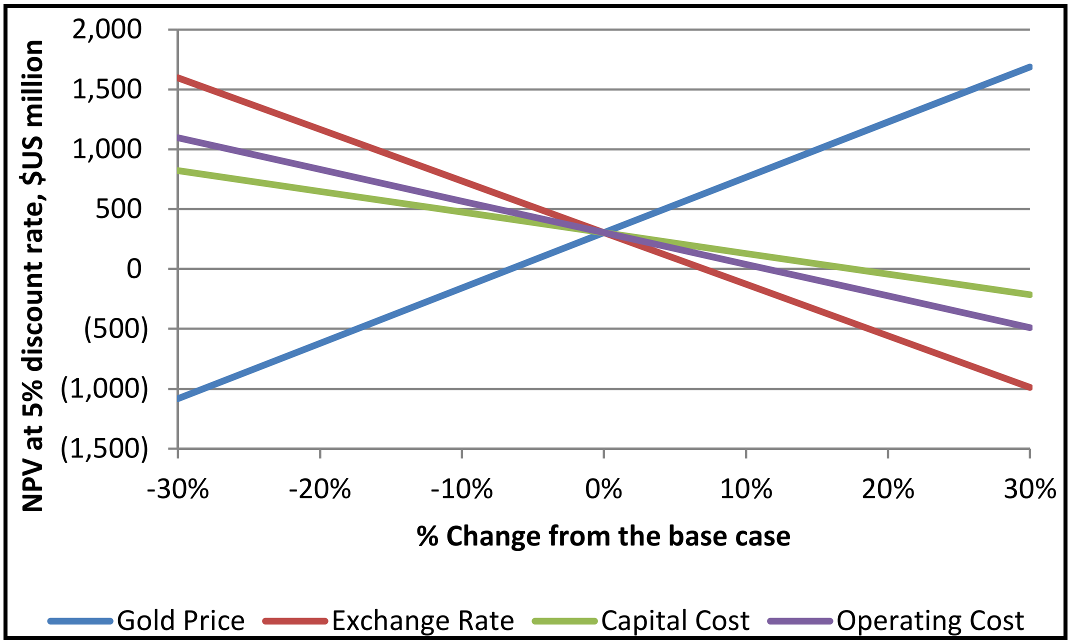
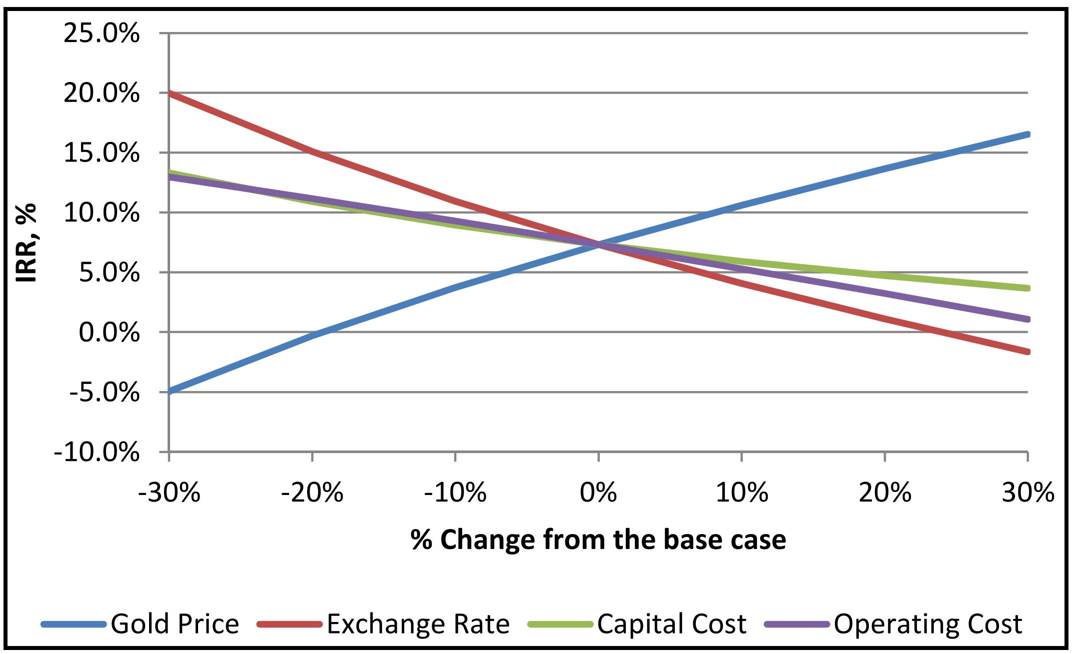


Figure 1.5 Base Case Sensitivity to IRR



1.17 Project Development Plan

It is estimated that the Project would take approximately six and half years to complete permit applications and construction activities, starting with the completion of this PFS. A high-level project schedule is provided in Section 18.0 of this PFS.

1.18 Conclusions

The Courageous Lake deposit represents a large gold resource that is amenable to open pit mining and conventional milling consisting of flotation concentration, concentrate POX, and gold extraction by cyanidation. Mineralization occurs as lenses and discontinuous sheets in silica ± carbonate altered felsic volcanics. A total of 13 distinct mineral zones have been identified by surface and underground diamond drilling.

RMI estimated mineral resources using data that have been verified by comparing electronic assay grades against signed certificates. The estimated block grades were classified into Measured, Indicated, and Inferred categories using distance to drilling data. It is the opinion of RMI that the gold resources stated in Table 1.2 satisfy the requirements of NI 43-101.

The metallurgical test work indicated that the mineralization responds well to the process consisting of conventional flotation, concentrate pressure oxidation and cyanidation. The gold extraction by cyanidation is high, ranging from 94 to 99%, when the flotation concentrate is pressure oxidized under the standard conditions practiced in the pressure oxidization industry. The overall gold recovery is projected to be approximately 89.4% on average.

1.19 Opportunities and Recommendations

There are hydropower options for the Project's power supply. The opportunities represent a reliable, sustainable, and clean energy source that would significantly reduce the requirement for diesel fuel at the site. A prefeasibility level assessment is currently underway with the fieldwork being carried out in the summer of 2012. When the report is completed in early 2013, the applicability of this option will be better understood and feasibility level studies will be considered.

Under the current design, access to the Project is by winter ice road, which is limited to less than three months per year. It is during this period that almost all of the project's supplies are transported to site. The Tibbitt to Contwoyto Winter Road Joint Venture investigated extending the winter road seasonal use by at least another month with a 150 km extension from the permanent road access at Tibbitt Lake to Lockhart camp. While this would result in some reduction in both operating and capital costs for Courageous Lake, an all-season access road from the Bathhurst Inlet would provide considerably more benefit to Courageous Lake economics. Site access improvements would significantly reduce on-site storage requirements, especially fuel oil and reagents such as lime. Seabridge will continue to investigate these options as the project moves forward.

The size and geometry of the Courageous Lake orebody, as well as the high capital impact of throughput and mine life, make the impact of the discounted cash flow economics important in determining an optimized economic pit limit. The current study, capital costs, and 15-year mine life are a good basis to evaluate the discounted cash flow cases. It would be difficult to use a Gemcom Whittle™ type of analysis, since the orebody does not produce even expansion increments as it deepens, and the fixed component of capital and operating costs is high due to the high Arctic location. Instead, different cases will need to be designed and full cash flows calculated, to determine meaningful economic comparisons. This analysis can also include combined open pit and underground options.

Based on the work carried out in the PFS and the resultant economic evaluation, this study should be followed by either an updated PFS or Feasibility Study in order to further assess the economic viability of the Project.