33 Alternative Means of Undertaking the KSM Project

33.1 Introduction

This chapter describes the processes and criteria that Seabridge Gold Inc. (Seabridge) and its consultants have used to select preferred options from among a variety of alternatives means of developing the KSM Project (the Project). The assessment of alternatives demonstrates the key decisions that Seabridge has made to undertake mining activities that, in aggregate, minimize adverse effects and maximize beneficial environmental, cultural, and socio-economic effects, while also remaining technically and economically feasible. For this report, "alternatives" refer to the functionally different design specifications or facility locations considered as potential options for the Project.

This evaluation of the alternative means of undertaking the proposed Project meets the requirements of Section 16(2b) of the *Canadian Environmental Assessment Act* (1992) and the Comprehensive Study Scope of Assessment (CEA Agency 2010) as a factor to be considered in the scope of the comprehensive study environmental assessment, in addition to satisfying the information requirements as outlined in the British Columbia (BC) Application Information Requirements (AIR) document (BC EAO 2011). As described in the Operational Policy Statement Addressing "Need for" "Purpose of," "Alternatives to," and "Alternative Means" under the *Canadian Environmental Assessment Act,* "alternative means" are the various technically and economically feasible ways to implement a project (CEA Agency 2007).

Throughout the process of developing the KSM Project since 2008, Seabridge has made numerous decisions and taken steps to make iterative improvements on available development options based on Project economic, technical, environmental, and social criteria. Only the major decision-making processes to the components suggested by the AIR are reported on in this chapter, including:

- mining method;
- mine production rates and development schedule;
- tailing management;
- process plant location;
- access routes;
- ore handling;
- ore concentrate transport;
- gold recovery method;
- waste rock disposal;
- water management, including:

  - discharge direction and timing from the Tailing Management Facility (TMF);
  - Water Storage dam (WSD) type;
  - mine contact water treatment;
  - selenium treatment; and
- ore comminution.

In addition to the above alternatives assessments, a summary of the decision-making process that did not involve consideration of alternatives is also provided for: closure and reclamation, employee work schedules, employee living conditions, and power supply.

33.2 Methods

The assessments in this chapter were conducted using a decision-making framework to systematically evaluate alternatives to identify the best means of undertaking the Project. This approach is similar to that used in the Mt. Milligan (AMEC 2008) and Victor Diamond (AMEC 2004) alternatives assessments, and it is also based on formal decision-making theory as outlined in the Guidebook to Decision-making Methods commissioned by the US Department of Energy for decisions on nuclear management (Baker et al. 2001; Fülöp 2005).

33.2.1 Basic Decision-making Process

The decision-making framework outlined below provides the general background context for the specific decision-making tools used in different sections of this assessment report.

33.2.1.1 Decision-making Steps

The assessment process for the Project follows an analytical reasoning system involving the steps described below (Baker et al. 2001; Fülöp 2005).

<u>Step 1: Define the Problem</u>

This step involves identifying a clear purpose for the alternatives assessment to be conducted, identifying relevant background information, assumptions, and issues involved for the specific component being assessed. The background and purpose of each assessment in this chapter is presented at the start of each assessment.

<u>Step 2: Identify Requirements</u>

In this step, requirements that the chosen alternative *must* meet are identified based on applicable law, policy, codes, industry standards, stakeholder input, and criteria set for the Project by the proponent, AIR/Comprehensive Study Scope of Assessment, or best professional judgment. In the interests of maintaining brevity, requirements are not typically listed for the individual assessments, but the table in Appendix 33-A provides a representative list of the kinds of legislative requirements used for the Project alternatives assessments. Obvious requirements are also typically used to screen a preliminary long list of candidates in order to narrow down the alternatives to be assessed, and during the assessment, requirements further help to eliminate those alternatives found to be unfeasible.

Step 3: Identify Goals

This step goes beyond what the alternatives must achieve to identifying what the *preferred* criteria are, which will enable further ranking of alternatives. For instance, when all options meet environmental requirements, setting the goal to minimize environmental effects allows a preference-based ranking to choose the best candidate. Similarly to the requirements in Step 2, the table in Appendix 33-A also lists some of the goals based on ecological benchmarking and best practice studies that were used in the alternatives assessments. The performance objectives tool (Step 6) incorporates requirements and goals into the performance objectives themselves.

Step 4: Identify Alternatives

In this step, the feasible alternatives to be assessed are identified. For some of the more important assessments, a pre-feasibility discussion will also be included to outline how feasible alternatives were initially chosen as well; otherwise it is assumed that basic technical and economic considerations were applied in order to identify preliminary feasible alternatives. In this report the identification of alternatives will be done in a clear section prior to their assessment.

Step 5: Identify Attributes (Criteria)

In complex decision making, often no one alternative will emerge immediately that will achieve all the stated objectives (requirements and goals) of a project, and so it becomes necessary to set specific indicative measures against those objectives to allow for a systematic method to assess and compare alternatives. These measures against the larger goals and requirements are the criteria or *attributes* used in the decision-making process. Attributes should ideally meet the following criteria:

- **Complete** – Attributes are identified for all objectives initially, so as to meet the principle of a complete assessment that does not neglect certain areas.
- **Meaningful/relevant** – Attributes should provide a qualitative/quantitative measure of central operational relevance to stated objectives (i.e., not be irrelevant).
- **Few in number** – Attributes should consist of what is necessary and sufficient to make a reasonable decision that will satisfy objectives.
- **Comparable** – Attributes should provide a means to discriminate between alternatives (e.g., if it turns out that the attribute identified will be the same for all alternatives, then it should not be used for the given assessment).
- **Non-redundant** – Sometimes an attribute will be identified that applies to multiple objectives; in this case, the final comparison should list that attribute only once to avoid double counting (e.g., the attribute of road length may serve as a measure of what would minimize environmental adverse effects, lower costs, and reduce technical work requirements, but in the final assessment it should be presented only once to avoid double counting).

Identifying attributes typically requires expertise to identify the necessary criteria that will allow for a meaningful comparison and selection of alternatives based on the variety of requirements and goals that apply to a given assessment. At times, goals and attributes may be complementary, as in the case of road length (discussed above), or involve trade-offs, in which case professional judgment and, if necessary, consultation may be necessary to further rank goals against each other.

In this report, attributes typically appear in the sections on the comparison of alternatives, and are listed in the final comparison summary tables for assessments employing the performance objectives tool.

Step 6: Select a Decision-making Tool

There are several tools that can be used in formal decision making. For the KSM Project, three main tools were used: (1) a performance objectives approach for most of the major Project decisions, (2) a software-assisted approach for decisions related to the mine production rate and development schedule, and (3) a multiple accounts analysis (MAA) approach for TMF siting and the selection of the TMF access road. The performance objectives methodology, which is used in most assessments, is outlined briefly below in Section 33.2.2. The MAA approach (used in Sections 33.5 (Tailing Management) and 33.7.3 (Processing and Tailing Management Access) is outlined in Section 33.5, and the software-assisted approach is described in Section 33.4 (Mine Production Rates and Development Schedules).

The final sections (33.15 to 33.18: Closure and Reclamation, Employee Work Schedules, Employee Living Conditions, and Power Supply, respectively)—which were suggested for inclusion by the AIR—did not involve alternatives assessments. Instead, best management practices (BMPs) were used to determine the means to carry out these components for the Project, as outlined in their respective sections.

Step 7: Apply the Tool

This step culminates in evaluating alternatives against the attributes chosen in Step 5. Methods can be qualitative, quantitative, or both. Regardless of the tool used, this step involves characterizing and then ranking and evaluating the attributes for each of the alternatives in a systematic manner in order to ultimately identify the most preferable alternative.

This step is broken down in the report between the descriptions of the alternatives identified and the alternative comparison sections. The attributes of the alternatives are compared in this step and ranked in order to first eliminate candidates with unfeasible attributes and then to select the preferred candidate. This step is represented in the comprehensive summary tables that are found for the alternatives evaluated by the performance objectives tool.

Step 8: Validate Solutions against Purpose Statement

It is important to review the results of a decision-making process against the original purpose to ensure that it has been properly addressed. The final solution should meet the desired outcome condition, satisfy requirements, and best achieve the goals of the decision makers and stakeholders involved.

For the KSM Project, this review step took place for different alternatives as a process of cross-checking between Seabridge, numerous experienced firms retained for expert and scientific work on the Project, and/or the Working Group, and/or other consultation processes. For some sections involving substantial decisions about the Project (i.e., decisions involved in the MAA assessments), this validation step resulted in a more iterative consultation process incorporating feedback from the Working Group and government. To simplify this document, the main decision-making process is the one reported on; the long and detailed iterations are excluded, in line with the decision-making principle of relevance described below.

33.2.1.2 Decision-making Principles

In order to arrive at and communicate defensible and credible conclusions for the alternatives assessments for the KSM Project, there were also several principles[1] applied to the basic decision-making process. These principles include the following, where applicable:

- **Conservatism** – In order to help manage risk and uncertainty, estimates of beneficial effects err on the low side and adverse effects err on the high side for a given alternative.
- **Completeness** – Primary information used to characterize attributes to assess alternatives should be as complete as possible before being assessed.
- **Consistency** – Similar methodologies should be applied in the assessment of alternatives to the extent possible.
- **Relevance** – The data and information used in the assessment of alternatives should be necessary and sufficient, reliable, and relevant to the decision-making process.
- **Accuracy** – The best available data and information should be gathered and analyzed by the relevant experts in their field for use in the assessment.
- **Engagement** – Appropriate and timely access and input to the alternatives analysis process should be provided for relevant community members and Aboriginal groups.
- **Transparency** – Preliminary data, methodology, assessment, and decisions should be transparently reported as part of the environmental assessment.

33.2.2 Performance Objectives and Attribute Ranking System

Four **performance objectives** were selected for the Project that reflect *Canadian Environmental Assessment Act* (1992) requirements and goals to select alternative means that are technically and economically feasible and minimize environmental effects and related social effects (CEA Agency 2007):

1. **Environmental performance objective** – To meet regulations for and minimize adverse effects (and/or maximize positive effects) on valued components (VCs) in terrestrial, atmospheric, and aquatic systems affected by the Project.

[1] These principles have been adapted from economic accounting conventions, life cycle analysis, and ISO standards for greenhouse gas accounting.

2. **Social performance objective** – To meet applicable regulations for and minimize adverse effects (and/or maximize positive effects) primarily on social VCs (e.g., cultural, Aboriginal, economic, heritage, archaeological, health, and aesthetic components) as well as personnel considerations (e.g. occupational health and safety [OH&S]).
3. **Technical performance objective** – To meet Project design criteria as well as industry and/or regulatory standards and best practices.
4. **Project economic performance objective** – To be supported by Project economics, minimize costs, and/or allow for a positive return on investment.

33.2.2.1 Summary Comparison Table System

A summary comparison table is provided in each performance objective assessment that evaluates attribute characteristics for each alternative against the four performance objective categories. The summary comparison tables list each attribute as either an advantage (✓) or disadvantage (x). Attributes are further characterized as being **preferred**, **acceptable**, **challenging**, or **unfeasible**—depending on how well they meet the requirements and goals for each performance objective—using the rationale and colour scheme provided in Table 33.2-1.

Table 33.2-1. KSM Project Alternatives Attribute Rating System

Attribute Ranking against Environmental and Social Performance Objectives	
● Preferred	Attribute has the least adverse effects on VCs *without mitigation* when compared to other alternatives' attributes; may also provide positive benefits.
● Acceptable	Attribute minimizes adverse effects on VCs *with mitigation.*
● Challenging	Attribute has significant adverse effects on VCs, and there are technical, financial, or other *barriers to mitigation.*
● Unfeasible	Attribute has unacceptable adverse effects on VCs that *could not be* reasonably *mitigated.*
Attribute Ranking against Technical Performance Objective	
● Preferred	Attribute is the most likely to be effective to implement, with the lowest risk, and contingencies (mitigation) in place to address risks.
● Acceptable	Attribute is likely to be effective to implement, with contingencies to address risks.
● Challenging	Attribute's effectiveness faces significant barriers to implement, or to reduce risk to acceptable levels, even with contingencies.
● Unfeasible	Attribute's effectiveness faces unacceptable risk, even with contingencies, or is unfeasible to implement.
Attribute Ranking against Project Economic Performance Objective	
● Preferred	Attribute has the lowest costs or gives the best return on investment.
● Acceptable	Attribute has reasonable costs or gives an acceptable return on investment.
● Challenging	Attribute has high costs leading to budgetary issues.
● Unfeasible	Attribute is not economically viable under Project budgets.

Notes:
This table provides an interpretation key for the summary tables presented in the chapter, and it is suggested that it would be useful for the reader to print it out for that purpose.
Source: Adapted from Baker (2001) and AMEC (2004, 2008).

After the attributes are rated using the above method, each alternative is then evaluated as a whole, receiving an overall rating of **preferred**, **acceptable**, **challenging**, or **unfeasible** depending on how its attribute ratings compare with those other alternatives, as follows:

- If any attribute for an alternative is **unfeasible**, this rating serves as a fatal flaw, and the alternative as a whole is rated as **unfeasible** and is then eliminated from further consideration.
- For an alternative to be rated as **preferred** overall, it must contain at least one attribute characterized as **preferred** along with, at worst, **acceptable** ratings for *all* its other attributes (i.e., alternatives with any attributes rated as ***challenging/unfeasible*** will not receive ***preferred*** rank).
- When preference between alternatives does not clearly emerge from the assessment, priorities based on the Project requirements and goals as well as professional judgment are used to select alternatives, and a rationale for the decision is provided in the accompanying text.

33.3 Mining Method

33.3.1 Purpose and Background

The mining method chosen for a project affects several other aspects of mine development such as production rates, development schedules, and waste rock volume. The two main methods for recovering ore from hard rock mines are open pit and underground mining. Both methods use drilling, blasting, and heavy equipment, but have different environmental, social, technical, and economic considerations.

Open pit mining is the industry standard practice for metal mining in BC (especially for low grade deposits with similar characteristics to the KSM Project), as reflected in the KSM Prefeasibility Study Update 2011 (Wardrop 2011). The decision to undertake underground mining instead of open pit mining is constrained by technical and economic considerations based on the deposit position, type, and grade of ore. These factors influence the potential production rates that can be achieved and ultimately determine the feasibility of underground mining. Seabridge used Gemcom's Footprint Finder software to evaluate deposit resources and determine the economic viability of underground mining. The potential effects of the mining method on the surrounding environment and human systems were also taken into account.

33.3.2 Alternatives Identification

The Project consists of four separate ore deposits—the Kerr, Sulphurets, Mitchell, and Iron Cap zones (Chapter 4, Figure 4.4-2). For each zone, Seabridge undertook analyses of deposit ore types, respective cut-off grades, production rates, and economic returns that could be achieved to mine the ore through either open pit or underground mining.

33.3.2.1 Open Pit Mining

Open pit mining is ideal for extraction of ore bodies that extend from the surface to considerable depths and have substantial horizontal dimensions with relatively little overburden. The method is flexible, allowing for large variations in production schedules at relatively short notice, and can be highly mechanized, making open pit mining the most productive mining method. Given favourable stripping ratios and climatic conditions, open pit mining produces ore at a fraction of the cost of underground mining. The method requires fewer workers, and has a lower accident frequency rate than underground operation. Open pit mines are developed by excavating rock, starting with overburden in the first phase, and then along a series of regularly spaced horizontal lifts/benches to access the ore in the second phase. The amount of overburden often accounts for the higher amount of waste rock produced by this method in comparison to underground mining. Access roads and ramps connect the benches, which allow haulage trucks to remove materials from the pit as it is deepened. Mining of the pit requires careful planning so that sufficient overburden and waste are stripped from the area to permit mining to proceed at an even pace. If the pit is developed below the groundwater table, groundwater must be pumped out to allow mining to proceed.

The lower ore grades and placement of the Kerr, Sulphurets, Mitchell, and Iron Cap deposits make open pit mining the base case for the KSM Project. The extent of pit limits for open pit mining is based on gold prices, which determine cut-off grades of ore found in each deposit. The open pit limits were determined in this way for the Kerr, Sulphurets, Mitchell, and Iron Cap deposits, which are planned to be developed through typical hard rock bulk mining methods involving large truck/shovel operation, at a production rate of 130,000 tonnes per day (tpd).

33.3.2.2 Underground Mining

Underground mining is generally more selective, producing less waste rock than open pit mining and posing fewer surface risks, such as avalanches. However, underground mining is also associated with greater equipment needs, longer worker hours to retrieve ore, and additional expenditures for air ventilation, electricity, and water pumping, resulting in higher overall costs.

There are several types of underground mining, including block caving, panel caving, and sublevel caving. Block caving is a bulk underground mining method used for massive low-grade ore bodies (such as those at the KSM Project) that are steeply dipping and have high friability. Other underground caving methods are panel caving and sublevel caving. The benefits of block caving are reflected in a recent study that found that 41% of international block caving mining is conducted in North America, typically on copper (29% of all caving by mineral) and gold (15% of all caving by mineral; Woo, Eberhardt, and Van As 2009).

The large tonnages and relatively low grades of the KSM Project deposits dictate that low cost bulk mining methods must be used in order to extract the copper and gold mineralization profitably. Block caving was determined to be the most effective and appropriate underground mining method to consider as an alternative to open pit methods for the Project. Underground mining would significantly reduce the pit limits involved in the open pit only scenario, leading to less surface disturbance and waste rock production than open pit mining.

33.3.3 Comparison of Alternatives

The underground and open pit mining attribute characteristics were compared against the four performance objectives as described in the following text and summarized in Table 33.3-1.

Seabridge evaluated mining methods for each of the four deposits individually, reflected in Table 33.3-1 where certain attributes are characterized separately for different deposits, or different sections of a deposit, as was the case with the upper and lower Mitchell deposit.

33.3.3.1 Technical and Project Economic Considerations

The main technical aspects that influence decisions on whether to utilize underground versus open pit mining methods include surface topography, depth to the top and bottom of the ore zone, plunge and dip of the deposit, ground conditions surrounding the ore zone, present and future production requirements, method of mining and stope development, equipment fleet and ventilation requirements (Association for Mineral Exploration British Columbia 2009). In general, due to simpler engineering requirements (with related improved economics), open pit mining is technically preferred over block caving, as indicated in Table 33.3-1. Testing for the feasibility of conducting block caving revealed that, while the Iron Cap and Mitchell deposits had geometries with good caving potential, the Sulphurets and Kerr deposit geometries are not conducive to block caving (Wardrop 2011). For this reason the Sulphurets and Kerr deposits have been marked as **unfeasible** for block caving in Table 33.3-1.

Financial criteria that were considered in the evaluation between open pit and underground mining focused on a comparison of capital expenditures (CAPEX) and operating expenditures (OPEX). CAPEX reflects start-up equipment and excavation estimates, while OPEX includes labour and production costs. Studies of CAPEX and OPEX projections for the Project concluded that due to the type and location of four ore bodies, open pit mining would minimize costs compared to block caving, and that block caving would be prohibitive for all but the Iron Cap and lower part of the Mitchell deposits (Wardrop 2011).

The result of the technical assessment is that both the Kerr and Sulphurets deposits are considered unfeasible to mine though block caving due to incompatible geometries. The Mitchell deposit could technically be mined through open pit or block caving methods, but due to financial constraints, open pit mining would have to be used down to a pit floor elevation of 390 m to recoup CAPEX, after which open pit mining could be continued, or block cave mining could follow from the underground as an alternative to open pit (Golder 2012a). The Iron Cap deposit is technically amendable to either open pit or block caving and would be economically viable to be mined either way, although open pit is more economically favourable.

33.3.3.2 Environmental and Social Considerations

The key environmental attribute considered when comparing open pit and underground block caving methods is the difference in the Project footprint between methods, and resultant differences in (1) waste rock volume and related metal leaching and acid rock drainage (ML/ARD), and (2) disruption of biotic systems, as listed in Table 33.3-1. Block caving also reduces above-ground noise, fugitive dust, and particulate emissions compared to open pit mining (potentially improving worker health and safety), and is more amenable to surface reclamation than open pit mining. One of the environmental challenges associated with the mining method for the Project is the glacier ice over the Iron Cap deposit; this ice would have to be removed if open pit mining were pursued, while block caving would allow the ice to be left in situ.

The surface footprint of the Project is largely driven by the spatial extent of the open pits and rock storage facilities, as shown in Figures 33.3-1 (old 2011 open pit only alternative) and 33.3-2 (new combined open pit/underground alternative). Figure 33.3-3 illustrates the difference in rock storage facilities and pit limits between the two alternatives.

Table 33.3-2 lists the overall quantitative footprint difference between the pits and rock storage facilities (RSFs) for the Project, with an overall footprint reduction of about 710 hectares (ha) resulting from the switch to underground mining for the Iron Cap and lower Mitchell deposits. The exposed height of pit wall in the Mitchell Pit is also reduced due to the addition of block caving of this deposit—from a 1,600 m pit wall height to 1,200 m (25% reduction). This overall reduction in surface disturbance and amount of exposed rock (the latter decreasing associated ML/ARD conditions) is anticipated to reduce potential adverse effects to vegetation, wildlife, soil, water quality, and downstream aquatic life and fish.

Note that the subsidence zone for the Mitchell Underground Works (Figure 33.3-2) is projected to only disturb surface area that is already within the open pit diameter. Modelling for the Iron Cap subsidence zone has resulted in a projected size of about 96 ha (Figure 33.3-2), where slumping may increase surface disturbance and amount of rock exposure; however, this surface disturbance will be much less than that from the previous Iron Cap Open Pit, and the disturbance of subsidence slumping effects will not lead to mass clearing of vegetation and soil and burning the way open pit mining does, and may even increase some habitats. The Sulphurets RSF footprint has also been eliminated in the new footprint (Figure 33.3-3), but a small portion of the original space will be used as a temporary laydown area.

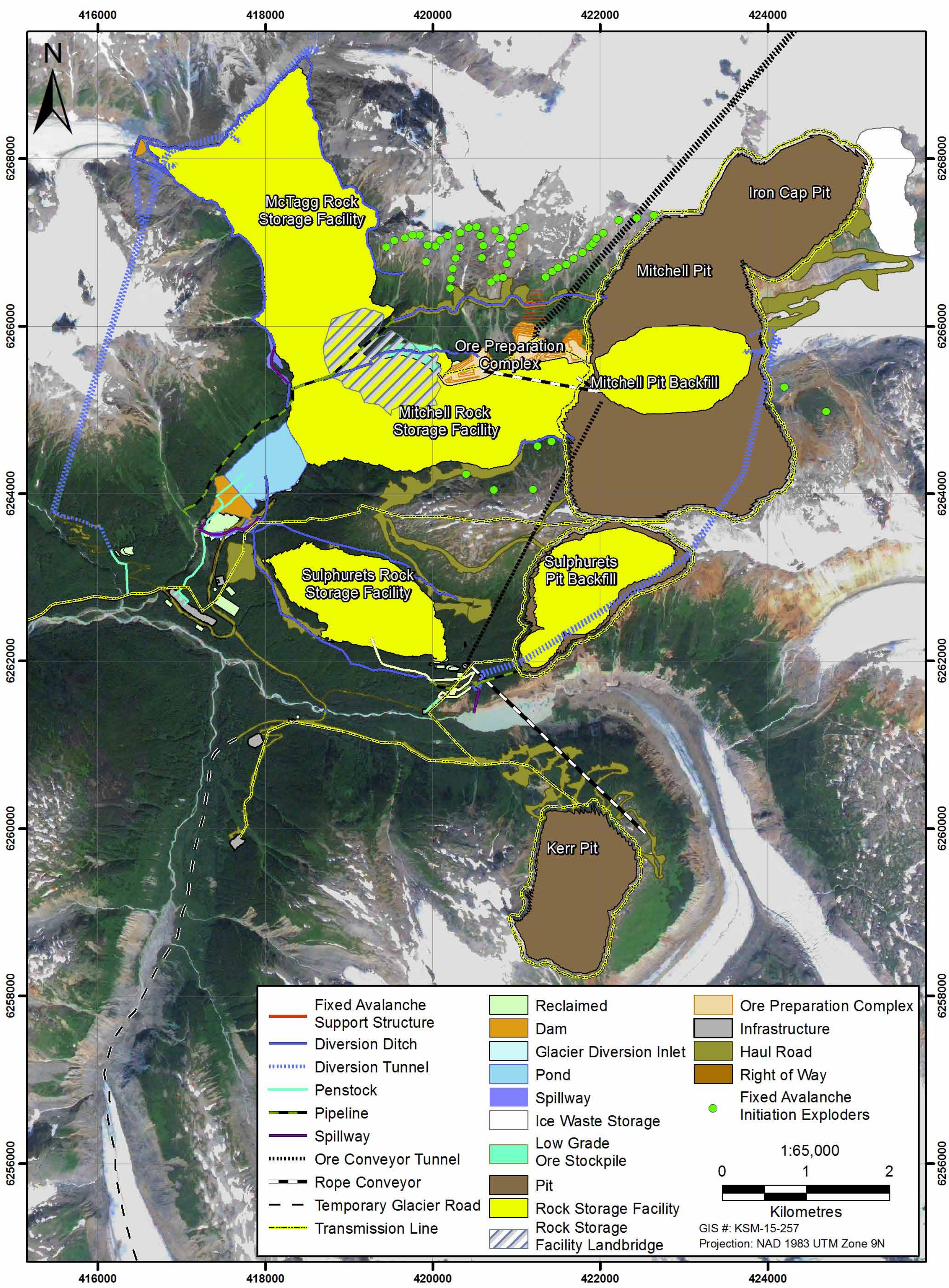
The potential for ML/ARD at mine sites is correlated with the volume of waste rock and the areal extent of rock surfaces exposed to weathering (water and oxygen) conditions (refer to Section 33.11 for additional detail). Waste rock volumes are largely a function of strip ratio, which is improved by 40% (i.e., lowered from 2.7 to 1.5) by employing the 2012 hybrid scenario of underground and open pit mining methods, resulting in a reduction of an estimated 2.6 Bt of waste rock (see Table 33.3-3), and thereby avoiding the potential for additional ML/ARD effects. Based on these environmental considerations, block caving is the environmentally preferred mining method for the Project.

Social attributes considered for mining method alternatives included air quality and noise effects on personnel, wages, aesthetics, and potential effects on fisheries resources. Open pit mining is the preferred method when considering direct OH&S of mine personnel. Conversely, block caving is preferred over open pit mining when considering water quality, since block caving produces lower waste rock volumes, leading to the generation of fewer untreated water quality contaminants. Therefore, the potential for effects on recreational, commercial, or Aboriginal traditional downstream fisheries resources would possibly be reduced through the use of block caving. There is limited fishing known to occur in the Canadian portion of the Unuk River due to rugged terrain and lack of access. Fishing may be more prevalent in the United States portion of the Unuk River, though fisheries resources on the United States side of the river are less likely to be affected by potential changes in water quality, since this area is further afield of the Mine Site.

Table 33.3-1. Summary Comparison of Mining Method Alternatives for Kerr, Sulphurets, Mitchell and Iron Cap Deposits

Alternative	ATTRIBUTE RATINGS AGAINST FOUR PERFORMANCE OBJECTIVES				OVERALL ALTERNATIVE RATING
	Environmental	**Social**	**Technical**	**Project Economic**	
Open Pit	x ● Larger surface footprint and habitat disturbance with more potential effects on biota; reduced to acceptable levels through mitigation x ● Larger waste rock and tailing volume x ● Potential ML/ARD changes to surface water from waste rock; mitigate to acceptable levels through water treatment x ● More noise, dust, and air quality changes from blasting with potential effects on aquatic and terrestrial life; reduced to acceptable levels through mitigation x ● **Iron Cap only**: Would be problematic to strip and store large volume of ice from above mineralized zone x ● Pits and RSFs not amenable to reclamation to previous state after closure, but some quality habitat can be recovered and a pit lake will form	✓ ● More attractive work environment and more opportunities for local work force x ● Reduced wage compared to underground as less skill required ✓ ● Fewer OH&S risks than for underground mining ✓ ● Fewer air quality and noise risks from blasting and vent fan noise to personnel in outside open spaces x ● Less appealing aesthetic view results in general from open pit mining x ● More waste rock may lead to increased effects on water quality and any downstream commercial, recreational, or traditional subsistence fishery resources	✓ ● Simpler engineering requirements for design and equipment ✓ ● Easier to accommodate scheduling changes x ● Mine water balance more influenced by precipitation, surface and groundwater ingress, and high sediment loads; will require pumping x ● More waste rock requiring handling as well as ice	✓ ● Fewer person-hours and Project life span reduces general costs per unit of production ✓ ● Lower infrastructure and equipment costs ✓ ● Lower general costs to mine most Project ore x ● Increased costs for waste excavation, loading, and hauling x ● **Iron Cap**: Increased costs for ice excavation, loading, and hauling	**ACCEPTABLE** (Mitchell, Kerr, Sulphurets) **CHALLENGING** (Iron Cap)
	Acceptable (Mitchell, Kerr, Sulphurets); Challenging (Iron Cap deposit only)	Acceptable (All deposits)	Preferred (All deposits)	Preferred (All deposits)	
Block Caving	✓ ● Lower surface footprint so less habit disturbance and fewer potential effects on biotic systems ✓ ● Waste rock and tailing volume is significantly reduced (by about 2 billion tonnes) ✓ ● Less ML/ARD effects and related treatment than from open pit due to less waste rock volume ✓ ● Fewer noise, dust, and air quality effects from blasting therefore fewer effects on aquatic and terrestrial life, although vent fan noise requires mitigation x ● Potential subsidence may cause topographic habitat disturbance and expose rock, leading to acid rock drainage, but ML/ARD risks would still be less than for open pit ✓ ● Less surface area affected, even with subsidence, and more amenable to reclaim to condition that looks less disturbed	x ● Less attractive environment for labour force and fewer skilled workers available ✓ ● Higher wage rate available to skilled workers x ● Higher OH&S risks than for open pit, but can be managed with BMPs x ● Higher air quality (dust/noise) OH&S risks from blasting in underground mining spaces; reduced to acceptable levels with ventilation as mitigation x ● Vent fan noise OH&S effects; reduce to acceptable levels through mitigation ✓ ● Block caving's reduction in disturbed footprint results in fewer potential adverse effects to views compared to open pit ✓ ● Less waste rock may lead to reduced effects on water quality and any downstream commercial, recreational, or traditional subsistence fishery resources	x ● **Sulphurets and Kerr**: deposit geometry is not conducive to block caving ✓ ● Mitchell and Iron Cap deposits have suitable geometry for block caving x ● More complicated engineering requirements for design and equipment x ● Not as flexible to accommodate scheduling changes ✓ ● Mine water balance more stable but will also require pumping x ● Subsidence poses risks to infrastructure that must be predicted and mitigated	x ● **Upper Mitchell**: Costs of block caving (equipment, labour, mine life, etc.) are very high compared to open pit mining x ● Block caving costs are generally higher per unit production than open pit mining ✓ ● Specific cost-benefit analysis indicates that Iron Cap and lower Mitchell deposits can be profitably mined through block caving	**PREFERRED** (Iron Cap and Lower Mitchell) **CHALLENGING** (Upper Mitchell) **UNFEASIBLE** (Sulphurets and Kerr)
	Preferred (All deposits)	Preferred (All deposits)	Acceptable (Mitchell and Iron Cap); Unfeasible (Sulphurets and Kerr)	Acceptable (All deposits except Upper Mitchell); Challenging (Upper Mitchell)	

Notes:
✓ = advantage, x = disadvantage, ● = Preferred, ● = Acceptable, ● = Challenging, ● = Unfeasible; See Table 33.2-1 for attribute ranking specific definitions.
ML/ARD= metal leaching and acid rock drainage; OH&S=occupational health and safety; BMP=best management practice; RSF=rock storage facility.



SEABRIDGE GOLD
KSM PROJECT

OLD Footprint - 2011 KSM Project Open Pit Mining - Pit Limits and Waste Rock at End of Operation

Figure 33.3-1

Rescan
Engineers & Scientists

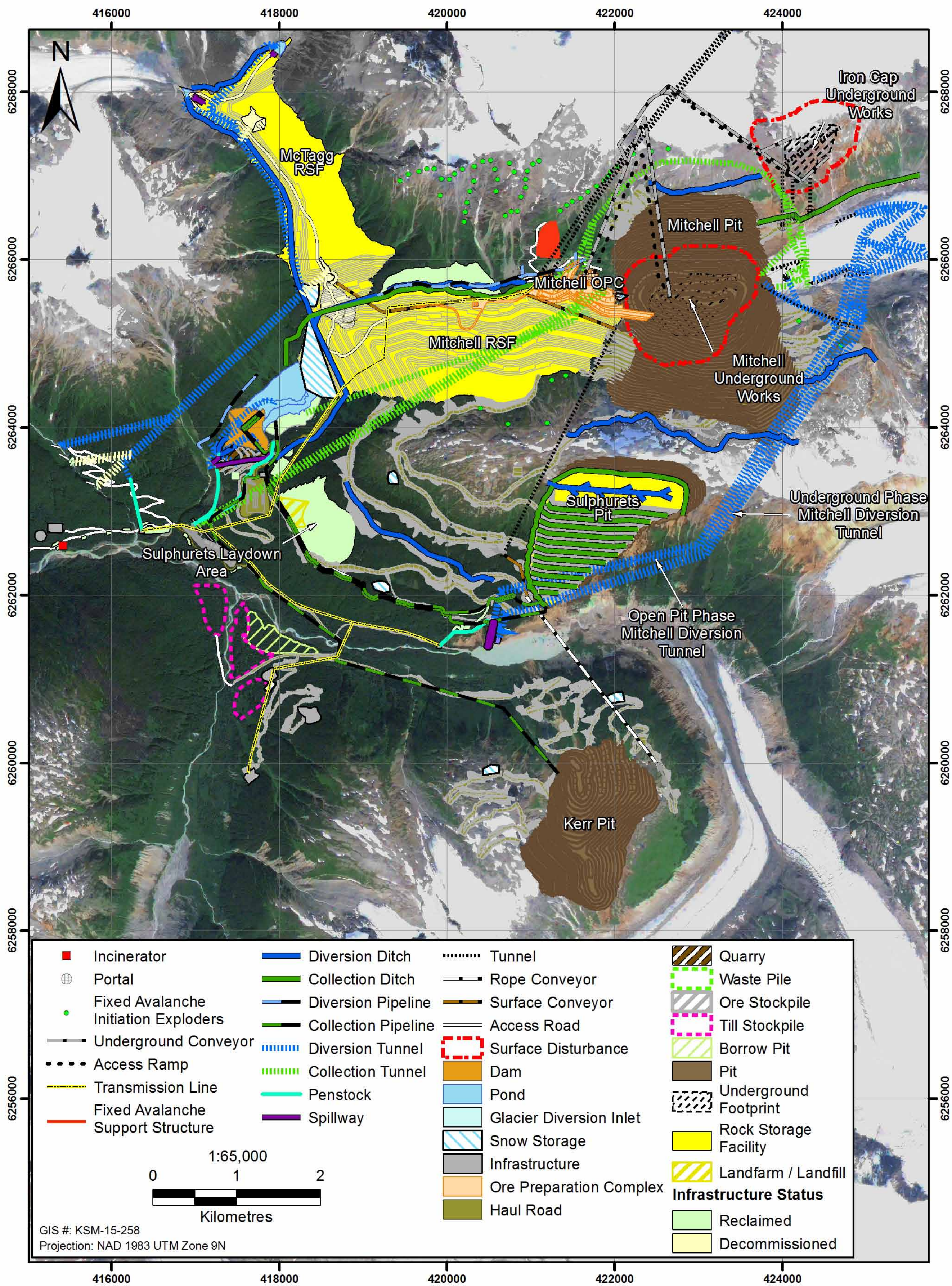


Figure 33.3-2

NEW Footprint - 2012 KSM Project Combined Open Pit and Block Cave Mining - Pit Limits and Waste Rock at End of Operation

SEABRIDGE GOLD
KSM PROJECT

Rescan
Engineers & Scientists

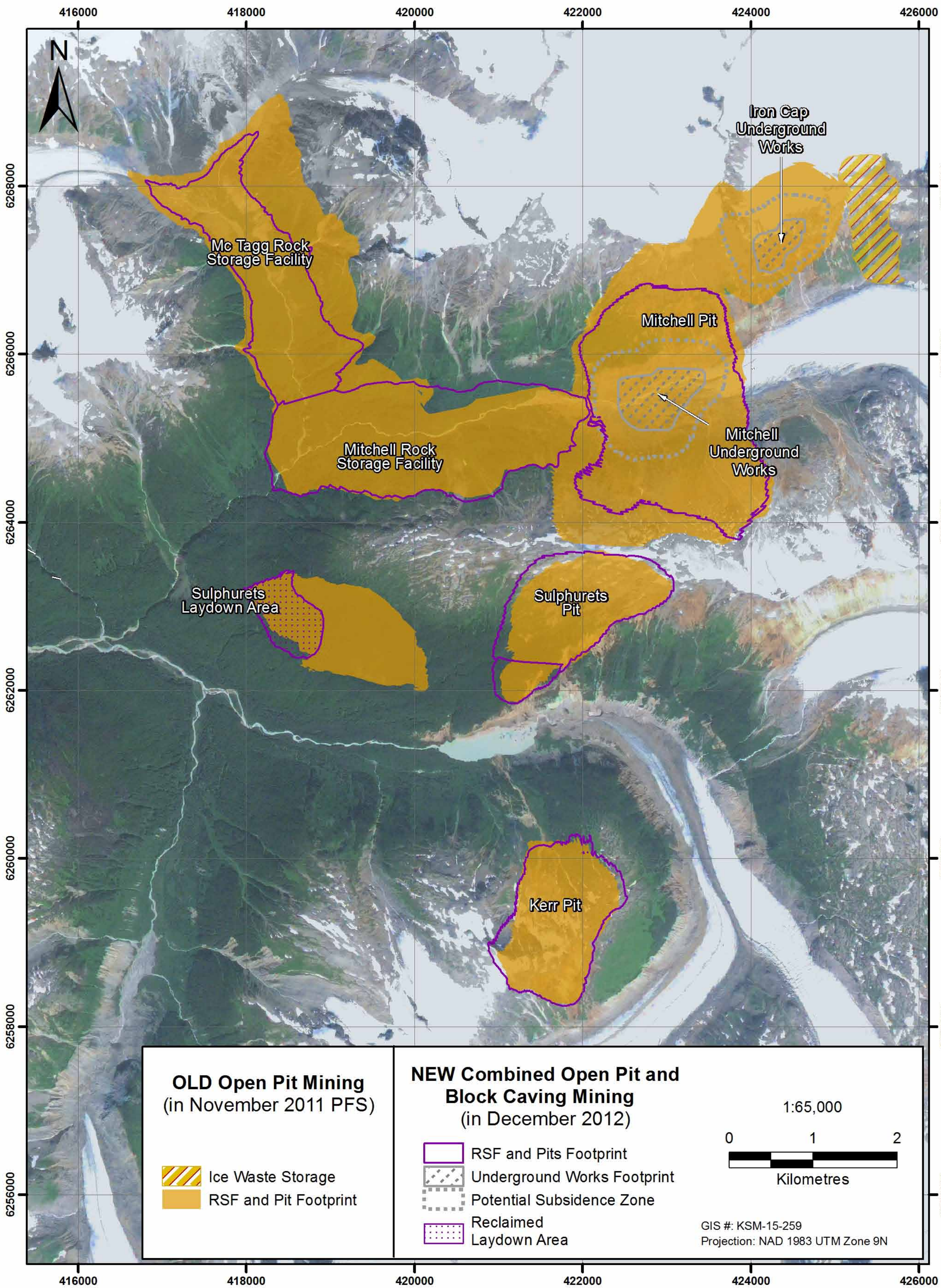


Figure 33.3-3

Old and New Footprint Comparison - Pit Limits and Waste Rock Footprint at End of Operation

SEABRIDGE GOLD
KSM PROJECT

Rescan
Engineers & Scientists

Table 33.3-2. KSM Project Mining Method Pit Footprint Comparison

OLD Footprint*	Area (ha)	NEW Footprint**	Area (ha)	Difference (ha)
Mitchell Pit	700.7	Mitchell Pit	486.9	213.83
Iron Cap Pit	207.8	(No longer pit)	-	-
Sulphurets Pit	188.6	Sulphurets Pit	220.7	(32.09)
Kerr Pit	189.5	Kerr Pit	203.1	(13.66)
Mitchell RSF	370.3	Mitchell RSF	424.6	(54.27)
McTagg RSF	477.6	McTagg RSF	268.9	208.74
Sulphurets RSF	179.7	No longer RSF	-	-
Total Pit and RSF footprint	**2,314.3**	**Total Pit and RSF footprint**	**1,604.2**	**710.03**

Notes:
***2011 Prefeasibility Study (Wardrop 2011) footprint with only open pit mining as shown in Figure 33.3-1.**
****2012 updated Project footprint with open pit and underground mining, shown in Figure 33.3-2.**
Source: Rescan 2013 GIS calculations from Project footprints

Table 33.3-3. Comparison of Stripping Ratios and Production Factors for Open Pit Only and Combined Mining Methods

		Over Life of Mine		
Production Components	**Units**	**Open Pit Mining Only (2011 PFS)**	**Combined Mining (2012 PFS)**	**Difference**
Ore from open pit to mill	Mt	1,126	1,196	70
Open pit ore to stockpile, then reclaim	Mt	1,066	337	-729
Mitchell underground ore to mill	Mt	n/a	438	438
Iron Cap underground ore to mill	Mt	n/a	193	193
Mill feed*	Mt	2,192	2,164	-28
Total waste moved**	Mt	5,923	3,287	-2,636
Total tailing	Bt	2.19	2.16	-0.03
Total strip ratio (total waste moved/mill feed)	t/t	2.7	1.5	-1.2

Notes: n/a = not applicable; PFS = Prefeasibility Study; Sources: Wardrop (2011, 2012a)
*** Mill feed is from open pit and underground mines).**
**** Total waste moved is from open pits and is higher than total waste mined (3.03 Bt) due to re-handle.**

33.3.4 .Selected Alternative

As shown in Table 33.3-1, a hybrid method employing both block caving and open pit mining has been chosen for the Project. The selected alternative consists of developing three open pits (upper Mitchell, Kerr, and Sulphurets) and two underground mines (lower Mitchell and Iron Cap) over a 51.5 year life of mine (LOM)[2].

[2] LOM commences with the start of the ore mill feed, which is also the start of the operation phase.

While open pit mining is generally preferred over block caving for most technical, economic, and social attributes overall, block caving is preferred for environmental reasons and was pursued as an alternative to minimize potential environmental effects. Block caving was found to be unfeasible for the Kerr and Sulphurets deposits due to incompatible geometries, and it was financially challenging to conduct in the upper Mitchell deposit; therefore, these deposits will have to be mined through open pit methods. Block caving was found to be economically and technically feasible for the lower Mitchell and Iron Cap zones and will be pursued for these two deposits in favour of the environmental performance objective. The decision to pursue block caving for these two deposits, and open pit for the rest, was then explored using Gemcom's Footprint Finder and PCBC software to determine potential mine production rates and schedules, as discussed in the next section (Golder 2012b, 2012c).

33.4 Mine Production Rates and Development Schedule

33.4.1 Purpose and Background

Once the portion of a deposit that is economical to mine is determined and the mining method is selected (Section 33.3), effort must be made to plan and schedule the extraction of the mineable reserve. Optimizing production rates is a core aspect of mining, as production rates influence cost per production unit, recovery, and product grade, which in turn bear on other decisions pertaining to mine development scheduling, pit sequencing, mining rate, mine design, and equipment requirements. Long-term mine planning is conducted iteratively as reserve estimates are refined during production and production rates are optimized as commodity prices change.

The decision to engage in block cave mining for some of the KSM Project deposits (Section 33.3) influences mine production rates and schedules. In particular, due to the low-grade ore deposits involved in the Project, high volume throughputs must be achieved in order to pay back the higher CAPEX involved in block cave mining.

33.4.2 Evaluation Method

Mine development schedules are built from production rates using software such as MineSight® Strategic Planner (MineSight), a long-range mine scheduling tool. Typically, analysis of a mine-mill system to determine optimal production rates and schedules is done by calculating the present worth of all expenses and profits relative to year zero of a mine, set as the year when the mine starts production. The project proponent can then decide what is required as a minimum return before undertaking the project investment. If alternatives are available, the alternative with the greatest positive net present value (NPV) is chosen. For large ore deposits, such as those present at the KSM Project, other factors (such as mining logistics, marketing, and financing) may also be used as criteria in decision-making.

Evaluation of a mineral deposit is typically conducted in two stages (MineSight 2009). The first stage consists of the following tasks, typically conducted at the pre-feasibility stage of a project:

- determining parts of the deposit that are economic to mine at different metal prices and operating costs (i.e., determining ultimate pit limits);
- determining the best place to begin mining and the best direction;

- developing a mining sequence extending to the limits of economic ore; and
- estimating capital costs and the mine schedule to the limits of economic ore at different production rates to analyze NPV, cash flows, and the internal rate of return.

If the first phase of planning reveals that a more detailed assessment would be worthwhile, the second phase consists of these further tasks that are generally refined iteratively toward the feasibility stage of a project:

- designing detailed mining pushbacks (i.e., divisions of mine pits into smaller mining phases) that extend to the limits of economic ore, design waste dumps, tailing facilities, roads, etc.;
- calculating ore reserves and stripping requirements for the pushbacks and final pit design;
- creating a detailed LOM schedule using designed pushback tonnages and grades; dump capacities; optimal production rate and cut-off grade strategy; haulage costs based on cycles times between pushbacks and dumps; detailed capital and operating costs; and recoveries and prices;
- determining mine equipment requirements annually;
- calculating annual cash flows, NPV, and internal rate of return; and
- creating charts and end-of-year maps of the LOM schedule.

Generally, economies of scale can be realized at higher production rates and lead to reduced unit operating costs. Higher production rates also enable CAPEX to be recouped earlier, providing more assurance on return on investment, which improves the project NPV. However, higher tonnage throughputs also require more capital, and economies of scale can still apply where some access and construction issues have a high fixed component regardless of the size of the project. In addition, rates of production are restricted by physical and operational constraints and flexibility issues (for example, power availability restrictions can limit the production of a mine).

33.4.3 Selected Alternative

33.4.3.1 Production Rates

Engineering studies using MineSight have identified that the optimal production rate for the KSM Project will be a mill feed of 130,000 tpd. This is 10,000 tpd higher than the mill feed in the 2011 prefeasibility study (Wardrop 2011), which was 120,000 tpd and based on only open pit mining. As mentioned in Section 33.4.1, in order to implement block cave mining—which is preferred for environmental reasons such as reducing surface disturbance and waste rock volume (Section 33.3)—higher production rates must be implemented in order to make up for the low grades and need to recover CAPEX in a timely fashion. Lower throughputs would not be economically viable.

MineSight also varies production slightly annually to maximize the NPV returns for the Project while meeting specific ore production targets. Further evaluation of grind size and recovery will be conducted to confirm the optimal production rate as the mine progresses.

33.4.3.2 Development Schedule

The block cave development schedule for mining the Mitchell deposit was designed with Gemcom's PCBC software; for Iron Cap, Surpac's Mineshed software was used. Overall development schedules for the Project were then designed with MineSight. The 2012 KSM Prefeasibility Study (Appendix 4-C) outlines the results of the updated development schedule associated with the mining method and production rate changes. Detailed pit phases were developed from the results of the LG[3] sensitivity analysis, integrating detailed pit slope criteria and highwall roads. MineSight generates a schedule that accounts for LOM waste storage limitations and long-term haulage requirements. The development schedule generated by MineSight was primarily based on:

- detailed pit phases for the Mitchell, Kerr, and Sulphurets open pit mines, and the Mitchell and Iron Cap block caving operation;
- development requirements, size, and capacity of each of the open pit and underground operations;
- varying production annually from the five areas to maximize the NPV returns for the Project;
- balancing the schedule to account for LOM waste rock storage limitations and haul requirements due to the limited space available for non-rock ore storage;
- adding open pits based on where their added production would provide the best contribution to Project economics, and then adjusting open pit targets to meet process plant capacity; and
- reducing the total production in the later years as the open pit reserves become exhausted and mill production is limited to the capacity of production from block cave mines.

Table 33.4-1 outlines the selected development schedule for both the open pit and block cave mining of the four deposits for the Project. For the Project schedule, pre-production development (involving construction and pre-stripping activities) starts earlier, marking the start of the construction phase, and is completed in about five years. The construction phase is followed by mill start-up in Year 1, which commences the operation phase of the Project. From this schedule, the LOM to develop the proven and probable reserves of the Project at a rate of 130,000 tpd will be 51.5 years. Following operation, the closure and post-closure phases are anticipated to last three years and 250 years, respectively.

Table 33.4-1. KSM Project Mine Development Schedule

Mining Type	Deposit	Operation Phase – Ore to Mill (Years)
Open Pit	Mitchell (upper)	1 to 23
	Sulphurets (starter)	1 to 6
	Sulphurets (final)	23 to 27
	Kerr	27 to 50
Block Caving	Mitchell (lower)	26 to 51.5
	Iron Cap	32 to 51

Note: some pre-stripping activities will also take place at some of the deposits prior to the start of the operation phase.
Source: Appendix 4-C.

[3] LG: Lerchs-Grossman is mining pit design optimization methodology

33.5 Tailing Management

33.5.1 Purpose and Background

When a fish-bearing waterbody is proposed to be designated as a tailing impoundment area, as is the case for the KSM Project, a regulatory amendment to Schedule 2 of the Metal Mining Effluent Regulations (SOR/2002-222) is required. As outlined in the *Guidelines for the Assessment of Alternatives for Mine Waste Disposal* (the Guidelines) published by Environment Canada (2011), an alternatives assessment is required to identify the best location for the tailing impoundment area, also known as a TMF. The Guidelines prescribe the required process of identifying, assessing, evaluating, ranking, and selecting the best location between the available options, following an MAA approach. A summary of the TMF MAA is provided in the sections below (refer to Appendix 33-B for the full report).

In order to address potential concerns, Seabridge was proactive in initiating a tailing management assessment two years prior to submitting the Application for an Environmental Assessment Certificate/Environmental Impact Statement (Application/EIS). Often a TMF assessment is conducted after the Application/EIS approval. The intent of this early action by the Proponent was to ensure that the siting of the TMF was done in a timely manner with appropriate consultation that allowed for the best environmental outcome.

33.5.2 Method

The TMF alternatives assessment process involves seven steps to select a TMF site by a MAA process of systematic analysis and elimination (Figure 33.5-1). The main evaluative step in the MAA commences with the development of a multiple accounts ledger, which is an explicit list of all the potential adverse effects associated with each TMF alternative that generates a clear and measurable description of those effects. The seven steps of the MAA are outlined below (and described in full in Appendix 33-B):

- Step 1 – Identify Candidate Alternatives: identify preliminary TMF candidates near the Mine Site deemed feasible based on basic topographic, geologic, accessibility, precedence, and technical threshold criteria.
- Step 2 – Pre-screening Assessment: screen the number of potential TMF sites by applying a fatal flaw analysis to eliminate alternatives that are not feasible.
- Step 3 – Alternative Characterization: a non-evaluative characterization of the TMF alternatives not eliminated in the previous step.
- Step 4 – Multiple Accounts Ledger: systematically evaluate each TMF option based on the characterization parameters developed in Step 3 using a valuation system based on best professional judgment, and considering issues raised by Aboriginal groups; local, provincial, and federal government agencies; stakeholders; and Seabridge.
- Step 5 – Value-based Decision Process: conduct a final value-based evaluation to identify the preferred TMF candidate. This is done by scoring and weighting the indicators developed in Step 4 and applying a quantitative analysis to develop weighted merit ratings for each TMF candidate.

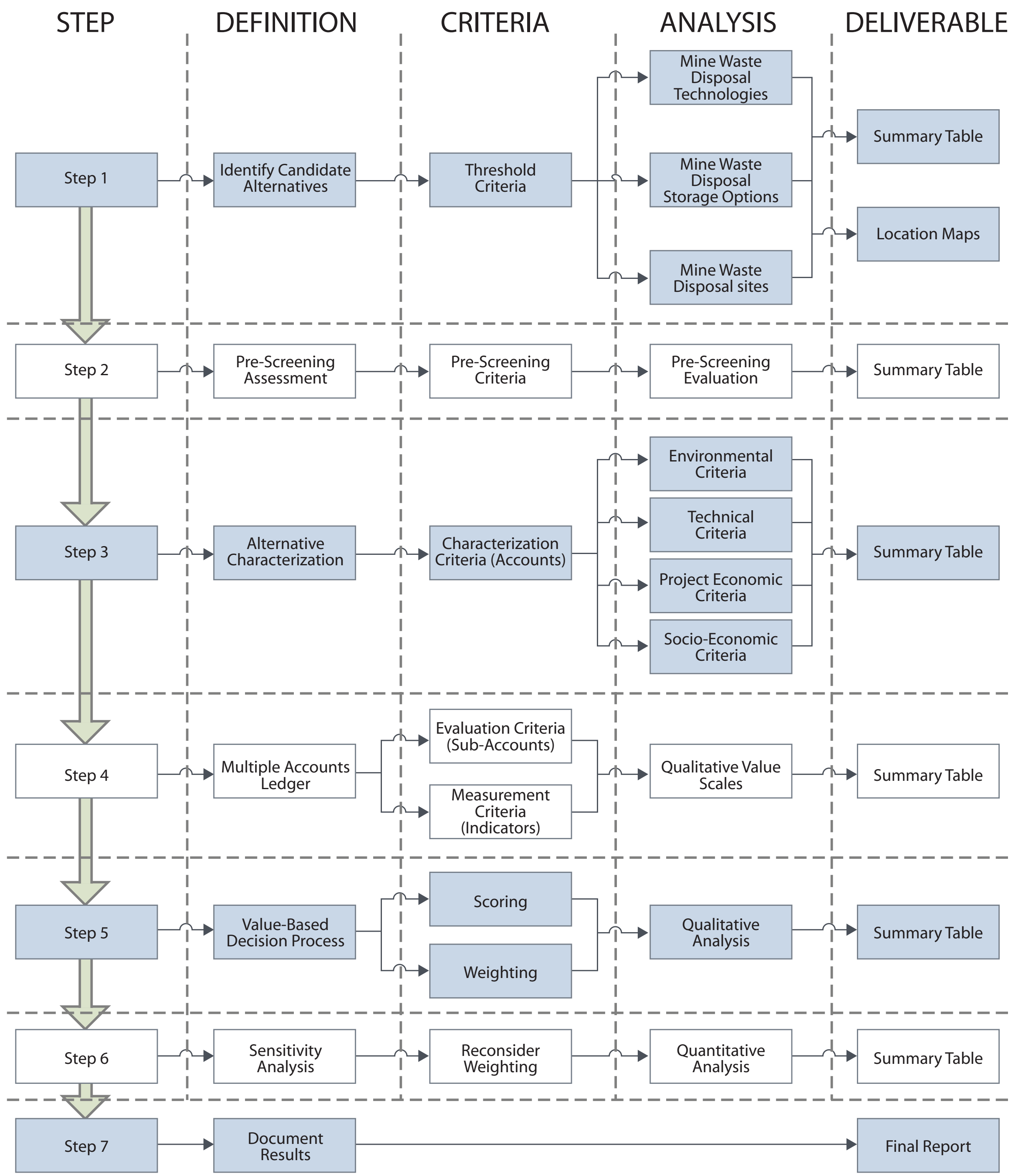


Figure 33.5-1

Environment Canada Guidelines for the Assessment of Alternatives for Mine Waste Disposal Flowchart

- Step 6 – Sensitivity Analysis: consider different value systems when weighting accounts, sub-accounts, and indicators.
- Step 7 – Document Process: transparently report on the TMF alternatives analysis process.

33.5.3 Results of the KSM Project Tailing Management Facility Alternatives Assessment

The results of the seven-step MAA process are summarized below (and presented in full in Appendix 33-B).

Step 1: Identify Candidate Alternatives

Seabridge conducted an initial screening of all potential tailing sites in a 50 by 50 km area surrounding the Mine Site. Threshold criteria were used by Seabridge to exclude sites in the 2,500 km^2 area that were not feasible due to basic topographic, accessibility/cost, and technological limitations.

The following threshold criteria were applied to determine reasonable potential TMF options for the Project.

- *Exclusion based on topography* – The region surrounding the Project is characterized by rugged terrain that limits the options for locating the TMF. Valley topography was a primary factor in excluding options. Many valleys are too steep or too small to qualify as appropriate options.
- *Exclusion based on accessibility* – Feasibility of the tailing disposal and storage sites varied based on relative accessibility associated with the mountainous topography rather than being primarily a function of distance from the Mine Site. High elevation regions and areas with limited or challenging access options were excluded as the technological challenges and/or cost of using these sites would be prohibitive.
- *Exclusion based on technological limitations* – Tailing disposal technologies considered as options included conventional impoundments, subaqueous or saturated storage, submarine storage, and in-pit tailing storage. Underground storage was not considered because underground mining will occur too late in the Project LOM to be of use for tailing storage.

Dry stacking, paste tailing, thickened tailing, and co-disposal of tailing and waste rock were all considered as tailing disposal technologies but were all rejected as being unfeasible.

Dry stacking tailing disposal has been used for low tonnage mines, typically in flat and dry climates. In general, the method relies on the tailing being dry enough to be self-supporting. Dry stacking is not a feasible option in the mountainous, wet, and seismically active area the Project is located in, as the stack would be require a very large footprint with storage piles up to 300 m high, as well as being susceptible to seismic liquefaction. The operation of a dry stacking facility of the Project's daily throughput is also unprecedented. The construction, operation, and closure costs of a dry stacking tailing facility would be in the order of $11 billion. No advantages would exist with respect to ML/ARD and water management for dry stacking over conventional subaqueous disposal.

Paste tailing disposal is commonly used for underground mines as backfill support. Paste tailing are produced by partial dewatering of the tailing to produce a thick slurry (toothpaste consistency) that can still be pumped. When used for underground backfill, 2 to 5% cement is added to improve the strength of the paste. Paste tailing are not self-supporting, and storage of paste tailing for the Project requires containment dams that would be essentially similar to the dams required for conventional tailing storage. The construction, operation, and closure costs of a paste tailing facility would be in the order of $6 billion, and no precedent exists for such a large-scale project. No advantages would exist with respect to ML/ARD and water management for paste tailing over conventional subaqueous disposal.

Thickened tailing includes the use of high-capacity thickeners to increase the tailing density from approximately 60% solids by weight to 75%, and thickening would therefore have negligible benefit to density or tailing management. The high-density thickener would add approximately $1 billion to the Project costs, and no advantages would exist with respect to ML/ARD and water management for thickened tailing over conventional subaqueous disposal.

Co-disposal is frequently examined as a tailing and waste rock disposal technique; however, it is rarely possible to implement. Favourable conditions for co-disposal would consist of a facility large enough to store both waste rock and tailing that would need to be located at the Mine Site due to the large volumes of waste rock involved. This condition is not met at the Project site.

Fourteen potential TMF candidate alternative sites were identified for MAA evaluation, illustrated in Figure 33.5-2:

1. Upper Treaty TMF;
2. West Teigen Lake TMF;
3. Bowser Lake TMF;
4. Segmented Bowser Lake TMF;
5. Knipple Lake TMF;
6. Ted Morris Creek Valley TMF;
7. McTagg Creek Valley TMF;
8. Sulphurets Creek Valley TMF;
9. In-pit Tailing Storage TMF;
10. Burroughs Bay Submarine Disposal TMF;
11. Scott Creek Valley TMF;
12. Combined Sulphurets Creek Valley and Ted Morris Creek Valley TMF;
13. Unuk Valley TMF; and
14. Upper Treaty Creek Valley TMF.

Step 2: Pre-screening Assessment

The following pre-screening criteria were applied to each of the initial 14 TMF candidate alternatives:

- Do government policies recommend against specific deposition methods?
- Are geological foundations insufficient for safe construction and operation of containment dams?
- Do water management issues preclude safe operation of the TMF?
- Will the TMF result in negative life of Project economics?
- Does the proposed facility have insufficient capacity for the entire proposed mine life?
- Are engineering issues prohibitive given current technology?
- Do geological hazards preclude the safe operation of the TMF?

Of the above 14 candidate alternatives for potential TMFs, four potential tailing management alternatives—one individual site and three combinations of two sites—met all the TMF siting pre-screening criteria (see Appendix 33-B for the full analysis):

- Upper Treaty TMF (1);
- Scott Creek Valley TMF combined with West Teigen Lake TMF (11 and 2);
- Unuk Valley combined with West Teigen Lake TMF (13 and 2); and
- Upper Treaty Creek Valley combined with West Teigen Lake TMF (14 and 2).

Step 3: Alternative Characterization

Step 3 expands the scope and detail of the characterization of each candidate alternative using Project-specific criteria as recommended in the Guidelines (Environment Canada 2011). The criteria fall under the following four broad categories, referred to as "accounts":

- Environmental characterization – This account describes the local and regional environment surrounding each proposed TMF. Elements such as climate, geology, hydrology, hydrogeology, water quality, and potential effects on fish and wildlife are considered.
- Technical characterization – This account describes the engineered elements of each alternative such as storage capacity, dam size and volume, diversion channel size and capacity, dumping techniques, haul distances, seepage dam requirements, tailing discharge methods, pipeline grades and routes, closure design, discharge and/or water treatment infrastructure, and supporting infrastructure such as access roads.
- Project economic characterization – The account describes the life of Project economics. All aspects of the mine waste management plan are considered including investigation, design, construction (inclusive of borrow development and royalties where applicable), operation, closure, post-closure care and maintenance, water management, associated infrastructure (including transport and deposition systems), compensation payments, and land use or lease fees.

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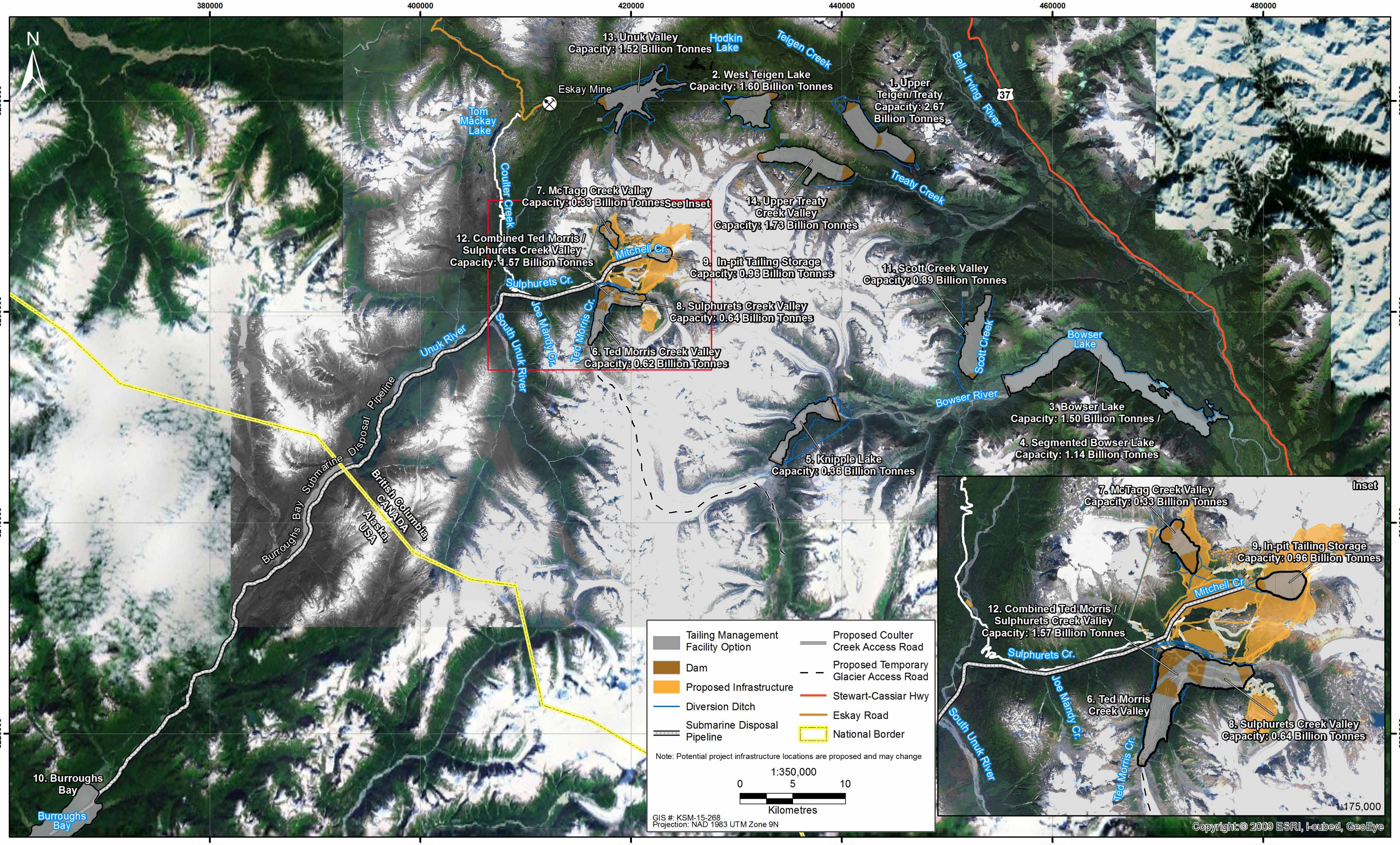

Figure 33.5-2

SEABRIDGE GOLD
KSM PROJECT

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KSM Project - Candidate Alternatives for Tailing Management Facility Locations

- Socio-economic characterization – This account describes how each proposed TMF may impact commercial land users and Aboriginal interests. Elements considered include characterization and valuation of land uses, cultural significance, presence of archaeological sites, and employment and/or training opportunities.

Each account considers short- and long-term issues associated with construction through operation, mine closure, and, ultimately, post-closure maintenance and monitoring. Detailed characterization data and summary tables are provided in Appendix 33-B.

Step 4: Multiple Accounts Ledger

As per section 2.5 of the Guidelines (Environment Canada 2011), a multiple accounts ledger (Table 33.5-2) identifies criteria from the alternative characterization that differentiate alternatives so that TMF options can be evaluated relative to one another. This ledger is derived from the characterization data. The multiple accounts ledger consists of two elements: sub-accounts (i.e., evaluation criteria) and indicators (i.e., measurement criteria).

Table 33.5-2. KSM Project Tailing Management Facility Alternatives Analysis Multiple Accounts Ledger

Account	Sub-account (Evaluation Criteria)	Indicator (Measurement Criteria)
Environmental	Aquatic Habitat Loss	Ecological quality index of affected wetlands. Lake surface area directly affected. Stream length directly affected. River length directly affected.
	Direct Loss Fisheries	Number of fish species directly affected. Extent of HADD to fish habitat.
	Downstream Fisheries	Distance from toe of containment dam to first occurrence of downstream salmon. TMF contribution to catchment area at first occurrence of downstream salmon. Downstream chinook salmon (*Oncorhynchus tshawytscha*) values. Downstream sockeye salmon (*O. nerka*) values. Downstream coho salmon (*O. kisutch*) values.
	Terrestrial Habitat Loss	Presence of rare and endangered ecosystems. Presence of rare and endangered plant species. Presence of high value grizzly bear (*Ursus arctos*) habitat. Presence of high value mountain goat (*Oreamnos americanus*) habitat. Presence of high value moose (*Alces alces*) habitat. Presence of high value American marten (*Martes americana*) habitat. Presence of rare and endangered wildlife species, excluding grizzly bear, mountain ungulates, moose, and American marten.
	Groundwater Quality and Quantity	Changes to groundwater quantity. Changes to groundwater quality.

(continued)

Table 33.5-2. KSM Project Tailing Management Facility Alternatives Analysis Multiple Accounts Ledger (completed)

Account	Sub-account (Evaluation Criteria)	Indicator (Measurement Criteria)
Environmental *(cont'd)*	Surface Water Hydrology	Potential impacts to downstream sediment transport. Potential runoff loss using watershed area (to salmon habitat).
	Surface Water Chemistry Immediate Receiving Environment[1]	Water quality (copper) in the immediate receiving environment. Water quality (zinc) in the immediate receiving environment. Water quality (sulphate) in the immediate receiving environment.
Technical	Water Management	Catchment size. Undiverted catchment size. Glacierized area in catchment. Number of diversion dams required. Diversion of main channel required. Feasibility of diversion construction.
	Containment Infrastructure Design	Number of containment dams required. Total containment dam volume. Number of facilities.
	Foundation Conditions	General Foundation Conditions. Earthquake Foundation Conditions.
	Construction, Operation, and Closure Requirements	Ease of construction. Ease of operation. Closure requirements.
Socio-economic	Aboriginal interests	Cultural and/or spiritual significance of site. Importance for Aboriginal land and resource use activities (hunting/trapping/fishing/plant gathering). Importance as access route to traditional harvest, cultural, and/or spiritual site. Aboriginal territories that overlap directly with TMF site. Nisg̲a'a Lands (i.e., Nass Wildlife Area, or Nass Area) overlapped.
	Commercial Land Uses	Number of guide outfitting tenures overlapped. Number of commercial recreation tenures overlapped.
	Employment	Local contracting and employment effects resulting directly from containment dams.
	Archaeology	Archaeological importance.
Project Economics	Estimated Costs	Capital costs. Operating costs. Closure costs.

Notes:
[1] Copper, zinc, and sulphate were chosen as indicators of surface water chemistry because, when the TMF report was completed, these were the chemicals of highest public concern, or were likely to exceed regulations.
HADD = harmful alteration, disruption, or destruction.

To allow the accounts and sub-accounts to be measured and compared, the indicators must be measureable. As per the Guidelines, a six-point scale was used, because it provides sufficient range to differentiate without being overly detailed, and it is an even number scale that eliminates the tendency to select the "middle-of-the road" value. Qualitative (i.e., value-based) scales were developed (e.g., very high, high, low, etc.) when precise measurability was not possible, as per the Guidelines (Environment Canada 2011). Value scales were developed to have the following characteristics:

- *Reliable* – External reviewers must be able to rate an alternative according to the value scale and assign the same score.
- *Value relevant* – The value scale must be directly relevant to the indicator being scored.
- *Justifiable* – Any external reviewer should reach the conclusion that the value scale is reasonable and representative.

Each indicator listed in Table 33.5-2 has a scoring descriptor (table or textual), as described in Appendix 33-B.

Step 5: Value-based Decision Process

The value-based decision process involves the creation of scoring and weighting scales for all relevant criteria (account, sub-account and indicators), set previously in the multiple accounts ledger, and applying them to the four alternatives. Values weighting was done on a six-point scale with six being the most highly valued and one the least. This value-based ranking methodology was done in concordance with the Guidelines (Environment Canada 2011) in order to differentiate the benefit or loss associated with each site. Weightings assigned to each indicator within a sub-account and within each account are outlined in Tables 33.5-3 to 33.5-9. The weighting scales are relative, not absolute, within a particular account or sub-account. As such, these weightings are only directly comparable within a particular account or sub-account and not among accounts or sub-accounts. Please see Appendix 33-B for a full explanation of weighting methodology and the tables summarized below.

Table 33.5-3. Account Weighting for Base-case Value Scenarios

Account	Weighting Base-case
Environment	6
Technical	3
Socio-economic	3
Project Economic	1.5

Table 33.5-4. Account: Environmental

Sub-account	Weighting Value
Aquatic Habitat Loss	3
Direct Loss Fisheries Value	6
Downstream Fisheries Value	1
Terrestrial Habitat Loss	3
Groundwater Quality and Quantity	5
Surface Water Hydrology	5
Surface Water Chemistry Immediate Receiving Environment	5

Table 33.5-5. Sub-account Indicator Weightings of the Environment Account

Sub-account	Indicator	Weighting Value
Aquatic Habitat Loss	Ecological quality index of affected wetlands	6
	Lake surface area directly affected	4
	Stream length directly affected	4
	River length directly affected	4
Direct Loss Fisheries Value	Number of fish species directly affected	2
	Extent of HADD to fish habitat	6
Downstream Fisheries Value	Distance from toe of containment dam to first occurrence of salmon	1
	Downstream chinook salmon values	6
	Downstream sockeye salmon values	6
	Downstream coho salmon values	6
Terrestrial Habitat Loss	Presence of rare and endangered ecosystems	6
	Presence of rare and endangered plant species	6
	Presence of high value habitat for grizzly bear	2
	Presence of high value habitat for mountain goat	5
	Presence of high value habitat for moose	3
	Presence of high value habitat for American marten	1
	Presence of rare and endangered wildlife species, excluding grizzly bear, mountain ungulates, moose, and American marten	6
Groundwater Quality and Quantity	Changes to groundwater quantity	3
	Changes to groundwater quality	3
Surface Water Hydrology	Potential impacts to downstream sediment transport	3
	Potential runoff loss using watershed area (to salmon habitat)	3
Surface Water Chemistry Immediate Receiving Environment	Water quality (copper) in the immediate receiving environment	6
	Water quality (zinc) in the immediate receiving environment	6
	Water quality (sulphate) in the immediate receiving environment	1

Table 33.5-6. Account: Socio-economic

Sub-account	Weighting Value
Aboriginal Interests	6
Commercial Land Uses	3
Employment	3
Archaeology	6

Table 33.5-7. Sub-account Indicator Weightings of the Socio-economic Account

Sub-account	Indicator	Weighting Value
Aboriginal Interests	Cultural and/or spiritual significance of site	6
	Importance for Aboriginal land and resource use activities (hunting/trapping/fishing/plant site gathering)	6
	Importance as access route to traditional harvest, cultural, and/or spiritual site	5
	Traditional territories that overlap directly with TMF	4
	Nisg̲a'a Lands that overlap directly with TMF	4
Commercial Land Use	Number of traplines overlaps	5
	Number of guide outfitting tenures overlaps	5
	Number of commercial recreation tenures overlaps	3

Table 33.5-8. Account: Technical

Sub-account	Weighting Value
Water Management	6
Containment Infrastructure Design	4
Foundation Conditions	3
Construction, Operating and Closure Requirements	6

Table 33.5-9. Sub-account Indicator Weightings of the Technical Account

Sub-account	Indicator	Weighting Value
Water Management	Catchment size	6
	Undiverted catchment size	4
	Glacierized area in catchment	5
	Number of diversion dams required	3
	Diversion of main channel required	3
	Feasibility of diversion construction	4
Containment Infrastructure Design	Number of containment dams	2
	Total dam volume	2
	Number of facilities	6
Foundation Conditions	General foundation conditions	5
	Earthquake foundation conditions	4
Construction, Operation, and Closure Requirements	Ease of construction	2
	Ease of operation	2
	Closure requirements	6

The base case weighting of each of the main accounts is provided in Table 33.5-3 with the environmental account rated the highest, the technical and socio-economic accounts given equal median weights and the economic account the lowest.

Environment Account

Table 33.5-4 provides the weighting values established for the environmental sub-accounts, showing direct loss of fisheries value as the highest weighting. Then, Table 33.5-5 lists the weighting values determined for each of the indicators for these environmental sub-accounts.

Socio-economic Account

Table 33.5-6 provides the weighting values established for the socio-economic sub-accounts. Then, Table 33.5-7 lists the weighting values determined for each of the indicators for these socio-economic sub-accounts.

Technical Account

Table 33.5-8 provides the weighting values established for the technical sub-accounts. Table 33.5-9 lists the weighting values determined for each of the indicators for these technical sub-accounts.

Project Economics Account

Table 33.5-10 provides the weighting values established for the economic sub-account indicators. Only one sub-account exists for the Project economics account (i.e., estimates costs), so no sub-account weighting table is necessary for this account.

Table 33.5-10. Sub-account: Estimated Costs

Indicator	Weighting Value
Capital costs	6
Operating costs	3
Closure costs	2

Using the weightings listed in Tables 33.5-3 to 33.5-10, combined with the indicator scores derived from the characterization data, as described in Step 4, a qualitative score for each of the candidate TMF alternatives was calculated. Calculation methodology followed the Guidelines (Environment Canada 2011), and full calculation tables are provided in Appendix G of Appendix 33-B. The result of these calculations, i.e., the results of the MAA for the KSM Project TMF alternatives assessment, is shown in Table 33.5-11.

Table 33.5-11. KSM Project Tailing Management Facility Multiple Accounts Analysis Results

	Upper Teigen / Treaty	Scott Creek Valley - West Teigen Lake	Unuk Valley - West Teigen Lake	Upper Treaty Creek - West Teigen Lake
Base case	4.5	2.5	2.4	2.2

The value-based MAA decision process result indicated that the Upper Treaty TMF is the most appropriate TMF alternative (i.e., resulting in the highest value from the MAA process). The remaining three sites are significantly less preferable, and roughly equivalent to each other.

Step 6: Sensitivity Analysis

A sensitivity analysis was performed for the KSM Project TMF alternatives assessment according to the Guidelines (Environment Canada 2011). For the KSM Project TMF MAA, the following sensitivity analyses were performed:

- all accounts were weighted equally;
- only environmental and socio-economic accounts were considered;
- only the environmental account was considered;
- only the technical account was considered;
- only the Project economics account was considered;
- only the socio-economic account was considered;
- all accounts and sub-accounts were weighted equally;
- all accounts, sub-accounts, and indicators were weighted equally;
- downstream fisheries and water quality indicators sub-accounts were weighted more significantly; and
- downstream fisheries sub-account was weighted more significantly.

The result of all the sensitivity analyses that were conducted was that the Upper Treaty TMF alternative consistently emerged as the preferred option. Full analytical results are presented in Appendix G of Appendix 33-B.

Step 7: Document Results

This document summarizes the full report provided in Appendix 33-B that fully documents the TMF alternative assessment process undertaken by Seabridge for the KSM Project, in conjunction with consultation with Nisga'a Nation and Aboriginal groups, as well as local, provincial and federal government agencies. The TMF alternatives analysis results were presented to the environmental assessment Working Group for the Project on September 15, 2011, and again on March 29 and 30, 2012, in Smithers, BC.

33.6 Process Plant Location

33.6.1 Purpose and Background

The location selected for the Project's process plant and its related facilities must be suitable for its primary activities: crushing and grinding ore (comminution), recovery of precious metals, and release of tailing waste to the TMF. Since these activities for the Project are divided between the main Mine Site and a separate Processing and Tailing Management Area (PTMA; assessed in Section 33.5), the potential locations considered in this assessment were restricted to feasible sites in proximity to either the Mine Site or one of the main TMF locations as shown in Figure 33.6-1. The alternatives assessment for selecting the location of a process plant was conducted based on a comparison of environmental, social, technical, and Project economic performance objectives.

33.6.2 Alternatives Identification

Several criteria determine the ideal siting for ore processing plants and related facilities. To process up to 130,000 tpd, a process plant would require an approximately 50 ha footprint at a location that minimizes the need to transport ore and tailing. Siting decisions must also consider the types of specific activities to be carried out at the facility (i.e., whether the primary crushing and grinding of ore occurs on-site or is done remotely; and whether tailing is thickened on-site or at the TMF). Flat or moderately sloping terrain is preferable to a steep or hummocky location; however, a sloping site can sometimes harness gravity to move materials between process stages. The location must also:

- provide a solid foundation to support heavy infrastructure and equipment;
- be free of geohazards;
- provide a water source for milling needs;
- be accessible year-round to heavy truck traffic (delivering reagents and grinding media and hauling concentrate to market);
- have electric power up to 150 MW, and emergency backup diesel generators; and
- support a construction camp of about 700 people and an operating camp for a staff of about 250 people in the vicinity.

Extraction of ore from the Kerr, Sulphurets, Mitchell, and Iron Cap deposits will take place in the Mine Site, primarily within the Sulphurets and Mitchell creeks valleys. These valleys provide the closest potential process plant locations to the Mine Site, but due to the generally poor foundation conditions and high geohazard ratings of available sites in theses valleys, and the unfeasibility of storing Project tailing in the Mine Site, alternative process plant sites were sought for comparison that would complement the Treaty TMF site alternatives. Table 33.6-1 identifies the plant sites assessed and the rationale for their inclusion based on preliminary technical feasibility criteria, and Figure 33.6-1 illustrates the sites as well as their corresponding TMF options.

Table 33.6-1. Alternative Locations for the KSM Project Process Plant

Process Plant Alternative	Location	Selection Rationale
Plant Site 1	Teigen Creek Valley	Most feasible location next to the Treaty TMF option
Plant Site 2A	Kaypros Creek Valley near West Teigen Creek	A feasible location next to the West Teigen Lake TMF option
Plant Site 2B	Kaypros Creek Valley near West Teigen Creek	A feasible location next to the West Teigen Lake TMF option
Plant Site 3	Upper reaches of Unuk River Valley watershed	A feasible location next to the West Teigen Lake TMF option
Plant Site 4	Sulphurets/Mitchell Creek valleys in the Mine Site	Most feasible location at the Mine Site

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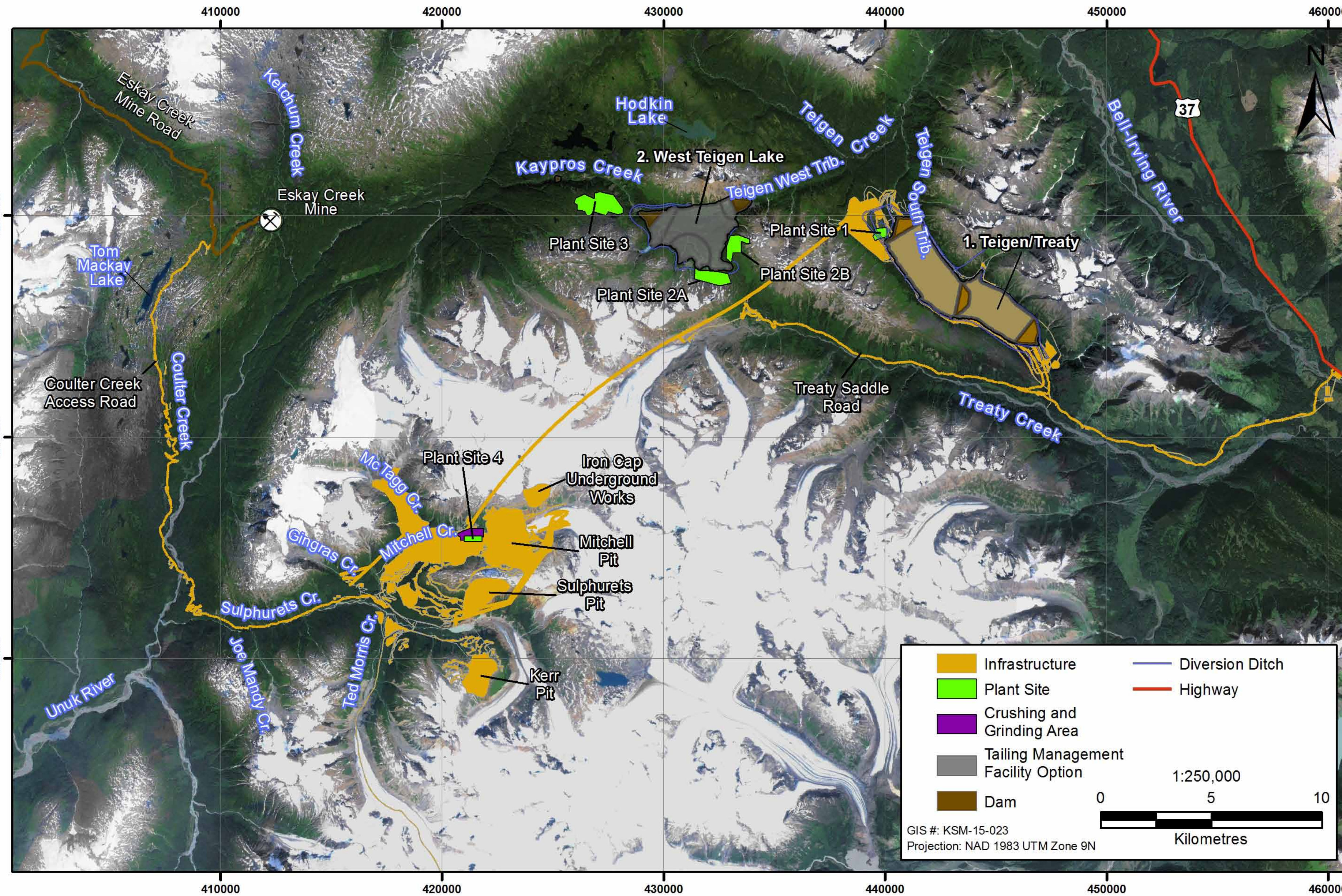

Figure 33.6-1

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KSM Project Process Plant Location Alternatives

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33.6.2.1 Plant Site 1

The Treaty location chosen for Plant Site 1 is a gently sloping (less than 10% grade) bare rock plateau, at elevations ranging from 1,050 m to 1,160 m. The bench forming this plateau is approximately 1 by 2 km. The plant and stockpile areas consist of exposed, uniformly steeply dipping, north-south striking sedimentary rock units, which would provide good foundation conditions. Plant area drainage for this option could also be routed to the TMF.

33.6.2.2 Plant Sites 2A and 2B

Plant sites 2A and 2B are located about 16 km northwest of the Mitchell Deposit, along the south end of the West Teigen Lake TMF option. This location is in the upper reaches of Kaypros Creek, which is a tributary to the Unuk River (Figure 33.6-1). Parts of the area drain towards West Teigen Creek to the east (part of the Bell-Irving River drainage system), while the western portion drains towards Kaypros Creek.

These sites were investigated, along with Plant Site 3, in order to reduce the length of the ore transport tunnel and because of their proximity to the West Teigen Lake TMF option. Plant sites 2A and 2B are located on steeply dipping hard sandstone bedrock that is under half a metre to several metres of silty, sandy soil of low to moderate plasticity. These foundations should be adequate to support a plant and related facilities. The near vertical dip may make it favourable for ripping. The first site (Plant Site 2A) is located south of the upper reaches of Kaypros Creek at an elevation ranging from 1,140 m to 1,170 m. It provides a potential development area of about 35 ha on a treeless north-facing side hill bench. It may be somewhat exposed to avalanche hazards. Ample water is available from Kaypros Creek, located in a 60 m deep gully immediately north of the site.

Ore would be transported to Plant Site 2A via a 16.1 km-long tunnel from Mitchell Creek with a northern portal elevation of about 1,140 m. Tailing could then be pumped, or delivered by gravity, in slurry form to the TMF, as the West Teigen Lake TMF option considered alongside this site is located immediately downslope of the plant site. The tailing pipeline to the TMF from this plant could either follow a road alignment along West Teigen Creek or go through a 5.1 km tunnel to avoid elevation changes and avalanche hazards present on the West Teigen Creek route.

Plant Site 2B is also located in the upper reaches of Kaypros Creek and is situated northeast of Plant Site 2A (Figure 33.6-1). Like Plant Site 2A, Plant Site 2B is located on steeply dipping hard sandstone bedrock that is under half a metre to several metres of silty, sandy soil of low to moderate plasticity. This foundation is considered adequate to support a plant and associated facilities, and the near vertical dip may make it favourable for ripping. The available area is about 50 ha in a long, narrow configuration on two benches in an area of alpine forest with variable topography at elevations ranging from 1,100 m to 1,170 m.

33.6.2.3 Plant Site 3

Plant Site 3 is located on a rugged hummocky plateau southwest of Kaypros Creek, and west of the West Teigen Lake TMF option in the upper reaches of the Unuk River drainage system. The site ranges from elevations of about 1,200 m to over 1,260 m, and includes numerous small lakes and ponds.

This plant site was considered as part of a concept that involved crushing and grinding the ore near the mineral deposits and pumping it through a slurry pipe to Plant Site 3. In order to reduce the length of tunnel required for the pipeline, Seabridge investigated tunnels from McTagg Creek through to the Unuk River Valley, with the route then following the valley to the plant site.

33.6.2.4 Plant Site 4

Within the Sulphurets and Mitchell Creek valleys, other facilities and infrastructure are also required to be located close to the pits, such as ore and waste rock storage, soil and overburden storage, mobile equipment maintenance facilities, personnel camps, roads, explosives manufacturing facilities, and water treatment facilities. The Plant Site 4 location is the most appropriate for a milling facility in the Sulphurets/Mitchell valleys as it is immediately adjacent to the Mitchell Pit near the tunnel portal, where a tailing transport system could be constructed from Plant Site 4 to the TMF.

33.6.3 Comparison of Alternatives

The alternative process plant location attributes are compared in Table 33.6-2, based on stated environmental, social, technical, and Project economic performance objectives (Section 33.2.2).

33.6.3.1 Technical and Project Economic Considerations

The primary technical consideration for process plant siting is to ensure adequate foundations, topography, and a site free of hazards. A study of the placement of a Plant Site 4 revealed that this site would be encroached upon by space requirements for the Mitchell RSF and that it was subject to higher geohazard risks than the other alternatives. For these reasons, Plant Site 4 is considered technically unfeasible.

After ensuring suitable foundations and a location safe from geohazards, the main technical and economic consideration in process plant technical site placement is that of distance regarding the need to convey ore and tailing. As shown in Figure 33.6-1, Plant Site 1, is the closest to the Treaty TMF (which is the chosen TMF site, as outlined in Section 33.5), minimizing tailing transport needs; however, it is 23 km away from the Mine Site, and thus requires conveying ore over that distance. Plant sites 2A, 2B, and 3 are about 16 km from the Mine Site, but are adjacent to the West Teigen Lake TMF, which was not chosen. This means that these sites would also have to transport tailing all the way to the Treaty TMF, which presents challenges compared to Site 1. Plant Site 4, at the Mine Site, would not require long distance ore transport, but would require tailing transport over the 23 km distance to the PTMA.

Route length is another key technical and economic attribute to choose from feasible plant sites. Shorter route lengths minimize construction costs as well as haul costs for reagents and ore concentrates during operation, and shorter route lengths also minimize environmental effects.

Plant sites 1, 2A/2B, and 3 would be designed to share access routes with either the Treaty or West Teigen Lake TMF options to minimize costs and environmental effects. Of these, the shortest access route from Highway 37 would be to the Treaty TMF and adjacent Plant Site 1 (Figure 33.6-1[4]) with plant sites 2A, 2B, and 3 having progressively longer routes from this TMF respectively. A preliminary review indicated that access to sites 2A, 2B, and 3 stemming from the Eskay Creek Mine road (Figure 33.6-1), would not be feasible due to challenging topography. Plant sites 2A, 2B, and 3 are also proximal to the West Teigen Lake TMF site that was not selected, which would mean that these sites would require lengthy route extensions from the Treaty TMF site, presenting several additional technical challenges. For instance, plant sites 2A, 2B, and 3 would likely all share an access route travelling through a canyon along the west tributary of Teigen Creek, where significant avalanche hazards may lead to road closures in the winter. Extra storage for grinding media, reagents, and concentrate to ensure continuous operation in the event of a road closure would be required for these sites. Therefore, of plant sites 1, 2A, 2B, and 3, Plant Site 1 faces the fewest barriers regarding access construction and operation. Access to Plant Site 4 would share the Coulter Creek access road (CCAR) to the Mine Site followed by a service road shared with other mining activities. While this route does not pose extra construction challenges, it would not be practical for reagent and ore concentrate hauling purposes, as it is the longest route and also faces geohazard risks such as avalanches.

Water is not a limiting factor for any of the plant sites considered, and neither is power supply, as the Mine Site and PTMA will both derive power from the Northwest Transmission Line (NTL); however, the length of transmission line construction to Plant Site 3 would be the highest, followed by plant sites 2A and 2B.

As a result of comparing technical and economic attributes, Plant Site 1 is considered to be preferred overall. Plant sites 2A, 2B, and 3 are considered to be technically and economically challenging, and Plant Site 4 unfeasible for the reasons discussed above.

33.6.3.2 Environmental and Social Considerations

As shown in Table 33.3-2, Plant Site 4 has the most attributes with a preferred environmental rating compared to all the sites as it minimizes many potential environmental effects due to it being situated in the Mine Site itself. An exception is that it has higher risks of potential haul truck reagent and ore concentrate spills due to the longer transport route with more geohazard risks, as well as higher tailing spill risks due to the need to pump tailing 23 km from the Mine Site to the TMF for this option.

Atmospheric greenhouse gas (GHG) and air quality emissions, as well as noise levels, are anticipated to be similar for every process plant site regardless of location. There are minor differences in dispersion factors between sites, with more exposure, faster wind speeds, and better atmospheric mixing likely to occur at, for example, plant sites 2A and 3. However, air quality effects to surrounding human communities are not anticipated to be material due to the remote location of the Project.

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[4] Although roads alignments are not shown in Figure 33.6-1, creeks that roads would follow are indicated, as well as assessed in the next section.

Table 33.6-2. Summary Comparison of Process Plant Location Alternatives

Alternative	ATTRIBUTE RATINGS AGAINST FOUR PERFORMANCE OBJECTIVES				OVERALL ALTERNATIVE RATING
	Environmental	**Social**	**Technical**	**Project Economic**	
Plant Site 1 (Treaty Valley)	✓ Shares the shortest access road with TMF, reducing related environmental effects ✓ Atmospheric: Strong wind dispersion not anticipated, so air effects localized to a few hundred metres x Accident risks to water quality could affect Bell-Irving River downstream x There are more wetlands in the area than for other sites, but site design avoids wetland loss and degradation ✓ Location on bedrock plateau above 1,000 m minimizes effects on wildlife x Some potential effects from noise on mountain goats and bears	✓ More construction jobs for site accommodation complex and road may be created for this option compared to Plant Site 4 ✓ Large available area allows adequate spacing of camp and process plant to reduce noise effects on staff x Additional human presence to and from process plant may increase wildlife disturbance with resultant effects on hunting/trapping x Presence of process plant may reduce visual attractiveness for heli-ski runs in the area x Process plant location may affect Skii km Lax Ha traditional land use (i.e., reported trail and harvesting)	✓ Closest to the selected TMF site (33.5) ✓ Power is easily available from existing transmission line to the PTMA ✓ Good foundation conditions and less than 10% grade on bare rock plateau ✓ Area drainage can easily be routed to TMF and easily available water from TMF diversions ✓ Low geohazards: avalanche hazards not present in process plant plateau area x Requires transport of ore via a 23-km underground tunnel	✓ Lowest construction costs ✓ Lowest operation costs (i.e., from reagent and concentrate transport hauling along 18 km Treaty Creek access road) ✓ Can share same access road as TMF, lowering costs x Requirement for tunnel transport increases construction and operation costs, but these area also necessary to situate TMF off-site x Redundancies in facility construction for camps, but facilities can be shared for TMF and process plant personnel	**PREFERRED**
	Acceptable	Acceptable	Preferred	Preferred	
Plant Site 2A (Kaypros Creek)	x Will require more new road construction compared to plant sites 1 and 4 x Will affect extra undisturbed natural systems as not adjacent to TMF site x Atmospheric: Strong wind may disperse air contaminants farther than other sites, though likely no material effects x Potential accident risks to water quality may affect Unuk River downstream, with potential cross border issues x Plant and access route will affect mountain goats ✓ Location on non-forested land likely minimizes wildlife effects x Road and pipelines may cause HADD for Kaypros Creek ✓ Direct footprint will not interfere with fish habitat	✓ Large available area allows adequate spacing of camp and process plant to reduce noise effects on staff x Site separated from TMF makes staff accommodation more isolated and transport more difficult due to increased safety risks x Additional human presence to and from process plant may affect wildlife disturbance and hunting/trapping x Presence of process plant may reduce visual attractiveness for heli-ski runs in the area x Process plant location may affect Skii km Lax Ha traditional land use (i.e., reported trail and harvesting)	x Not adjacent to the selected TMF (Section 33.5) presenting extra transport requirements x Requires extra transmission line construction x Relies on access via Teigen Creek road option, which was not selected as TMF access route (Section 33.7), so would require construction of new road with resultant HADD ✓ Low avalanche geohazard risk to process plant site ✓ Avalanche geohazard risks along access route x Some resourcing redundancy for personnel camps ✓ Foundations suitable for process plant; steeply dipping hard sandstone bedrock under thin silty/sandy soil	x Higher access construction costs than plant sites 1 and 4 x Higher operating costs than Plant Site 1 due to longer access ✓ Reduces cost of ore transport compared to Plant Site 1 and tailing pipeline would flow by gravity x Higher materials and supplies transport costs than Treaty option x Avalanche geohazard risk along access route means increased capacity for on-site storage for reagents so that road closure does not halt production	**CHALLENGING**
	Challenging	Acceptable	Challenging	Acceptable	

(continued)

Table 33.6-2. Summary Comparison of Process Plant Location Alternatives (continued)

Alternative	ATTRIBUTE RATINGS AGAINST FOUR PERFORMANCE OBJECTIVES				OVERALL ALTERNATIVE RATING
	Environmental	Social	Technical	Project Economic	
Plant Site 2B (Kaypros Creek)	x Will require more new road construction compared to plant sites 1 and 4 x Will affect extra undisturbed natural systems as not adjacent to TMF site x Strong prevailing winds will disperse any air contaminants, but more moderate than 2A because of buffering effect of nearby forest x Plant and access route may affect mountain goats and/or other wildlife more than sites 1, 2A, and 4 x Road and pipelines may cause HADD for Kaypros Creek or Teigen Creek, and potential downstream effects of accidents to Unuk River with cross-border implications ✓ Direct footprint will not affect fish habitat	✓ Large available area allows adequate spacing of camp and process plant to reduce noise effects on staff x Site separated from TMF makes staff accommodation more isolated and transport more difficult due to increased safety risks x Additional human presence to and from process plant may affect wildlife disturbance and hunting/trapping x Presence of process plant may reduce visual attractiveness for heli-ski runs in the area x Process plant location may affect Skii km Lax Ha traditional land use (i.e., reported trail and harvesting)	x Not adjacent to the selected TMF (Section 33.5) presenting extra transport needs ✓ A variety of elevations available for site x Requires extra transmission line construction ✓ Foundations suitable for process plant; steeply dipping hard sandstone bedrock under thin silty/sandy soil ✓ Low geohazard risk to process plant x Access has same avalanche risks as Plant Site 2A x Relies on access via original Teigen Creek road, which was not chosen as TMF access route (Section 33.7) x Some resourcing redundancy for personnel camps	x Higher access road construction costs than plant sites 1 and 4 x Higher operation costs than Plant Site 1 due to longer access road for ore concentrate and reagent hauling ✓ Reduces cost of ore transport compared to Plant Site 1, and tailing pipeline would flow by gravity ✓ Lower elevation than Plant Site 2A, making ore transport less expensive	**CHALLENGING**
	Challenging	Acceptable	Challenging	Acceptable	
Plant Site 3 (Upper Unuk Valley)	x Will require the most new road construction of any site causing the most habitat fragmentation x Will affect extra undisturbed natural systems, as not adjacent to TMF site x Strong wind in the area may disperse air contaminants farther than other sites x Accident risks to water quality may affect Unuk River downstream with potential cross-border issues x Ungulate winter range (UWR) in the vicinity of the process plant, and site most likely to affect undisturbed wildlife than other sites x Road and pipelines may cause HADD for Kaypros Creek or Teigen Creek, and accidents may affect Unuk River with cross-border implications ✓ Direct footprint will not affect fish habitat	✓ Some more construction jobs for site accommodation and road may be created for this option compared to Plant Site 4 ✓ Large available area allows adequate spacing of camp and process plant to reduce noise effects on staff x Additional human presence to and from process plant may affect hunting/trapping due to wildlife disturbance and habitat fragmentation x Site separated from TMF makes staff accommodation more isolated and transport more difficult due to increased safety risks x Presence of process plant may reduce visual attractiveness for heli-ski runs in the area x Process plant location may affect Skii km Lax Ha traditional land use (i.e., reported trail and harvesting)	x Not adjacent to the selected TMF (Section 33.5), presenting extra transport needs x Requires longest extra transmission line construction x Construction will require extensive site preparation x Access to site will be the most challenging to establish and longest to construct ✓ Low geohazard risk to process plant x Access route has same avalanche risks as Plant Site 2A x Some resourcing redundancy for personnel camps	x Highest access construction costs than all other options from longest access route x Higher operation costs than plant sites 1, 2A, and 2B from reagent and concentrate hauling ✓ Reduces cost of ore transport compared to Plant Site 1, and tailing pipeline would flow by gravity x Higher materials and supplies transport costs than plant sites 1, 2A, and 2B	**CHALLENGING**
	Challenging	Acceptable	Challenging	Challenging	

(continued)

Table 33.6-2. Summary Comparison of Process Plant Location Alternatives (completed)

Alternative	ATTRIBUTE RATINGS AGAINST FOUR PERFORMANCE OBJECTIVES				OVERALL ALTERNATIVE RATING
	Environmental	Social	Technical	Project Economic	
Plant Site 4 (Sulphurets / Mitchell)	✓ ● Avoids need for separate access road and larger ore transport tunnel ✓ ● Will minimize air quality effects compared to other sites as will be minor compared to mining activities (i.e., blasting) and contained within Mine Site area x ● Accident risks to water quality are minimized due to extensive diversion and water storage systems ✓ ● No wetland disturbance anticipated ✓ ● Less likely to affect wildlife compared to other process plant sites, as existing disturbance from mining will be greater ✓ ● Direct footprint will not affect fish habitat x ● More spill risks for materials and supplies transport (i.e., reagents) due to steep grades on access road compared to other options; may introduce cross-border complications for spills located in Unuk River watershed x ● More spill risks for tailing which would have to be pumped 23 km from the Mine Site	✓ ● A centralized large camp would likely be more attractive to workforce and have more amenities ✓ ● Known low likelihood of disturbance of archaeological or heritage sites due to recent glaciation x ● Larger camp may make social issues such as substance abuse more challenging to manage ✓ ● Presence of process plant is minimal compared to other facilities and infrastructure, minimizing the relative visual effect compared to other sites ✓ ● Not likely to create material increases in adverse effects on hunting compared to the rest of the Mine Site infrastructure x ● Concentration of facilities in Mine Site may elevate noise stress for more staff ✓ ● Less likely to affect downstream fishing ✓ ● Presence of process plant will not noticeably affect visual attractiveness for heli-ski runs in the area compared to changes to visual quality from major components such as pits and RSFs ✓ ● There is no known traditional use of trails/harvesting in the Mine Site area, so no effects to traditional use from this plant site	x ● Placement unfeasible due to geographic lack of suitable space ✓ ● Electric power is easily available at the Mine Site x ● Steep grades, rugged terrain, and generally poor foundation x ● Significant avalanche and debris flow geohazard risks, and little land with suitable foundations x ● The only viable location is adjacent to Mitchell Pit, but this land must be used for waste rock storage	x ● Highest construction costs for process plant due to site levelling, foundation preparation, and geohazard mitigation x ● Highest operational reagent and concentrate haul costs due to longest haul distance (58 km Eskay Creek Mine road + 32 km extension + 36 km further along Highway 37 = 126 km) compared to 18 km Treaty Creek access road) ✓ ● Lowest ore transport costs from pits to process plant of all options, but would still need to transport tailing to TMF ✓ ● Slightly lower costs for single accommodation camp	**UNFEASIBLE**
	Preferred	Preferred	Unfeasible	Acceptable	

Notes:
✓ = advantage, x = disadvantage, ● = Preferred, ● = Acceptable, ● = Challenging, ● = Unfeasible; See Table 33.2-1 for attribute ranking specific definitions
HADD = Harmful alteration, disruption, or destruction, as defined under the *Fisheries Act* (1985a); TMF = tailing management facility

Atmospheric emissions will also be the highest for the process plant site with the longest access route, which is Plant Site 4, followed by 3, 2A/2B, and then 1, the latter with the least vehicle-related emissions. Site 4 has the advantage of constraining air quality and noise effects to the Mine Site rather than creating effects over a wider regional area. Aquatic and wetland habitat at each of the sites, and along site access routes varies per site. Plant Site 3 faces the largest challenges with crossing waterbodies and potential harmful alteration, disruption, or destruction (HADD) compared to other sites.

The remote location of all the potential process plant sites minimizes the potential for social effects from construction and operation. Process Plant sites 1, 2A/2B, and 3 may interfere with the views currently enjoyed by recreational tourism heli-skiing operations in the region, or with trapping or hunting opportunities in these areas, reducing business and other development opportunities. These kinds of effects from Plant Site 4 in the Mine Site will be relatively small in comparison to other mine components (e.g., pits, rock storage facilities, crushing, and grinding). Plant sites 1, 2A/2B, and 3 are also situated in areas where the Skii km Lax Ha have reported traditional land use (trails and harvesting activities), which could lead to effects on traditional use in these plant site areas. However, there is no reported traditional land use in the Mine Site area of Plant 4, so no effects on traditional use are anticipated for this process plant.

Potential effects on personnel vary depending on the process plant and associated TMF site chosen. It is anticipated that plant sites 1, 2A/2B, and 3 will share camps between TMF and process plant staff for construction and operation. Plant 4 has the advantage of being able to combine camps to reduce redundancies and potentially provide the most amenities for staff, but this may also come with related increases in potential issues from crowding, substance abuse, and more noise exposure to more personnel from mining activities.

For the reasons discussed above, Plant Site 4 is considered as the preferred location to minimize potential environmental and social effects.

33.6.4 Selected Alternative

The process plant location deemed the most appropriate for the Project, as summarized in Table 33.6-2, is Plant Site 1. While preferable against environmental and social performance objectives, Plant Site 4 was found to not be technically feasible. Plant Site 1 is feasible from an engineering perspective, has no significant environmental effects that cannot be acceptably mitigated, is acceptable from a social perspective, and is financially viable. Coupled with the adjacent TMF and Mitchell-Treaty Twinned Tunnels (MTT) ore transport system from the Mine Site, Plant Site 1 provides the most cost effective and environmentally certain alternative for both ore concentrate production and tailing management. It also reduces potential tailing, reagent, or ore concentrate accident spill risks compared to Plant Site 4.

Baseline studies revealed that the general habitat of the Plant Site 1 location in the Treaty divide contains wetland areas. As a result, since originally selecting this site, Seabridge has adjusted the placement of the Process Plant within the PTMA to a location where potential adverse effects to wetlands will be avoided and minimized (see Chapter 16 for full wetland effects assessment).

33.7 Access Routes

33.7.1 Purpose and Background

The KSM Project will require the following access from Highway 37 to transport personnel, equipment, materials, and ore concentrates:

- road access to the Mine Site; and
- road access to the PTMA.

The Project is currently accessible only via helicopter. While the access route to the Mine Site is being constructed, a temporary route will be required to expedite construction activities. Helicopters will continue to be required to facilitate access to the two main areas of the Project. Alternatives for the transport of materials and supplies to the Project besides roads (i.e., oceanic shipping) were not considered. Seabridge identified alternative routes, illustrated in Figure 33.7-1, to access the Mine Site and the PTMA from preliminary feasibility studies that were conducted for the Project. The primary feasibility constraints used to identify access alternatives included topography, geohazards, avoiding fish habitat HADD, total route length, and construction costs.

The assessments for the main Mine Site road and temporary access route based on the performance objectives methodology are provided below in Section 33.7.2. An MAA approach (method described in Section 33.5 and in the full MAA report provided in Appendix 33-B) was used for the PTMA access road assessment, as summarized in Section 33.7-3.

33.7.2 Mine Site Access Alternatives Assessment

33.7.2.1 Alternatives Identification

Road access to the Mine Site will be necessary to transport personnel and cargo to and from the site. Cargo will consist of both non-hazardous and hazardous materials. Non-hazardous cargo will include personnel, materials, equipment, and supplies. Potentially hazardous materials will include diesel fuel and lubricants, ore concentrates, lime, reagents, and explosives. Mining equipment will include electric and diesel shovels, haul trucks, bulldozers, loaders, graders, blasthole drills, and explosives trucks. In general, equipment required for open pit mining is larger and heavier and requires more robust route conditions for safe transport than equipment for underground mining.

Traffic volumes are anticipated to be the same regardless of route chosen. The total average annual one-way trips from the Mine Site to Highway 37 are an estimated 2,883 during construction (5 years) and 1,004 during operation (51.5 years), with trips dropping to zero for the closure (3 years) and post-closure (250 years) phases, as traffic from the Mine Site will be re-routed through the MTT and along the access road from the PTMA (Rescan 2012). The Mine Site access road will be required to support safe and consistent use by heavy-duty traffic year-round for all Project phases.

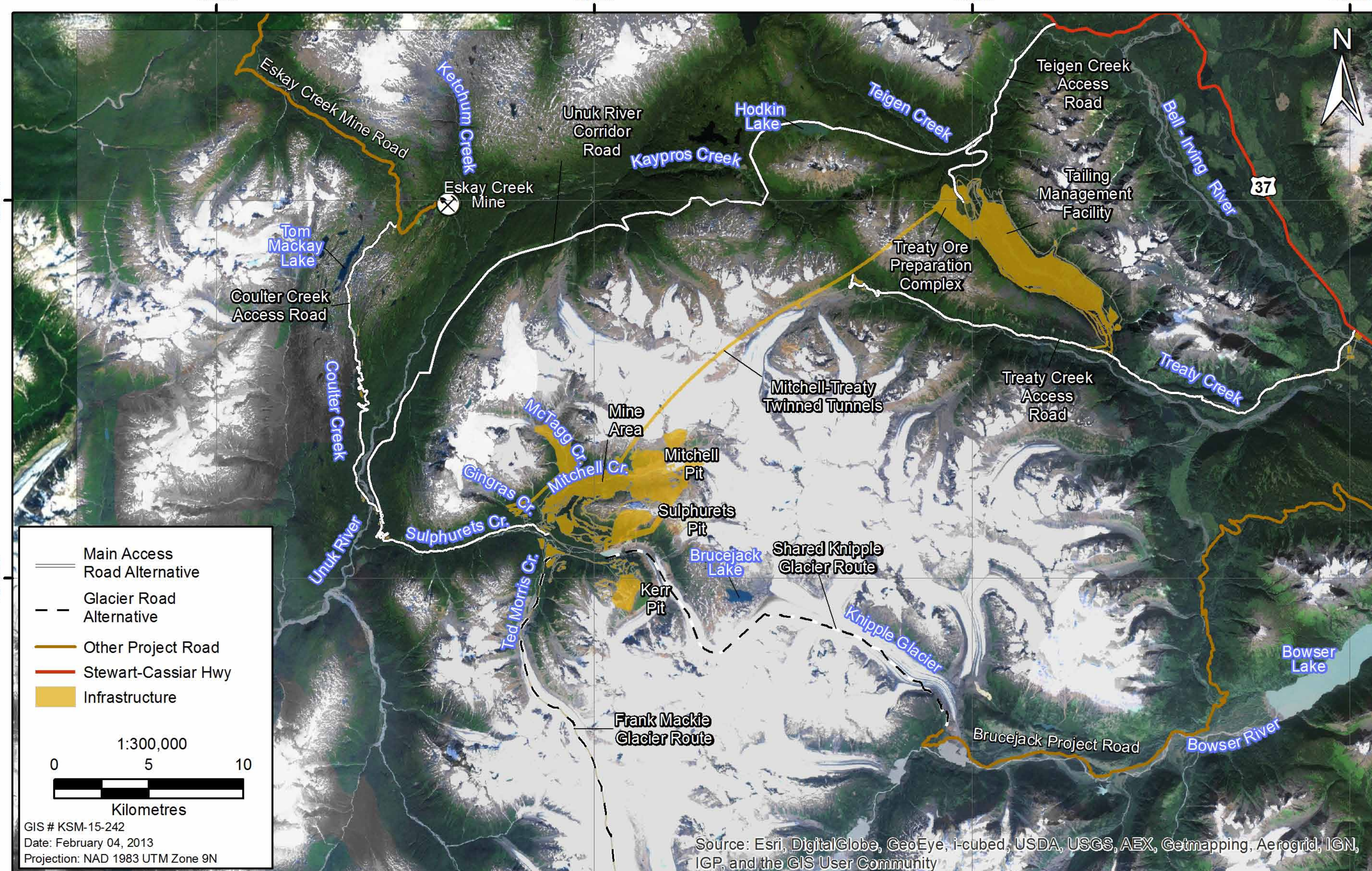

Figure 33.7-1

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At the start of the Project there will be no road to the site; however, heavy earthmoving equipment, water treatment supplies, a portable construction camp, construction materials and supplies, explosives, and diesel fuel will need to be transported to a staging area in the Sulphurets Creek drainage to start preliminary on-site construction work. For this reason, a temporary access route to the Mine Site for use during the first two or three years of the construction phase will be required to establish construction headings to support the development of the permanent Mine Site access road, diversion channels, construction of the transport tunnel, and pre-stripping of the deposit areas. The establishment of a construction heading for a permanent access road will reduce the construction period by a year and will achieve comparable schedule reductions for the tunnel if a second heading is established.

As illustrated in Figure 33.7-1, the alternative Mine Site access routes identified from preliminary feasibility studies are:

- the CCAR (permanent road);
- the Unuk River Corridor road (permanent road);
- the shared Knipple Glacier route (temporary or permanent route; partially shared with the Brucejack Project route controlled by Pretium Resources Inc. [Pretivm]); and
- the Frank Mackie Glacier access route (temporary route).

33.7.2.1.1 Coulter Creek Access Road

The existing Eskay Creek Mine road was constructed in 1994 from Highway 37 to service the Eskay Creek Mine, located about 18 km (straight line distance) northwest of the Mitchell Pit, as shown in Figure 33.7-1. The road was in daily use until the mine's 2008 closure, and it is being maintained for mine deactivation. The road is held under a Special Use Permit from the BC Ministry of Forests, Lands, and Natural Resource Operations by Barrick Gold Corporation and has been constructed to the ministry's standards to handle heavy truck traffic.

Extending southwest from the Highway 37 intersection, the 105 km CCAR will consist of 35 km of new road starting at mile 70 on the Eskay Creek Mine road. The proposed CCAR will follow an existing mine road south to Tom Mackay Lake and then continue south along a forested ridge east of Coulter Creek before continuing along a rocky ridge to the Unuk River, where a bridge will be required. The road then veers east through a series of switchbacks from km 23 to 25, into Sulphurets Creek Valley, and then finally crosses the Mitchell Creek Valley into the Mine Site.

A previous alignment was considered for CCAR, commencing at the Eskay Creek Mine, descending directly eastward from that point into the Unuk River Valley. From there, it would cross the Unuk River and proceed down the east bank to the mouth of Sulphurets Creek. It would continue up the north side of Sulphurets Creek, following the same alignment as described in the foregoing paragraph (dotted line on Figure 33.7-1). This route was measured as 32 km to the Mitchell deposit. This alignment was deemed to have fatal flaws based on preliminary feasibility criteria. Due to the steep descent into the Unuk River Valley and the need to follow the river for a longer distance, which would have higher potential to affect aquatic habitat and fish, this route was not considered further.

33.7.2.1.2 Unuk River Corridor Road

The permanent Unuk River corridor road (Unuk road) alternative will require new construction totalling 73 km in length. This route extends from Highway 37 north of Bell II, sharing 19 km with the Teigen Creek access route alternative to the PTMA, as shown in Figure 33.7-1. The road then continues up the south side of the Teigen Creek Valley, looping around the north side of Hodkin Lake and then down through the Kaypros Creek and Unuk River valleys to the Sulphurets Creek Valley, where it shares the same 27 km alignment with the CCAR from this point.

33.7.2.1.3 Shared Knipple Glacier Route

The third alternative access route to the Mine Site is the shared Knipple Glacier route, which consists of a partly shared route with the Brucejack Mine exploration camp controlled by Pretivm to the east of the KSM Project, as illustrated in Figure 33.7-1. This route was considered by Seabridge for temporary access, as well as potentially for permanent access to the Mine Site.

The shared Knipple Glacier route consists of a previously used route to transport equipment to the Brucejack Lake area to support underground exploration of the Sulphurets-Bruceside property in the late 1980s. Pretivm has recently reactivated this route for further Brucejack Project exploration work, including putting in a working road (labelled "Brucejack Project Road" in Figure 33.7-1) that heads westward to the area of Brucejack Lake from Highway 37, heading north around Bowser Lake, and then turning into a glacier route from the foot of the Knipple Glacier up to the Brucejack Lake area. The existing route to the Brucejack Project is about 75 km long and involves about a 15 km stretch of glacier ice travel up the Knipple Glacier. New elements of the route for use by the KSM Project will extend the glacier road from the vicinity of Brucejack Lake into the Sulphurets Creek Valley and up into the Mine Site as shown in Figure 33.7-1. The new route will require construction of a bulldozer trail down the steep face of the glacier upstream of Sulphurets Lake. The total length of this route would be roughly 87 km from Highway 37, with about 22 km of glacier travel.

33.7.2.1.4 Frank Mackie Glacier Access Route

The Frank Mackie Glacier access route (illustrated in Figure 33.7-1 and in Section 33.9, Figure 33.9-1) is considered for temporary access for the Project. This route stems from an existing 37 km gravel road from Stewart, BC, through Hyder, Alaska, alongside the Salmon Glacier to the old airstrip near the Granduc mill site. This route will start at the Berendon Glacier to access and then cross the Frank Mackie Glacier up the Mine Site via Ted Morris Creek Valley (see Figure 33.9-1). Most of the route length for this option (32.8 km) will involve glacier travel, with a total length of about 38 km. It will take approximately two weeks to construct this route.

33.7.2.2 Comparison of Alternatives

The alternative access route attributes are compared in Table 33.7-1, based on stated environmental, social, technical, and Project economic performance objectives.

33.7.2.2.1 Technical and Project Economic Considerations

Regarding temporary access to the Mine Site, the relatively warm climate in the area, high winter snowfall quantity, and topography makes the construction of an "ice road," where regular highway vehicles could be employed, unfeasible for both routes. Instead, a "snow road" would have to be built, where tracked equipment pulling skid-mounted sleds will be used to haul materials, equipment, and supplies as needed. Maximum weight and sizes of loads is limited to a maximum weight of 33 t and a width of 2.3 m. While the transportation of equipment over glacial ice has proven to be effective in this region using these methods, geohazard risks (i.e., from crevasses and avalanches) from adjacent slopes are present and will require special glacial travel management plans and protocols to be developed and implemented.

Shorter route lengths with less challenging terrain are preferred for the Project as they reduce capital and operating expenses as well as construction and operation time. Although a significant portion of the shared Knipple Glacier route has already been re-activated for use by the Brucejack Project, this route would still require upgrading to meet the additional needs of the KSM Project in transporting heavier machinery for use in open pit mining as well as underground mining, likely resulting in more off-ice construction activities than the temporary Frank Mackie Glacier access route. The shared Knipple Glacier route also would involve a much longer haul distance than the Frank Mackie Glacier access route from Stewart.

As shown in Table 33.7-1, the shared Knipple Glacier route for use as either temporary or permanent access to the Mine Site has also been determined to be technically unfeasible—particularly for long-term access—due to the treacherous topography that would need to be constructed from Brucejack Lake to the Project, and the continued reliance on this route over the Project life.

While glacier travel is considered feasible for the Project on a temporary basis, its continued use would increase travel risks associated with glacier use for transport for the Project, particularly during operation when reagents are being transported to the Project and ore concentrates are being shipped out on a regular basis. Over the life of the Project, these risks would multiply, and are therefore considered to be unfeasible compared to the safer options of the CCAR or Unuk road. For this reason, the shared Knipple Glacier route is considered unfeasible as a whole for use by the Project, leaving the Frank Mackie Glacier access route as the only technically and economically viable temporary access alternative for the Project.

CCAR is identified as the preferred alternative compared to the Unuk road in Table 33.7-1 because the technical and economic attributes of CCAR are rated either preferred or acceptable, while the Unuk road poses both technical and economic challenges to construct and operate. While the total route length would be shorter and grades lower for the Unuk road, construction of 73 km of new road would lead to longer construction schedules (about a year longer) and almost double capital expenses ($93.8 M instead of $51.5 M). Additionally, the rough terrain and requirement for several more bridges along the Unuk road pose significant challenges for construction crews (accessing this route alignment for sampling and surveying purposes has been very challenging to date).

Table 33.7-1. Summary Comparison of Mine Site Access Alternatives

Alternative	ATTRIBUTE RATINGS AGAINST FOUR PERFORMANCE OBJECTIVES				OVERALL ALTERNATIVE RATING
	Environmental	Social	Technical	Project Economic	
Coulter Creek Access Road (Permanent; 35 km new; 105 km combined from Highway 37)	✓ ● Less new construction (35 km) with lower associated habitat disturbance, and atmospheric emissions x ● Longer overall route associated with higher atmospheric emissions, and higher wildlife disturbance/collision risk x ● Fish habitat compensation will be required ✓ ● Only one crossing of Unuk River means lower disturbance of aquatic habitat and fish; Coulter Creek is too steep to support fish for much of its length x ● May disturb mountain goat ungulate winter range (UWR)	x ● Lower employment during construction due to shorter new road construction ✓ ● Lower risk of disrupting heritage and archaeological sites with less new construction x ● Greater safety risk due to steep grades between Iskut and Unuk rivers ✓ ● Shorter new road length minimizes adverse aesthetic effects to heli-ski operations in the area ✓●There may be a minor disruption of land use where Skii km Lax Ha have a reported traditional use trail at the Unuk River crossing	✓ ● Shorter new construction along easier route facilitates construction schedule ✓ ● Construction estimated to take 2 years x ● Steeper grades required for section between Iskut and Unuk rivers ✓ ● Requires fewer major bridges x ● Avalanche hazards in Sulphurets Canyon require avalanche mitigation measures	✓ ● Construction cost reduced to almost half of Unuk River alternative ✓ ● Shorter construction period ✓ ● May share Iskut road with other users, thereby reducing maintenance costs x ● Longer overall route to Highway 37 will increase hauling costs	**PREFERRED**
	Acceptable	Preferred	Preferred	Preferred	
Unuk River Corridor Road (Permanent; 73 km new from Highway 37	✓ ● Shorter overall road length, associated with lower air quality, noise, and GHG emissions, and lower wildlife disturbance/collision risk x ● More new road construction (73 km) with associated habitat disturbance and higher atmospheric emissions x ● Additional fish habitat compensation will be required x ● Additional disturbance of aquatic habitat and fish (e.g., salmon) from two bridges along Teigen Creek and its tributaries, and two bridges over major tributaries to the Unuk River (including Kaypros Creek) x ● May disturb UWR for mountain goats; effects can be reduced to acceptable levels with mitigation	✓ ● Higher employment during construction due to longer new road construction x ● Higher risk of disrupting heritage and archaeological sites with more new construction. x ● Greater safety risks in section of road shared with PTMA access route due to high traffic levels x ● Longer new route length increases potential aesthetic adverse effects to heli-ski operations in the area x●There may be disruption of land use where Skii km Lax Ha have reported a traditional use trail along the Unuk River valley	x ● Longer new construction along more challenging route may pose delays x ● Construction estimated to take 3 years x ● Requires more major bridges construction x ● Long sections of road require steep side cast construction due to steep slopes x ● Road will be challenging to keep open during winter due to heavy snow and steep terrain x ● Same avalanche hazards as CCAR in Sulphurets Canyon, plus additional avalanche hazards in Teigen Canyon	x ● Very high construction cost x ● More new construction required with fewer headings, therefore higher construction phase costs ✓ ● Shorter overall route to Highway 37 will decrease hauling	**CHALLENGING**
	Challenging	Acceptable	Challenging	Challenging	

(continued)

Table 33.7-1. Summary Comparison of Mine Site Access Alternatives (completed)

Alternative	ATTRIBUTE RATINGS AGAINST FOUR PERFORMANCE OBJECTIVES				OVERALL ALTERNATIVE RATING
	Environmental	Social	Technical	Project Economic	
Shared Knipple Glacier Route (Permanent or temporary use; 87 km total, 22 km glacier from Highway 37)	✓ ● Involves the least new road construction, although may require road upgrades along much of route increasing air emissions and ecological disturbance x ● Geohazard risks involved with glacier travel increases the chance of spills; BMPs bring risk to acceptable levels, but the risks increase over the life of the Project x ● Longer temporary route length potentially affecting fish habitat	x ● Local concerns, including for traditional use, about Bowser Lake fisheries being adversely affected x ● OH&S concerns regarding glacier travel (i.e., crevasses) become more pronounced with usage over Project life x ● May adversely affect commercial recreation operators such as heli-skiing	✓ ● Some parts of the alignment are already established by Pretivm for Brucejack Project exploration, reducing road work to be done; but will need upgrades to handle larger equipment for open pit mining at KSM Project x ● Geohazards from glacier crevasses and avalanches, though manageable with BMPs, increase risks significantly compared to normal road travel, especially for large equipment required for open pit mining x ● Route extension from Brucejack Lake area is over treacherous terrain, which presents large construction challenges and travel hazards, especially over life of Project x ● Longer of two temporary routes, which would impede scheduling	✓ ● Much of route already reactivated by Pretivm for the Brucejack Project, but reduced construction costs may be offset by upgrade work x ● Extending route over treacherous terrain from Brucejack Lake area increases construction and operation costs x ● Longer of the two temporary route options increases operation costs	**UNFEASIBLE**
	Acceptable	Challenging	Unfeasible	Acceptable	
Frank Mackie Glacier Access Route (Temporary use; 38.4 km (32.8 km glacier); 37 km from Stewart)	x ● Will lead to atmospheric emissions; more than other option during construction but less during operation ✓ ● Less sensitive fish habitat along route x ● Requires stream crossings, but effects can be minimized to acceptable levels through mitigation ✓ ● Less wildlife and vegetation habitat along route as consists mostly of glacier travel ✓ ● Reduces reliance on helicopters, which minimizes disturbance to wildlife such as mountain goats	✓ ● Avoids fisheries concerns in Bowser Lake, including for traditional use x ● May adversely affect commercial recreation operators such as heli-skiing x ● OH&S safety concerns regarding glacier travel (i.e., crevasses) but can be mitigated with BMPs	✓ ● Less off-ice road construction/re-activation ✓ ● Shorter route improves scheduling and construction time (two weeks) ✓ ● Relatively easy terrain for road construction from Granduc mill site to toe of glacier x ● Geohazard risks from crevasses and avalanches, but can be mitigated with BMPs, especially over short term ✓ ● Viability of route has been recently demonstrated	✓ ● Lower estimated construction costs ✓ ● Lower operation costs due to shorter length from Stewart	**PREFERRED**
	Preferred	Preferred	Acceptable	Preferred	

Notes:
✓ = advantage, x = disadvantage, ● = Preferred, ● = Acceptable, ● = Challenging, ● = Unfeasible; See Table 33.2-1 for attribute ranking specific definitions
GHG = greenhouse gases; TMF = tailing management facility, OH&S = occupational health and safety; BMP = best management practice; UWR = ungulate winter range

33.7.2.2.2 Environmental and Social Considerations

Environmental attributes considered to compare access routes for the Project include potential effects to fish and fish habitat due to bridge crossings, soil erosion, and sedimentation of water courses from disturbed road surfaces, loss of wetlands, habitat loss and fragmentation, disturbance to ungulate winter range for mountain goats, vehicle-related air emissions during transportation, and increased risk of collisions and sensory disturbance to wildlife. Social attributes considered in the assessment included OH&S risks, minimizing adverse effects to traditional land use, minimizing potential adverse effects to downstream fisheries (including for traditional use), and minimizing potential adverse effects to recreational tour and heli-ski operators in the area.

For temporary access to the Mine Site, the Temporary Frank Mackie Glacier access route is considered to be the preferred route based on reduced net disturbance to terrestrial and aquatic systems, as well as fishery resources used for commercial, recreational, and traditional use purposes in the Bowser Lake area, as shown in Table 33.7-1. The shared Knipple Glacier route is considered acceptable based on environmental attributes, but poses challenges for use based on social considerations due to potential concerns on risks to Bowser Lake fishery resources from the use of this road either temporarily or permanently by the Project (i.e., from spill risks of reagent and ore concentrate transport along Bowser River). Extended use of the shared Knipple Glacier route would also prolong glacier travel by the Project, increasing spill and OH&S risks over the Project life that can be managed with BMPs over the short term, but would go up with prolonged use.

A comparison of the environmental attributes for CCAR and Unuk road, as listed in Table 33.7-1, indicates that the CCAR is the preferred Mine Site access road, primarily due to the reduced distance of new road and bridge construction, and because the additional amount of potential fish habitat disturbance and compensation poses challenges to the Unuk road alternative. The social attributes for CCAR are also preferred compared to Unuk road as shown in Table 33.7-1. In particular, the CCAR minimizes potential adverse effects to traditional land use along the Unuk River Valley where there is a reported foot trail used by the Skii km Lax Ha (Chapter 30, Appendix 30-B, Figure 4.4-1), as well as minimizing effects to heritage and visual aesthetic attributes associated with travel and recreation in the area.

33.7.2.3 Selected Alternative

The selected temporary glacier access alternative to the Mine Site is the Temporary Frank Mackie Glacier access route, primarily as it received preferred or acceptable ratings for all the four performance objective categories, and ranked as preferred overall. The alternative shared Knipple Glacier route was eliminated due to technical construction barriers, but also because it posed challenges based on social attributes.

The CCAR was selected as the road alternative for long-term access to the Mine Site because the social, technical, and Project economic attributes were more favourable than those for the Unuk River access road, which faces environmental, social, and Project economic challenges. The CCAR has the main advantage of requiring less new construction, with associated environmental, social, technical, and Project economic benefits, in particular minimizing the potential for aquatic habitat disturbance.

33.7.3 Processing and Tailing Management Area Access Alternatives Assessment

Road access to the PTMA for the KSM Project will be necessary to deliver equipment and materials for the construction and operation of the TMF, Treaty Process Plant, and ore transport tunnel; deliver reagents, grinding media, and fuel during operation; and to provide trucks passage to haul ore concentrates to market. The PTMA access road will also enable employee access to the PTMA, as well as allow regular transport of supplies to support the accommodation complex. Because the Project will be running year-round, the PTMA access road will need to be available for safe and consistent use by heavy-duty traffic all year. The total average annual one-way trips anticipated from the PTMA to Highway 37 are 5,362 during construction and 30,154 during operation, 5,834 for closure, and 2,288 post-closure (Rescan 2012).

An MAA, *Assessment of Alternatives for Teigen/Treaty Tailing Management Facility Access Road* (Rescan 2012), was conducted to determine the preferred route to the PTMA (provided in full in Appendix 33-B).

33.7.3.1 Alternatives Identification

The following three access route options were assessed in the MAA:

1. Teigen access road;
2. Treaty access road; and
3. Teigen South/Treaty West access road.

33.7.3.1.1 Teigen Access Road

This route includes a new road parallel to Teigen Creek, connecting the Treaty Process Plant to Highway 37 about 14 km to the northeast, and a second route, the Tunnel Spur access road, that would run between the Process Plant and the intermediate portals of the MTT, running along West Teigen Creek.

33.7.3.1.2 Treaty Access Road

This new road would stem from Highway 37 about 19 km south of Bell II and head west along the north side of the Treaty Creek Valley for 17.9 km to the Mitchell-Treaty Saddle Area. The road would then turn north, along the Treaty Creek spur road, parallel to the north tributary of Treaty Creek for about 12 km to the Treaty Process Plant and TMF.

33.7.3.1.3 Teigen South/Treaty West Access Road

This route would consist of a new 14 km road parallel to Teigen Creek connecting the Treaty Process Plant to Highway 37, and a second road between the Treaty Process Plant and the intermediate portals of the MTT paralleling first South Teigen Creek, then North Treaty Creek, and finally Treaty Creek.

33.7.3.2 Comparison of Alternatives

Three route options were characterized and compared based on environmental, technical, social, and economic factors. A multidisciplinary team selected the evaluation criteria for the MAA as described in Appendix 33-B. The characterization criteria are Project-specific but fall under the following four broad categories, referred to as "accounts":

- *Environmental characterization* – This account describes the local and regional environment surrounding each proposed PTMA access road alternative. Elements such as climate, geology, hydrology, hydrogeology, water quality, and potential effects on fish and wildlife are considered.
- *Technical characterization* – This account describes the engineered elements of each alternative such as total road length, road length crossing potentially unstable terrain, road length susceptible to landslides, road length susceptible to snow avalanches, road grade, road elevation, road length susceptible to flooding, fish-bearing stream crossings, road length with metal leaching and/or acid rock drainage concern, number of bridge and major culverts, volume of soil and rock excavation, and volume of fill construction.
- *Project economic characterization* – The account describes the life of project economics. All aspects of the road construction, operation, and closure are considered.
- *Social and archaeological characterization* – This account describes how each proposed PTMA access road alternative may influence local and regional land users. Elements considered include characterization and evaluation of land use, cultural significance, presence of archaeological sites, and employment and/or training opportunities.

Each account considers short- and long-term issues associated with construction through operation, mine closure, and, ultimately, post-closure maintenance and monitoring. Detailed characterization data and summary tables are provided in Appendix 33-B.

A multiple accounts ledger (Table 33.7-2) identifies criteria that differentiate alternatives so that TMF options can be evaluated relative to one another. This ledger is derived from the characterization data. The multiple accounts ledger consists of two elements: sub-accounts (i.e., evaluation criteria) and indicators (i.e., measurement criteria).

Table 33.7-2. KSM Project Tailing Management Facility Alternatives Analysis Multiple Accounts Ledger

Account	Sub-Account	Indicator
Environmental	Rare and Endangered Ecosystems and Species	Presence of rare and endangered terrestrial ecosystems
		Presence of rare and endangered wildlife species; excluding grizzly bear, mountain ungulates, moose, and American marten
		Presence of rare and endangered fish species
	Terrestrial Habitat	Mountain goat habitat
		Moose habitat
		Western toad habitat

(continued)

Table 33.7-2. KSM Project Tailing Management Facility Alternatives Analysis Multiple Accounts Ledger (completed)

Account	Sub-Account	Indicator
Environment *(cont'd)*	Fisheries Value	Number of road crossings affecting fish-bearing streams
		Number of fish species potentially affected
Socio-economic and Archaeological	Archaeology	Number of archaeological sites
		Importance of archaeological sites
	Aboriginal Interests[1]	Nisga'a Nation stated preference
		Tahltan stated preference
		Skii km Lax Ha stated preference
		Gitanyow stated preference
	Commercial Land Use Stakeholders	Number of traplines affected
Technical	Road Operation	Total road length
		Road length with 6 to 10% grade
		Road length with > 10% grade
		Road elevation
	ML/ARD Potential	High ML/ARD potential
		Possible ML/ARD potential
	Excavation and Fill Volumes	Soil excavation volumes
		Rock excavation volumes
		Fill volume
	Geohazards	Terrain stability
		Landslides
		Snow avalanches
	Associated Structures	Bridge structure
		Major culverts
Economic	Total Road Cost	Total road cost

Note:
[1] During preparation of the *Assessment of Alternatives for Teigen/Treaty Tailing Management Facility Access Road* (Rescan 2012), Gitxsan Nation and wilp Skii km Lax Ha did not explicitly state a preference as to the PTMA access road, and are not included in this sub-account.

Following the same methodology as used above in Section 33.5.2 and as further described in Appendix 33-B, each indicator listed in Table 33.7-2 has a scoring descriptor, and was assigned relative weightings (Tables 33.7-3 to 33.7-9). The base-case weighting of each of the main accounts is provided in Table 33.5-3 with the environmental account rated the highest, the technical and socio-economic accounts given equal median weights, and the economic account the lowest (as per the Guidelines [Environment Canada 2011]).

Table 33.7-3. Account Weighting

Account	Weighting Base-case
Environment	6
Technical	3
Socio-economic	3
Project Economic	1.5

Table 33.7-4. Account: Environment

Sub-account	Weighting
Rare and Endangered Ecosystems and Species	5
Terrestrial Habitat	2
Fisheries Value	6

Table 33.7-5. Sub-account Indicator Weightings of the Environment Account

Sub-account	Indicator	Weighting Value
Rare and Endangered Ecosystems and Species	Presence of rare and endangered terrestrial ecosystems	5
	Presence of rare and endangered wildlife species, excluding grizzly bear, mountain ungulates, moose, and marten	2
	Presence of rare and endangered fish species	6
Terrestrial Habitat Loss	Mountain goat habitat	4
	Moose habitat	6
	Western toad habitat	1
Fisheries Value	Number of road crossings affecting fish-bearing streams	6
	Number of fish species potentially affected	3

Table 33.7-6. Account: Socio-economic and Archaeology

Sub-account	Weighting
Aboriginal Interests	6
Archaeology	4
Commercial Interests	1

Table 33.7-7. Sub-account Indicator Weightings of the Socio-economic Account

Sub-account	Indicator	Weighting Value
Archaeology	Number of archaeological sites	2
	Importance of archaeological sites	5

Table 33.7-8. Account: Technical

Sub-account	Weighting
Road Operation	2
ML/ARD Potential	2
Excavation and Fill Volumes	5
Geohazards	6
Associated Structures	2

Table 33.7-9. Sub-account Indicator Weightings of the Technical Account

Sub-account	Indicator	Weighting Value
Road Operation	Total road length	5
	Road length with 6 to 10% grade	1
	Road length with > 10% grade	4
ML/ARD Potential	High ML/ARD potential	3
	Possible ML/ARD potential	2
Excavation and Fill Volumes	Soil excavation volume	3
	Rock excavation volume	4
	Fill volume	3
Geohazards	Terrain stability	4
	Landslides	5
	Snow avalanches	6
Associated Structures	Bridge structures	3
	Major culverts	2

33.7.3.2.1 Environment Account

Table 33.7-4 provides the weighting values established for the environmental sub-accounts. Then, Table 33.7-5 lists the weighting values determined for each of the indicators for these environmental sub-accounts.

33.7.3.2.2 Socio-economic Account

Table 33.7-6 provides the weighting values established for the environmental sub-accounts. Then, Table 33.7-7 lists the weighting values determined for the archaeology sub-account; there was only one indicator for each of the other two sub-accounts, so these indicators did not need to be weighted against each other in the table.

33.7.3.2.3 Technical Account

Table 33.7-8 provides the weighting values established for the technical sub-accounts, and the weighting values determined for each of the indicators for these environmental sub-accounts are identified in Table 33.7-9.

33.7.3.2.4 Economic Account

There was only one sub-account (estimated cost) with one indicator (total cost) for the economic account, so there was no weighting performed for this account.

33.7.3.3 Selected Alternative

Using the weightings provided in Tables 33.7-3 to 33.7-9, combined with the indicator scores derived from the characterization data, a qualitative score for each of the candidate access road was calculated. Calculation methodology followed the Guidelines (Environment Canada 2011), and full calculation tables are provided in Appendix H of Appendix 33-B. The result of these calculations, i.e., the results of the MAA for the PTMA access road alternatives assessment, is provided in Table 33.7-10.

Table 33.7-10. KSM Project Processing and Tailing Management Area Access Road Multiple Account Analysis Results

	Teigen Creek Access Road	Treaty Creek Access Road	Teigen South/Treaty West Access Road
MAA Value	2.4	3.8	2.6

The result of the value-based MAA decision process was that the Treaty Creek access road (TCAR) is the most appropriate PTMA access road alternative (i.e., resulting in the highest value from the MAA process). The remaining two alternatives are less preferable, and roughly equivalent to each other.

33.8 Ore Handling

33.8.1 Purpose and Background

Both the alignment and method to transport ore from the Mine Site to the Process Plant in the PTMA were considered for the Project. The ore transport route is constrained by the location of the PTMA, which has been selected to be the Treaty Creek TMF site, as outlined in Section 33.5. The resulting alignment of the ore transport route is illustrated Figure 33.8-1, so this assessment therefore focuses on the assessment of alternative methods of ore handling for the Project.

33.8.2 Alternatives Identification

The PTMA is located is approximately 23 km northeast of the Mine Site, spanning a range of high, glaciated mountains. Transporting ore using haul trucks over land across or around these mountains is not technically or economically feasible due to the distance and challenging terrain. The only viable option is to transport ore through a tunnel (MTT), from the Mine Site to the PTMA as shown in Figure 33.8-1.

During the operation phase, an average of 130,000 tpd of ore will need to be conveyed through the 23 km MTT. Ore can be transported through tunnels via trucks, slurry pipelines, or conveyance systems. Truck transport was rejected due to the requirement for two-way traffic through the MTT, which would make the tunnel dimensions too large, leading to significant technical construction challenges and unfeasible economics. The remaining ore transport alternatives are a twin ore slurry pipeline or an ore conveyor belt system.

33.8.2.1 Ore Slurry Pipeline

This alternative consists of a twin slurry pipeline, shown in Figure 33.8-2, to transport ore from a portal near the Mitchell Pit to the Treaty Process Plant. To grind the ore fine enough to form a slurry amenable to piping will require the addition of a secondary processing plant with a grinding circuit and pumping station near the Mitchell Creek portal. Ore slurry transport systems also have higher water requirements than conveyance systems, which will also require the inclusion of a water return pipeline to recycle water in order to minimize water use for this option.

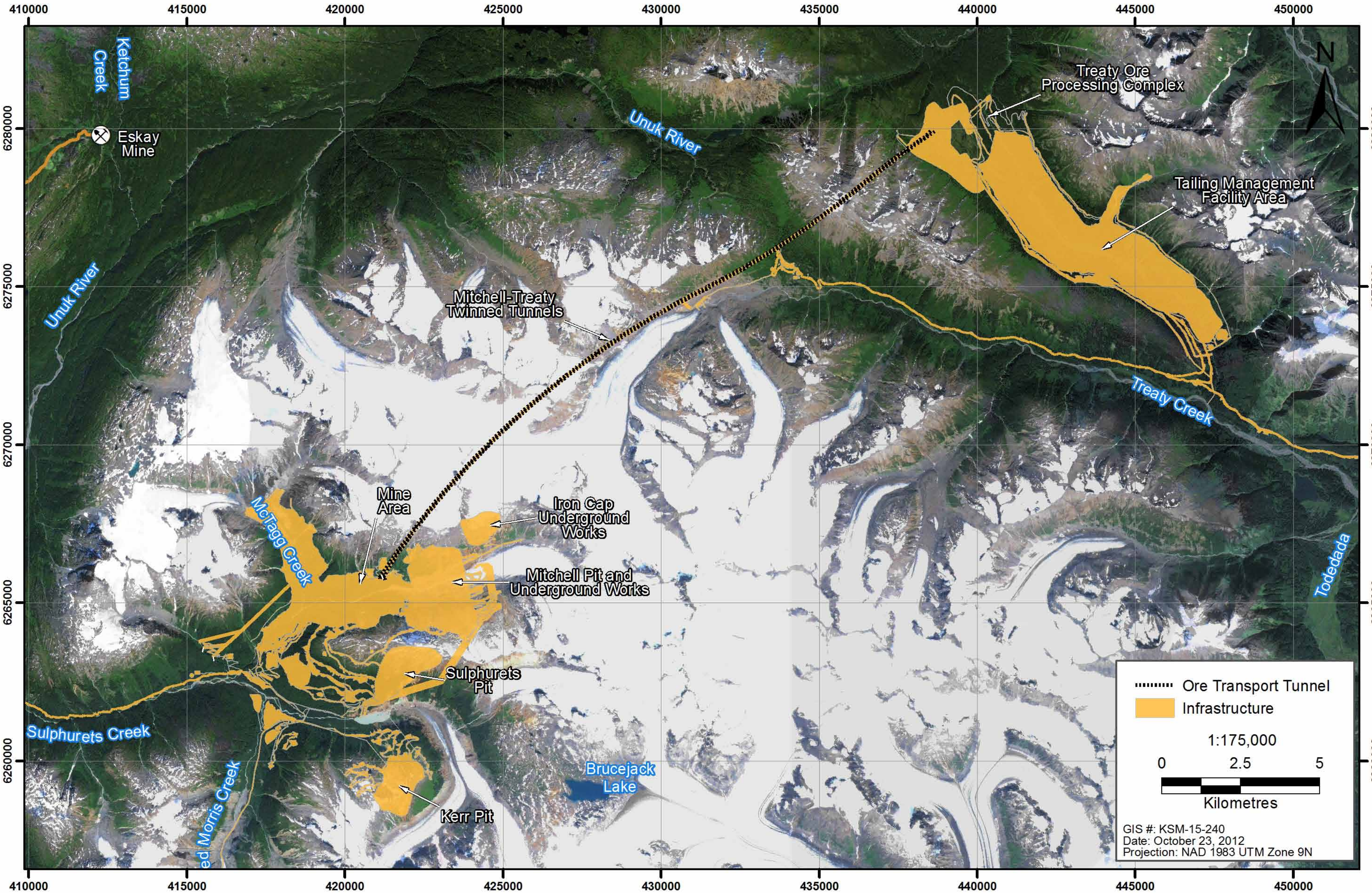
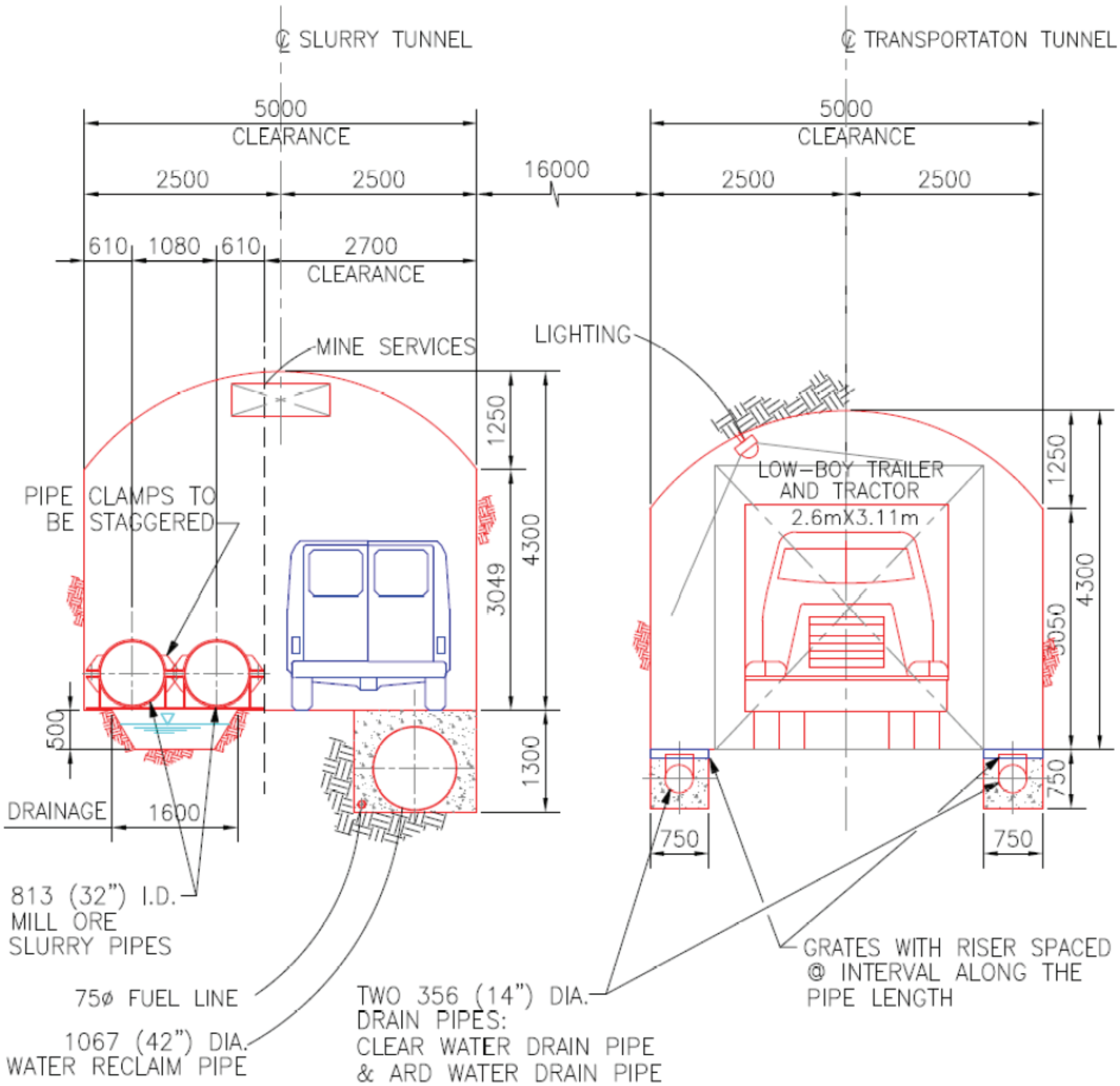


Figure 33.8-1

SEABRIDGE GOLD
KSM PROJECT

Ore Transport Route

Rescan
Engineers & Scientists



Source: From Klohn Crippen Berger (2011).

Figure 33.8-2

Cross-section of the Ore Slurry Pipeline

33.8.2.2 Ore Conveyor Belt

The use of a conveyor (Figure 33.8-3) to transport ore for the Project will require a primary crusher near the start of the conveyor. The crusher will reduce the size of run of mine ore to about 150 mm. The crusher will be equipped with a bag house to control fugitive dust. A short conveyor will transport the crushed ore to the tunnel portal on both sides of the tunnel. The conveyor will be covered for protection from snow and wind, reducing the potential for windblown fugitive dust. The larger part of the conveyor will be located in the tunnels, where fugitive dust dispersion will not be a concern, but transfer points will use dust control measures. The section of the conveyor outside of the tunnel portals will be covered for weather and wind protection. The conveyor will discharge to a coarse ore stockpile near the Treaty Process Plant.

33.8.3 Comparison of Alternatives

Trade-off studies were conducted by Wardrop for Seabridge on the relative advantages and disadvantages of using an ore slurry or conveyor belt system for the Project (Wardrop 2011 and Appendix 4-C). This comparison is summarized in Table 33.8-1.

33.8.3.1 Technical and Project Economic Considerations

Several of the technical attributes for the ore conveyor were found to be preferred over the twin ore slurry pipeline system for the Project as shown in Table 33.8-1. In particular, the ore slurry method would require that a redundant processing plant be built at the Mitchell site for primary and secondary ore grinding, as well as a pumping facility and extra piping for water reclamation. These additions would significantly increase operational complexity and power requirements, with the slurry system increasing power demand to about 287 kV from an estimated 138 kV for the conveyor system.

The poor geotechnical conditions (i.e., high geohazards) at the Mitchell site would also pose significant risk to the safe construction and operation of an ore comminution facility. With the conveyor system, however, the ore comminution could take place at the Process Plant at the TMF site rather than at the Mine Site, reducing redundancy, saving power and water, and reducing geohazard risks.

Economic estimates by Wardrop (2011 and Appendix 4-C) indicate that direct CAPEX of the ore slurry pumping option will cost less ($323 M for tunnelling and $180 M for ore slurry transfer system) than ore conveyance ($344 M for tunnelling and $274 M for conveyor transfer system).

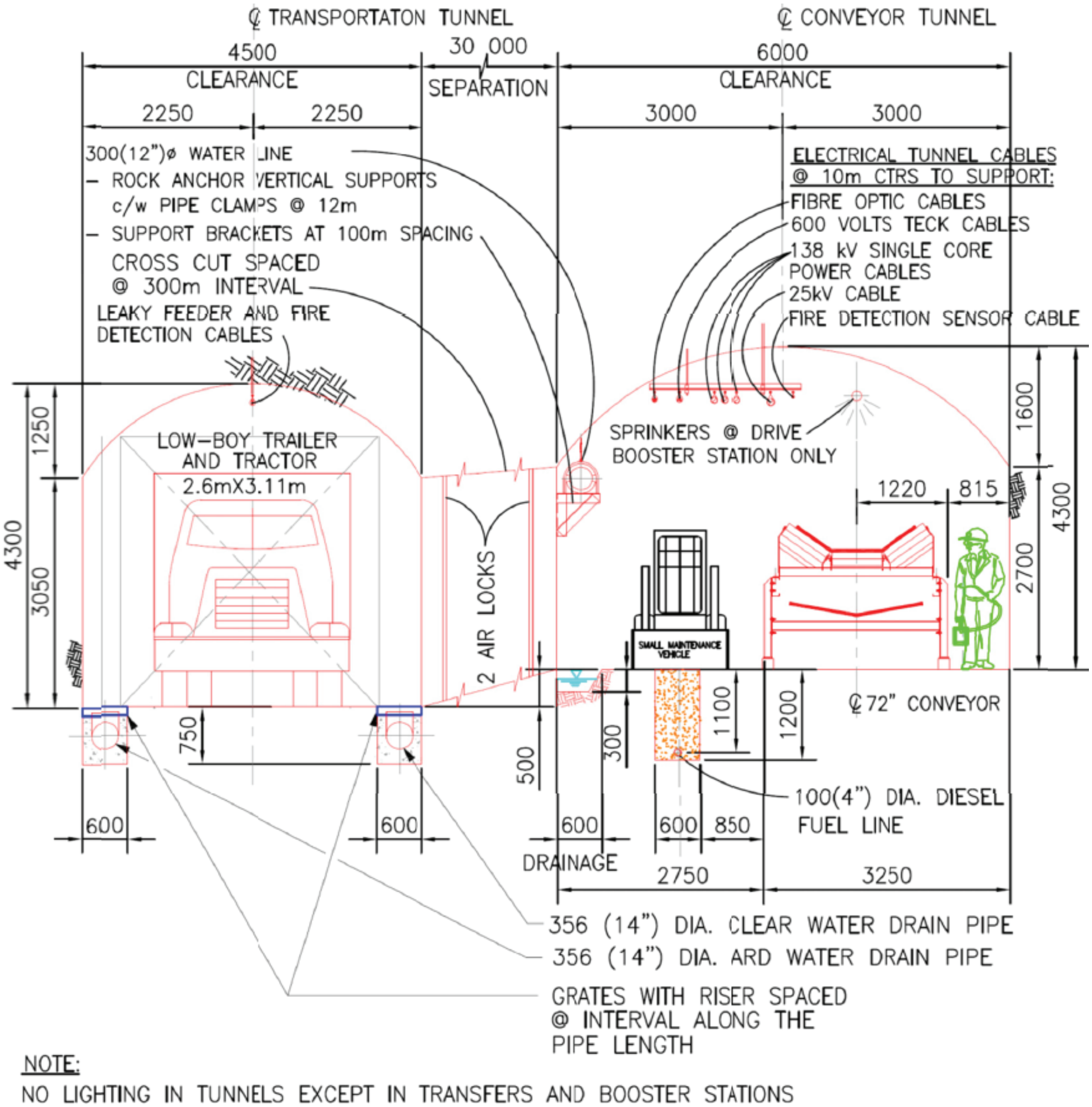
33.8.3.2 Environmental and Social Considerations

Each ore transport system has advantages and disadvantages in how it meets environmental and social performance objectives. The major environmental advantages of ore conveyance are reduced water supply and related extra energy for pumping demand compared to the ore slurry method, and reduced risks of dry ore conveyance compared to liquid slurry. There are slightly higher OH&S risks from fugitive dust in the tunnel from ore conveyance compared to the ore slurry pipelines, but dust can be brought to acceptable levels with ventilation.

Table 33.8-1. Summary Comparison of Ore Transport Alternatives

Alternative	ATTRIBUTE RATINGS AGAINST FOUR PERFORMANCE OBJECTIVES				OVERALL ALTERNATIVE RATING
	Environmental	Social	Technical	Project Economic	
Ore Conveyor (23 km)	✓ ● Higher tunnel fugitive dust, noise, and air emissions in general, but mitigated through ventilation systems ✓ ● Requires significantly less water supply from watershed at Mitchell site ✓ ● Less dust at Mitchell site due to less comminution of ore for conveyor belt ✓ ● In the event of a spill, low risk for contamination and easier to clean compared to a slurry spill ✓ ● Amenable to reclamation as can be dismantled and removed on closure and any dust piles recovered and removed	x ● Higher fugitive dust, noise, and other air emissions in the tunnel pose higher OH&S risks	✓ ● Only requires one process plant at PTMA ✓ ● Lower power needed at Mitchell site ✓ ● Simpler flow sheet and processing requirements x ● Requires larger diameter tunnel with more associated construction time x ● Potentially greater maintenance requirements x ● Higher ventilation requirements to maintain air quality in the tunnel x ● Higher risk of stopping production line, as only one conveyor x ● Higher fire risk	x ● Overall capital expenditure (CAPEX) is higher for ore conveyor transfer system through MTT: $274M x ● Tunnelling costs for ore conveyance option are a bit more: $344M	**PREFERRED**
	Preferred	Acceptable	Preferred	Acceptable	
Twin Ore Slurry Pipeline (23 km)	✓ ● Although slightly higher at Mitchell site due to extra crushing activities, lower overall fugitive dust, noise, and other air quality emissions x ● Higher water supply needs ✓ ● Less fugitive dust and other atmospheric emissions in general; also mitigated through tunnel ventilation systems x ● Increased GHGs from higher electricity and diesel requirements x ● More fugitive dust at Mitchell site due to extra comminution to prepare ore slurry x ● Higher spill risk and potential water impacts from ore slurry compared to dry ore on conveyor ✓ ● Amenity to reclaim would be the same as for conveyor if pipeline above ground, but if buried, would cause less disturbance as could leave in situ	✓ ● Lower dust, noise, and other air emissions in the tunnel pose reduced OH&S risks	✓ ● Requires slightly smaller diameter tunnel, lowering construction time ✓ ● Two pipelines lowers risk of halting production x ● Mitchell site has slightly higher power requirements for pumping slurry and much higher power requirements for additional plant to house grinding circuit x ● Requires construction and operation of two main process plants, posing more complicated flow sheet and process x ● High geohazard risks at Mitchell site puts process plant at risk x ● Poor geotechnical conditions at Mitchell Ore Preparation Complex site for process plant construction x ● Requires additional water return piping system ✓ ● Lower fugitive dust, fire, OH&S risk, less ventilation and other systems required	✓ ● Overall CAPEX is lower for ore slurry transfer system through MTT: $180M ✓ ● Tunnelling costs for ore slurry option are a bit less: $323M	**CHALLENGING**
	Acceptable	Preferred	Challenging	Preferred	

Notes:
✓ = advantage, x = disadvantage, ● = Preferred, ● = Acceptable, ● = Challenging, ● = Unfeasible; PTMA=Processing and Tailing Management Area; See Table 33.2-1 for attribute ranking specific definitions
OH&S = operational health and safety



Source: From Tetra Tech Wardrop (2012).

Figure 33.8-3

Cross-section of Ore Conveyor Tunnel

The footprint of the tunnel portals and related waste rock storage areas will be relatively small for both options. In the Mitchell Valley in particular, the footprint of the tunnel and related infrastructure will be small relative to disturbances caused by mining and waste rock disposal. The tunnel construction will be similar for both ore transport options as the alignment will be the same, except that the conveyor system will require a larger tunnel, thereby creating more waste rock. None of the proposed tunnel portals for the original scenario are in areas of fish habitat. Mitchell Creek has no fish and poor natural water quality. Neither the upper reaches of Treaty Creek near the proposed portals and conveyor trestle nor the Teigen Creek tributaries near the Plant site are fish-bearing. Both systems are relatively similar in their amenity to reclamation, although a buried slurry pipeline could remain buried and be less intrusive to reclaim in general.

33.8.4 Selected Alternative

The ore conveyor belt system was chosen by Seabridge as the preferred method of transporting ore for the Project; the twin ore slurry pipeline was excluded from use for the Project, primarily to reduce risk and to avoid processing ore twice.

33.9 Ore Concentrate Transport

33.9.1 Purpose and Background

The KSM Project will produce ore concentrate that will need to be transported during the operation phase from the Process Plant site to the nearest port for shipment to overseas smelters. The closest ports to the PTMA are in Stewart and Prince Rupert and are shown in Figure 33.9-1. Combined road and rail transport methods were identified by early engineering studies for the Project as potential options for hauling concentrate. Concentrate can sometimes be transported via pipeline, but this alternative was deemed unfeasible for the Project as the shortest pipeline option to the Granduc Mine staging area is still too far to be technically and economically viable.

For all routes, ore transport trucks will depart from the Treaty Process Plant along the TCAR. Since this access road will be constant for all concentrate transport alternatives, and the decision for its placement has been covered in Section 33.6, it will not be included in this assessment. The starting point for comparing all concentrate transport routes will therefore be the junction of the TCAR with Highway 37, as indicated in Figure 33.9-2, and since there is no rail access to this area, trucking along Highway 37 is the only option for the beginning portion of the ore concentrate transport route.

Highways 37, 37A, and 16 are suitable for a range of vehicles. All of the road sections to be used by concentrate haul trucks are hard-surfaced (either paved or seal-coated). The speed limit along these highways is 70 to 100 km/h and it is assumed that concentrate trucks will travel at approximately 70 km/h on average. These highways are also typically double-lane, though lane markings may not be present and single-lane bridges are common.

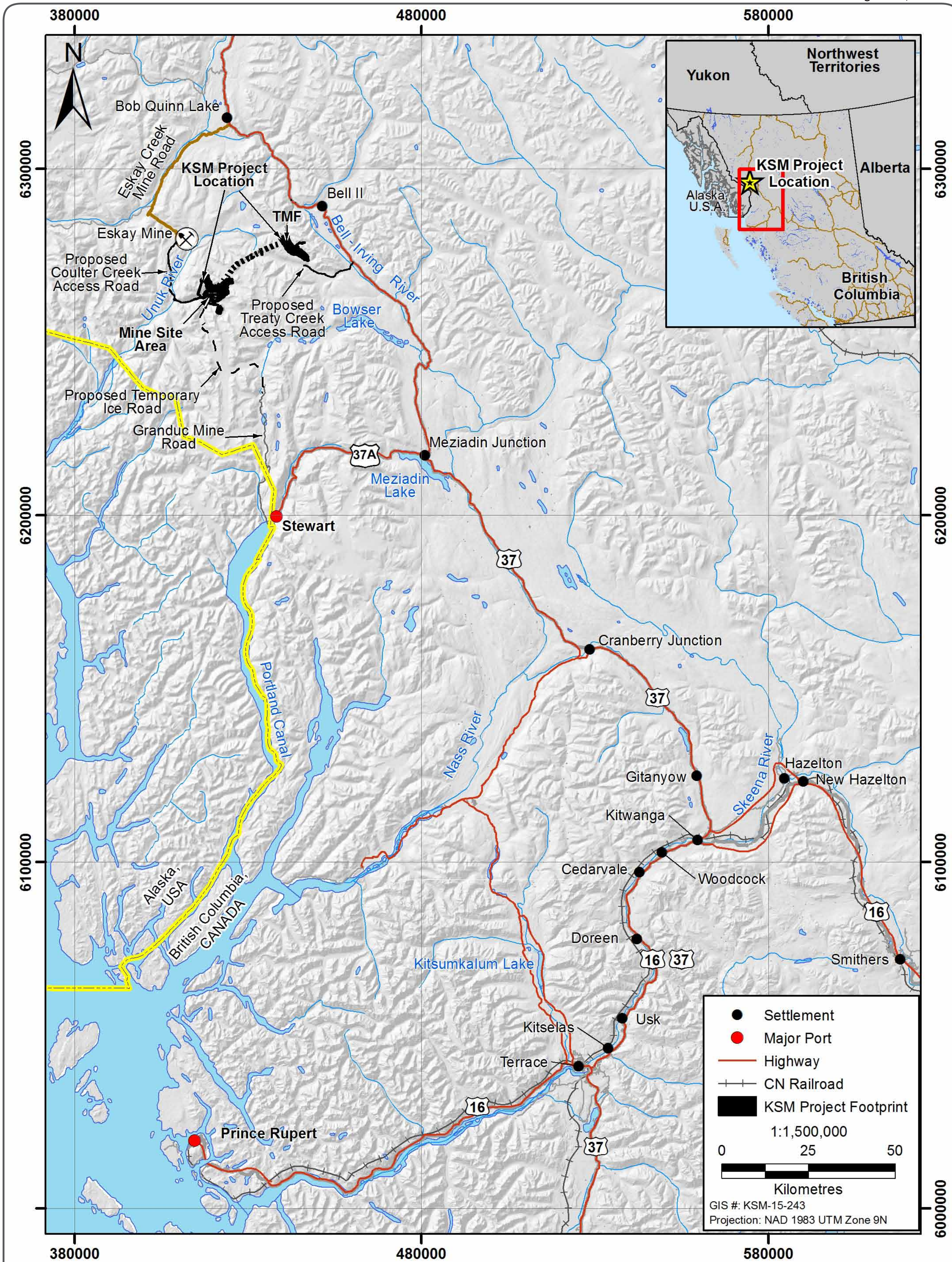


Figure 33.9-1

KSM Project Routes to Nearest Ports

SEABRIDGE GOLD
KSM PROJECT

Rescan
Engineers & Scientists

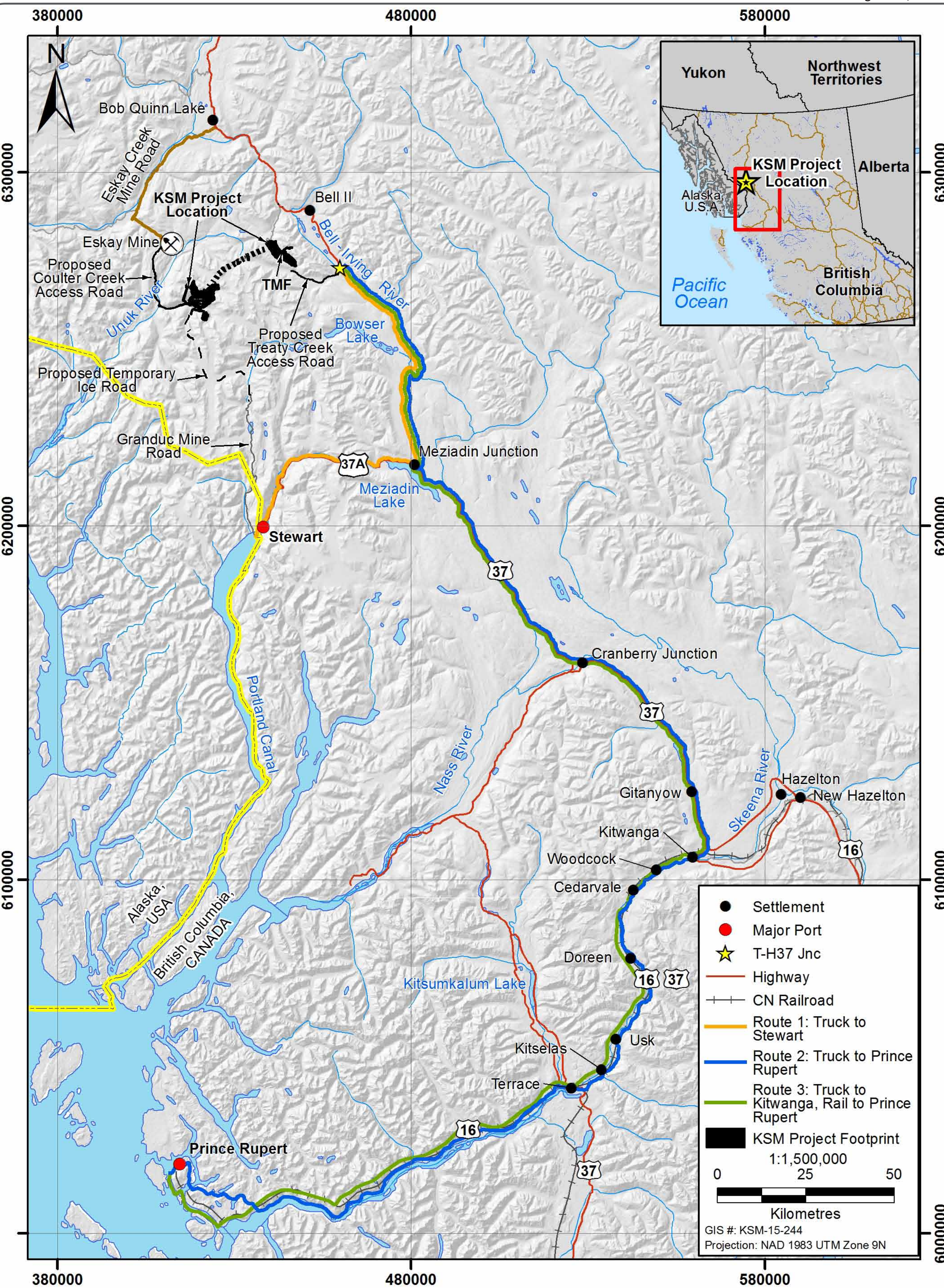


Figure 33.9-2

KSM Project Concentrate Transport Alternative Routes

SEABRIDGE GOLD
KSM PROJECT

Rescan
Engineers & Scientists

Road conditions along highways 37, 37A, and 16 depend on the season and the weather; potholes and broken sealcoat are common year-round. The BC Ministry of Transportation and Infrastructure is responsible for repair and improvements to all highways. Construction and maintenance occurs throughout the year, and highways are sanded and snowploughed in the winter (BC MOTI 2012), so it is assumed that all highways will be accessible to concentrate trucks year-round, barring avalanches or other conditions that may temporarily close the highway.

Regardless of the route taken, B-train vehicles are identified as the most suitable truck for transporting ore concentrate. B-trains are popular for hauling as they have more directional (yaw) and lateral (roll coupling) stability than other configurations, granting them higher relative safety ratings and higher payloads in Canada than other rigs. B-trains typically handle 30 to 40 t loads, and newer units can transport up to 50 t. For concentrate transport for the KSM Project, 50 t trucks are assumed. In BC, B-trains have a maximum Gross Combined Vehicle Weight of 63.5 t (BC MOTI 2011). B-trains used for the Project will be below this weight limit at 50 t.

33.9.2 Alternatives Identification

As illustrated in Figure 33.9-2, all concentrate transport routes considered in this assessment begin at the junction between the TCAR and Highway 37, 19 km south of Bell II. From this point, the alternatives determined to be the most feasible for transporting concentrate are:

1. Route 1 – Truck concentrate to Port of Stewart via highways 37 and 37A.
2. Route 2 – Truck concentrate to Port of Prince Rupert via highways 37 and 16.
3. Route 3 – Truck concentrate to the railhead in Gitwangak (Kitwanga in Figure 33.9-2) via Highway 37, then via CN Rail to the Port of Prince Rupert.

33.9.2.1 Route 1

For this route, from the TCAR-Highway 37 junction, concentrate haul trucks will drive south along Highway 37 to the Meziadin Junction, then turn west along Highway 37A to the Port of Stewart. The Port of Stewart is within the District of Stewart, located at the head of the Portland Canal, a 150 km fjord that is free of ice year-round. At the port, Stewart Bulk Terminals operates a concentrate bulk loader and two storage sheds capable of holding over 30,000 t.

33.9.2.2 Route 2

To go to the Port of Prince Rupert, trucks would first take the same path as Route 1 from the TCAR-Highway 37 junction to the Meziadin Junction, continue down Highway 37 to Gitwangak, and then turn east on Highway 16 through Terrace and on to Prince Rupert. The Port of Prince Rupert is larger than that at Stewart, but is more targeted toward train container shipments. It also hosts the deepest harbour in North America that is ice-free year-round, and is the safest regarding navigational risk (CN Rail 2012).

33.9.2.3 Route 3

This route involves a combined road-rail approach, involving trucking concentrate to the railhead where it would be transferred at Gitwangak and then transported by train to the Port of Prince Rupert. For transport via rail, CN Rail is equipped with ore gondolas for concentrate transportation. Ore gondolas can carry loads between 92 and 96 t, have a 65 m^3 capacity, and are about 36 feet long. For the purposes of this report, 96 t ore gondolas are assumed. Due to the larger capacity of railcars, concentrate will be stockpiled at the railway loadout facility and then transferred to ore gondolas and shipped, resulting in fewer train trips than for trucks.

33.9.3 Comparison of Alternatives

Concentrate transport routes via truck and rail to the ports located at Stewart and Prince Rupert are evaluated below based on how their attributes compare against the environmental, social, technical, and Project economic performance objectives for the Project, and they are summarized in Table 33.9-1.

33.9.3.1 Technical and Project Economic Considerations

During operation, the Treaty Process Plant will produce concentrate 24 hours per day, 365 days per year, totalling an estimated annual average of 321,840 t/year (wet basis) of copper-gold (Cu-Au) concentrate (Appendix 4-C). Cu-Au concentrate will account for about 99% of the ore concentrate produced by the Project, and will make up about 45% of total Project traffic (Rescan 2012). Assuming 50-t B-train trucks, Cu-Au hauling will require 6,437 average annual return trips per year during operation, or about 18 trips per day. The number of trips will be the same regardless of route taken via truck. To prevent dust generation from ore concentrate en route, trucks will be covered.

Driving distances and times between major stops are indicated in Table 33.9-2. Considering that the starting point for all routes at the TCAR-Highway 37 junction is 19 km south of Bell II, the total driving distance to Prince Rupert is 468 km, which is over three times that to Stewart at 134 km. The total driving distance for each route may also vary slightly depending on the location of storage warehouses to store concentrate near the port prior to shipping.

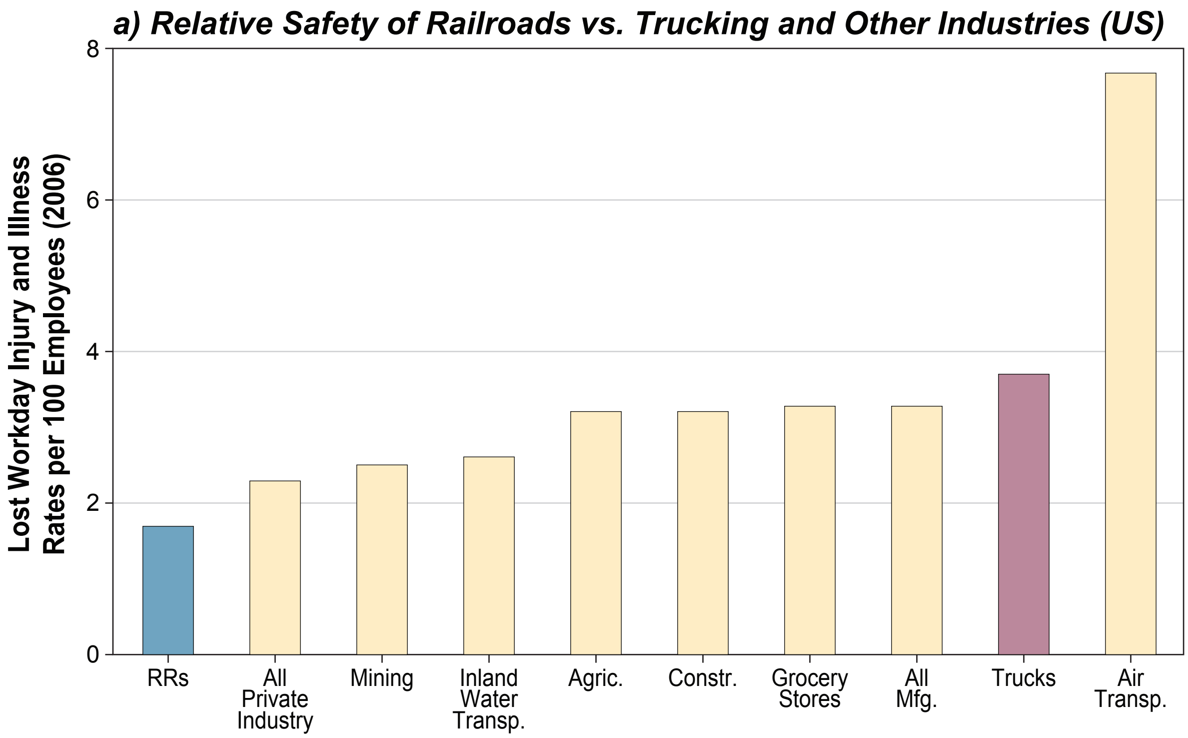
For the rail option, trucks would need to transport concentrate to the closest CN railhead, approximately 226 km from the TCAR-Highway 37 junction at Gitwangak, from which point concentrate would be stockpiled and then loaded onto the ore gondolas for shipment to Prince Rupert. It is assumed that the rail distance to Prince Rupert is about equal to the road route (242 km) as the railway parallels Highway 16, mostly along the other side of the Skeena River, as shown in Figure 33.9-2. Rail transport will presumably be slower than highway transport, but it is also significantly safer, reducing the risk of accidents and related spills, as shown in Figure 33.9-3.

Table 33.9-2 illustrates how the shorter distance of hauling ore concentrate is preferable from a technical and economic perspective. The increased distance to Prince Rupert almost triples driving times compared to hauling ore concentrate to Stewart, with associated significant increase in expenses. With extra transfer times and slower transport speeds, the total haul time to transport ore concentrate via train from Gitwangak would be greater than that of trucking to Prince Rupert, as well as incurring extra expense for storage and handling fees.

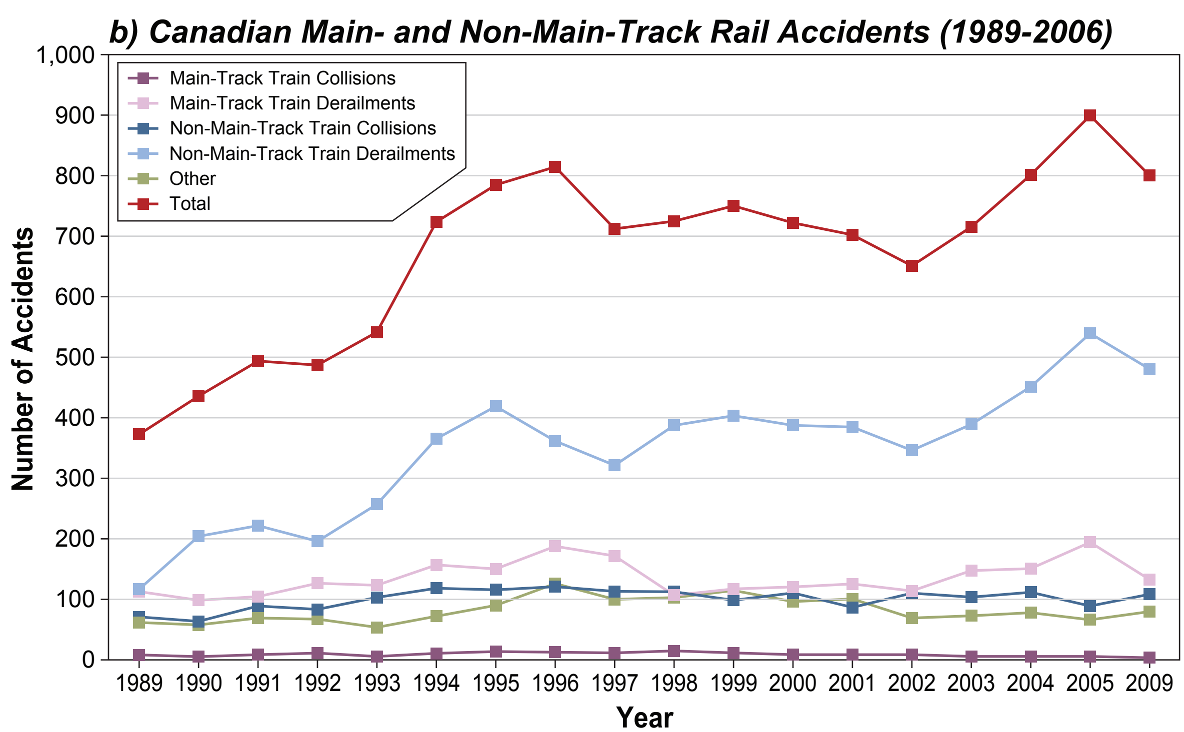
Table 33.9-1. Summary Comparison of Ore Concentrate Transport Alternatives

Alternative	ATTRIBUTE RATINGS AGAINST FOUR PERFORMANCE OBJECTIVES				OVERALL ALTERNATIVE RATING
	Environmental	Social	Technical	Project Economic	
Truck to Stewart (160 km total)	x Trucking emits more GHGs/km than rail ✓ Much shorter total route length reduces air emissions compared to other options x Trucking in general has higher rate of wildlife collisions per trip than train (combined haul) ✓ Reduces risk of wildlife collisions compared to other options due to much shorter length ✓ Lowest number of water crossings and slightly longer length in proximity to water than trucking to Gitwangak ✓ Lowest risk of invasive species transport	✓ Concentrate haul traffic may increase local economic activity (gas, pit stop restaurants, etc.) along the route ✓ Opportunities for local residents to be employed as contract drivers x Potential nuisance effects of haul trucks in Stewart higher than for Prince Rupert ✓ Least potential for nuisance, noise, air quality, and safety effects to populations, including to Aboriginal communities and their traditional use of lands (i.e., harvesting), along the route	✓ Haul cycle times are considerably less than to Prince Rupert ✓ Existing copper bulk concentrate loadout facilities ✓ Lower traffic accident risks than route to Prince Rupert as lower traffic flows x Trucking has generally higher accident risk levels than transport by train x Higher grades along route ✓ Existing storage shed facility at Stewart	✓ Lowest cost estimate ✓ Port is about three times closer than other nearest port x May require investment in concentrate storage facilities	**PREFERRED**
	Preferred	Acceptable	Preferred	Preferred	
Truck to Prince Rupert (495 km)	x Longest trucking route length, about three times that to Port of Stewart with associated increased air emissions x Significant increase of wildlife collisions and disturbance above the baseline x Highest number of water crossings and longest length in proximity to natural waterbodies x Highest risk of invasive species transport	x Highest potential air quality effects on surrounding communities x Highest potential nuisance and traffic accident risk effects on surrounding communities along longer route ✓ Highest potential for increasing local economic activity (gas, pit stop restaurants, etc.) along the route ✓ Most job opportunities and potential driving time for locally contracted drivers x Highest potential for nuisance, noise, air quality, and safety effects to populations, including to Aboriginal communities and their traditional use of lands (i.e., harvesting), along the route	x Haul cycle times are about three times greater than to Stewart x Port has no bulk concentrate loadout facilities. Would have to promote their design and construction. x Generally higher accident traffic risks than route to Stewart due to higher traffic levels on Hwy 16 x Trucking has generally higher accident risk levels than train ✓ Slightly better grades along Hwy 16	x Total cost estimate is economically prohibitive x Requires significant investment in design and construction of bulk loadout facilities	**UNFEASIBLE**
	Challenging	Acceptable	Unfeasible	Challenging	
Truck to Gitwangak, Rail to Prince Rupert (245 km truck, 248 km rail)	✓ Rail transport emits lower GHG emissions/km than transport by truck x Air emissions lower than that of trucking to Prince Rupert, but still higher than trucking the much shorter distance to Stewart ✓ Rail transport has lower wildlife collisions than trucking ✓ Using train reduces potential wildlife effects and air emissions versus trucking, but still higher than trucking to Stewart due to 245 km trucking distance x Reduces risk to waterbodies along train route x Moderate risk of invasive species transport x Higher surface disturbance as construction of loading facility would be required in Gitwangak	✓ Reduced potential air quality and nuisance effects between Kitwanga and Prince Rupert x Reduced economic benefits to local amenity businesses compared to trucking to Prince Rupert x Use of train slightly reduces employment opportunities and driving time for truck drivers x Use of trains and loading facility may disturb residents of Gitwangak x Potential for nuisance, noise, air quality, and safety effects to populations, including to Aboriginal communities and their traditional use of lands (i.e., harvesting), along the route; higher effects than trucking to Stewart, but lower than trucking to Prince Rupert	x Haul cycle times would be even longer than that to Prince Rupert x Would require design, construction, and maintenance of storage sheds in Gitwangak ✓ Rail transport reduces accident risk compared to trucking	x Cost estimate is as challenging, if not more so, than trucking all the way to Prince Rupert x Investment required in storage sheds and loading and unloading facilities at Gitwangak	**CHALLENGING**
	Acceptable	Acceptable	Challenging	Challenging	

Notes: ✓ = advantage, x = disadvantage, ● = Preferred, ● = Acceptable, ● = Challenging, ● = Unfeasible; See Table 33.2-1 for attribute ranking specific definitions



Source: US Bureau of Labour Statistics, cited in AAR 2008.



Source: Railway Safety Act Review Secretariat, 2007.

Table 33.9-2. Haul Distances from Bell II to Stewart and Prince Rupert

From	To	Distance (km)	Highway	Driving Time*
Bell II to Stewart				
Bell II	Meziadin Junction	87	37	1 h 14 min
Meziadin Junction	Stewart	66	37A	1 h 27 min
Bell II	**Stewart**	**153**	**37 & 37A**	**2 h 41 min**
Bell II to Prince Rupert				
Bell II	Meziadin Junction	87	37	1 h 14 min
Meziadin Junction	Gitwangak	158	37	2 h 2 min
Gitwangak	Prince Rupert	242	16	3 h 21 min
Bell II	**Prince Rupert**	**487**	**37 & 16**	**6 h 37 min**

Notes:
Distances to Gitwangak are found searching for distances to Kitwanga; distances may vary slightly from other Application/EIS chapters due to using Drive BC as measuring technique for this study.
*** Average driving time estimated for normal highway traffic (70 to 100 km/h as posted)**
Source: DriveBC (2012)

Table 33.9-3 lists some of the differences between the Port of Stewart and the Port of Prince Rupert. The ports differ mostly in size, and the relative environmental effects from the shipment of Project concentrate through both ports are estimated to be similar. Aside from being the closest port, Stewart offers a key advantage in that it and the Port of Vancouver are the only two ports in BC equipped for concentrate bulk loading, as well as private loading and storage facilities. Although with larger capacity than the Port of Stewart, the Port of Prince Rupert is more specialized for rail container transfer and is equipped with bulk terminals for coal, grain, and petroleum coke, but not concentrate—and for this reason, is considered technically unfeasible.

Table 33.9-3. Comparison of Ports of Stewart and Prince Rupert

Port Factors	Port of Stewart	Port of Prince Rupert
Distance from city	In the community	In the community
Type of port	Ocean	Ocean
Channel depth	46 m	44 m
Berth length / depth	244 m / 23 m	360 m / 17 m
Major ship traffic	Copper concentrate bulk terminals	Mostly rail container; bulk terminals for coal, grain, and petroleum coke; cruise ships
Loading rates	700 to 800 t concentrate/h	Bulk 9,000 t/h; container 500,000 TEUs/yr[1]
Bulk concentrate loadout facilities	Yes	No
General cargo facilities	No	Yes, but not set up for concentrate shipment
Train containerized facilities	No	Yes

(continued)

Table 33.9-3. Comparison of Ports of Stewart and Prince Rupert (completed)

Port Factors	Port of Stewart	Port of Prince Rupert
Storage facilities	2 existing	Bulk storage capacity of 24 million t; 6 train storage tracks; container yard can handle 9,000 TEUs
Highway serving port	Yes, 37A	Yes, 16
Railway serving port	No	Yes
Expansion plans	Port has identified expansion plans of 1.84 ha sheet pile and fill wharf	Quadruple capacity to 2 million TEUs
Waterway	Portland Canal ice-free year-round	Tuck Inlet, Morse Basin, Wainwright Basin, and Porpoise Harbour ice-free year-round

Notes:
[1] TEU = Twenty foot equivalent
Source: CNR (2012), District of Stewart (2009)

Trucking prices for Cu-Au ore concentrate are estimated at $32.94 ($/wet metric tonne) while transport by rail is estimated to be higher at $41.15/t for Cu-Au concentrate (Appendix 4-C). The CN railcar price calculator for gold ore concentrate also provides a quote from Gitwangak (Kitwanga) to the Prince Rupert terminal at $4,795 per gondola (CN Rail 2013).

33.9.3.2 Environmental and Social Considerations

Table 33.9-4 provides a comparison of the alternative ore concentrate transport routes based on environmental and social attributes. Environmental attributes include: route length, with longer roads producing higher air emissions (as indicated by GHG estimates) and risk of wildlife collisions and disturbance (i.e., moose, bear [*Ursus* spp.], crossbills [*Loxia* spp.], and pine siskins [*Spinus pinus*]); number of water crossings and the length of road proximal to natural waterbodies and wetlands, as indicators of the level of risk of accidental spills impacting aquatic and wetland systems; and preventing the spread of invasive species. Social attributes include the number of settlements and the population along routes to compare potential effects from air emissions and noise or visual nuisance factors on sensitive receptors to concentrate traffic, and identification of amenity services that may benefit from increased truck traffic. It is assumed that country foods (including for traditional use) and drinking water contamination that could affect human health will be negligible for concentrate transport as trucks will be covered to prevent fugitive dust (Rescan 2012).

As shown in Table 33.9-4, the additional length of hauling ore concentrate to Prince Rupert by truck or by train is associated with increased air emissions, and exposure to waterbodies along the route, making transport of ore concentrate to Stewart preferable to minimize environmental effects.

Table 33.9-4. Environmental and Social Comparison of Ore Concentrate Transport Alternatives

Assessment Category	Attribute	Route 1 via Truck	Route 2 via Truck	Route 3 via Rail	
		Truck H37 Jct.[§] -Stewart	Truck H37 Jct.-PR	Truck H37 Jct.-Gitwangak	Rail Gitwangak-PR
General	Distance (km)	134	468	226	242
Environmental Effects	GHG emissions (CO_2e t/year)*	787	2,762	1,334	1,398
	Number of water crossings**	212	525	281	n/a
	Route length proximal to water (m)**	4,850	28,700	4,150	n/a
	Route length proximal to wetland (m)	950	1900	1900	n/a
	Invasive species, special provisions	No	Yes	No	n/a
Social Effects	Number of settlements along route	2	13	4	n/a
	Population along route***	529	30,542	2,125	n/a

Notes:
[§] H37 Jct. = The junction between TCAR and Highway 37, located 19 km south of Bell II
*** Emissions calculated from CN Rail GHG calculator tool (CN Rail n.d.); CO_2e (t) = Greenhouse gas emissions in carbon dioxide equivalents (tonnes)**
**** Water and wetland values determined from map analysis from iMapBC (n.d.) web maps at 1:20,000 scale; proximal interpreted as within 30 feet**
***** Canada Census 2011 (Statistics Canada 2012)**
n/a = not applicable
All values are in metric
PR = Prince Rupert

Table 33.9-4 also includes the number of settlements and estimated population along the route (per census data) as indicators of potential social effects. More affected settlements along the route could correspond to greater nuisance, noise, and air quality effects to surrounding populations (including to Aboriginal communities and their traditional use of lands along roadways), especially for truck transport, making the shorter distance to Stewart preferable regarding these attributes. Amenity providers would stand to benefit though from increased traffic. Potential job opportunities also vary between train versus truck transport, as it is assumed that train transport would reduce haul driver requirements.

In general, train transport involves several advantages over truck transport. Since train transport involves adding ore concentrate cars to existing train loads, this minimizes the risk of wildlife collisions as a whole. The reduced risk of accidents associated with train transport (Figure 33.9-3) also minimizes the risk of ore concentrate spills. Train routes are also already established, and so addition of cars for ore concentrate transport to existing trains would not be likely to increase visual or nuisance factors to surrounding populations. Since train transport will only involve adding extra cars to existing train runs, it is assumed that, aside from incrementally different air emissions (calculable using CN Rail's GHG online calculator as reported in Table 33.9-4), there are minimal increased environmental risks due to concentrate transport than already exist from regular train travel. Ore concentrate dust can be irritating to respiratory systems though, so routes that minimize transfer points also minimize the risk of exposure to personnel. Train transport would also reduce potential air quality, noise, or nuisance effects to settlements, including to Aboriginal communities and their traditional use of land along routes, as train routes and schedules are already established. In spite of these advantages of train as an ore transport alternative, the extra haul lengths involved with this option for the Project make both trucking or hauling via train (after Gitwangak transfer station) to Prince Rupert challenging.

33.9.4 Selected Alternative

The concentrate route deemed the most appropriate for ore concentrate transport from the Project, as summarized in Table 33.9-1, is Route 1 to the Port of Stewart. This route involves transporting Cu-Au concentrate by truck to the Port of Stewart to be transferred to oceangoing vessels for overseas shipment to smelters in Asia. Trucking ore concentrate to Stewart will significantly improve Project scheduling efficiencies, reduce expenses, and minimize potential effects on environmental and human systems largely due to the route to Stewart being over three times shorter than to Prince Rupert. In addition, the Port of Stewart is equipped with bulk concentrate loadout facilities while the Port of Prince Rupert is not, making the latter option unfeasible.

33.10 Gold Recovery Method

33.10.1 Purpose and Background

Mining activities in the Mine Site of the KSM Project will produce low-grade ores, which will be transported to and processed in the Treaty Ore Processing Complex in order to produce gold. Selecting a safe and effective method to recover gold from ore is a key consideration in the overall Project design. Processing of lower grade hard rock ores involves physical and chemical methods to recover precious metals, generally involving the following steps:

- **Step 1** – *Crushing and grinding* (comminution) of ore into sand sized or smaller grains to break apart the minerals of interest from the surrounding rock.
- **Step 2** – Initial recovery through a preliminary flotation circuit through the addition of reagents to cause minerals of interest bound in the ore to selectively either float or sink when air is pumped through a slurry of ground ore and water.
- **Step 3** – *Gravity concentration* is a technique that can augment flotation extraction of gold in cases such as when gold in the ore is refractory, unliberated, or too coarse for flotation. Gravity recoverable gold tests can be performed to test if gravitational techniques would add to gold recovery for a given ore (Laplante and Dunne 2002).
- **Step 4** – *Regrinding* of ground ore that is not selected by the flotation process to further release finer-grained minerals of interest in subsequent flotation sessions. Regrinding at this step avoids having to grind all the initial ore to a smaller grain size.
- **Step 5** – *Pre-treatment* prior to further processing, such as roasting the ore or pressure oxidation, to break down the sulphide minerals (i.e., pyrite [$FeSO_4$]).
- **Step 6** – *Secondary recovery* involving using a lixiviant (i.e., cyanide solution) to extract gold and silver from solution (*leaching* process). Sometimes pre-treatment may alternatively be applied to this selectively extracted mineral, rather than the whole ore, thereby minimizing treatment volumes.
- **Step 7** – *Separation* of the gold from the solution usually through the adsorption of the gold onto activated carbon (carbon-in-leach [CIL]). After elution from the activated carbon, the gold is produced by *precipitation* or *electrodeposition* (e.g., electrowinning).

Metallurgical testing performed on KSM Project mineralization assessed the effects of varying the process parameters involved in the above steps, such as the fineness of the grind and the type and amount of reagents used in the flotation and leaching processes, to adjust the recovery of each metal to maximize the overall value. Steps 1, 4, and 7 above did not involve the consideration of any alternatives for the Project, so will not be considered further. The proposed flotation circuit (Step 2) for the Project is projected to yield a copper-gold (Cu-Au) concentrate that should recover between 76 to 88% of the copper and 50 to 62% of the gold from the mill feed (Appendix 4-C). This process will lead to flotation tailing being produced and reported to the TMF, which will be 90% of the total tailing by weight, corresponding to 90% of the Project ore being processed through this step to produce gold (Appendix 4-AC). The resulting Cu-Au ore concentrate will be shipped out from the Project to overseas smelters to produce gold for market sale. Flotation circuits commonly use biodegradable reagents that produce relatively benign outputs. Since there are no significant environmental differences between reagents used in this step, flotation will not be assessed further.

Gravity concentration (Step 3) is a physical method to recover gold that has the advantage that it avoids any potential toxic effects of chemical methods. Pre-treatment (Step 5) is actually a sub-step of Step 6 that is sometimes applied. This assessment will include whether these two steps are applicable for use at the KSM Project in the pre-assessment, but will focus on alternatives used in Step 6 to increase the gold recovery of the mine.

The 50% to 62% gold recoveries mentioned above would not be sufficient on their own to make the KSM Project an economically viable mine. Extra processing (steps 6 and 7) to further increase the gold produced by the Project is therefore necessary. The reason further processing is typically required at hard rock mines is that, as a noble metal, gold is very stable in water—being unreactive in pure water, and over a wide pH range. As a result of gold's stability in aqueous solutions, particles of gold that are bound within the structure of ore from hard rock mines are challenging to recover through initial flotation and/or gravity processes alone, and even corrosive (oxidizing) agents such as sulphuric acid are ineffective at dissolving gold in the absence of a complexing ligand (also called *lixiviant* or *complexant*).

To dissolve gold, a hydrometallurgical process can be used whereby a liquid medium containing a lixiviant is used to selectively extract the gold from the ore. The ligand in the complexant chemically removes the gold from the molecular lattice of ore minerals, and then forms relatively stable[5] complexes with the gold species in solution. As a result of this complexation reaction, gold can be dissolved in relatively mild oxidizing solutions, typically using cyanide as a lixiviant in aerated aqueous cyanide solutions (Marsden and House 2006). This technique liberates microscopic particles of gold, previously bound to ore, available for further recovery processes, making once unprofitable mines profitable. Due to its technical effectiveness and reasonable price as a lixiviant, most large-scale mines of hard rock ores—which have been the basis of commercial gold recovery since the beginning of the 20th century—use cyanide leaching (*cyanidation*) as the hydrometallurgical gold extraction standard of practice. For the KSM Project, the extra CIL cyanidation circuit (involving steps 6 and 7), could increase gold recovery for the KSM Project from 70% to 79% according to test studies (Appendix 4-C). Note that since 90% of the ore for the KSM Project is only processed through flotation prior to being shipped out as ore concentrate, only 10% of the ore from the Project will involve cyanidation, affecting only 10% of the total tailing by weight (Appendix 4-AC).

The main disadvantage of cyanide in industrial use is the risks associated with its toxicity. For this reason, the Environmental Code of Practice for Metal Mines (Environment Canada 2009) recommends that metal mines considering the use of cyanide follow the International Cyanide Management Code, which stipulates in Section 4.2 of their Principles and Standards of Practice (Appendix 33-C) to take measures to minimize cyanide use (ICMI 2012). Toward this end, Seabridge undertook an assessment of alternative means of gold extraction besides cyanide leaching for potential use by the Project that would generate competitive economic returns and that would balance the relative recoveries of each metal against the overall cost, potential environmental effects, and health and safety risks according to the performance objectives approach.

33.10.2 Alternatives Identification

The six lixiviants listed below were researched and identified by Seabridge as alternative lixiviants for use in the Project. Cyanidation was considered as the standard of practice for gold extraction at modern metal mines. The five other alternatives were selected due to their prevalence in research literature as potential substitutes for cyanide as complexing ligands.

[5] See Section 33.10.4 for discussion of the thermodynamic stability of lixiviants.

33.10.2.1 Cyanidation

The cyanidation process for gold extraction was first patented in 1897 and first used commercially in 1899 at a mine in New Zealand. By 1988 cyanide was used in approximately 90% of global gold producing mines (Yarar 2002). Of the approximately 1.1 million metric tonnes of hydrogen cyanide produced annually worldwide, only about 6% is used to produce cyanide reagents for gold processing, while the remaining 94% is used in industrial applications including production of plastics, cosmetics, pharmaceuticals, pesticides, and for use in anticaking agents in road salt (Paschka, Ghosh, and Dzombak 1999). Cyanide is a substance also commonly found in nature. For instance, cyanide is found in more than 1,000 higher plants species and other organisms (Eisler 2000).

Heap and vat cyanide leaching are two main types of cyanidation processes used in gold mining. Heap leaching (or valley fill leaching) is typically used on ores containing less that 0.04 Troy oz/t, while vat leaching (or percolation, carbon-in-pulp or agitated CIL) techniques are usually used on ores with more than 0.04 oz/t of gold (US EPA 1994). In 1998, vat leaching operations produced about 70% of gold recovered from cyanidation, with the remaining 30% produced through heap leaching (Amey 1998). Heap leaching techniques cost less to implement than vat leaching, but recover less gold, are not as effective in cold climates, require more space to operate, and are subject to more risks of leaking of pregnant solution from liners placed at the bottom of the heap. Risks to personnel from both heap and vat processing methods are both relatively low, especially in comparison to techniques such as mercury amalgamation (Eisler 2003). For the above reasons, CIL vat leaching was considered more appropriate for use by the Project, and will be the method assessed.

33.10.2.2 Thiourea

Thiourea, $CS(NH_2)_2$, is an organosulphur compound. It occurs as white or almost colourless crystals at room temperature, is soluble in cold water and alcohol, and is stable under normal temperatures and pressures (US DHHS 2011). Intensive academic research and several industrial pilot studies were conducted on the use of thiourea as a gold lixiviant in the 1980s and 1990s (Tremblay et al. 1996). The most common uses for thiourea are in the production of thiourea dioxide (30%), in leaching of gold and silver ores (25%), and in the production of light-sensitive photocopy (diazo) paper (US DHHS 2011). Thiourea is also used in textile processing and in the production of flame retardant resins.

33.10.2.3 Thiosulphate

Thiosulfate ($[S_2O_3]^{2-}$) is a colourless crystalline compound that acts to form a strong complex with gold (Muir and Alymore 2004). Thiosulphate is mostly produced from liquid waste products of sodium sulphide or sulphur dye manufacture and so is not as widely available as cyanide. Sodium thiosulphate was historically used in photographic processing as a fixer and has medical applications such as in the treatment of cyanide poisoning and for use in chemotherapy. It is also used to make hand warmers and chemical heating pads. Ammonium thiosulphate is often employed as a less expensive alternative to sodium thiosulphate for use in gold extraction, using ammonium thiosulphate (ATS), ammonia (NH_3), and copper (Cu) species in solution to facilitate the reaction.

33.10.2.4 Thiocyanate

Thiocyanate (also called rhodanide) is the anion $[SCN]^-$. Thiocyanates are used in various applications, in particular in the textile and fibre, agriculture, metal and steel, and construction industries. Thiocyanate for use as a gold extraction technology is just emerging from the research stage, and involves high temperature processing at low pH (Gos and Rubo 2001). Similar to thiosulphate, ammonium thiocyanate is sometimes used instead.

33.10.2.5 Bromine

Bromine (Br) is a chemical element in the halogen group. Halogens react with metals to form metal halides. Specifically, liquid bromine reacts with metal whereby it is reduced to a bromide anion (Br^-), and the metal is oxidized to a metal cation. Bromine is less reactive than chlorine but more reactive than iodine. The use of bromine to leach gold was identified in 1846 (Yannopoulos 1991). The use of halogen-halide systems for gold extraction actually pre-dates cyanidation (la Brooy, Linge, and Walker 1994).

33.10.2.6 Chlorine

Chlorine (Cl), similar to bromine, is a halogen that forms metal halides containing the chloride anion (Cl^-). Chlorine is a commonly used industrial product and a naturally occurring substance. Chlorine is found in the earth's surface materials, mostly in the form of sodium chloride (salt) in sea water and in natural deposits such as carnallite and sylvite. Like bromine, chlorine was used in the 1800s, prior to the development of the cyanide process, to leach gold from ores and concentrates.

33.10.2.7 Excluded Gold Recovery Techniques

33.10.2.7.1 Gravity

Gravity (Step 3, Section 33.10.1) is a traditional technique to physically separate gold from ore. For gold to undergo gravity separation, it must exist as small nuggets or flakes. Elemental gold has a specific gravity of 19, compared to a range of 2 to 3 for most common rock types. This difference in specific gravity can be used to separate gold from other minerals after comminution. The main advantage of gravity separation is that, as a physical rather than chemical technique, it does not pose any toxicological risks.

After initial grinding and processing of ore, sometimes gravitational techniques can augment flotation, but this process becomes less efficient with the finer grain sizes typical of the KSM Project deposit ores. For instance, ores in which gravity recoverable gold is fine (less than 105 µm) and ranging from 8% to 33% often show that full scale gravity recovery is generally below 10% and can be as low as 2% (Laplante and Dunne 2002). Gravitational techniques applied to gold that is incorporated in the crystal structure of other minerals, such as in the Project's hard rock ores, would need to be supplemented with the use of a lixiviant anyhow, so would not provide a replacement to cyanide. Gravitational methods would also require costly, labour-intensive gravitational circuits that would not likely yield enough extra gold to offset expenses. For these reasons, gravitational techniques are considered technically inappropriate and economically prohibitive, and are therefore **unfeasible** for use by the Project.

33.10.2.7.2 Pre-treatment

Pre-treatment (Step 5, Section 33.10.1) is sometimes used as a precursor to the use of lixiviants (usually prior to cyanidation), to further expose gold. Exposure can often be achieved by crushing and grinding of the host ore, but gold that is extremely fine-grained, or that is incorporated within the crystalline structure of sulphide minerals such as pyrite and arsenopyrite (refractory ore), might not be liberated by grinding alone. For such ore, pre-treatment can remove excess sulphide minerals, exposing extra gold. Processes available for pre-treatment all involve oxidation of sulphur including (1) bio-oxidation, using sulphur-consuming bacteria in a water solution; (2) pressure oxidation, using oxygen and heat under pressure in an autoclave; (3) roasting, using heat and air to burn away sulphur from dry ore; and (4) chemical oxidation, using nitric acid at ambient pressure and temperature, which has only been used on a limited basis. These pre-treatment methods all produce compounds such as sulphur dioxide gas (an acid rain precursor) and therefore require elaborate scrubbing systems that sometimes produce sulphuric acid as a by-product. This requires considerable increases in capital and operating expenses for equipment and the implementation of safe handling procedures for by-products.

Analysis of KSM Project ores indicates that, while pre-treatment may generate a nominal increase in gold recovery, the additional capital and operating costs and the necessary disposal of detrimental sulphur compounds would offset any potential additional economic gains, making this option **unfeasible**.

33.10.2.7.3 Mercury Amalgamation

Mercury amalgamation is a technique that was traditionally used for gold recovery, and is still used by small artisanal miners in remote areas, but its use by large-scale gold mining was discontinued years ago due to concerns relating to mercury's toxicity and persistence leading to significant environmental and health and safety risks. Due to current issues from past mining of mercury accumulation in the environment, during the lifetime of the Project, the use of mercury in mining may also be prohibited (US Geological Survey 2000; Gos and Rubo 2001; Ziegler-Syklakakis et al. 2003; Xia 2008; US DHHS 2011). Due to these risks posed by mercury, this lixiviant is considered **unfeasible** and was not considered as a gold recovery method by Seabridge.

33.10.3 Comparison of Alternatives

The six alternative gold extraction lixiviants were assessed against performance objectives as summarized in Table 33.10-1.

33.10.3.1 Technical and Project Economic Considerations

Technical and economic attributes used to compare gold lixiviants include (1) thermodynamic stability, (2) the speed and effectiveness of the process, (3) lixiviant recyclability, (4) ease and complexity of controlling processing parameters, (5) whether the technique is suitable for large-scale application or just coming out of research and/or pilot phases, (6) availability of precursors, and (7) net, capital, and operating expenses.

Table 33.10-1. Summary Comparison of Gold Extraction Method Alternatives

Alternative	ATTRIBUTE RATINGS AGAINST FOUR PERFORMANCE OBJECTIVES				OVERALL ALTERNATIVE RATING
	Environmental	Social	Technical	Project Economic	
Cyanidation	x Acutely toxic substance, with associate adverse environmental effects in the event of a spill; risks made acceptable with BMPs demonstrated in wide industry application ✓ Degrades quickly ✓ Not persistent in environment; amenable to reclaim as it naturally degrades with exposure to light and water	x OH&S exposure risks reduced by following BMPs demonstrated in wide industry application ✓ Fast degradation and non-persistence ensure that even in event of a spill there are no long-term effects on drinking water or fisheries	✓ Proven technique; industry standard ✓ Highest thermodynamic stability ✓ Fast reaction rates ✓ Good control of process parameters ✓ Method most appropriate for KSM Project ores	✓ Medium price ✓ Wide availability ✓ Lowest overall cost of operation using this method ✓ Highest recovery ensures profitability ✓ Recyclability reduces reagent costs	**PREFERRED**
	Acceptable	Acceptable	Preferred	Preferred	
Thiourea	x Originally thought to pose a safer alternative to cyanide, but recent research indicates poses risk to aquatic life x Moderate to high aquatic toxicity x Safe water quality guidelines have not been established yet x Persistent in water and resistant to biodegradation x Ability to treat and reclaim at industrial scale may be hindered by environmental persistence	x Originally thought to pose a safer alternative to cyanide, but research indicates same or higher OH&S BMPs as cyanide may apply x Research indicates poses high health risks as a human toxin, carcinogen, and may also potentially cause adverse reproductive foetal effects x OH&S handling procedures not as fully developed, which increases exposure risks	✓ Some industrial-scale application ✓ Third highest thermodynamic stability but still orders of magnitude lower than cyanide ✓ Up to 10 times faster gold dissolution than cyanide x Hard to control process parameters x Suitable for refractory ores, which doesn't apply to KSM Project	✓ Good availability ✓ Likely similar equipment costs as cyanide x Very high reagent costs - 2 to 4 times price of cyanide x Limited recyclability, which increases reagent costs	**UNFEASIBLE**
	Challenging	Unfeasible	Unfeasible	Challenging	
Thiosulphate	✓ Sodium thiosulphate is the least toxic to aquatic life of all reagents ✓ Lower LD_{50} for fish than cyanide for both sodium and ammonium thiosulphate x Likely use of ammonium thiosulphate (least expensive form) would increase toxicity risks, as high reagent consumption and buildup of by-product, ammonia, in TMF x May be less toxic; but by-products, ammonia, sulphide, and bisulphide, are toxic x Sulphate by-product and may contribute to metal leaching and acid rock drainage ✓ Not persistent in environment	✓ Sodium thiosulphate likely benign, minimizes OH&S risks x Likely that less costly form, ammonium thiosulphate would be used due to large volumes required, raising OH&S risks to be similar to those of cyanide x OH&S transport and handling procedures not as fully developed, which increases exposure risks	x Method still coming out of research phase and pilot phase; faces barriers to implementation at industrial scale ✓ Second to highest thermodynamic stability, but still orders of magnitude lower than cyanide x Hard to control process parameters, in particular recovering complexed gold x More suitable for refractory/preg-robbing ores, not those at KSM Project ✓ No corrosion concerns for equipment	✓ Available x High costs for reagent and detoxification x Gold recovery not as assured compared to cyanide x Likely not recyclable, and high reagent consumption, increasing reagent costs	**CHALLENGING**
	Challenging	Acceptable	Challenging	Acceptable	

(continued)

Table 33.10-1. Summary Comparison of Gold Extraction Method Alternatives (completed)

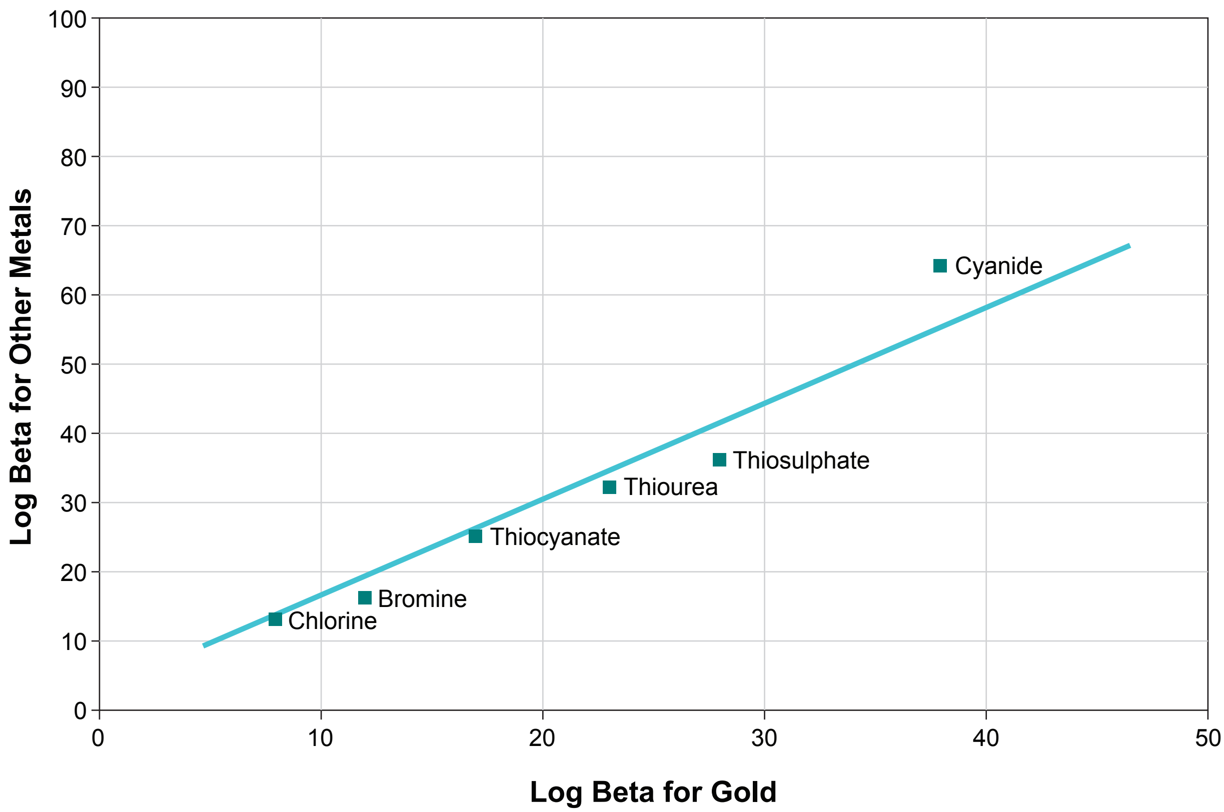
Alternative	ATTRIBUTE RATINGS AGAINST FOUR PERFORMANCE OBJECTIVES				OVERALL ALTERNATIVE RATING
	Environmental	Social	Technical	Project Economic	
Thiocyanate	✓ Lower toxicity than cyanide if sodium thiosulphate used x If ammonium thiocyanate used, raised environmental risks similar to cyanide from ammonia x Degrades to sulphates, which may exacerbate metal leaching and acid rock drainage conditions	✓ Lower toxicity than cyanide if sodium thiocyanate used x If ammonium thiocyanate used, raised OH&S risks similar to cyanide due to production of ammonia, which TDG classifies as a “Toxic Gas” x OH&S transport and handling procedures not as fully developed, which increases exposure risks	x Not proven industrial-scale technique x Third lowest thermodynamic stability ✓ Suitable for several ore types x Hard to control process parameters ✓ Can be used at wide range of pH (but best in acidic conditions)	x Limited availability x Likely economically unviable due to high temperature processing increasing operation expenses and supply costs x Gold recovery not assured x Potential loss of silver during processing ✓ Partly recyclable	UNFEASIBLE
	Challenging	Acceptable	Unfeasible	Unfeasible	
Bromine	x Bromine is a strong water contaminant ✓ Bromide is not toxic at low concentrations x Considered unfeasible for industrial-scale application due to bromide by-product toxicity at higher concentrations to aquatic life and persistence up to 10 years, which may affect amenability to reclaim	x Have as high or higher OH&S risks than cyanide (i.e., bromism) x Handling challenges to prevent exposure to bromine, which TDG lists as a “Toxic Gas”	x No longer used in industrial mining ✓ Proven technique at small scale x Second to lowest thermodynamic stability ✓ Faster gold dissolution than cyanide ✓ Suitable for most ore types x Corrosion concerns for equipment	✓ Available x Net costs likely prohibitive x Gold recovery not assured x Likely requires expensive equipment due to higher temperature processing	UNFEASIBLE
	Unfeasible	Challenging	Challenging	Challenging	
Chlorine	x Environmental risks of exposure to chlorine compounds x Chlorine is a acutely toxic to aquatic life ✓ Chloride by-product is not toxic at low concentrations x By-product chloride is less toxic than cyanide, but persistent and mobile in water, and poses chronic and acute adverse effects to freshwater aquatic life at increasing concentrations	x Higher OH&S risks than cyanide due to risks of chlorine gas leaks during high temperature processing x Handling challenges to prevent exposure to chlorine, which TDG lists as a “Toxic Gas”	x No longer used in industrial mining ✓ Proven technique at small scale x Lowest thermodynamic stability ✓ Faster gold dissolution than cyanide x Process controls and handling are challenge at large scale ✓ Suitable for most ore types x Corrosion concerns for equipment and high temperature processing increase risk of gas leaks	✓ Widely available ✓ Low reagent supply costs x Likely higher net costs than cyanide x Gold recovery not assured x Likely higher operation costs than cyanide and bromine due to high corrosiveness and high temperatures	CHALLENGING
	Challenging	Challenging	Challenging	Challenging	

Notes:
✓ = advantage, x = disadvantage, ● = Preferred, ● = Acceptable, ● = Challenging, ● = Unfeasible; See Table 33.2-1 for attribute ranking specific definitions
BMP = best management practice; TDG = Transport of Dangerous Goods; OH&S = occupational health and safety

One of the primary technical determinants of the utility of a lixiviant is its thermodynamic stability: a chemical property that measures the relative stability of the water-soluble gold complex that the lixiviant forms with gold in solution (Gos and Rubo 2001). The thermodynamic stability of the six lixiviants is shown in Figure 33.10-1. Even if a lixiviant is a very rapid gold complexant, if it has a low thermodynamic stability, it faces higher risks of losing gold from the complex back to solution again, leading to poorer economic gold recovery as a result. The cyanide anion, CN^-, is very reactive with metal cations in aqueous solutions, making it a powerful lixiviant that selectively and effectively dissolves gold from gangue material. As shown in Figure 33.10-1, cyanide also has the highest thermodynamic stability, orders of magnitude higher than all of the alternative lixiviants. Many of the other alternative lixiviants act as more rapid gold complexants than cyanide, but they all have lower thermodynamic stabilities, reducing their gold recovery and efficiency overall, and making cyanide the most reliable for gold recovery overall. Cyanide is also widely available, can be selectively recycled or destroyed, is the standard of practice of gold extraction at the industrial scale for hard rock mines, and is economically viable.

Thiourea acts as a rapid gold complexant, with leaching rates reported that are 10 times faster than cyanide (Hiskey and DeVries 1992). The lixiviant faces challenges in controlling processing parameters, can have poor selectivity for gold, and requires acidic leaching conditions that makes the gold processing less efficient (Gos and Rubo 2001). High reagent consumption with low recyclability (1 kg/t to 12 kg/t of thiourea), the requirements for sulphuric acid and/or peroxide, gold passivation problems, high total dissolved solids problems, and high capital costs associated with the acidic leaching conditions have hindered further development of this lixiviant for larger gold production scales (Marsden and House 2006). Thiourea is now considered a proven technology for leaching gold at smaller scales, but was developed for and is more suitable for refractory ores (i.e., sulphide and carbonaceous ores), not those that are characteristic of the KSM Project. For these reasons, thiourea is considered technically **unfeasible** for use by the Project. Thiourea also has a low global annual production and availability, about 10,000 t in 1993 (Ziegler-Syklakakis et al. 2003), and related higher supply costs than other lixiviants, so although equipment costs may be similar to that for cyanidation, operation costs would likely be very high. Thiourea reagent consumption is at least twice that of cyanide under ideal operating circumstances, and costs around four times as much (Rezai et al. 2003).

In general, thiosulphate has good performance leaching gold (i.e., greater than 99%; Gos and Rubo 2001) but not as high as that of cyanide (SGS 2012). Thiosulphate also has the second to highest thermodynamic stability of the lixiviants (Figure 33.10-1), although it is orders of magnitude less than that of cyanide. The reaction mechanism of the ammonium thiosulphate leaching system with copper (as present in KSM Project ores) is more complicated than cyanidation and most other leaching processes, which results in challenges to apply this technology (Wan 1997). Thiosulphate is sometimes proposed for use in areas where the use of cyanide is banned, or for difficult ores, such as those containing large amounts of cyanide-consuming copper, refractory sulphides, or carbonaceous material (Marsden and House 2006). KSM Project ores are not known to be refractory nor include significant carbonaceous materials.



Adapted from Gos & Rubo 2001

SEABRIDGE GOLD
KSM PROJECT

Thermodynamic Stability
of Gold Lixiviants

Figure 33.10-1

Rescan
Engineers & Scientists

A thiosulphate leaching process, followed by resin-in-pulp gold extraction, has been developed "to the point where it is a technically and economically viable alternative to cyanidation for some gold bearing ore bodies," in particular, preg-robbing[6] ores (SGS 2012), but again these are not characteristic of the Project. Thiosulphate leaching is an alkaline process, and so there are not the same issues with potential corrosion of equipment and related operation costs, as with some of the other lixiviants that require acid conditions. Process parameters are a challenge to control for thiosulphate, and it also decomposes readily, limiting its recyclability (Gos and Rubo 2001).

Thiocyanates are not as available as other gold extraction lixiviants (Gos and Rubo 2001). The biggest disadvantage with thiocyanate is that it is still coming out of the research stage with limited application, so it is considered **unfeasible** for industrial-scale application. Its limited commercial availability and the higher acidity and temperatures for some applications mean that operating costs for thiocyanate would likely be much higher than those for cyanide, making it an economically **unfeasible** alternative (Gos and Rubo 2001). It also has the third to lowest thermodynamic stability of all lixiviants considered (Figure 33.10-1), reducing final gold recovery of this technique. More research is also to optimize conditions, minimize reagent consumption, and develop consistently reliable methods for gold recovery from solution (Gos and Rubo 2001; Marsden and House 2006; J. Li et al. 2012). The gold-thiocyanate system offers some technical advantages over gold cyanidation. For instance, compared to some other complexants, thiocyanate is also suitable for most ore types, and recyclability is possible if temperatures are not too high to cause excess destruction of the ligand. Various trade-offs complicate the usage of thiocyanate as a gold complexant such as gold dissolution increasing with thiocyanate concentration and temperature which is beneficial, but thiocyanate consumption also increases as these parameters increase requiring higher supply. The addition of ferric ion as an oxidant can increase the rate of leaching; however, this would also involve the trade-off of producing a contaminated gold-bearing leach solution that would need further treatment (J. Li et al. 2012). Another potential disadvantage of thiocyanate leaching is that silver can form a relatively insoluble product, silver thiocyanate, which would mean that a significant portion of the silver value in the KSM Project ores would likely not be recoverable with the thiocyanate process.

Halide/halogen (i.e., bromine and chlorine) lixiviant systems are well known for leaching gold, and likely have reasonable supply economics. Bromine production has increased since the 1960s to 556,000 t produced globally in 2007 (Lyday 2007), but would face higher supply costs than chlorine. Bromine-bromide systems to dissolve gold are very oxidizing, and dissolution rates are faster than those of cyanide and oxygen under ambient conditions (Marsden and House 2006). Gold recovery is not assured for these techniques though as bromine has the second to lowest and chlorine the lowest thermodynamic stability of all the lixiviants (Figure 33.10-1), causing this attribute to be considered **unfeasible.** This (along with health risks) likely explains why modern industrial applications do not exist for bromine (Gos and Rubo 2001). Some companies are pursuing chlorine-chloride processes for metal leaching and a few are in operation. Typically

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[6] These are ores with high carbonaceous components that preferentially absorb gold as well as gold-ligand complexes.

these operations are intended to treat refractory base metal ores rather than those typical of the KSM Project, though. The Platsol process is an example of a relatively new proprietary chloride-assisted total pressure oxidation process that has been tested (Marsden and House 2006). Halide systems are strongly oxidizing and capable of dissolving many sulphide minerals; however, the dissolution rates are strongly dependent on the concentrations of the complexant and oxidant, and require higher temperatures (e.g., 150°C to 180°C) in some applications to achieve higher recoveries, which increase operation costs. The Platsol process requires even higher temperatures (200°C to 225°C) and pressures (Marsden and House 2006). Halogens would require higher capital equipment investment than that of cyanide due to their corrosive nature and the need for a closed system, and current large-scale applications of bromine systems are not known to be in use (Gos and Rubo 2001). Costs are further exacerbated by high lixiviant consumption and the requirement of specialized transport and processing equipment to minimize OH&S and environmental risks (Gos and Rubo 2001; Muir and Alymore 2004; Marsden and House 2006).

33.10.3.2 Environmental and Social Considerations

The primary impetus for the assessment of alternative lixiviants is the known toxicity of cyanide. Hence, the environmental and social performance objectives associated with the assessment of alternative lixiviants are mainly associated with investigating whether there is a lixiviant that minimizes potential OH&S and downstream environmental toxicity risks on ecological and/or human systems. These risks would only occur in the event of accidental release, which should be prevented with appropriate BMPs in shipping and on-site handling, packaging, and system equipment engineering design (see Chapter 35, Accidents and Malfunctions). The main attributes used to assess the level of risk of adverse human or environmental health toxic effects are lixiviant *toxicity level*, *persistence*, and *chances of release* in the environment.

Government regulatory systems for chemical substances—like the Workplace Hazardous Materials Information System (WHMIS) for on-site handling (Health Canada 2010) and the Transport of Dangerous Goods (TDG) Regulations (Transport Canada 2011)—usually use factors such as those listed above to develop safety ratings classes, handling guides, and emergency procedures guides that will be used by the Project. If a substance is not classified under these systems, this doesn't necessarily imply a lack of toxic risk. The standards systems are based on demonstrated "weight of evidence" compiling wide and systematic research on risk factors, so it can take several years for new substances to be classified with these programs. Regardless of lixiviant used, the Project would be required by the *Hazardous Products Act* (1985b) to have specific Material Safety Data Sheets (MSDS) produced in concordance with the Canadian WHMIS system for every chemical used (Health Canada 2008), which would include health and ecological toxic risk factors, storing and handling guidance, and emergency procedures, to ensure the safe handling and transport of all Project substances.

33.10.3.2.1 Toxicity and Persistence

The level of toxicity of substances is typically derived from lethal dose (LD) studies (research on chronic effects is more challenging to conduct and less prevalent). Indicators used as proxies for toxic risk level in this assessment are taken from studies on the lethal dose (LD_{50}) and lethal concentration (LC_{50}) of the substance to cause mortality in 50% of a test population. LD_{50} values can pertain to OH&S and oral or inhaled risk to terrestrial organisms, while LC_{50} values pertain to aquatic organisms. It is important to note that independent LD_{50} and LC_{50} findings are from different studies performed under different conditions and with separate species, and so serve as a rough metric of lixiviant toxicity level rather than as the comprehensive guide the TDG provides. Table 33.10-2 provides a comparison of the six lixiviants based on their toxicities according to several TDG Regulations Schedule 1 (Transport Canada 2012) controlled substances, MSDS produced for the lixiviants by various chemical manufacturers, and primary research.

The lixiviant and by-product substances listed in Table 33.10-2 that are considered toxic under TDG Regulations are those in Class 2.3 for gases (chlorine and ammonia) and Class 6.1 (cyanide compounds, bromine). Substances are ranked by TDG as Class 6.1 (Toxic Substances):

- *due to oral toxicity if its LD_{50} (oral) is less than or equal to 300 mg/kg;*
- *due to dermal toxicity if its LD_{50} (dermal) is less than or equal to 1,000 mg/kg; or*
- *due to inhalation toxicity:*
  - *by dust or mist if dust or mist is likely to be produced in a transport accident and its LC_{50} (inhalation) is less than or equal to 4 mg/L, or*
  - *by vapour if its LC_{50} (inhalation) is less than or equal to 5,000 mL/m^3* (Transport Canada 2011).

Substances are ranked as Class 2.3 “Toxic Gases” by TDG that:

- *are known to be toxic or corrosive to humans according to CGA P-20, ISO Standard 10298 or other documentary evidence published in technical journals or government publications, or*
- *have an LC_{50} value less than or equal to 5,000 mL/m^3* (Transport Canada 2011).

In addition to the TDG toxicity classification, Table 33.10-2 provides potential adverse OH&S and ecological risks with persistence information, and toxicological information based on LD_{50} and LC_{50} indicators for mammals (ingested and inhaled) and fish, the latter indicating level aquatic toxicity and potential downstream effects in the event of a spill. The results of this table are summarized in Table 33.10-1.

The compiled attributes on the direct human health and downstream ecological toxicity effects of the alternative lixiviants and their common degradation or by-products in Table 33.10-2 provides a basis to compare the alternative lixiviants to cyanide. The result is that chlorine/chloride, bromine/bromide, thiourea, thiocyanate, and ammonia-based systems also pose pronounced direct OH&S and downstream toxic risks—and so are not considered suitable replacements for cyanidation on the sole basis of reduced toxicity. Further, due to the potentially adverse carcinogenic and foetal reproductive effects of thiourea, combined with its environmental persistence, this lixiviant is considered **unfeasible** for use by the Project. Bromine is also considered **unfeasible** due to its demonstrated OH&S risks and potentially high adverse aquatic effects combined with its bioaccumulation potential and persistence of its derivative product, bromide, in the environment.

While posing toxicological risks in the event of a spill, cyanide- and ammonia- based systems are widely used in a variety of industrial applications with demonstrated effective BMPs, which bring down OH&S and downstream risks (for cyanide, this includes the International Cyanide Management Code [ICMI 2012]; Principles and Standards of Practice of the International Cyanide Code are provided in Appendix 33-C). Hence, these lixiviants are considered **acceptable** for use by the Project. Of all the lixiviants compared in Table 33.10-2, only sodium thiosulphate appears to be a significantly safer alternative based on the available information, so this lixiviant is preferred based on stated environmental and OH&S performance objectives.

33.10.3.2.2 Chances of Release/Exposure

The chances of transport-related spills of lixiviants are characterized as unlikely or rare for the Project because transport of lixiviants will be infrequent and accident rates are relatively low for transport routes due to the Project's remote location (Rescan 2012). Transport containment systems for potentially hazardous substances will be set by TDG Regulations (Transport Canada 2011) to minimize release in the event of an accident, and the chances of a truck spill leading to lixiviant spilling into a waterway en route are very low.

The risks of accidental release during on-site processing are also very low due to WHMIS handling guidelines and BMPs, but risks do go up based on solid, liquid, and gas chemical states and temperatures, with solids being the easiest to contain under cool conditions, and gases the hardest to contain at high temperatures. For this reason, chlorine, bromine, and ammonium thiocyanate are considered at most risk of processing leaks—and related OH&S exposure risks—due to their combined corrosive nature, high temperature processing, and gaseous state. The lack of information on the risks—and related best practices handling/containment protocols—of some substances, in particular thiourea, may also increase the risks of workers being exposed to these substances (Hazardous Substances Data Bank 2011). The chances of an accidental spill of tailing, no matter the reagent used, are assumed to be about the same for the Project.

33.10.4 Selected Alternative

Cyanidation is selected as the most appropriate method of gold extraction for the Project. Although sodium thiosulphate minimizes the OH&S risks and some of the environmental risks compared to cyanide, the technical and economic challenges of implementing thiosulphate listed in Table 33.10-1, preclude it for use by the Project. Cyanide therefore remains the preferred lixiviant due to its technical and economic attributes, and the ability to mitigate OH&S and environmental risks with the use of well-established BMPs, including cyanide recycling, destruction, and treatment to ensure that water quality guidelines are met. To implement the use of cyanide for use by the Project, the Proponent has committed to use best practices, consistent with the International Cyanide Management Code for cyanide transportation, storage, and use in ore processing (ICMI 2012). Appendix 33-C presents the excerpted Principles and Standards of Practice from the International Cyanide Management Code.

Table 33.10-2. Comparison of Alternative Lixiviant Toxicity

Substance	TDG Regulations (Transport Canada 2012)	Toxicological Indicators			Potential Effects in the Event of Accidental Release	
	Schedule 1 Class Substances (Class / Label)	LD_{50} Acute Oral (mg/kg)	LD_{50} Inhalation (ppm)	LC_{50} Ecological (mg/L)	Potential Direct Occupational Health & Safety Effects to Personnel (on site)	Potential Effects to Environments and Human Populations (downstream)
Lixiviant 1: Sodium Cyanide	6.1 Toxic Substances	6.44 (rat) (WISER 2012)	-	15-81[7] (96 h, cyanate rainbow trout)	Highly toxic by ingestion, skin absorption, and inhalation. (EPA 1994b)	Forms hydrogen cyanide in water; rapid volatilization. See hydrogen cyanide (EPA 1994b).
By-product: Hydrogen Cyanide	6.1 Toxic Substances 3 Flammable Liquids	3.7 (mouse) (WISER 2012)	142 (rat) (WISER 2012)	.057 (96 h rainbow trout) (WISER 2012)	Highly toxic; health consequences at low doses. Potentially fatal if inhaled or swallowed (CALGAZ 2005).	Highly toxic, but not persistent. Volatilization half-life: 3 hours (model river); 3 days (model lake). No bioconcentration. (CALGAZ 2005) CN_{WAD} 30-day average max. water quality guideline for freshwater aquatic life is 5 µg/L and 10 µg/L max at any time (BC MOE 1986).
Lixiviant 2: Chlorine	2.3 Toxic Gases 8 Corrosives	-	137 (mouse) 260-344 (rat) (WISER 2012)	.005 - .16 *Daphnia* spp. .1 - .82 fish spp. 96 h (EPA 1994a)	Gas inhalation irritating to nose, throat, and lungs at lower concentrations, with symptoms leading pulmonary edema and death at high concentration. Liquid form, highly corrosive to skin. Chronic exposure at 0.4-9.0 ppm can cause respiratory effects, or wearing of teeth enamel (Cleartech Industries 2010).	Acutely lethal to fish and aquatic organisms at low doses, but does not bioaccumulate. Free chlorine reacts with dissolved organic compounds, which can form harmful by-products. (EPA 1994a) Rapidly forms hypochlorous acid and chloride ion (Cl-) in water (see chloride below for effects).
By-product: Chloride	Not listed	1,000-3,000 (rat) for various salts. (WHO 1996)	-	1,204-13,085 96 h: Cladocerans and invertebrates (BC MOE 2006)	Not toxic. Adult body contains 81.7 g, loses about 530 mg/day, so needs daily intake of about 9 mg/kg body weight (WHO 1996).	Mobile, persistent, transported to sea. Natural fresh water levels range from 1-100 mg/L. Adverse effects to freshwater species: acute 735–4,681 mg/L; chronic around 150 mg/L (BC MOE 2003).
Lixiviant 3: Bromine	8 Corrosives 6.1 Toxic Substances	2,600 (rat) (WISER 2012)	85.2 (rat) 750 (mouse) (WISER 2012)	.31 - .52 (various spp.) (Acros Organics 2004; WISER 2012)	Toxic gas by inhalation, moderately toxic by ingestion. Corrosive to skin (Denisen 1994).	Harmful to aquatic life. Elemental bromine in water is very reactive with dissolved organic compounds, forming highly toxic and carcinogenic by-products (Flury and Papritz 1993).
By-product: Bromide	Not listed	2,700–5,430 (mammals) (Flury and Papritz 1993)	-	2.3 - 7.8 Lowest reported for invertebrates, fish and amphibians (Flury and Papritz 1993)	Generally not harmful at very low doses but bioaccumulation from chronic low exposure can lead to bromism (Denisen 1994).	Bromide persistence: up to 10 years. Concentrations greater than 1 mg Br-/L may have potential for adverse aquatic environment effects. If water containing Br- is chlorinated for drinking, can form toxic and carcinogenic compounds (Flury and Papritz 1993). In BC, working guideline: drinking water should not surpass 50 µg/L (monthly mean; BC MOE 2006). Levels up to 45 mg/L have been reported in freshwater from industry (Flury and Papritz 1993).
Lixiviant 4a: Sodium Thiocyanate	Not listed	232-837 (rat) (EMD 2009)	-	83-250 Various spp. and ages (EMD 2009)	Very hazardous in case of ingestion, skin contact, or inhalation (Sciencelab.com Inc. 2012b). Flaccid paralysis, muscle contraction, nausea, and vomiting recorded from higher dose exposure (Santa Cruz Biotechnology Inc. 2009).	Degradation products likely more toxic (Sciencelab.com Inc. 2012b). Requires similar water treatment as cyanide due to cyanate; some studies show degrades to ammonia and sulphate in 4 days (Santa Cruz Biotechnology Inc. 2009). Harmful to aquatic organisms; toxic to micro-organisms at high doses. See sulphate and ammonia, below.
Lixiviant 4b: Ammonium Thiocyanate	Not listed	See ammonia and sulphate	-	See ammonia and sulphate	See ammonia and sulphate	See ammonia and sulphate
Lixiviant 5a: Ammonium Thiosulphate	Not listed	See ammonia and sulphate	-	See ammonia and sulphate	See ammonia and sulphate	See ammonia and sulphate.
By-product: Ammonia	2.3 Toxic Gases 8 Corrosives	350 (WISER 2012)	303 (mouse) (WISER 2012)	.97 (rainbow trout 24 h); 8.2 (fathead minnow 96 h) (Air Products 1999)	Toxic gas can cause severe eye, skin, and respiratory tract burns. Can cause central nervous system effects progressing to death following 5 minute at 5,000 ppm exposure (Air Products 1999).	Ammonia (NH_3) is toxic to fish with provincial water quality criteria concentrations being pH and temperature dependent. For instance, at pH 7.0 and temperature of 0 ºC, the 30 day average chronic concentration (of N) is 2.08 mg/L (BC MOE 2001b).
By-product: Sulphate		200 (magnesium sulphate) (Health Canada 1987)		Recent reports indicate the lowest effect level to be 205 mg/L (BC MOE 2000)	Generally not toxic to humans. At concentrations over 500 mg/L, has laxative effect, and above 1,000 to 2,000 mg/kg body weight has a cathartic effect, resulting in purgation of the alimentary canal (Health Canada 1987).	Sulphates released into tailing may exacerbate metal leaching and acid rock drainage, and pose water treatment challenges. The maximum draft water quality guideline to not lead to adverse downstream effects set for the Project is 250 mg/L.
Lixiviant 5b: Sodium Thiosulphate	Not listed	5,000 (rat) (Acros Organics 2012)	-	24,000 (96 h mosquito fish) (Acros Organics 2012)	Irritant. May cause eye, skin, and respiratory tract irritation; prolonged contact may cause dermatitis (Acros Organics 2012).	See sulphates comments above for ammonium thiosulphate. No drinking water guidelines.
Lixiviant 6: Thiourea	Not listed	125 (rat) (Fisher Scientific 2006)	-	600 (fathead minnow) (Gos and Rubo 2001)	Harmful if swallowed; adverse reproductive and foetal effects; chronic thyroid damage (Fisher Scientific 2006). Carcinogen (Fisher Scientific 2006; Xia 2008; US DHHS 2011; Sciencelab.com Inc. 2012a).	Harmful to aquatic life. Persistent: generally resistant to aquatic biodegradation; half-life of 171 days in sunlit natural water; and persisted 15 weeks in one soil study. Potential carcinogen if in drinking water (Fisher Scientific 2006).

Notes: LC_{50} = Lethal concentration to produce death in 50% of test population; sources included within table
Arrows in the table indicate chemical pathways from lixiviants used to by-products.

33.11 Waste Rock Disposal

33.11.1 Purpose and Background

Determining the location and method of waste rock disposal is one of the key decisions for metal mines. Waste rock consists of overburden and other rock materials, ranging from soil and fine sand to large boulders that lie over and around ore deposits as well as other areas, which requires appropriate disposal once it is excavated. Waste rock is either barren of precious metals or may have concentrations below cut-off grades, so what is originally classified as waste may change over a project lifetime based on metal commodity prices.

Over its life, the Project will generate about 3.03 Bt of waste rock (Appendix 4-C). Waste rock will be generated by extraction activities from open pit mining of the Kerr, Sulphurets, and upper Mitchell deposits, with lesser volumes from underground mining, numerous rock cuts for activities such as the construction of roads, tunnels, and diversion channels. Waste rock volumes have been significantly reduced for the Project by changing the mining method from open pit to combined open pit and underground block cave mining for the Iron Cap deposit and the lower section of the Mitchell deposit, as described in Section 33.3. This change has lowered the Project's strip ratio from 2.7 to 1.5 by eliminating 2.6 Bt of waste rock (Table 33.3-3) from the original volume projected in the 2011 pre-feasibility study (Appendix 4-C). This substantial reduction accounts for almost half of the original waste rock predicted for the Project, and related reductions in potential environmental effects from RSF footprints and acid mine drainage.

Identifying suitable waste rock disposal locations requires careful consideration as RSFs can occupy large footprints, reach high elevations, and contribute to ML/ARD. The main attributes that are considered for RSF disposal are minimizing habitat disturbance and loss, preventing and minimizing potential ML/ARD, minimizing haul distances, and finding a sufficiently large area to contain the waste rock.

33.11.2 Alternatives Identification

Even with the waste rock reductions from the switch to underground mining, space for waste rock disposal is very limited near the Mine Site. A number of alternative locations were examined to optimize safety and economy and to minimize potential adverse effects to surrounding environmental and human systems.

33.11.2.1 Pre-assessment

Technical criteria—such as foundation conditions, topographic features, and geohazard risks—provide preliminary constraints to the potential sites that can be used for waste rock disposal. RSF layouts were designed by Moose Mountain Technical Services, with assessment of stability and geotech guidance from Klohn Crippen Berger Ltd. (KCB). Site and waste rock characterization from initial desk and field studies help identify such constraints. Desk-based investigations of potential RSF locations for the Project began in 2004, when AMEC reviewed the viability of several different RSFs and recommended a few for further investigation. This led to some waste disposal sites being suggested, that assumed that waste rock would likely be not acid generating (Rescan 2008). Subsequently, geotechnical, hydrogeological and hydrological characterization site investigations were conducted by KCB in 2008, 2009, and 2010, and Rescan Environmental Services Ltd. also began baseline studies in 2008. Geohazards at the Mine Site were then conducted by BCG Engineering Inc. (BGC), and results were used for the selection of sites to avoid higher risk areas and to mitigate geohazards (Appendices 9-A and 9-D). These investigations led to the further assessment of RSF locations, the results of which are summarized in Table 33.11-1.

Table 33.11-1. Waste Rock Alternatives Pre-assessment

Alternative	Pre-assessment Summary	Result
Mitchell Valley RSF	*Feasible* to build a valley fill RSF in Mitchell Valley based on capacity, topography, and hydrological parameters.	Carry forward
McTagg Valley RSF	*Feasible* to build a valley fill RSF in McTagg Valley based on capacity, topography, and hydrological parameters.	Carry forward
Gingras Valley RSF	*Feasible* to build a valley fill RSF in Gingras Valley based on capacity, topography, and hydrological parameters.	Carry forward
Sulphurets RSF	*Feasible* to build side hill fills in several locations in Sulphurets Valley based on physiographic parameters.	Carry forward
Co-disposal in Treaty TMF	Co-disposal would submerge waste rock, preventing against ML/ARD, but *unfeasible* due to insufficient capacity (2.3 Bt) of the TMF to store all the tailing plus waste rock volume.	Eliminate
Mitchell Creek Valley isolated PAG waste rock storage	Most of the waste rock was later found to be PAG, making isolating PAG *unfeasible*. Also, this option requires that PAG waste rock be transported over several kilometres in rugged terrain which is technically *challenging*.	Eliminate
Backfill into Mitchell Pit after closure	After the mine plan changed to underground mining, it became *unfeasible* to backfill over Mitchell underground mining operations for engineering and safety reasons.	Eliminate
Backfill into Sulphurets Pit after closure	*Feasible* as option to segregate just the Kerr waste rock (high-PAG and high selenium) to facilitate contact water treatment.	Carry forward
Back fill into Kerr Pit after closure	*Unfeasible* due to distance from other pits, scheduling and water management challenges.	Eliminate

Source: Rescan (2008); Wardrop (2011, 2012a); KCB (2009, 2012)

Decisions regarding project waste rock disposal alternatives took into consideration meeting the BC Mine Waste Rock Pile Research Committee Interim Guidelines (Piteau Associates 1991), guidelines developed by the BC Waste Dump Research Committee (1991 to 1995), and the Health, Safety and Reclamation Code for Mines in British Columbia (BC MEMPR 2008). One of the key factors used in selection of waste rock disposal sites is determining if waste rock will be potentially acid generating or not (PAG and NPAG, respectively; Price 1997). PAG waste rock disposal typically involves extra engineering provisions to appropriately manage contact water to prevent ML/ARD. In addition, to prevent spilling PAG rock, the handling/transport of PAG rock over rugged terrain, such as involved in the Project, should be avoided. Testing performed on the four Project deposits revealed that the Iron Cap, Kerr, and Mitchell deposits have high amounts of waste rock predicted to be PAG (Appendix 10-A). For this reason, the locations chosen to store waste rock from these deposits should be able to contain seepage to minimize ML/ARD, incorporate measures to minimize the generation of and treat contact water, and minimize transport distance to reduce dusting and spill risks. Testing of waste rock from the Sulphurets deposit found more NPAG rock compared to other deposits (Appendix 10-A), which means that this waste could also be used in constructing mine components rather than needing engineered disposal (Appendix 4-J).

One of the preferred methods to prevent the formation of ML/ARD is co-disposal of waste rock and tailing under subaqueous conditions, such as in a TMF (A. Li et al. 2011). For this reason, co-disposal was originally proposed for the Project based on preliminary estimates that the Project would generate about 1 Bt of tailing and 1 Bt of waste rock (Rescan 2008). It was later found that the total tailing would actually be 2.3 Bt (see Section 4.5.3.10.1 in Chapter 4, Project Description), with a contingency for expansion, so as reported in Table 33.11-1, co-disposal will be unfeasible for the Project.

33.11.2.2 Resultant Rock Storage Facility Alternatives

The results of the pre-assessment narrowed the RSF candidates to five potential sites (Figure 33.11-1):

- Mitchell RSF;
- McTagg RSF;
- Sulphurets Valley RSF;
- Backfill Kerr waste into Sulphurets Pit; and
- Gingras Valley RSF.

33.11.2.2.1 Mitchell Rock Storage Facility

The Mitchell RSF will be located in the Mitchell Creek Valley a short distance from the Mitchell, Sulphurets, and Iron Cap deposits in the Mitchell Creek Valley. The layout involves placing waste rock across the valley floor (with NPAG waste rock reporting from Sulphurets Pit forming the under drains) to 810 masl on the north margin of the valley, and then raising bottom up benches on the south slope of the valley (Appendix 4-J). This RSF will eventually extend into the lower reaches of McTagg Creek, joining with the McTagg RSF as shown in Figure 33.11-1. The Mitchell RSF is designed to store 1.58 Bt of waste rock to a final elevation at 1,200 masl (Chapter 4, Project Description, Section 4.5.1.7).

33.11.2.2.2 McTagg Rock Storage Facility

The McTagg RSF was considered as an option to potentially dispose Mitchell, Iron Cap, Kerr, and/or Sulphurets waste. The McTagg RSF alternative is similar to the Mitchell RSF in that it will involve filling a valley to 900 masl on the north and west margins of the McTagg Valley, and to 1,020 masl along the eastern margin (Appendix 4-J). The westward extension reaches into the neighbouring valley to the toe of the West Glacier and east into the East McTagg Glacier area. This RSF is planned for storage of 0.76 Bt of waste rock (Chapter 4, Project Description, Section 4.5.1.8), though it has an estimated 2.23 Bt capacity. As a result of the reduction in total mine rock with the inclusion of underground mining, the footprint of the RSFs will be reduced, with no need to place mine rock in the presently glaciated areas of the Upper McTagg Valley. The McTagg RSF will eventually merge with the MRSF.

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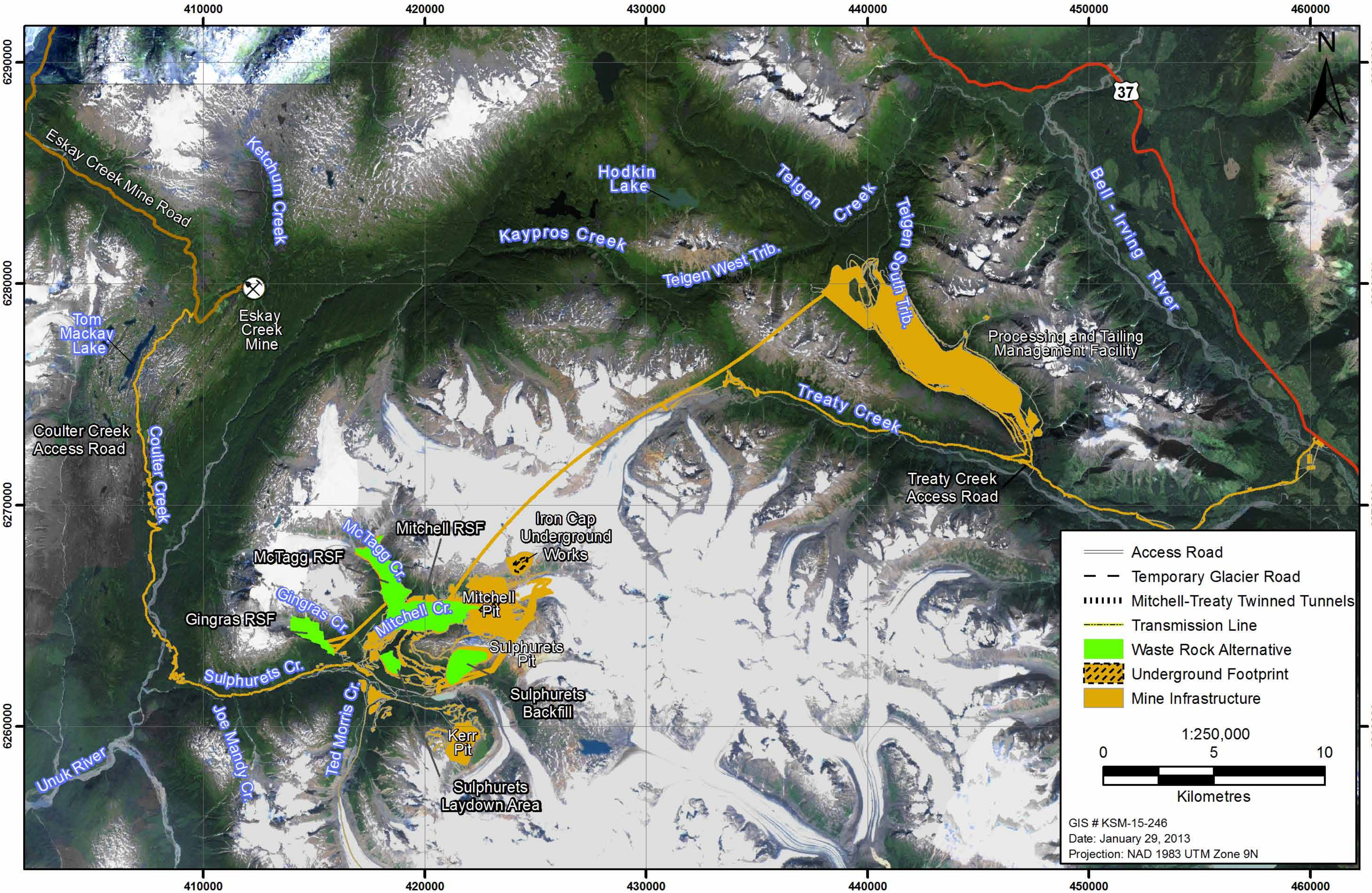


Figure 33.11-1

Figure 33.11-1

SEABRIDGE GOLD
KSM PROJECT

KSM Project Waste Rock Alternative Disposal Sites

Rescan
Engineers & Scientists

33.11.2.2.3 Sulphurets Rock Storage Facility

The Sulphurets RSF was primarily identified to store waste from the Sulphurets Pit (Wardrop 2011). For this alternative, waste rock would be stored as a side hill formation in the Sulphurets Valley. Two potential side hill locations were identified for Sulphurets including a 168 Mt RSF on the south side of the west ridge of the Sulphurets deposit, and an 80 Mt RSF on the side of the Sulphurets Valley west of the Kerr deposit.

33.11.2.2.4 Backfill into Sulphurets Pit

In the fall of 2012, geochemical testing revealed that the selenium content of waste rock from the Kerr Pit would cause significant increases in the selenium content of contact water contained in the Water Storage Facility (WSF) if Kerr waste rock was placed for disposal in the McTagg RSF. The option to backfill 0.66 Bt of Kerr waste rock (Chapter 4, Project Description, Section 4.5.1.5.5) into the Sulphurets Pit was developed to mitigate related water quality challenges (Appendix 4-J). For this alternative, waste rock is proposed to be backfilled into Sulphurets Pit starting in Year 27, after the mining activities in Sulphurets Pit are finished; contact water will be treated prior to reporting to the WSF and undergoing further treatment in the Water Treatment Plant (WTP; see Sections 33.12.3 and 33.12.4).

33.11.2.2.5 Gingras Rock Storage Facility

The Gingras RSF was proposed to dispose of waste reporting from Kerr, Sulphurets, and/or Mitchell deposits. For this alternative, a valley fill RSF would be developed in the Gingras Creek Valley as shown in Figure 33.11-1. The RSF would extend into the Sulphurets Creek Valley downstream of the WSF and is estimated to contain 450 Mt of waste rock at a final elevation of 1,335 m. All RSFs will require road access, and in this case, a double-lane haul road would have to be built into the valley, passing through exceptionally steep and treacherous terrain.

33.11.3 Comparison of Alternatives

Table 33.11-2 provides a summary of the assessment of the four RSF alternatives based on how their attributes align with the four Project performance objectives.

33.11.3.1 Technical and Project Economic Considerations

The main technical attributes to meet technical and economic performance objectives used in the assessment are maximizing waste rock storage capacity, reducing haul distance/cycle time, avoiding/mitigating foundation issues or geohazards, and determining whether there are any particular technical or economic barriers or advantages of a given RSF site.

One of the primary considerations in waste rock placement for the Project is the need to minimize the distance that PAG rock is hauled. Mitchell is preferred overall as it minimizes distance from most of the waste rock for the Project, in particular compared to the Gingras Valley location. The Sulphurets RSF location is the closest to the Sulphurets and Kerr pits, and would reduce costs and cycle time considerably in the first part of the Project when waste would have to be trucked the extra distance down into the Mitchell Valley. Backfilling Kerr rock into the Sulphurets Pit would also shorten cycle times compared to the alternative of having Kerr waste rock report to the McTagg RSF. The Gingras Valley option was found to have the longest haul distance (including hazardous terrain that would increase risks of spilling PAG rock), which would raise construction costs and render cycle times so high as to not be supported by Project economics. For these reasons the Gingras RSF alternative has been rated as unfeasible overall.

Using the Sulphurets RSF in the Sulphurets Valley would significantly reduce cycle times and haul costs for waste rock reporting from the Sulphurets and Kerr pits. However, site investigations found that the hydrological conditions of Sulphurets Valley, in association with the long term characteristics of waste rock could lead to downstream adverse effects that would be challenging to manage. For this reason, this location is considered to be unfeasible.

Backfilling of Kerr waste into Sulphurets Pit was found to be feasible and preferred compared to placement in the McTagg RSF as shown in Table 33.11-1. Although backfilling into the Sulphurets RSF would require extra infrastructure for management of contact water, this alternative will facilitate managing Kerr waste rock contact water selenium levels through isolation and treatment prior to reporting the WSF, and will enable improved water quality management for the Project compared to the McTagg RSF option (see Section 33.12.4 on selenium treatment alternatives).

Slope stability was not a primary consideration in selecting RSF locations, but has been an factor involved in the RSF design and safety of selected RSFs based on BMPs. RSF geotechnical stability during construction and closure was analyzed by KCB using SLOPE/W© 2007 software (Appendix 4-C). Site factors affecting RSF stability—site topography, foundation conditions, climate, hydrology, and waste rock strength—were assessed as indicative measures of dump stability hazard and exposure risk. Both the selected Mitchell RSF and McTagg RSF have been classified as a moderate failure hazard based on the Mined Rock and Overburden Piles, Investigation and Design Manual (Piteau Associates 1991).

The Mitchell RSF will overlie some glacial sediment and Holocene alluvium and colluvium, with depths ranging from 2 m upslope to about 120 m in the valley bottom (KCB 2011). In order to consolidate this foundation material, and for areas where lacustrine sand and clay deposits were found, mitigation measures have been developed to ensure foundation strength and RSF stability (Appendices 9-A, 9-D, and 4-J). The Mitchell RSF is also subject to landslide and avalanche geohazards, which can be brought to acceptable levels through mitigation. In addition, placement of mine rock will provide buttressing to unstable slopes, improving stability and reducing landslide risks as the RSF is developed, and avalanche hazards will reduce as the RSF surface rises on the south side of the valley, although some avalanche risk may still remain on the north side of the Mitchell RSF, which would be mitigated per monitoring regulatory requirements and OH&S BMPs.

In order to increase strength and provide stability for all RSFs, tills will be removed from foundations and adjacent areas, and the dumping sequence for each will begin by building the RSF bottom-up lifts. Slope stability will be ensured by following relevant guidelines and best practices, and monitoring the RSFs will also be an important component of safety regarding stability. Mine staff will inspect RSFs daily—with the use of electric piezometers, inclinometers, surface survey monuments, and potentially radar or optical scanning instruments—looking for indications that would provide advanced warning of a possible RSF failure (Appendix 4-C).

Table 33.11-2. Comparison of Waste Rock Alternative Locations

Alternative	Environmental	Social	Technical	Project Economic	OVERALL ALTERNATIVE RATING
	ATTRIBUTE RATINGS AGAINST FOUR PERFORMANCE OBJECTIVES				
Mitchell RSF (for waste rock from Sulphurets, Mitchell, and Iron Cap)	✓ ● Good containment and routing of contact water to WSF ✓ ● Good ability to control water infiltration to dump (i.e., through diversion structures) prevents ARD generation x ● Western toad observed in vicinity of proposed RSF x ● Contributes to loss of areas suitable as wildlife habitat ✓ ● Very low loss of winter habitat for mountain goat ✓ ● Can be partially reclaimed to forest on closure	✓ ● Reduced distance lowers haul driver risks	✓ ● Has the largest capacity (2.3 Bt) ✓ ● Closer to and faster cycle time[5] to most waste rock sources ✓ ● Reasonable haul route x ● Appropriate storage areas to salvage and store till for till cap will be very difficult to find in the tight constraints of the valley x ● Foundation in some areas reduces stability; mitigation through toe berm in Year 1 and pre-consolidation will assure adequate stability x ● Geohazards present but can be managed with mitigation	✓ ● Closest to Mitchell Pit, reducing haul costs and cycle times, increasing the financial viability of the Project x ● Waste has to be trucked down the valley, then back up—inefficient	**PREFERRED**
	Acceptable	Acceptable	Preferred	Acceptable	
McTagg RSF (Considered for waste rock from Mitchell, Iron Cap, Kerr, and Sulphurets)	✓ ● Good containment and rerouting of contact water to WSF ✓ ● Good ability to control water infiltration to dump (i.e., through diversion structures) prevents ARD generation x ● Kerr waste rock: existing selenium content presents challenges for water treatment with associated potential downstream adverse effects x ● Contributes to loss of areas suitable as wildlife habitat ✓ ● Can be partially reclaimed to forest on closure	x ● Slightly longer distance increases haul risks	✓ ● Similar capacity (0.76 Bt) to Gingras (second highest) ✓ ● 36 minute cycle time ✓ ● Haul distance is manageable (9.2 km for McTagg RSF from the Mitchell Pit rim) x ● **Kerr waste only**: Requires water treatment for selenium, but high volume of contact water reporting to McTagg RSF makes isolation of contact water from Kerr waste not possible	x ● Slightly farther from pits, increasing haul costs compared to Mitchell Valley and cycle times x ● Waste has to be trucked down the valley, then back up—inefficient	**CHALLENGING** (Kerr waste only) **ACCEPTABLE**
	Acceptable / Challenging (Kerr waste only)	Acceptable	Acceptable / Challenging (Kerr waste only)	Acceptable	
Sulphurets RSF (Considered for waste rock from Sulphurets)	x ● Local hydrogeological conditions, in association with the long-term characteristics of the waste rock, are considered to create unacceptably high potential adverse downstream effects ✓ ● Low wildlife habitat loss compared to other RSFs: Low loss of areas suitable as high value winter moose habitat and hoary marmot, and no grizzly bear denning nor mountain goat habitat lost ✓ ● Can be fully reclaimed to forest during operation	✓ ● Short distance reduces haul risk	x ● Reduced capacity compared to Mitchell, McTagg, and Gingras x ● Reduced capacity compared to other RSFs ✓ ● Used only for Sulphurets waste, reducing cycle time	✓ ● Closest to the Sulphurets and Kerr pits, reducing haul costs and cycle times, increasing the financial viability of the Project ✓ ● Temporary storage in Sulphurets RSF during about the first 20 years would reduce costs significantly compared to direct hauling to Mitchell or McTagg RSFs	**UNFEASIBLE**
	Unfeasible	Acceptable	Acceptable	Preferred	

(continued)

Table 33.11-2. Comparison of Waste Rock Alternative Locations (continued)

Alternative	ATTRIBUTE RATINGS AGAINST FOUR PERFORMANCE OBJECTIVES				OVERALL ALTERNATIVE RATING
	Environmental	Social	Technical	Project Economic	
Sulphurets Pit Backfill (for waste rock from Kerr)	✓ ● Good containment and rerouting of contact water, which can also be minimized using a cover ✓ ● Kerr waste rock selenium content in waste rock and resulting contact water can be better isolated than in McTagg RSF ✓ ● Lowest wildlife habitat loss, since in pit ✓ ● Can be partially reclaimed on closure	✓ ● Short haul distance reduces risks	x ● Reduced capacity (0.66 Bt) compared to Mitchell, McTagg, and Gingras, but only required for Kerr waste rock ✓ ● Short haul distance x ● Requires extra infrastructure to be built for diversion structures and rerouting contact water separately	✓ ● Closer to Kerr Pit than McTagg RSF x ● Although closer, haul elevation is expensive and adverse economically	**PREFERRED** (For Kerr waste rock placement over McTagg)
	Preferred (compared to McTagg)	Acceptable	Acceptable	Acceptable	
Gingras RSF (for waste rock from Kerr, Sulphurets, and Mitchell)	x ● Longer haul route increases air quality emissions compared to other alternatives x ● Contributes to loss of areas suitable as wildlife habitat ✓ ● Can be partially reclaimed to forest on closure x ● Would require an additional dam to segregate contact water, resulting in increased environmental disturbance	x ● Longer distance increases haul risks	✓ ● Similar capacity to McTagg (second highest) x ● 49 minute cycle time is one of the longest x ● Appropriate storage areas to salvage and store till for till cap will be very difficult to find in the tight constraints of the valley. x ● Rough terrain around valley and steepness makes the placement of fill material and long road construction/operation a challenge, especially to transport PAG rock	x ● Longest haul distance (12.2 km) and longest cycle time could not be economically supported by the Project x ● The requirement of an additional dam to segregate contact water would increase expenses	**UNFEASIBLE**
	Acceptable	Acceptable	Challenging	Unfeasible	

Notes:
✓ = advantage, x = disadvantage, ● = Preferred, ● = Acceptable, ● = Challenging, ● = Unfeasible; See Table 33.2-1 for attribute ranking specific definitions
WSF = water storage facility; ARD = acid rock drainage; cycle time = time to haul waste rock to dump and back

33.11.3.2 Environmental and Social Considerations

Acidic drainage from past mining of sulphidic geological materials globally has resulted in costly cleanup for industry and governments. As a result, the main environmental attribute considered to compare waste rock disposal sites is the ability to control and treat contact water to manage water quality effectively. As listed in Table 33.11-1, other attributes were also considered such as minimizing short- and long-term habitat loss and air emissions.

To meet receiving environment water quality guidelines, contact water will be collected from each RSF, and rerouted to the WSF for treatment prior to release to Mitchell Creek just before its confluence with Sulphurets Creek. Natural surface and glacial flows will be diverted and rerouted around the RSFs via lined diversion channels built up progressively, or through the Mitchell and McTagg diversion tunnels, and discharged to the Sulphurets Valley. Contact water collected in the WSF will require treatment in perpetuity to ensure mine effluent will meet end of pipe pollution control objectives and receiving environment water quality guidelines.

Due to the above water management features and mitigation, most RSFs are considered acceptable from a water management perspective. The exception is that storing high selenium content Kerr waste rock in the McTagg RSF is considered challenging due to issues to effectively manage the potential ML/ARD and selenium levels likely to be generated in contact water. For this reason, backfilling into Sulphurets Pit is the preferred disposal alternative for Kerr waste. In order to further prevent the formation of contact water with the waste rock backfilled into Sulphurets Pit, the Proponent plans to place a partial cover over the waste rock on closure that will prevent infiltration of water into the waste dump. In addition, contact water reporting from this waste rock dump will be separately collected and routed to a selenium treatment plant prior to reporting to the WSF for general treatment with the rest of the mine contact water. This will reduce costs as well as increase the effectiveness of water treatment for selenium.

The Sulphurets RSF minimizes habitat disturbance, but was found to have high water flow and foundation issues that would make managing contact water in this location problematic. For this and the earlier stated technical reason, this alternative is considered unfeasible.

Dust and other air quality and GHG emissions will be created when hauling and dumping waste rock for the RSFs. In general, longer haul distances will lead to more air emissions. Aquatic habitat and fish can be affected both directly from habitat loss as well as water quality. No fish habitat is located in the direct footprint of any of the RSFs and wetlands are also not predicted to be adversely affected by RSFs for the Project.

End land use objectives and reclamation requirements for the Project are mostly based on BMPs rather than alternatives. The objective for closure is to replicate similar landforms and habitats as existed pre-mining. Chapter 27 describes the closure and reclamation plan for the RSFs, and Section 33.14.2 discusses options considered for the closure and reclamation of the RSFs.

Social considerations of RSFs were primarily regarding longer routes likely leading to more work for personnel, counterbalanced by increased accident risks with longer (and more treacherous) distances.

33.11.4 Selected Alternatives

As summarized in Table 33.11-1, Seabridge has selected the following three waste rock disposal locations for the Project:

- Mitchell RSF;
- McTagg RSF; and
- Backfill into Sulphurets Pit—for Kerr waste rock.

As a result of Seabridge's decision to switch to underground mining for part of Mitchell and all of the Iron Cap deposits, the total footprint of the above RSFs will be reduced compared to the 2011 prefeasibility study (Wardrop 2011), with no need to place mine rock in the presently glacierized areas of Upper McTagg Valley (Appendix 4-J). The selection of the adjacent Mitchell RSF and McTagg RSFs will provide long-term waste rock storage for the Project and can be confined to the Lower Mitchell and McTagg valleys. This will increase the costs of waste haulage from the further mining areas, but will reduce the areas of disturbance, as well as reclamation and waste treatment requirements. Although these two RSFs have the capacity to store all the Project waste rock, the option to backfill Kerr waste rock into Sulphurets Pit was also chosen to meet water quality objectives.

33.12 Water Management

Ensuring that water quality meets requirements to ensure the safety of the receiving environments is one of the most important objectives in sustainable mine development. Mine water management consists of controlling both the water quality and quantity into and out of the Mine Site where activities may either use or influence natural water systems. Effective mine water management is central to efficient mine operation as well as to the mitigation of potential adverse effects on water quality and quantity.

The Project will affect and be affected by the water quantity and quality flowing in and out of both the Mine Site and PTMA. Suitable water management strategies for both sites are required for the use, diversion of, and treatment of water. The production rates (130,000 tpd) and size of the diverted catchment areas of the Project add to the level of complexity of water management, as well as to the costs to design and operate water management infrastructure. Figure 33.12-1 illustrates the two main Project areas (the Mine Site and the PTMA) with respect to local surface waterbodies and flows.

To meet water quality and quantity regulatory and best practices guidelines, water management for the Project will consist of the following objectives:

- collecting freshwater for mine start-up;
- preventing contamination of on- and off-site resources by diverting surface water and groundwater flows and preventing infiltration/seepage;
- staging construction to minimize disturbance to surface water;
- mitigating potential water contamination from ML/ARD or other wastewater sources (i.e., treating effluent from tailing and waste rock areas);

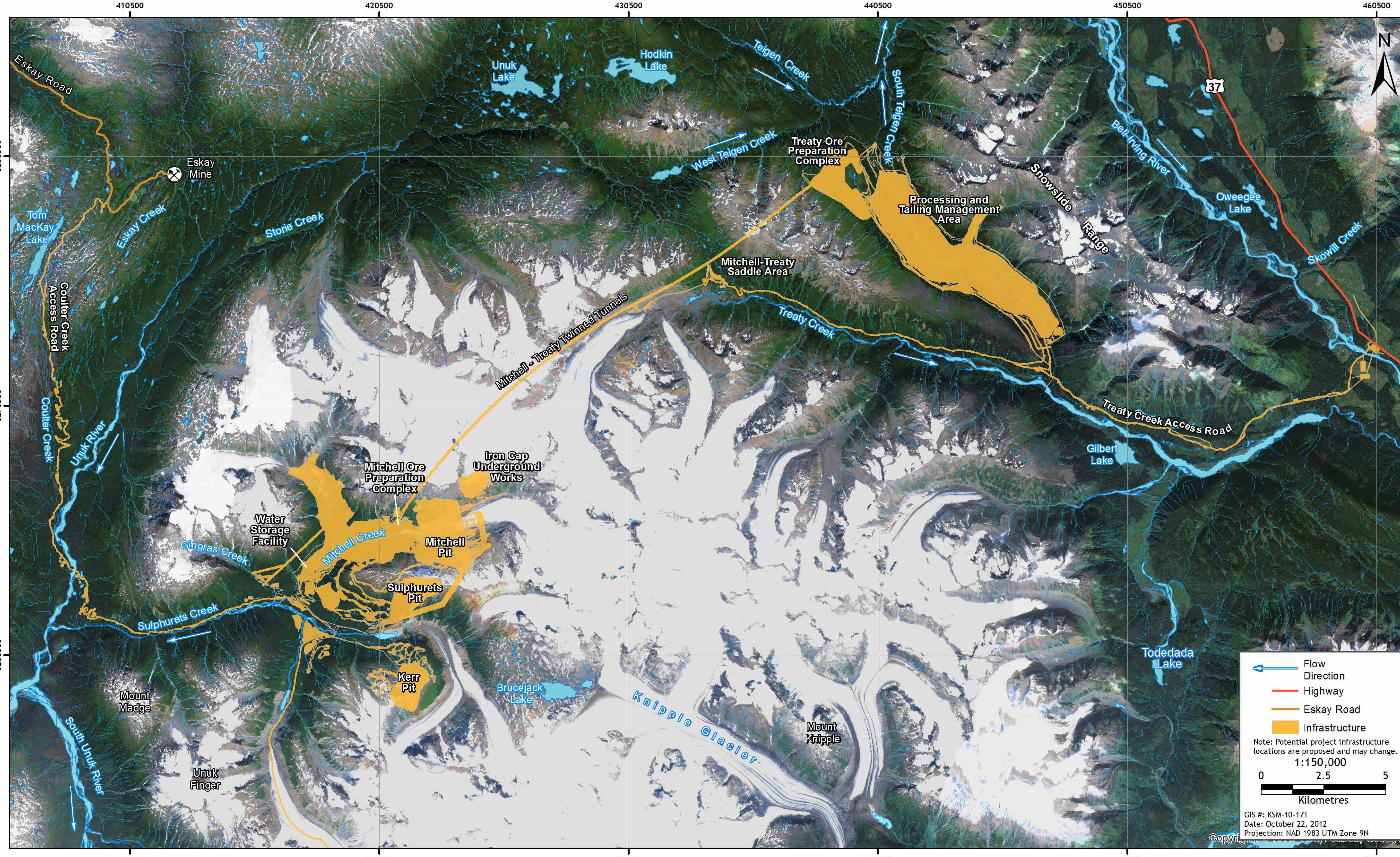


Figure 33.12-1

KSM Project Areas and Surface Water Features

- maintaining mine water balance through diverting inflows (i.e., runoff) and dewatering (i.e., pumping out excess water) that may cause issues to mine workings such as flooding;
- controlling erosion to reduce runoff sediment levels, or preventing damage to mine closure works;
- storing, reclaiming, and recycling of water (i.e., collecting and recycling runoff from disturbed sites) in order to reduce water use; and
- not discharging tailing supernatant or contact water during operations.

Potential alternatives regarding water management that are often considered at metal mines include diverting and storing water, tailing management, dewatering, discharge type and direction, water treatment, and the prevention of adverse effects to water quality and quantity during mine life and post-closure. The primary water management decisions that are summarized in this assessment are:

- TMF discharge direction;
- WSD fill types;
- mine contact water treatment techniques; and
- selenium treatment techniques.

33.12.1 Tailing Management Facility Discharge Direction

The assessment on TMF discharge direction was conducted in August 2012, and is provided in Appendix I of the TMF MAA located in Appendix 33-B of this chapter. The TMF is located about 23 km northeast of the Mine Site in the drainage divide between Teigen Creek and Treaty Creek. Both Teigen and Treaty creeks flow into the Bell-Irving River, which itself is part of the Nass River catchment. Figure 33.12-1 illustrates the two drainage options from the TMF that were available in this assessment, and Table 33.12-1 provides a summary of the main attributes considered in the assessment.

33.12.1.1 Alternatives Identification

After deliberation, including receiving input from Nisga'a Nation, Tahltan Nation, Gitxsan Nation, and Gitanyow First Nation through working group consultation, two main options for TMF discharge direction were identified:

- Option A – discharge direction north with closure flows routed north to Teigen Creek; and
- Option B – discharge direction south with closure flows routed south to Treaty Creek.

33.12.1.1.1 Option A: Discharge Direction North to South Teigen Creek

For Option A (Figure 33.12-2), closure flows will be routed north toward the south stem of Teigen Creek. To reduce excess water and allow progressive reclamation, tailing deposition would be staged with the North Cell operated during years 1 to 25 and the South Cell operated during years 26 to 52. Option A would consist of two primary dams, the North dam and the Southeast dam. Construction of a third dam, the Saddle dam, at the high point of the drainage divide between the North and Southeast dams, is proposed to allow for staging of the TMF into the North and South cells.

33.12.1.1.2 Option B: Discharge Direction South to Treaty Creek

For Option B (Figure 33.12-2), closure flows will be routed south to Treaty Creek, and the cell staging and dams are similar to Option A. This option also involves changing maintenance flows to South Teigen Creek by increasing the proportion of the diverted flows reporting to the toe of the North dam. During closure, the water surface elevations of the ponds associated with the CIL, North, and South cells are levelled to an elevation of 1,054 masl. A closure channel between the CIL Cell and South Cell ponds is excavated through the Saddle dam. Another closure channel between the North Cell and CIL Cell ponds is excavated through the Splitter dam. The 5 m deep water cover on the CIL Cell is retained. All of the diversion channels, except those associated with the seepage dams, are decommissioned, and runoff will report to the according cells. The East Creek Catchment diversion tunnel and pipeline are decommissioned.

The Southeast Spillway is constructed, and initially 70% of the South Cell excess flow is routed south to North Treaty Creek. Gates and weirs will allow flows to be regulated. Once water quality meets receiving environment conditions, excess flow from the CIL Cell and 30% of the South Cell excess flow and excess flows from the North Cell will be routed north to South Teigen Creek.

33.12.1.2 Comparison of Alternatives

The following were identified as being the main attributes affected by options A and B: water quality, fish and fish habitat, downstream fisheries, net cost, site conditions, geohazards, and infrastructure considerations such as dam height, and ease and effectiveness of design construction and operation. Table 33.12-1 summarizes the comparison, also provided in full in Appendix I of the TMF MAA located in Appendix 33-B.

33.12.1.2.1 Technical and Project Economic Considerations

The primary technical *similarities* between Option A and Option B are:

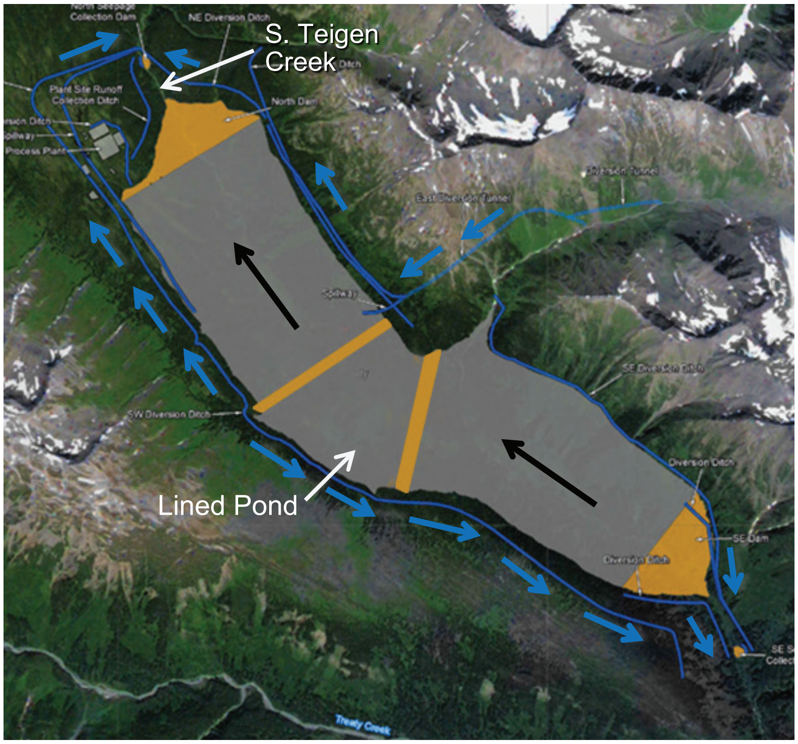
- overall footprint;
- dam height;
- overall site conditions;
- access and infrastructure routing;
- geohazard risk;
- catchment areas;
- seepage collection and water storage;
- probable maximum precipitation and freeboard; and
- tailing delivery and recovery.

Table 33.12-1. Summary Comparison of Tailing Management Facility Discharge Direction Alternatives

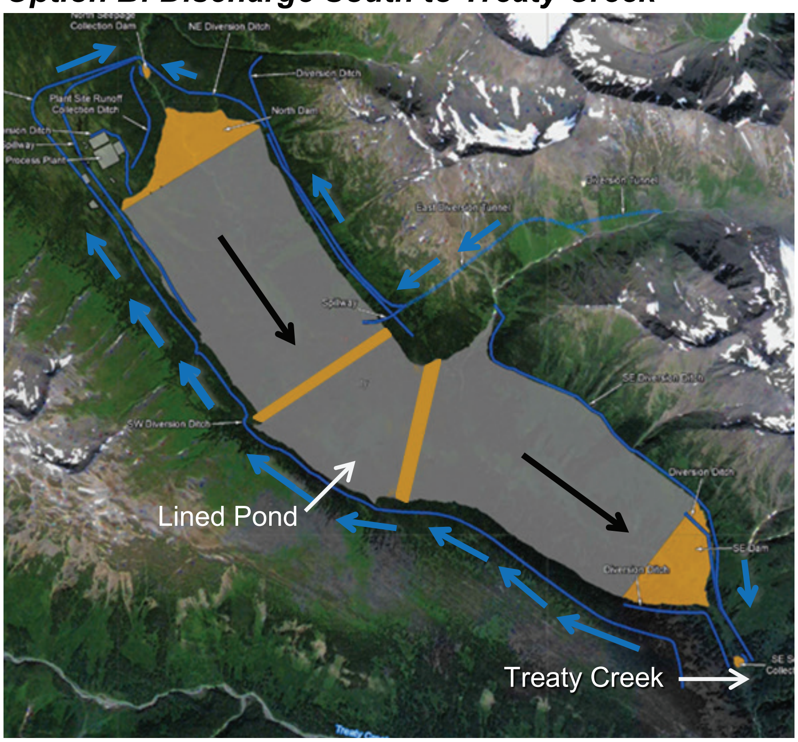
Alternative	ATTRIBUTE RATINGS AGAINST FOUR PERFORMANCE OBJECTIVES				OVERALL ALTERNATIVE RATING
	Environmental	Social	Technical	Project Economic	
Option A: Discharge North (to South Teigen Creek)	x ● South Teigen Creek contains two more species (bull trout [*Salvenlinus confluentus*] and rainbow trout [*Oncorhynchus mykiss*]) than Treaty Creek x ● Teigen Creek mainstem contains chinook and sockeye salmon, not found in Treaty Creek mainstem ✓ ● No bull trout present in Teigen Creek mainstem x ● Potential effects to an additional 5.1 ha of fish habitat compared to Option B x ● Important rearing habitat but little spawning habitat x ● Critical chinook and coho salmon spawning habitat potentially affected x ● Change of water flow may affect fish habitat in South Teigen Creek; however, fish rearing and overwintering habitat in South Teigen Creek will be maintained and downstream effects in Teigen Creek will be unlikely	x ● Treaty and First Nations have indicated concerns regarding discharge to South Teigen Creek regarding potential adverse effects on fish	✓ ● No substantial differences from Option B	✓ ● Cost is same as Option B: $2.2 billion	**ACCEPTABLE**
	Acceptable	Acceptable	Acceptable	Acceptable	
Option B: Discharge South (to Treaty Creek)	✓ ● Contains two fewer trout species than South Teigen Creek ✓ ● Contains two fewer salmon species than Teigen Creek x ● Bull trout present in Treaty Creek ✓ ● Potential effects to 5.1 ha less fish habitat than Option A x ● Abundant and important rearing, spawning, and overwintering habitat downstream ✓ ● No critical fish habitat affected ✓ ● Has higher base flows which allow mimicking of the natural hydrograph for discharge scenarios, resulting in significantly mitigating potential for downstream adverse water quality effects	✓ ● Treaty and First Nations have indicated that discharge to Treaty Creek is preferred for them compared to discharge to South Teigen Creek to minimize potential downstream effects on fish	✓ ● No substantial differences from Option A	✓ ● Cost is same as Option A: $2.2 billion	**PREFERRED**
	Preferred	Preferred	Acceptable	Acceptable	

Notes:
✓ = advantage, x = disadvantage, ● = Preferred, ● = Acceptable, ● = Challenging, ● = Unfeasible; See Table 33.2-1 for attribute ranking specific definitions

Option A: Discharge North to S. Teigen Creek



Option B: Discharge South to Treaty Creek



Discharge direction
Diversion flow direction

TMF Discharge Direction Alternatives

The primary technical *differences* between the two options are summarized as:

- Diversion designs for Option B have been advanced to improve maintenance flows to South Teigen Creek by increasing the proportion of the diverted flows reporting to the toe of the North dam.
- On closure, surplus water of suitable quality will be discharged to South Teigen Creek for Option A and to North Treaty Creek for Option B.

Option B (Discharge Direction South to Treaty Creek) is the more preferable TMF option from a technical perspective because it provides improved diversion flows. Economically, the two options are assessed to be the same, each at $2.18 billion.

33.12.1.2.2 Environmental Considerations

The main attributes taken into consideration that would be affected by options A and B were potential water quality effects to fish and downstream fisheries. Diversion designs for Option B were developed to improve regular maintenance flows during operation to South Teigen Creek by increasing the proportion of the diverted flows reporting to the toe of the North dam. Post-closure flows of surplus TMF water would be discharged to South Teigen Creek in Option A and to North Treaty Creek in Option B. Similarities between the two TMF drainage options are limited but include:

- Waterways that are potentially affected for both options contain two fish species of conservation concern.
- Waterways that are potentially affected for both options consist of mostly important fish habitat.

The summarized differences in potential fisheries effects from the two drainage options are as follows:

- South Teigen Creek and the Teigen Creek mainstem contain more fish species than North Treaty Creek and the Treaty Creek mainstem.
- Option A would potentially affect water flows for an additional 5.1 ha of fish habitat compared to Option B.
- Option A contains critical chinook salmon and coho salmon spawning habitat.

Option B (Discharge Direction South to Treaty Creek) would be preferable from a fisheries perspective over Option A (Discharge Direction North to South Teigen Creek) because South Teigen and Teigen creeks have more potentially affected fish habitat, a greater number of fish species, and potentially affected downstream critical salmon spawning habitat compared to North Treaty and Treaty creeks.

33.12.1.3 Selected Alternative

Option B (Discharge Direction South to Treaty Creek) is more preferable to Option A (Discharge Direction North to South Teigen Creek) because it has improved water management, as well as lower potential adverse downstream effects. Option A (Discharge Direction North to South Teigen Creek) was not chosen because Teigen Creek and its tributaries have more potentially affected fish habitat, more fish species, and critical salmon spawning habitat downstream, and its diversion flows were not as optimal as those possible with Option B.

33.12.2 Water Storage Dam Type

33.12.2.1 Purpose and Background

The proposed WSD on Mitchell Creek will collect Mine Site contact water in the WSF prior to treatment in the WTP located downstream of the dam. The WSD will be approximately 165 m high, and the design life is expected to be 1,000 years. Contact water will be acidic, with an expected pH in the range of 3.5 to 5.5. The WSD is designed to store mine contact water up to 63 Mm^3, which is the 1:200 year "wet year" level (Chapter 4, Project Description, Section 4.5.1.11.4). A spillway will be provided to safely discharge higher flows, up to the probable maximum flood.

33.12.2.2 Alternatives Identification

Two types of WSDs were investigated: an asphalt core rockfill dam (ACRD) and a roller compacted concrete (RCC) gravity dam (Appendices 4-AC and 33-E). An ACRD was recommended in 2011 for the site because asphalt (bitumen) has a very long life and is resistant to acidic environments. An RCC dam was selected for consideration as an alternative because this type of dam can provide the following benefits: a relatively rapid construction rate; a small footprint for site preparation; the spillway can be constructed on the dam rather than as a separate structure; and the dam has a relatively small quantity of concrete compared to rockfill and earthfill dams, which results in competitive cost despite the high unit rate for concrete. The cost estimates address only the major quantity items, and attributes common to both dams, such as the grout curtain, drainage facilities, and water delivery system to the WTP, are not included in this assessment.

33.12.2.2.1 Asphalt Core Rockfill Dam

Asphalt liners are used in industrial acidic water handling applications due to asphalt's acid resistance and long-term durability (Appendix 4-AC). Asphalt core placement is less disrupted by poor weather conditions than placement of clay and till cores. The plastic, self-healing nature of asphalt cores makes this type of dam resistant to leakage from settling of fill or earthquake deformation. The first asphalt core dam was built in Germany in 1961/1962. Since then, more than 150 ACRD dams have been built in many regions around the world but mostly in northern Europe, including several in Norway. Currently, two dams of similar height are under construction in China, and one is in the detailed design stage in Turkey.

Construction of the ACRD would require two years. The ACRD will comprise a central core of asphalt; two transition zones of gravel on each side of the core; central zones of non-acid reactive rock (non-reactive) in the upstream and downstream shells; and rockfill zones that could comprise reactive rock in the outer parts of the shells. Aggregate for the asphalt, as well as the transition fills, would be manufactured from non-reactive rock. A drain layer at the base of the downstream shell would also comprise non-reactive rock. A central zone of the foundation would be excavated to bedrock; beneath most of the shells, weak foundation soil would be removed but dense soil would be left in place. The steep rock slopes immediately above the creek would be excavated back to 45° in the central zone of the dam. The dam meets all stability requirements. Estimated deformation due to the Maximum Design Earthquake is less than 100 mm. Settlement of the dam over a 1,000 year life (extrapolated from case history data that only covers about 50 years) is 1,350 mm.

The estimated cost of the ACRD, using unit rates that reflect the use of large mining equipment, is $127 million. Use of the largest equipment planned for use at the mine would require that the permanent access road be in place. Using unit rates that might be expected for a large earthworks project, the total cost estimate is $235 million.

33.12.2.2.2 Roller Compacted Concrete Dam

RCC was developed in the 1980s, and there are now about 450 RCC dams worldwide. The cementitious content (cement, fly ash, and sometimes other pozzolanic materials) is relatively low compared to conventional concrete. RCC is spread and compacted using earthfill equipment. High placement rates can be achieved. The RCC is compacted in typically 300 mm thick layers. The upstream and downstream faces are formed. The upstream face is most often vertical; conventional concrete formwork is usually used. The downstream face is usually stepped; a variety of formwork has been used including conventional concrete formwork and concrete blocks. Several RCC dams of similar height have been constructed around the world (International Water Power & Dam Construction 2012).

Most RCC dams rely on an upstream face of conventional concrete, grout-enriched RCC (grout is added to RCC so the resulting mix can be vibrated as is conventional concrete), or unmodified RCC to provide an impervious barrier to water. However, many dams incorporate a geomembrane, most commonly polyvinyl chloride (PVC) on the upstream face to prevent seepage through the dam. PVC facings have been in place on dams since the early 1970s in the Italian Alps. The material formulation includes stabilizers to mitigate UV deterioration.

Construction of the RCC dam would require three years. The RCC dam will be constructed with earth-moving equipment. Conventional concrete would be placed on the upstream and downstream faces to provide freeze-thaw protection. An impervious PVC membrane, 2.5 mm thick, would be installed on the upstream face. Aggregate for RCC and conventional concrete would be manufactured from non-reactive rock. The dam is stable for all static and seismic load cases. The downstream slope, assumed to be 0.7H:1V for this conceptual layout, could likely be steepened somewhat and the RCC volume reduced accordingly.

Foundation preparation for the RCC dam includes excavation of all overburden beneath the dam footprint. The steep lower canyon slopes would be excavated to 45º for the full extent of the dam. The RCC dam will be resistant to overtopping by water, either due to avalanche-induced waves or during large floods. The dam could be lowered, compared to the ACRD. In addition, a stepped spillway can be constructed on the downstream face of the dam, at little incremental cost compared to the dam construction, eliminating the need for a separate spillway on the abutment. Large quantities of cement and fly ash would be required on an ongoing basis during RCC dam construction and a permanent access road would be required for delivery of these materials. The estimated cost of the RCC dam is $443 million. Constructing a spillway on the dam would result in $50 million savings compared to the ACRD with its separate spillway on the abutment.

Additional savings for the RCC dam would result from shorter diversion tunnel compared to that for the ACRD, especially if the dam crest is made narrower than 10 m. However, it is not expected that the total savings will be sufficient to result in a cost similar to that for the ACRD.

33.12.2.3 Comparison of Alternatives

Table 33.12-2 provides a summary of the relative advantages and disadvantages of the two WSD alternatives for the Project based on how their attributes align with the four performance objectives.

33.12.2.3.1 Technical and Project Economic Considerations

Relative benefits in terms of design issues, construction, cost, and risks are compared. Asphalt is resistant to acidic environments. The ACRD is cost effective, using local fill and mine equipment. In the case of the RCC dam, the long-term performance of the PVC in the acidic environment is unknown; allowance would be required for periodic replacement. The concrete is not resistant to acidic water. The RCC dam would take longer than the ACRD to construct.

In the case of the ACRD, the spillway will be a rock cut on the left abutment. It will be concrete-lined in the vicinity of the dam. The spillway channel will lead flows to discharge into the creek valley downstream of the seepage collection dam. It is currently envisaged that downstream of the dam vicinity, the spillway channel would comprise an excavation into rock. A concrete flip bucket will be required near the valley bottom to facilitate energy dissipation.

For the RCC dam, the spillway can be incorporated on the downstream slope. A stepped spillway would follow the downstream slope, and would use the steps that are commonly built into the dam to dissipate energy. The dam crest would be at the maximum reservoir level at the spillway section, and walls on the downstream slope would contain the flow. Some portion of the maximum design flow could be permitted to overtop the entire dam for a short time, rather than designing a spillway that can handle 100% of the design flood.

Estimated construction cost for the ACRD is $235.2 million. Estimated construction cost for the RCC dam is $442.9 million.

33.12.2.3.2 Environmental and Social Considerations

Leakage through and beneath the dam should be considered for all dams, and appropriate measures should be put in place to mitigate the risk. Leakage most commonly develops due to construction defects. For both the ACRD and RCC dam, leakage through the dam would be delivered via drains/galleries or the blanket drain to the seepage collection dam reservoir. The risk of seepage through the dam foundation is similar for the two alternatives. Both dams would include drainage adits in the abutments to intercept and monitor seepage. The adits and galleries also provide a means to grout the foundation, should it be found necessary in the future.

All leakage intercepted by the drainage system would be delivered to the seepage collection dam reservoir. The RCC dam would likely have greater labour requirements due to its longer construction time and periodic replacements.

33.12.2.4 Selected Alternative

The ACRD dam is the selected alternative due to the resistant nature of the asphalt core to acidic environments, the shorter construction time involved, and the lower construction and possibly lower operation costs involved. To facilitate using a technology that is novel to North America, European contractors who have experience and expertise on ACRD dams, could be hired into the Project contracting/construction teams.

33.12.3 Mine Contact Water Treatment

33.12.3.1 Purpose and Background

The objective of water treatment is to take dissolved phase elements and transform them into solid phases that can be removed from solution. This can be achieved through adsorption, co-precipitation, precipitation using chemical transformation, biological transformation, or phase partitioning.

Due to the geology of the Mine Site, mine contact water will contain concentrations of heavy metals and nutrients above recommended guidelines, as well as elevated pH. For the protection of aquatic organisms and habitat, as well as other dependent species and ecosystems (e.g., riparian ecosystems), this water must be treated before being discharged to the environment. Mine contact water exceeds BC Water Quality Guidelines (BC MOE n.d.) and other applicable regulations and legislation for multiple metals and other chemicals of concern.

The selected WTP must have sufficient capacity to treat 7.5 m^3/s during high flow periods. This capacity must be scalable, because treatment rates in late fall, winter, and early spring will be very low (0.10 to 0.25 m^3/s) due to low receiving environment stream flows. The large treatment flows at certain times of the year are dictated by the requirement to meet stringent discharge criteria downstream in Sulphurets Creek and down the Unuk River to the United States border and beyond.

33.12.3.2 Alternatives Identification

Two methods of water treatment were investigated: a low-density sludge (LDS) plant, and a high-density sludge (HDS) plant. These two alternatives are discussed and compared below.

33.12.3.2.1 Low-density Sludge Plant

A LDS plant is a WTP that uses lime neutralization to treat acid waters, which often contain metal ions such as zinc, manganese, copper, cadmium, lead, selenium, etc. (Kuyucak et al. 1999). Lime is used as either quicklime (CaO) or hydrated lime ($Ca(OH)_2$). Through the addition of lime to acidic mine contact water, the metal cations in solution react with the hydroxide ions in lime, and precipitate as metal hydroxides. This process is represented in the following reactions:

$$\text{Equation 1: } M^{++} + SO_4^{--} + Ca^{++} + 2(OH)^{-} + 2H_20 \rightarrow M(OH)_2 + CaSO_4 \bullet 2H_20$$

$$\text{Equation 2: } M^{+++} + 3(SO_4)^{--} + 3Ca^{++} + 6(OH)^{-} + 6H_20 \rightarrow 2M(OH)_3 + 3CaSO_4 \bullet 2H_20$$

As shown in equations 1 and 2, the products of these reactions are metal hydroxide precipitates and calcium sulphate (gypsum; SGS 2013). If the sulphate concentration of the waste water is high enough, there will be sufficient gypsum produced to exceed its solubility, and it will precipitate with the sludge.

Table 33.12-2. Summary Comparison of Water Storage Dam Alternatives

Alternative	ATTRIBUTE RATINGS AGAINST FOUR PERFORMANCE OBJECTIVES				OVERALL ALTERNATIVE RATING
	Environmental	Social	Technical	Project Economic	
Option A Asphalt Core Rockfill Dam	✓ ● Seepage and leakage risks are similar between options	x ● Shorter construction time and limited to no replacements may require lesser labour requirements, creating less jobs	✓ ● Asphalt is resistant to acidic environments; replacements may not be necessary ✓ ● Shorter construction time x ● Spillway constructed in adjacent bedrock x ● Proven technology but North American experience with asphalt core dams is very limited	✓ ● Order of magnitude construction cost is estimated at \$235.2 million, which is \$207.7 million less expensive than an RCC dam ✓ ● Operation costs may be lower because it is not necessary to replace dam core	**PREFERRED**
	Acceptable	Acceptable	Preferred	Preferred	
Option B Roller Compacted Concrete Dam	✓ ● Seepage and leakage risks are similar between options	✓ ● Longer construction time and periodic core replacements may require greater labour requirements, creating more jobs	x ● Long-term performance of the PVC in an acidic environment is unknown; allowance would be required for periodic replacement (25 to 50 years) x ● Longer construction time ✓ ● Spillway constructed in downstream slope of dam ✓ ● RCC dam is a well-understood technology	x ● Order of magnitude construction cost is estimated at \$442.9 million, which is \$207.7 million more expensive than an ACRD dam x ● Operation costs may be higher because of periodic dam replacements	**ACCEPTABLE**
	Acceptable	Preferred	Acceptable	Acceptable	

Notes:
✓ = advantage, x = disadvantage, ● = Preferred, ● = Acceptable, ● = Challenging, ● = Unfeasible; See Table 33.2-1 for attribute ranking specific definitions

A simple diagram of the LDS process is presented in Figure 33.12-3. Less complex variants of this process may involve no more than a lime silo, mixing tank, and settling pond. More complex variants of this process may involve the addition of neutralizing chemicals and flocculants to promote precipitation of treated chemicals (Kuyucak et al. 1999). Generally, in all LDS WTPs, no recycling of sludge occurs.

Generally, LDS systems are not costly to build but are inefficient (i.e., longer reaction times are required, and they produce a discharge with higher trace metal concentrations, if present). Although this technology is widely practiced, it generates LDS with 1% to 2% solids, and requires a large volume for settling/clarification ponds or clarifiers, and large areas for sludge disposal and storage. The amount of water recovered is low due to the large volume of LDS generated (Kuyucak et al. 1999). LDS systems are generally only suitable for relatively small flows or less complex acid mine drainage (Scousen, Hilton, and Faulkner n.d.).

33.12.3.2.2 High-density Sludge Plant

High-density sludge WTPs are built on LDS technology, but the key difference is that sludge is recycled to achieve higher effluent water quality (Figure 33.12-4).

The HDS process is normally run at a pH greater than 9.3, as most metals encountered will precipitate at or below this concentration of hydroxide ions. Oxidation of ferrous to ferric iron take place rapidly at this pH, with air being the most common oxidizing agent (Kuyucak et al. 1999). The final sludge waste ranges from 8% to 22% solids (Appendix 4-S). Because the resultant are granular and hydrophic in nature, attract heavy metals while repelling water, and settle rapidly and drain readily, high final solids contents (greater than 30%) can be achieved during disposal. This reduces the need for a large clarifier.

Generally, HDS systems are not costly to build and can be highly efficient. This technology is also scalable to accommodate a range of flow rates and high concentrations of metals of concern. HDS plants are highly efficient to operate, and staffing requirements are expected to be low.

33.12.3.3 Comparison of Alternatives

Table 33.12-3 provides a summary of the relative advantages and disadvantages of the two mine contact WTP alternatives for the Project based on how their attributes align with the four performance objectives.

33.12.3.3.1 Technical and Project Economic Considerations

Both the HDS and LDS plants would use well-established technologies, although LDS has a larger historical precedent. However, both technologies have been used effectively in real-world situations, including in BC.

The HDS plant would produce a higher density sludge (8 to 22% solids) than the LDS plant (less than 2% solids). This higher density translates to lower footprint requirements for sludge storage, as well as increased ease of disposal/storage. The HDS is also chemically and physically more stable.

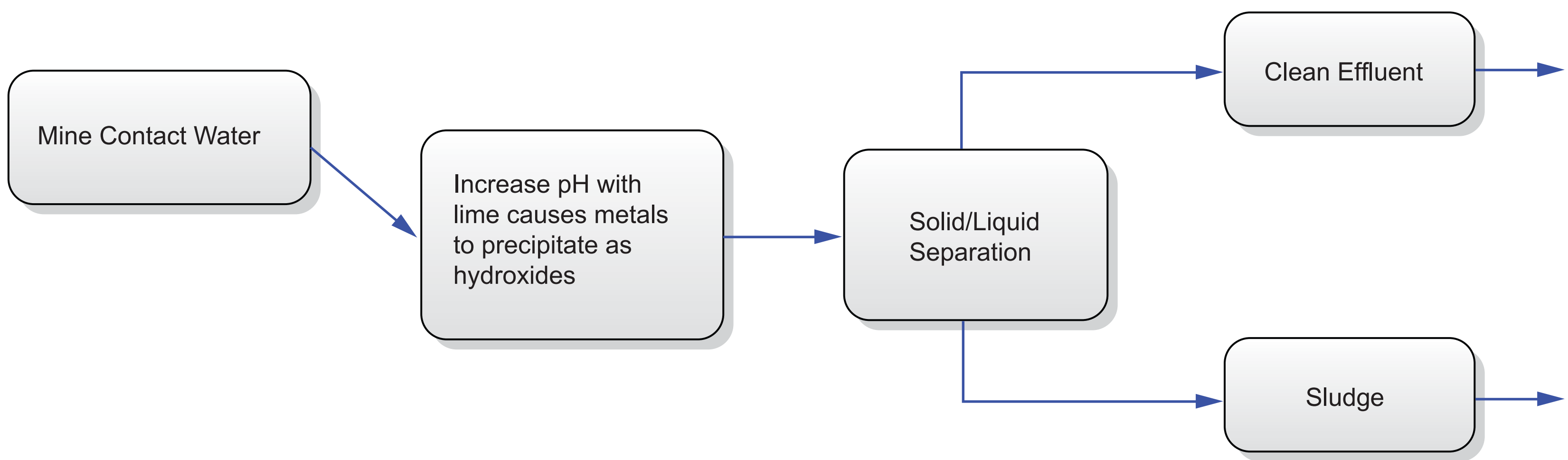


Figure 33.12-3

Low Density Sludge (LDS) Water Treatment Process

SEABRIDGE GOLD
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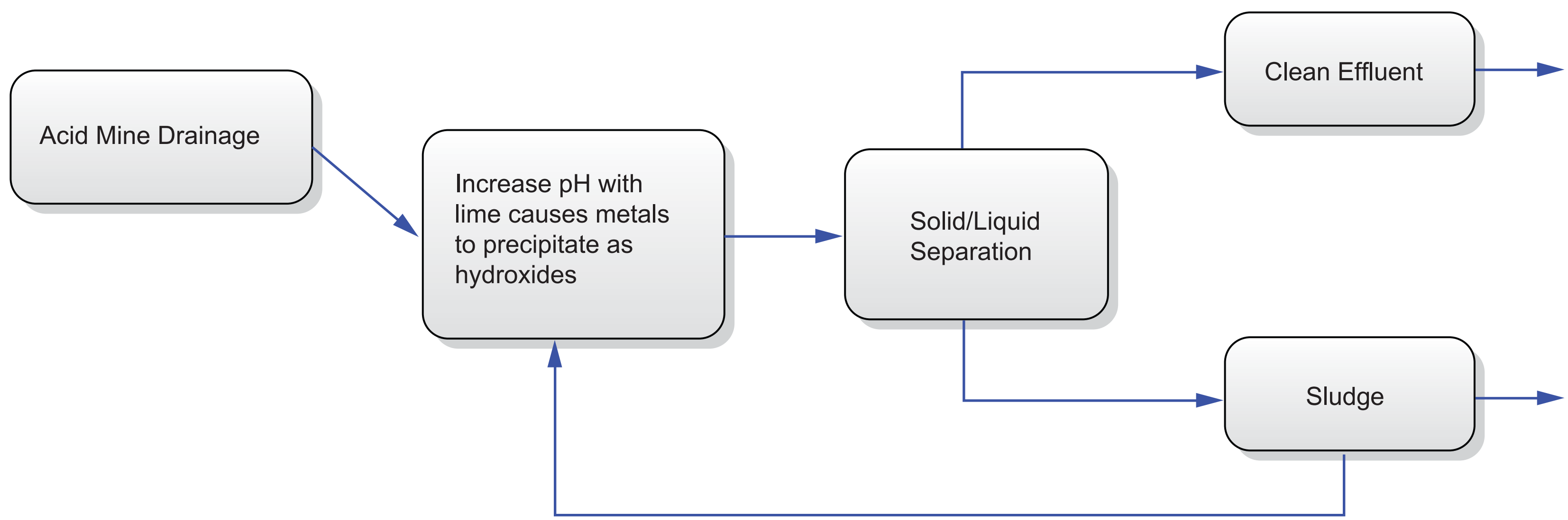
Table 33.12-3. Summary Comparison of Mine Contact Water Treatment Plant Alternatives

Alternative	Attributes and Ratings: Environmental	Social	Technical	Project Economic	OVERALL ALTERNATIVE RATING
Option A LDS Plant	x ● Higher overall footprint required x ● Much higher waste/sludge volumes x ● Effluent will still have poor water quality x ● Lime efficiency not possible	✓ ● No comparable attributes	✓ ● LDS is a well-known technology and has large historical precedent x ● LDS plants cannot process high volumes of waste water or chemically complex water x ● Sludge is chemically and physically less stable during its subsequent disposal/storage x ● Sludge particles are less granular and hydrophic in nature, tending to not attract heavy metals and repel water, and settle rapidly and drain readily. A large clarifier is required ✓ ● Good sludge viscosity	x ● Higher cost per unit volume of waste sludge pumping due lower waste density x ● Cost for treatment per unit volume of treated water (i.e., less water is recovered for the same quantity of lime consumed) is greater	**UNFEASIBLE**
	Unfeasible	Acceptable	Unfeasible	Acceptable	
Option B HDS Plant	✓ ● Lower overall footprint required ✓ ● Much lower waste/sludge volumes ✓ ● Produces much high quality effluent ✓ ● Lime efficiency increased due to recycling	x ● No comparable attributes	✓ ● HDS is a well-known technology and has been used in BC ✓ ● HDS plants can process high volumes of waste water ✓ ● Sludge is chemically and physically more stable during its subsequent disposal/storage ✓ ● Sludge particles are more granular and hydrophic in nature, tending to attract heavy metals and repel water, and settle rapidly and drain readily to achieve high solids content (> 30%) during disposal. Need for large clarifier is reduced x ● Sludge viscosity can be low ✓ ● Can be constructed as a series of independent compartmentalized units	✓ ● Lower cost per unit volume of waste sludge pumping due higher waste density ✓ ● Cost for treatment per unit volume of treated water (i.e., more water is recovered for the same quantity of lime consumed) is less	**PREFERRED**
	Preferred	Acceptable	Preferred	Preferred	

Notes:
✓ = advantage, x = disadvantage, ● = Preferred, ● = Acceptable, ● = Challenging, ● = Unfeasible; See Table 33.2-1 for attribute ranking specific definitions



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High Density Sludge (HDS) Water Treatment Process

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For HDS technology, the sludge particles are more granular and hydrophic in nature, and attract heavy metals while repelling water, and settle rapidly and drain readily to achieve high solids content (greater than 30%) during disposal. This reduces the need for a large clarifier. Because the particles resulting from LDS technology are less granular and hydrophobic in nature, they do not achieve the settling and drainage benefits of particles from the HDS process, and a large clarifier is required.

While estimated costs for both technologies have been developed, the HDS plant would be more cost efficient because of the greater process efficiencies, and the reduced pumping and storage requirements. The volume of lime required for both plants would be similar.

33.12.3.3.2 Environmental and Social Considerations

Environmentally, the LDS plant would require a much larger footprint due to the lower density sludge produced and the possible requirement of settling ponds. This large footprint makes an LDS plant challenging given the general lack of flat terrain in the Mine Site.

From an environmental perspective, the footprint for an HDS plant would be smaller than for a LDS plant primarily because the volume of waste material (sludge) produced would be significantly lower than for an LDS plant, as HDS solids concentrations are expected to be approximately 8 to 22%, while LDS solids concentration would be less than 5%. Further settling and concentration of sludge is possible using HDS technology, to achieve final solids concentrations of greater than 30%. The amount of water recovered is high due to the smaller volumes of HDS generated.

LDS technology is generally only applicable for low waste water volumes and chemically simple waste water. The high volumes of water treatment required, and the elevated concentrations for a wide range of chemicals, nutrients, and pH indicate that LDS technology is inappropriate for the Project.

33.12.3.4 Selected Alternative

The HDS plant was the selected alternative because of the reduced footprint required, the ability of HDS WTPs to treat high flow rates and chemically complex water, the greater settling ability of the sludge, and the increased chemical and physical stability of the sludge.

33.12.4 Selenium Treatment

The Proponent is committed to protecting water quality through the effective management and mitigation of water affected by the Project. Kerr Pit waste rock has been identified as a significant source of selenium, and contact water from this waste rock will require additional treatment, prior to treatment by the HDS WTP discussed in Section 33.13.3, to protect aquatic organisms and habitat, as well as other dependent species and ecosystems (e.g., riparian ecosystems).

Selenium is a naturally occurring metalloid often found in organic-rich sedimentary rocks in association with sulphide mineralization. At low levels it is micronutrient, in particular for ungulates such as mountain goats, where selenium deficiency can lead to physiological chronic performance issues and population declines (Flueck et al. 2012). Above certain threshold concentrations, selenium can have teratogenic effects (substances or agents that can interfere with normal embryonic development), particularly on egg laying species. The BC Water Quality Guidelines have limits of 2 µg/L of selenium in the receiving environment (BC MOE 2001a). Because the Mine Site geology is highly mineralized, natural background selenium concentrations within the Sulphurets Creek watershed have been found to exceed these guidelines (Chapter 14).

Selenium is more challenging than other constituents to treat in mine waste water because it is normally present in relatively dilute amounts (e.g., 50 to 100 µg/L), with associated high flows. Treating even the most easily treated elements at dilute amounts is challenging, and would require high residence times, or energy intensive systems.

Selenium is commonly found in four oxidation states in the environment:

- Se(II) - selenide, or Se^{--} forming compounds such as H_2Se;
- Se(0) – elemental selenium;
- Se(IV) - selenite, or SeO_3^{--}, $HSeO_3^-$, and H_2SeO_3;
- Se(VI) - selenate, or SeO_4^{--}, $HSeO_4^-$, and H_2SeO_4.

Se(IV) and organic selenium were effectively removed (greater than 85%) by the HDS process, while the concentration of Se(VI) was unaffected by this process. These results reflect that Se(IV) more readily sorbs to the iron oxyhydroxides produced during the HDS process and is therefore less mobile than Se(VI) (Martin et al. 2011). Se(VI) is often a problematic chemical in mine waste water because it absorbs weakly and is susceptible to displacement by other commonly occurring anions such as sulphate (Sobolewski 2005). Thus, Se(VI) tends to remain in mine waste water that has been treated with lime and flocculants, such as in the HDS plant.

33.12.4.1 Alternatives Identification

Multiple technologies exist to remove Se(VI) from mine contact water. These technologies include both active and passive treatment systems. Active treatment processes are generally those processes that include constructed basins, tanks, and mechanical and electrical equipment such as mixers and/or pumps, and often that include chemical feed equipment and instrumentation and controls. Passive treatment systems are generally considered to be those that rely less on mechanical and electrical equipment and automatic chemical feed, and more on naturally occurring processes. The distinction between these two types is not sharp, and many treatment systems for mine drainage represent a blend of both approaches. Active treatment methodologies include co-precipitation and adsorption (ion exchange), co-precipitation and adsorption (ferrous hydroxide treatment), co-precipitation and adsorption (ferrihyrite adsorption), zero-valent iron, and reverse osmosis. Passive treatment methodologies include treatment wetlands and biochemical reactors including anaerobic biochemical reactors, ABMet®, fluidized bed reactor, and BioSolve®. These treatment methodologies are discussed below. Methodologies involving the evaporation/crystallization of treatment water are also currently being investigated for selenium treatment in general (Sandy and DiSante 2010); however, these are not discussed here because the Project's climate would not be conducive to the effective application of these technologies.

Co-precipitation and Adsorption: Ion Exchange

Ion exchange technology involves an adsorption process in which undesirable ions (e.g., Se(VI)) are removed from solution through ion exchange with adsorbed anions on a solid matrix, called an anion exchange resin, placing more desirable ions into solution. The undesirable ions are then displaced off the exchange resin through back flushing, and are replaced with more benign anions such as chloride or hydroxide. Ion exchange technology concentrates the selenium into brine, and requires further treatment to precipitate the selenium. This process can be very efficient, with generally greater than 90% recovery rates (Sandy and DiSante 2010) and achievable effluent with selenium levels that are less than 1 µg/L, (Chapter 14, Appendix 4-V) but is susceptible to interference from competing ions in solution. This technology may also require high disposal costs of exhausted resins. As well, the brine will require further treatment to precipitate selenium. Ion exchange is a mature technology for other anions such as perchlorate and hexavalent chromium, and has also been demonstrated for industrial use for selenium.

Co-precipitation and Adsorption: Ferrous Hydroxide Treatment

Ferrous hydroxide treatment is a two-step reduction-oxidation (redox) and adsorption process where a ferrous iron salt is added to a continuously stirred reactor to reduce Se(VI) to Se(IV), which then co-precipitates/adsorbs onto the ferric hydroxide formed in the redox reaction. The iron hydroxides formed have a large surface area with a strong affinity for selenite, which forms a stronger surface complex with adsorption sites on the mineral surface than Se(VI). Ferrous hydroxide treatment is widely implemented at full-scale throughout the mining industry, and is relatively simple and low cost redox and physical adsorption technology (Sandy and DiSante 2010). At small scales this method can achieve effluent with selenium levels that are less than 0.020 mg/L and is a mature, patented technology. However, the treated selenium is susceptible to re-release from the solid matrix. The reduction reaction is not efficient, and this treatment method is only appropriate for small volumes of waste water. Due to the high flow volumes required for treatment for the Project, this technology is not appropriate for the Project and will not be discussed further.

Co-precipitation and Adsorption: Ferrihyrite Adsorption

Ferrihyrite adsorption is a pH-controlled co-precipitation and adsorption process in which a ferric iron salt is added to influent groundwater along with a coagulant to form ferrihydrite, which has a large surface area and a strong affinity for Se(IV). WTPs using this technology are typically designed to include a continuously stirred reactor, followed by clarification and setting. Se(IV) is removed with the precipitating solid. This technology has been established by the United States Environmental Protection Agency as the best demonstrated available technology for Se(IV) from treatment water (Sandy and DiSante 2010), it is not an effective technology for removing Se(VI) from mine water. As such, this technology is inappropriate for the Project and will not be discussed further.

Zero-valent Ion

Zero-valent ion (ZVI) technologies use iron rusting in water to create highly reducing conditions that reduce oxidized selenium species (e.g., Se(IV) and Se(VI)) to elemental selenium. Various treatment rates can be accommodated through the size of the iron particles used. These reactions occur rapidly and are less temperature dependent than biological processes. Ferric hydroxides formed during the rusting reaction also function as adsorbents for selenium species flowing through the treatment system. This technology can achieve effluent with selenium levels that are less than 0.005 mg/L. This technology has not progressed beyond pilot scale projects. There is the potential for long residence times, and the spent ZVI must be removed, disposed of, and replaced (Sandy and DiSante 2010). As well, under certain conditions dissolved oxygen and other oxyanions can oxidize the ZVI.

Reverse Osmosis

The reverse osmosis technology involves a high pressure membrane separation process that forces a solution through a membrane that is capable of excluding larger anions and cations such as Se(VI). The resultant effluent water has lower total dissolved solids concentration depending on the efficiency of the membrane system. Reverse osmosis technology concentrates the selenium into brine, and will require further treatment to precipitate the selenium. Reverse osmosis systems typically have an efficiency of 80 to 90% depending on the quality of the water entering the system. Membranes are susceptible to scaling from the precipitation of other ions in the system. This technology can achieve effluent with selenium levels that are less than 1 µg/L, and has been used in full-scale production in metals mine environments.

Treatment Wetlands

Treatment wetland technologies involve constructed wetlands with both marsh-type vegetation and open water. They are designed to use soils and vegetation to transform and fix selenium in sediments, although small amounts are also taken up in plant materials and/or volatilized. Up to 90% of the selenium can be removed, although this is dependent on the speciation of selenium. This technology can operate with minimal operator supervision, and operates passively without energy or chemicals (Sandy and DiSante 2010). This technology can achieve effluent with selenium levels that are less than 0.01 mg/L, although there is potential to achieve lower concentrations. Treatment wetlands are a mature technology, and are in full-scale use for petroleum hydrocarbon waste waters, although the Proponent is not aware of its use in metal mines. This technology would require a large and flat footprint for wetland construction (Sandy and DiSante 2010), and because the Mine Site is located in steep and rugged terrain, sufficient space is not available. As well, wetlands in the Project area would be frozen in winter and water treatment would, by necessity, only occur during the summer months. As such, this technology is inappropriate for the Project and will not be discussed further.

Biochemical Reactors: Anaerobic Biochemical Reactors

Anaerobic biochemical reactor (ABR) technology contacts a waste-water-containing dissolved metals/metalloids with a bioavailable carbon-based material. Biological and physical-chemical reactions occur in the ABR's solid substrate bed in which anaerobic and facultative heterotrophic bacteria are present. This technology can achieve effluent with selenium levels that are less than 0.005 mg/L, although there is potential to achieve lower concentrations. ABR technology is mature for other ions, although it is at the pilot scale stage for selenium. This technology requires a large footprint for the Project flow rates due to high minimum hydraulic residence time requirements. Because the Mine Site is located in steep and rugged terrain, sufficient space is not available. As such, this technology is inappropriate for the Project and will not be discussed further.

Biochemical Reactors: ABMet®

ABMet® technology uses an attached growth, down-flow, bed bioreactor using a granular activated carbon filter to support a microbial biofilm typically colonized by indigenous microbes from the influent water. Molasses is typically used as a carbon source for microbes. The biologically reduced elemental selenium that ABMet® technology produces is insoluble and is an integral part of the biological solids. This technology can achieve effluent with selenium levels that are less than 0.010 mg/L. ABMet® is full-scale for coal-fired power plants, and is being pilot tested at several metals mine sites. This technology requires a large footprint for the Project flow rates due to the low hydraulic loading rate requirements and high minimum hydraulic residence time requirements. Because the Mine Site is located in steep and rugged terrain, sufficient space is not available. As such, this technology is inappropriate for the Project and will not be discussed further.

Biochemical Reactors: Fluidized Bed Reactor

Fluidized bed reactor (FBR) technology involves influent water passing through a granular media composed of sand or granular activated carbon at high velocity to suspend or fluidize the media. This creates a mixing environment that increases the interaction between the biofilm-coated granular matrix and the contaminant, reducing the hydraulic retention time. Systems using FBR technology are typically inoculated with indigenous organisms in influent water. This technology can achieve effluent with selenium levels that are less than 0.010 mg/L, although there is potential to achieve lower concentrations. FBR technology is currently undergoing pilot-scale studies. This technology requires a large footprint for the Project flow rates due to the low hydraulic loading rate requirements and high minimum hydraulic residence time requirements. Because the Mine Site is located in steep and rugged terrain, sufficient space is not available. As such, this technology is inappropriate for the Project and will not be discussed further.

Biochemical Reactors: BioSolve®

BioSolve® technology is a fluidized bed reactor using sponge media to increase surface area and to promote the formation of biofilm. BioSolve® technology uses methanol as a carbon source to reduce biomass production, because fewer organisms are able to use methanol as a sole carbon source. Systems using BioSolve® technology are typically inoculated with indigenous organisms in influent water. This technology can achieve effluent with selenium levels that are less than 0.010 mg/L, although there is potential to achieve lower concentrations. BioSolve® technology is currently undergoing pilot-scale studies. This technology requires a large footprint for the Project flow rates due to the low hydraulic loading rate requirements and high minimum hydraulic residence time requirements. Because the Mine Site is located in steep and rugged terrain, sufficient space is not available. As such, this technology is inappropriate for the Project and will not be discussed further.

33.12.4.2 Comparison of Alternatives

Table 33.12-4 provides a summary of the relative advantages and disadvantages of Se(VI) water treatment alternatives for the Project based on how their attributes align with the four performance objectives.

Table 33.12-4. Summary Comparison Selenium, Se(VI), Treatment Technology Alternatives

Alternative	ATTRIBUTE RATINGS AGAINST FOUR PERFORMANCE OBJECTIVES				OVERALL ALTERNATIVE RATING
	Environmental	Social	Technical	Project Economic	
Option A Co-precipitate and adsorption: ion exchange	✓ Achievable effluent of selenium < 1 µg/L ✓ Relatively small footprint x Exhausted membranes must be removed, disposed of, and replaced	✓ No comparable attributes	x Concentrates selenium to a brine, and will require further treatment to precipitate ✓ Process is very efficient, with generally > 90% recovery rates x Resins are susceptible to interference from competing ions in solution x Mature technology for other anions such as perchlorate and hexavalent chromium, but is only in pilot scale stage for selenium ✓ Relatively short residence times (compared with other available technology)	x Membrane disposal may carry significant cost x Additional costs for brine treatment and sludge disposal ✓ Capital cost approximately $15 M x Annual operating costs estimated at $6-7 M[1] ✓ Potentially lower costs from reduction of required hydraulic capacity due to concentration and recycling from regeneration, and modularity of the process[2]	**PREFERRED**
	Preferred	Acceptable	Acceptable	Acceptable	
Option B Zero-Valent Ion	✓ Achievable effluent of selenium < 1 µg/L x Long residence times relate to large footprint requirements x Zero-valent ions must be removed, disposed of, and replaced	✓ No comparable attributes	✓ Can accommodate various treatment rates through the size of iron particles used ✓ Reactions occur rapidly and are relatively insensitive to temperature ✓ Ferric hydroxides formed during the rusting reaction also function as adsorbents for Se species flowing through the treatment system. x Long residence times required x Dissolved oxygen and other oxyanions can oxidize the zero-valent ions x Technology is at pilot scale only x May not be effective for higher selenium	x Sludge disposal may carry significant cost ✓ Capital cost approximately $13 to $16 M[3] ✓ Annual operating cost approximately $3 to $4 M[3]	**CHALLENGING**
	Challenging	Acceptable	Challenging	Preferred	
Option C Reverse Osmosis	✓ Achievable effluent of selenium < 1 µg/L ✓ Relatively small footprint x Exhausted membranes must be removed, disposed of, and replaced	✓ No comparable attributes	x Concentrates selenium to a brine, and will require further treatment to precipitate x Membranes are susceptible to scaling from the precipitation of other ions in the system x Operating issues will result from viscosity changes at extreme low temperatures ✓ Mature technology that has been used in full-scale production in metals mine environments	x Membrane disposal may carry significant cost x Additional costs for brine treatment and sludge disposal x Capital cost approximately $55 M[3] ✓ Annual operating cost approximately $3 to $4 M[3]	**CHALLENGING**
	Preferred	Acceptable	Challenging	Challenging	

Notes:
✓ = advantage, x = disadvantage, ● = Preferred, ● = Acceptable, ● = Challenging, ● = Unfeasible; See Table 33.2-1 for attribute ranking specific definitions
[1.] Based on treatment cost of 0.5 cents per litre treated and 60 L/s treatment rate
[2.] From BioteQ (2013); Appendix 14-V
[3.] Estimated costs derived from Sandy and DiSante (2010) and are based on an assumed 60 L/s treatment rate

<u>Technical and Project Economic</u>

The primary technical concerns among the three short-listed selenium treatment options are the requirement for further treatment to precipitate the selenium, the suitability of the technology for the Project terrain and climate, and the maturity of the technology.

ZVI is the only treatment option that would require no further treatment to precipitate the selenium into a sludge that can then be disposed of. Both ion exchange and reverse osmosis technologies concentrate the selenium into a brine that would require further treatment.

Although ZVI technology can scale to accommodate higher selenium with smaller sized iron particles, the technology still requires long residence times. For the high treatment volume required for the Project (7.5 m^3/s), these long residence times can equate to large footprint requirements, which may not be possible to accommodate given the challenging terrain of the Mine Site. Both ion exchange and reverse osmosis technologies require relatively small footprints, even at high treatment flow volumes.

ZVI technology also may pose operating issues resulting from viscosity changes at extreme low temperatures; a problem that may prevent this technology from being effective during the winter. Ion exchange and reverse osmosis technologies are not known to have this issue.

Reverse osmosis is the only short-listed technology that has been used in full-scale production in metals mine environments. Reverse osmosis is currently being used to treat water from 21 ha of leach pads at the Barrick Richmond Hill Mine (Sobolewski 2005) . The reverse osmosis system used treats 12.5 L/s with influent selenium of 12 to 22 µg/L to an effluent of about 2 µg/L. This technology is also used at a historic gold mine site in California to treat impounded water for reduction of selenium from approximately 60 µg/L to less than 5 µg/L (Golder 2009). This system experienced reduced recovery rates due to the high total dissolved solids of the influent.

ZVI technology has been shown to be effective at treating low levels of selenium in mining influenced water during pilot-scale studies (Golder 2009). Influent selenium ranged from 5 µg/L to 17 µg/L, and effluent selenium were consistently below 5 µg/L. Residence times were generally very long. A very small pilot-scale study investigated the viability of reacting stripped sour water at an oil refinery through columns of ZVI (iron filings). Influent selenium ranged from 250 to 500 µg/L and could be reduced by 79% (Shamas, Wagener, and Cooke 2009). Although effective, this process also had extended reaction times and pH sensitivities (Davis et al. 2009).

BioteQ conducted a study of ion exchange selenium treatment for Project mine waste water (BioteQ 2013 in Appendix 4-V). Their Selen-IX treatment process removed water with influent selenium of almost 100 µg/L to effluent selenium of less than 1 µg/L. This process was also highly specific to Se(VI). This process was found not to be susceptible to interference by other ions, and membrane refresh rates are high thereby minimizing the replacement rates for membranes.

As shown in Table 33.12-4, the capital cost estimates for ion exchange (Option A) and ZVI (Option B) technologies are about the same, while both are estimated to be lower than that of reverse osmosis (Option C). However, operating costs of ion exchange treatment are anticipated to be higher than that of the other two options. Although ion exchange is likely to be more expensive to operate, this is a conservative cost estimate which is likely be lower in practice; the high estimate is considered economically viable for the Project.

Environmental and Social

Environmental considerations for selenium treatment largely surround the ability of the technology to meet effluent standards, and the footprint requirements of the selenium WTP. While either at the pilot stage or at full maturity, all technologies can achieve selenium effluent concentrations of less than 1 µg/L. However, no technology has the ability to meet this effluent standard at Project flow rates of 7.5 m^3/s. Thus, the Proponent proposes to segregate the most selenium-enriched mine contact waste water and focus selenium treatment only on that water using a treatment flow rate in the plant of 0.06 m^3/s (60 L/s).

33.12.4.3 Selected Alternative

The selected selenium treatment technology is Option A—involving co-precipitation and adsorption with ion exchange—as shown in Table 33.12-4. This is the only technically feasible option for selenium treatment for the Project primarily due to the high water flow volumes required for treatment, and the terrain and climatic constraints at the Project Mine Site, combined with the high removal efficiency rates achieved by this technology.

33.13 Ore Comminution

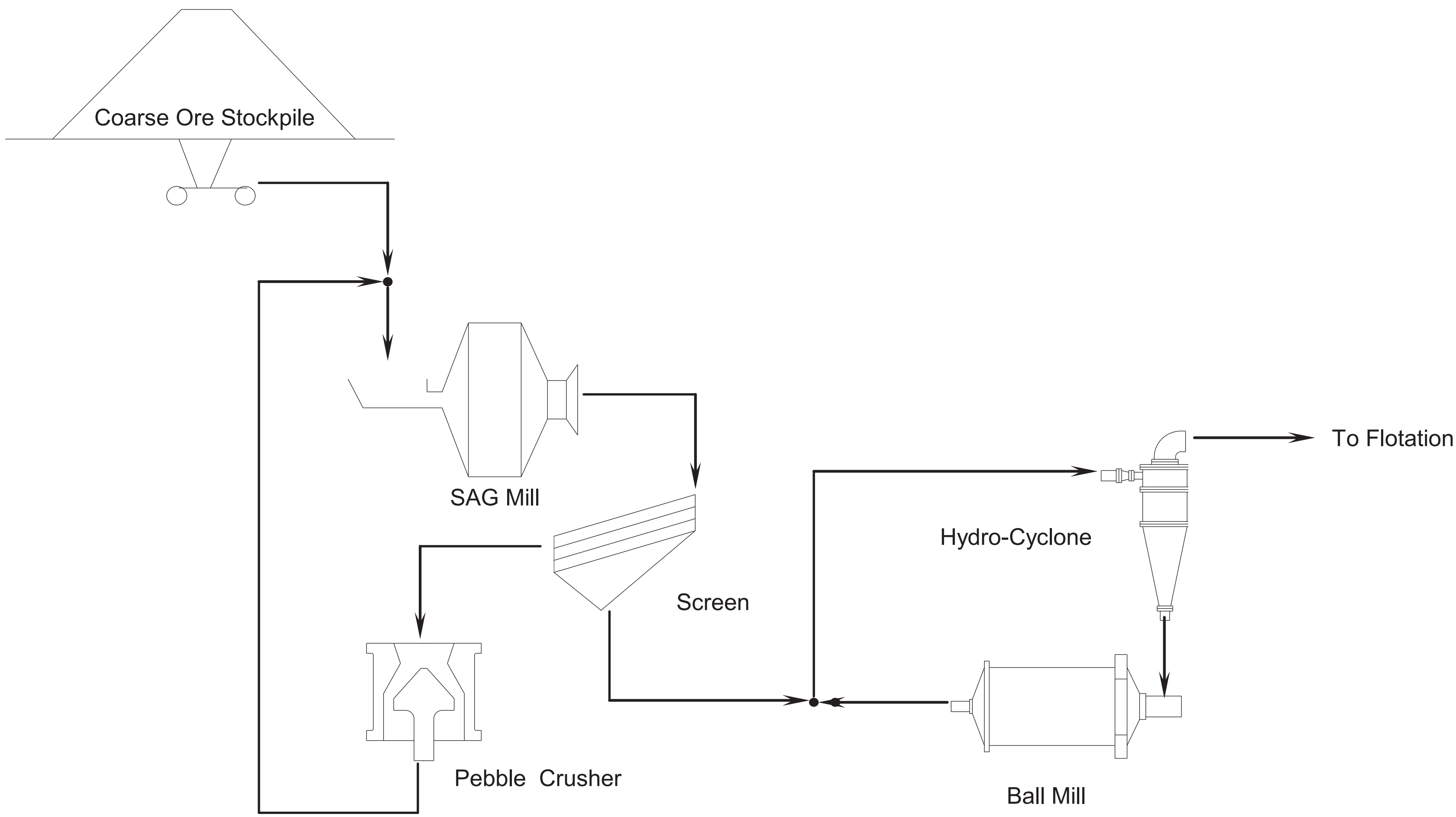
33.13.1 Purpose and Background

Ore comminution at hard rock mines involves the breaking down and pulverizing of ore to prepare it for treatment processes to recover precious metals. The grinding throughput for the Project will be 130,000 tpd at an availability of 94%, or an annual throughput of 47,450,000 tpa, for the duration of the operation phase. Grinding is required for ore comminution and, because of the large volume of ore involved in the Project, will require large inputs of energy, so options for this process were investigated for the Project.

33.13.2 Alternative Identification

Two ore comminution methodologies were investigated for the KSM Project: semi-autogenous grinding (SAG) mill-ball mill-pebble crushing (SABC) and crushing using secondary crushers and high pressure grinding rolls (HPGR) crushers followed by ball mill grinding (Wardrop 2012b; Appendix 33-D). Test work results, combined with industry experience, indicated that KSM Project mineralization is amenable to either method (Wardrop 2012b).

Autogenous mills are so-called because of the self-grinding of the ore. A rotating drum throws larger rocks of ore in a cascading motion, which causes impact breakage of larger rocks and compressive grinding of finer particles. SAG mills function similarly to autogenous mills, except with the addition of grinding balls to aid in grinding. The SABC mill involves a SAG mill plus a ball mill, in which rock pebbles create friction and attrition between the rock pebbles and the ore, and a pebble crusher (Figure 33.13-1). Rock pebbles are typically made of quartz or silica when product contamination by iron from steel balls must be avoided.



Simplified Flowsheet: Semi-autogenous Grinding Ball Mills Circuit Option

Figure 33.13-1

The proposed equipment used for the comminution circuit with SAG grinding mills will be composed of the following items at the Treaty Ore Processing Complex:

- two SAG mills, each of 12.2 m diameter and 6.7 m length (effective grinding length), installed power of 22,500 kW each;
- four ball mills, each of 7.6 m diameter and 11.9 m length, installed power of 14,000 kW each;
- four pebble crushers each with 750 kW installed power; and
- associated conveyors, feeders, screens, and pumps.

The estimated annual energy consumption for the SABC mill is presented in Table 33.13-1.

Table 33.13-1. Estimated Annual Energy Consumption of the Semi-autogenous Grinding Mill

Equipment Description	Annual Energy Consumption (MWh)
Two SAG mills	347,250
Four pebble crushers	15,671
Four ball mills	364,875
Screens, conveyers, pumps and others	92,946
Total	**820,742**

Source: Wardrop (2012b)

33.13.2.1 High Pressure Grinding Rolls

The HPGRs consist of two rollers with the same dimensions, which are rotating against each other with the same circumferential speed. The special feeding of bulk material through a hopper leads to a material bed between the two rollers. The bearing units of one roller can move linearly, and they are pressed against the material bed by springs or hydraulic cylinders. The pressures in the material bed are greater than 50 MPa. In general, they achieve 100 to 300 MPa. By this, the material bed is compacted to a solid volume portion of more than 80%.

The roller press has a certain similarity to roller crushers and roller presses for the compacting of powders, but the purpose, construction, and operation mode are different.

Extreme pressure causes the particles inside of the compacted material bed to fracture into finer particles and also causes microfracturing at the grain-size level. Compared to ball mills, HPGRs achieve 30 to 50% lower specific energy consumption, although they are not as common since they are a newer technology.

The comminution circuit with HPGR grinding rolls will consist of the following major equipment at the Treaty Ore Processing Complex (Figure 33.13-2):

- five cone crushers with four in operation and one in standby mode, each equipped with a 750 kW motor;
- four 24/17 model HPGR units, each with 5.8 MW installed power;

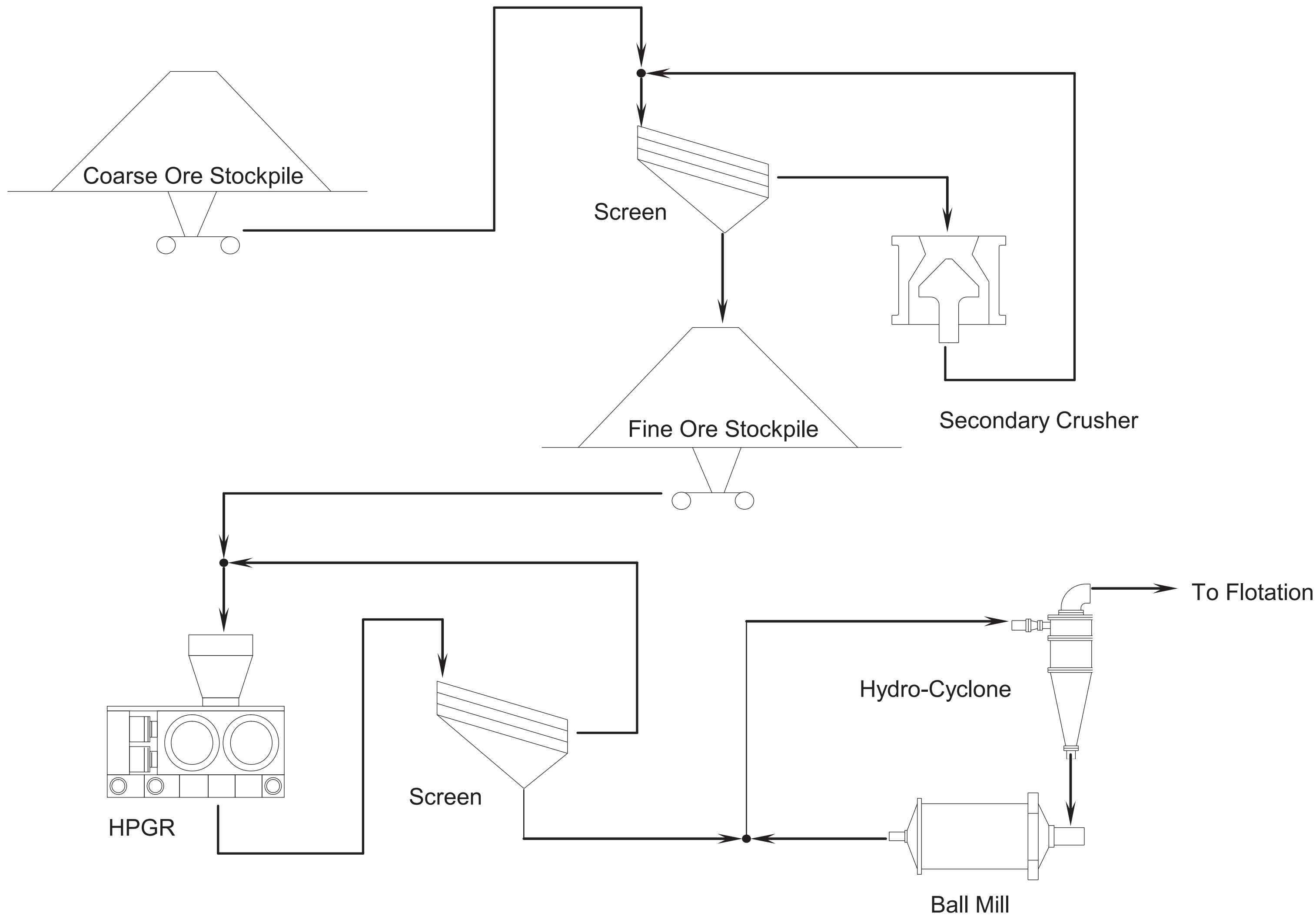


Figure 33.13-2

Simplified Flowsheet:
High Pressure Grinding Rolls Option

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- four ball mills, each with a 7.6 m diameter and 11.9 m in length, installed power 14,000 kW each; and
- associated conveyors, feeders, screens, pumps, and tanks.

The estimated annual energy consumption for the HPGR mill is presented in Table 33.13-2.

Table 33.13-2. Estimated Annual Energy Consumption of High Pressure Grinding Rolls Mill

Equipment Description	Annual Energy Consumption (MWh)
Four cone crushers	19,341
Four HPGR	147,588
Four ball mills	355,871
Screens, conveyers, pumps, and others	150,720
Total	**673,521**

Source: Wardrop (2012b)

33.13.3 Comparison of Alternatives

Table 33.13-3 provides a summary of the relative advantages and disadvantages of the two comminution alternatives for the Project based on how their attributes align with the four performance objectives.

33.13.3.1 Technical and Project Economic Considerations

The conventional SABC circuit has been widely used in various mineral process plants due to relatively inexpensive capital costs and large capacity. It is a proven circuit that is familiar to metallurgists and operators. The SABC grinding process, however, is relatively energy inefficient. Energy is mainly lost to the environment in the forms of heat and sound (Wardrop 2012b).

The development of the HPGR circuit has provided a more energy-efficient alternative to the SABC circuit. There are several significant benefits for using HPGR in the mining industry, including:

- significant energy cost savings;
- reduced grinding media consumption and operating costs;
- improved equipment delivery schedule compared to SAG mills; and
- potential benefits for downstream mineral recovery.

Capital and operating cost comparisons are presented in Tables 33.13-4 and 33.13-5.

The higher roller surface wear rate in HPGR initially deterred the mining industry from making use of HPGR technology. However, due to significant improvement in wear protection technologies in recent years, HPGR circuits have become more attractive for the comminution of hard ores (Wardrop 2012b).

Table 33.13-3. Summary Comparison of Ore Grinding Alternatives

Alternative	ATTRIBUTE RATINGS AGAINST FOUR PERFORMANCE OBJECTIVES				OVERALL ALTERNATIVE RATING
	Environmental	Social	Technical	Project Economic	
SABC Mill	x ● Annual energy consumption is approximately 821 GWh/a, approximately147 GWh/a more than the HPGR mill option ✓ ● Smaller physical footprint required	x ● Potential creation of relatively less jobs	✓ ● SABC circuit is a mature process and is more widely employed in the mining industry ✓ ● Precedent exists for large capacity operations operating in cold and wet climates	✓ ● Capital cost is $141 million less than for HPGR x ● Incremental electricity cost per year is $14.1 million	ACCEPTABLE
	Acceptable	Acceptable	Preferred	Acceptable	
HPGR Mill	✓ ● Annual energy consumption is approximately 673 GWh/a, approximately 147 GWh/a less than the SAG mill option x ● Larger physical footprint required	✓ ● Potential creation of relatively more jobs	x ● Relatively new technology x ● Potential crushing problems with wet/frozen ores x ● Potential wear issues ✓ ● Potential metallurgical benefits due to preferential mineral liberation	x ● Incremental capital cost over SAG mill option is $141 million ✓ ● Electricity cost savings per year is $14.1 million (simple payback in 10 years)	PREFERRED
	Preferred	Preferred	Acceptable	Preferred	

Notes:
✓ = advantage, x = disadvantage, ● = Preferred, ● = Acceptable, ● = Challenging, ● = Unfeasible; See Table 33.2-1 for attribute ranking specific definitions

Table 33.13-4. Capital Cost Summary Comparison

Description	Capital Cost (thousand CAN$)		
	SABC Option	**HPGR Option**	**HPGR vs. SABC**
Direct Works			
Plant Site	4,856	12,873	8,017
Crushing & Grinding	406,060	494,428	88,368
Other	9,685	12,565	2,880
Sub-total	**420,601**	**519,866**	**99,265**
Indirect Works			
Project Indirects	176,067	200,827	24,760
Contingency	82,244	99,138	16,894
Sub-total	**258,311**	**299,965**	**41,654**
Total Capital Costs	**678,912**	**819,831**	**140,919**

Source: Wardrop (2012b)

Table 33.13-5. Comminution Circuit Operating Cost Comparison

Description	Unit Cost (CAN $/t Milled)		
	SABC Option	**HPGR Option**	**HPGR vs. SABC**
Personnel			
Sub-total	0.109	0.126	**0.017**
Supplies			
Operating Supplies	0.003	0.003	0
Maintenance Supplies	0.245	0.262	0.017
Major Consumables	1.691	1.258	-0.433
Power Supply	0.934	0.696	-0.238
Sub-total	**2.873**	**2.218**	**-0.654**
Total Operating Costs	**2.981**	**2.344**	**-0.637**

Source: Wardrop (2012b)

Considerable time and resources have been spent in an effort to reduce wear rate, resulting in a new generation of superior rolls in the current market. These rolls use studs, segments, edge protection inserts, and advanced materials to reduce the surface wearing rate.

Mining companies are now beginning to incorporate HPGR technology in ore processes. Freeport-McMoRan Copper & Gold Inc. has introduced HPGR at the Cerro Verde Copper Mine in Peru. The comminution circuit at Cerro Verde uses four HPGR units processing 2,500 t/h per unit. The mill has a design capacity of 108,000 tpd. The Newmont Mining Corp. Boddington Gold Project in Australia has also included HPGR technology in their comminution circuit.

The HPGR option may result in a lower Bond work index number due to micro-fracturing created during HPGR crushing. The particle reduction by HPGR is caused by the compressive force applied to the ore when it passes through the HPGR. The resulting high pressure causes micro-cracks at the weak interfaces in the ore. These weak interfaces normally occur around the grain boundaries between different minerals. This creates benefits in the form of preferential liberation between target minerals and gangue (commercially worthless material) minerals.

33.13.3.2 Environmental and Social Considerations

The HPGR mill option will use 147 GWh per year of power less than the SABC mill option (Table 33.13-6). The SABC mill will use smaller equipment and will require a smaller footprint.

Table 33.13-6. Total Electrical Energy Cost for Mill-ball Mill-pebble Crushing and High Pressure Grinding Rolls Mill Options

Description	Annual Electrical Power Consumption (kWh)	Annual Electrical Energy Cost (thousand CAN$)
Comminution Circuit Energy Cost		
SABC Option	820,742,674	44,320
HPGR Option	673,521,008	33,002
Savings Subtotal	**147,221,666**	**11,318**
Remaining Operation Energy Cost (Flotation/Leaching/Mining/Other)		
SABC Option	528,928,007	28,562
HPGR Option	528,928,007	25,917
Saving Subtotal	**-**	**2,645**
Savings, Total	**147,221,666**	**13,963**

Source: Wardrop (2012b)

Due to the potential wear issues on the HPGR machines, this mill may potentially require more intensive maintenance, which would correspond to a larger potential workforce.

33.13.4 Selected Alternative

The HPGR mill is the selected alternative due to the significant energy and operating cost savings.

33.14 Closure and Reclamation

Once the Project has concluded mining activities during the operation phase (51.5 years), it will require closure, including decommissioning and reclamation of the various sites and infrastructure. Closure and post-closure phases for the Project are anticipated to last about 3 and 250 years, respectively. As required under the BC *Mines Act* (1996a) and its accompanying Health, Safety and Reclamation Code for Mines in British Columbia (BC MEMPR 2008), detailed mine and reclamation plans and specifications will be submitted later as part of the Mine Plan and Reclamation Program Permit application.

Amenity to reclamation and adverse effects mitigation has been considered as an integral part of the alternatives assessments throughout this document for all phases of the Project. High-level closure and reclamation plans for the Project are outlined in the recent prefeasibility study (Appendix 4-C) for both the Mine Site and PTMA, and in Chapter 27 of this Application/EIS. The development of these plans was mostly based on complying with the Health, Safety and Reclamation Code for Mines in British Columbia (BC MEMPR 2008) and BMPs such as progressive reclamation. As well, various approaches have been developed to return the site to wildlife habitat, as described below

33.14.1 Tailing Management Facility

Upon mine closure, tailing ponds are generally left as open water facilities. For the Project, Seabridge has adopted an alternative to use tailing to create more beach area, including creating beaches over much of the pond, resulting in less open water.

An extra addition to common practice that the Proponent will undertake is to spread till on and re-vegetate the beaches, as well as plant submerged vegetation to promote wetlands. These areas will provide foraging habitat for wetland-dependent species such as moose.

To help restore pre-mine flows to Teigen and Treaty creeks, the Proponent will invest in changing diversion structures upon closure that route the TMF pond water discharge direction. When the TMF pond water meets water quality guidelines, water will be routed either to Teigen or Treaty Creek by adjusting the elevation of inlet weirs or control gates at the spillways. This system will allow for spillway maintenance by temporarily routing water to either spillway. It is anticipated that much of flow will go to North Treaty Creek and a smaller portion used to maintain South Teigen Creek flows, depending on hydrological factors and fisheries requirements.

33.14.2 Rock Storage Facilities

The reclamation of RSFs consists of re-sloping to ensure long-term stability and to allow for re-vegetation. Slope stability is a function of factors such as foundation and waste rock sheer strength, which varies from site to site, and so the hazard and risks associated with RSF slopes, and the slope angles that RSFs should meet, are determined on an individual basis.

For the Project, Seabridge has opted to have slopes re-sloped to 26º so that the final slope angle meets reclamation criteria provided in the Interim Guidelines of the British Columbia Mines Waste Rock Pile Research Committee (1991). Till will be placed on the re-sloped areas to minimize water percolation and contact water volumes, and the area will be re-vegetated to provide for wildlife habitat. Depth of cover on reclaimed slopes at mines is typically 10 to 20 cm, which the Proponent considers insufficient to help reliably establish vegetation. Therefore, the Proponent has opted to place cover to a depth of 50 cm in order to provide a more suitable foundation for vegetation to establish. The slopes above 1,100 masl on the Mitchell RSF will be left at the angle of repose and will serve as escape terrain for mountain goats.

33.14.3 Coulter Creek Access Road

Often, access roads to mines remain open on closure to facilitate monitoring, water treatment, and other post-closure activities. In order to maximize restoration of the Project, Seabridge has opted to close the CCAR and restore it to wildlife habitat use in the post-closure phase, as described in Chapter 27, Closure and Reclamation. Only the TCAR will remain open to provide access to the PTMA, while the MTT will enable vehicle access to the Mine Site from the PTMA.

33.15 Employee Work Schedules

There were no major alternatives considered for employee work schedules for either construction or operation, which are yet to be finalized for the Project. The Project will operate 24 hours per day, 365 days per year. The Proponent is developing work schedules designed to maximize productivity, ensuring a safe work environment, and maximizing time for workers with their families. It is assumed that any changes made to schedules will also be environmentally neutral.

A variety of rotational schedules will be looked at by the Proponent including two weeks on and two weeks off, and three weeks on and one week off for most mine personnel, with different schedule options for management and administrative support. Employee schedules will be finalized as the Project progresses.

33.16 Employee Living Conditions

The Project will require residences for personnel during the construction, operation, and closure phases, because the Project is too remote for personnel to commute. Camp locations for the Project were selected based on minimizing distances, ease of access, site safety and suitability, and economics, as well as environmental factors. No major alternative assessments were conducted to select camp locations for the Project.

It is estimated that the Project will require 12 construction camps (Table 33.16-1) containing 40 to 700 people, including a camp for the temporary access road construction (to be relocated to Camp 11 upon completion). Some of the early camps, with access via the winter access road or helicopter, will be temporary foldaway or similar camp types for ease of transport. During operation (51.5 years), two camps will be required: a 250-person camp in the PTMA and a 350-person camp in the Mine Site. More information on Project camps can be found in Chapter 4.

Table 33.16-1. Employee Camps—Construction Phase

Camp #	Location	Capacity (person)
1	Granduc Staging	80
2	Ted Morris	50
3	Eskay Staging	50
4	Mitchell North	125
5	Treaty Plant	700
6	Treaty Saddle	120
7	Unuk North	40
8	Unuk South	40
9	Mitchell Initial	140
10	Mitchell Secondary	400
11	Treaty Marshalling Yard	60
12	Highway 37 Construction	60

Source: Wardrop (2012a)

33.17 Power Supply

Both the Mine Site and PTMA will require power sources to provide energy for mining activities. Design of the Project considered suitable sources of power, as well as the location of transmission lines. Power generation and transmission utilities in the province of BC are regulated by the British Columbia Utilities Commission, acting under the *Utilities Commission Act* (1996b). The majority of the power in BC is generated by BC Hydro, although there are an increasing number of independent power providers. The major transmission system in BC is also owned and operated by BC Hydro, which is the electric utility that would serve the KSM Project.

The viability of the Project is dependent upon being able to access low-cost electric power from the provincial electricity grid via the NTL. Grid connection would reduce sensitivity to changes in fuel costs, reduce traffic and accident risks on access roads, reduce air emissions associated with diesel power generation, and provide reliable and consistent power throughout the Project life. With the remoteness of the Project location, modular diesel generator sets will likely be required to supply power during construction. However, the Project's operational power demand is sufficiently large that power cannot be feasibly provided by on-site diesel generation.

The NTL will run near the Project in proximity to Highway 37, and will be accessible via an approximately 28.5-km long, 287 kV extension. This extension will be built from a switching station located in the near the TCAR junction with Highway 37. Prior to the selection of the TCAR (Section 33.7.4), another transmission line option was considered that would run along the Teigen Creek access road. A key factor in the viability of transmission line routes is the availability of road access to minimize construction required, maintenance costs, and habitat fragmentation and disturbance. Once the TCAR was selected, the Teigen alignment became the only viable transmission line extension route for the Project.

There are no alternatives besides the NTL for main power supply for the Project. In order to reduce and supplement power use from the NTL, Seabridge has made design changes to the Project. Although these changes did not involve alternatives evaluations, they are outlined briefly below, and are described in detail in the 2012 Prefeasibility Study (Appendix 4-C) for the Project.

One of the power challenges for the Project is that, even with planned power conservation measures, its power requirements—projected at 1,305 GWh per annum with average and peak flows of 149 MW and 171 MW respectively—will likely exceed the 150 megavolt-ampere (MVA) maximum contract power demand set by BC Hydro (Appendix 4-C). Customers that exceed this power demand must provide a large non-refundable capital contribution toward utility system transmission and generation reinforcement, applicable to the entire power load, not just exceedances.

Under its Power Smart program, BC Hydro has a tiered pricing system to incentivize designing energy conservation measures into new plants. The Project includes energy conservation design features, including using HPGR in lieu of SAG milling and other energy conservation design measures that may be certified by BC Hydro. If fully implemented, these and other energy conservation measures will eliminate higher tier pricing for the Project as well as provide the additional benefit of reducing GHG emissions.

To eliminate the substantial capital costs associated with generation reinforcement, the Project budget includes a gas-powered turbine to be installed in or near Terrace to feed peaking power into the BC Hydro system and thus eliminate the peaks in demand above 150 MVA. The turbine will use natural gas from the existing Pacific Northern Gas line that runs through Terrace. The capital cost of the installation has been included in the Project budget, and the cost of power for the Project has been adjusted to account for fuel, operation, and maintenance. The turbine can run unattended under automatic control, with generation being dispatched from the mine, and will be contracted to a third party.

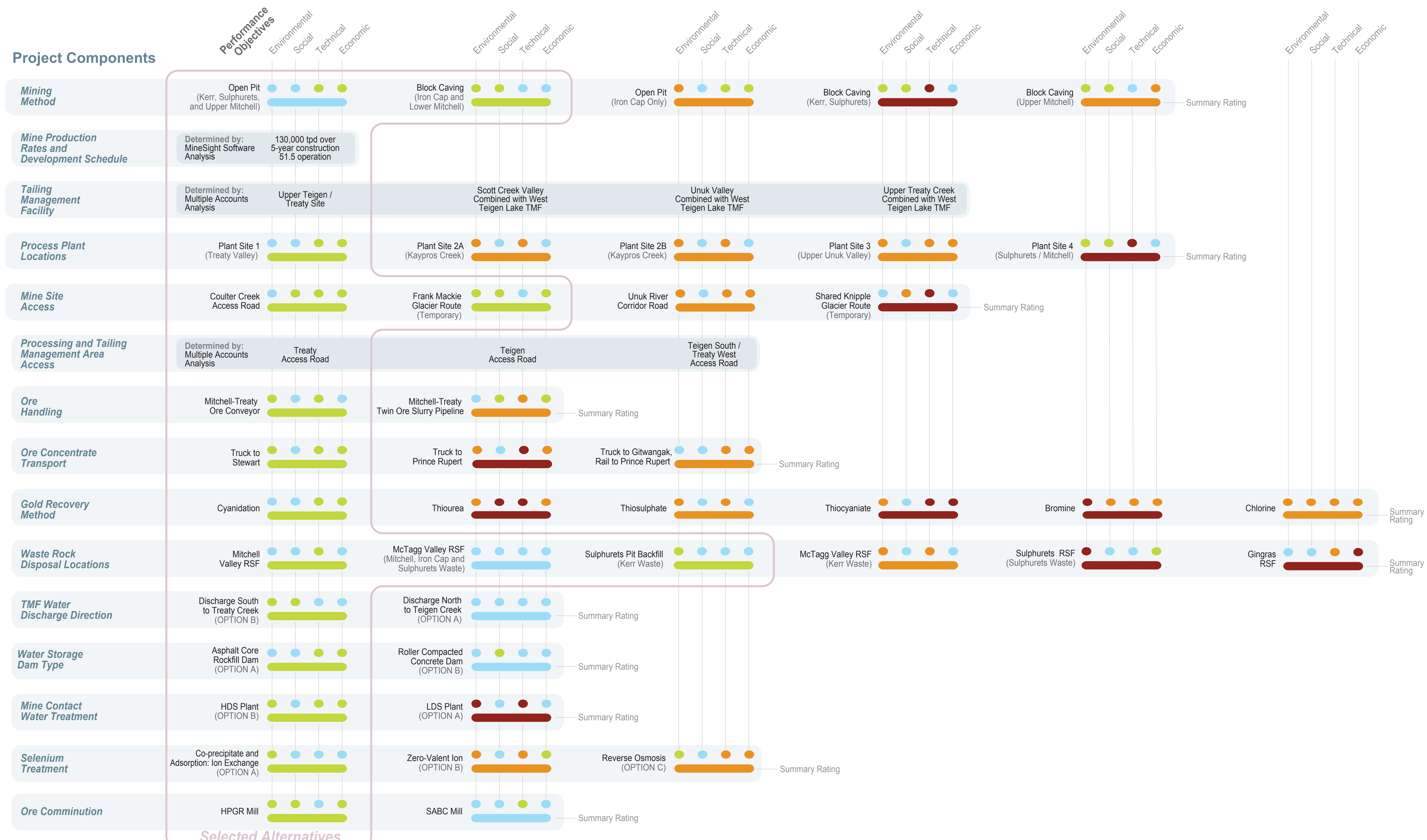
In addition to the gas-powered turbine, diverting of Mitchell and McTagg creeks into tunnels creates an opportunity for hydro-electric power generation for the Project. Such plants installed on the diversions are similar to run-of-river installations, in that they provide peak power during freshet flows. This generated power will be available for use during operation or sold back into the grid. During operation, the hydro-electric plants are estimated to reduce the power requirements of the Project. Upon closure, these plants will continue to operate, generating income, and offsetting water treatment costs.

W.N. Brazier Associates Inc. has assessed the hydroelectric capacities and revenue of the diversion hydroelectric facilities for Seabridge (Chapter 4, Appendix 4-X). The Sulphurets Power Plant from the Mitchell diversion will likely generate 5.5 MW, which will reduce power requirements of the Project by three to four percent. The McTagg Diversion will provide further energy savings when it comes into operation in Phase 2 (Year 10), and early estimates indicate that it may generate up to 15.5 MW. Further hydropower will be generated by the WTP Energy Recovery Facility at a relatively constant rate throughout each year, from contact water flowing from the WSF reservoir to the WTP.

The continued operation of the McTagg and Sulphurets power plants after closure is also estimated to generate about $2.25 M per year and $1.54 M per year respectively, which will support water quality treatment in the post-closure phase in perpetuity, along with posted bonds for treatment.

33.18 Summary of Alternatives Evaluation

This chapter has described the decision-making rationale behind all the Project components recommended for assessment in the AIR, as well as several more. For this assessment, Seabridge has undertaken to transparently demonstrate that the decision-making rationale behind the selected alternative for each of the Project components addressed has been conducted in a systematic, reasonable, and defensible manner—balancing technical and economic Project criteria with minimizing potential adverse effects on surrounding environmental and human systems. Figure 33.18-1 presents a summary of the entire alternatives assessment carried out for the Project. More detail on the development of selected Project alternatives is provided in the Project Description in Chapter 4.



- Preferred
- Acceptable
- Challenging
- Unfeasible

SEABRIDGE GOLD
KSM PROJECT
Rescan
Engineers & Scientists

Figure 33.18-1

Summary of KSM Project Alternatives Assessments

References

1985a. *Fisheries Act*, RSC. C. F-14.

1985b. *Hazardous Products Act*, RSC. C. H-3.

1992. *Canadian Environmental Assessment Act*, SC. C. 37.

1996a. *BC Mines Act*, RSBC. C. 293.

1996b. *Utilities Commission Act*, RSBC. C. 473.

Metal Mining Effluent Regulations, SOR/2002-222.

Acros Organics. 2004. *Bromine Material Safety Data Sheet*. http://alemis.us.airliquide.com/ChemSafe/MSDS/Image/129671_1.PDF (accessed January 2013).

Acros Organics. 2012. Sodium Thiosulfate MSDS.

Air Products. 1999. *Ammonia Material Data Safety Sheet*. Air Products and Chemicals Ltd. http://avogadro.chem.iastate.edu/MSDS/NH3_gas.pdf (accessed January 2013).

AMEC. 2004. *Victor Diamond Project: Environmental Assessment Comprehensive Study.* Prepared for De Beers Canada Exploration Inc. and submitted to the Canadian Environmental Assessment Agency: Toronto, ON.

AMEC. 2008. *Effects Assessment. Volume 5 of Mt. Milligan Copper-Gold Project Environmental Assessment*. Produced for Terrane Metals Corp.: Toronto, ON.

Amey, E. B. 1998. *Gold*. US Geological Survey Minerals Yearbook - 2003. http://minerals.usgs.gov/minerals/pubs/commodity/gold/gold_myb03.pdf (accessed August 2012).

Association for Mineral Exploration British Columbia. 2009. *Mineral Exploration Primer*. Notes from a workshop presented by David Wortman, prepared by J.V. Tully of Fluor Daniel Wright Engineers Ltd. http://web.archive.org/web/20090416234130/http:/amebc.ca/primer4.htm (accessed August 2012).

Baker, D., D. Bridges, R. Hunter, G. Johnson, J. Krupa, J. Murphy, and K. Sorensen. 2001. *Guidebook to Decision-making Methods*. Washington, DC: US Department of Energy.

BC EAO. 2011. *Application Information Requirements*. Approved by the British Columbia Environmental Assessment Office: Victoria, BC.

BC MEMPR. 2008. *Health, Safety and Reclamation Code for Mines in British Columbia*. British Columbia Ministry of Energy, Mines and Petroleum Resources - Mining and Minerals Division: Victoria, BC.

BC MOE. 1986. *Water Quality Criteria for Cyanide*. http://www.env.gov.bc.ca/wat/wq/BCguidelines/cyanide/cyanide.html (accessed September 2012).

BC MOE. 2000. *Ambient Water Quality Guidelines for Sulphate Overview Report*. http://www.env.gov.bc.ca/wat/wq/BCguidelines/sulphate/sulphate.html (accessed January 2013).

BC MOE. 2001a. *Ambient Water Quality Guidelines for Selenium*. http://www.env.gov.bc.ca/wat/wq/BCguidelines/selenium/selenium.html (accessed January 2013).

BC MOE. 2001b. *Water Quality Criteria for Nitrogen (Nitrate, Nitrite, and Ammonia)*. http://www.env.gov.bc.ca/wat/wq/BCguidelines/nitrogen/nitrogen.html#tab3 (accessed January 2013).

BC MOE. 2003. *Ambient Water Quality Guidelines for Chloride*. http://www.env.gov.bc.ca/wat/wq/BCguidelines/chloride/chloride.html (accessed January 2013).

BC MOE. 2006. *A Compendium of Working Water Quality Guidelines for British Columbia*. http://www.env.gov.bc.ca/wat/wq/BCguidelines/working.html (accessed January 2013).

BC MOE. n.d. *Water Quality Guidelines (Criteria) Reports*. BC Ministry of Environment. http://www.env.gov.bc.ca/wat/wq/wq_guidelines.html (accessed January 2013).

BC MOTI. 2011. *Vehicle Guidelines and Permits*. http://www.th.gov.bc.ca/cvse/ctpm/Chapter_5.pdf (accessed April 2012).

BC MOTI. 2012. *British Columbia Traffic Data Program. Traffic Data for Kitwanga*. http://www.th.gov.bc.ca/trafficdata/tradas/tradas.asp?loc=P-47-1EW (accessed June 2012).

BioteQ. 2013. *Report #: 2012-10-187-SEL-P1 - Laboratory Investigation of Hexavalent Selenium Removal from Kerr Waste Rock Leachate Water*. Prepared by BioteQ Environmental Technologies for Rescan Environmental Services Ltd.: Vancouver, BC.

CALGAZ. 2005. *Hydrogen Cyanide Material Safety Data Sheet*. http://www.conspec-controls.com/prodresources/msds/Hydrogen-Cyanide.pdf (accessed September 2012).

CEA Agency. 2007. *Operational Policy Statement: Addressing "Need for", "Purpose of", "Alternatives to" and "Alternative Means" under the Canadian Environmental Assessment Act*. http://www.ceaa-acee.gc.ca/default.asp?lang=En&n=5C072E13-1 (accessed September 2012).

CEA Agency. 2010. *Comprehensive Study Scope of Assessment Pursuant to Section 21(1) of the Canadian Environmental Assessment Act for the Proposed KSM (Kerr-Sulphurets-Mitchell) Project in Northwestern British Columbia Proposed by Seabridge Gold*. Prepared by the Canadian Environmental Assessment Agency, Environment Canada, Fisheries and Oceans Canada, Natural Resources Canada, Transport Canada. http://www.ceaa.gc.ca/050/documents/43478/43478E.pdf (accessed September 2012).

Cleartech Industries. 2010. *Chlorine Material Safety Data Sheet*. http://www.cleartech.ca/msds/chlorine.pdf (accessed January 2013).

CN Rail. 2012. *Port of Prince Rupert Container Terminal*. http://www.cn.ca/en/shipping-ports-prince-rupert-container-terminal.htm (accessed May 2012).

CN Rail. 2013. *Get a Carload Price eBusiness Tool*. http://www.cn.ca/en/shipping-prices-tariffs-get-price-freight.htm (accessed January 2013).

CN Rail. n.d. *Greenhouse Gas Calculator*. http://www.cn.ca/en/greenhouse-gas-calculator-tool.htm (accessed January 2013).

Davis, H., A. Congram, P. Nelson, G. Pulliam, and B. Davis. 2009. *A Coordinated Approach to Achieving NPDES Permit Compliance*. Paper presented at National Petrochemical & Refiners Association 2009 Environmental Conference: Denver, CO.

Denisen. 1994. *Bromine Material Safety Data Sheet*. http://www.denison.edu/offices/security/bromine_msds.html (accessed January 2013).

District of Stewart. 2009. *Investment Ready Community Profile*. http://www.districtofstewart.com/District/Stewart%20-%20Investment-Ready%20Community%20Profile%20(web).pdf (accessed April 2012).

DriveBC. 2012. *Traveller Information System*. http://www.drivebc.com/ (accessed August 2012).

Eisler, R. 2000. *Organics*. Vol. 2 of *Handbook of Chemical Risk Assessment: Health Hazards to Humans, Plants, and Animals*. Boca Raton, FL: Lewis Publishers.

Eisler, R. 2003. Health risk of gold miners: a synoptic review. *Environmental Geochemistry and Health*, 25: 325-45.

EMD. 2009. *Sodium Thiocyanate pure - Material Safety Data Sheet*. https://www.midlandsci.com/customer/miscne/specpages/msdsu_1.06627.pdf (accessed January 2013).

Environment Canada. 2009. *Environmental Code of Practice for Metal Mines*. http://www.ec.gc.ca/lcpe-cepa/default.asp?lang=En&n=CBE3CD59-1 (accessed January 2013).

Environment Canada. 2011. *Guidelines for the Assessment of Alternatives for Mine Waste Disposal*. Prepared for the Mining and Processing Division, Environment Canada: Gatineau, QC.

EPA. 1994a. *Chemical Summary for Chlorine*. Office of Pollution Prevention and Toxics, US EPA. http://www.epa.gov/chemfact/s_chlori.txt (accessed December 2012).

EPA. 1994b. *R.E.D. Facts: Sodium Cyanide*. http://www.epa.gov/oppsrrd1/REDs/factsheets/3086fact.pdf (accessed September 2012).

Fisher Scientific. 2006. *Thiourea Material Safety Data Sheet*. http://www.coleparmer.com/Assets/Msds/23420.htm (accessed January 2013).

Flueck, W. T., J. M. Smith-Flueck, J. Mionczynski, and B. J. Mincher. 2012. *The implications of selenium deficiency for wild herbivore conservation: a review*. European Journal of Wildlife Research. http://www.deerlab.org/Publ/pdfs/61.pdf (accessed May 2013).

Flury, M. and A. Papritz. 1993. Bromide in the Natural Environment: Occurance and Toxicity. *Journal of Environmental Quality*, 22 (4): 747-58.

Fülöp, J. 2005. *Introduction to Decision Making Methods. Working Paper 05-6.* . Laboratory of Operations Research and Decision Systems, Computer and Automation Institute, Hungarian Academy of Sciences. http://academic.evergreen.edu/projects/bdei/documents/decisionmakingmethods.pdf (accessed April 2012).

Golder. 2009. *Literature Review of Treatment Technologies to Remove Selenium from Mining Influenced Water*. Prepared for Teck Coal Ltd. by Golder Associates Inc.: Calgary, AB.

Golder. 2012a. *December 13, 2012 Report on Mitchell Deposit Geotechnical Characterization*. Prepared for Seabridge Gold Inc. by Golder Associates Ltd.: Toronto, ON.

Golder. 2012b. *Report on Pre-feasibility Block Cave Mine Design—Iron Cap Deposit*. Prepared for Seabridge Gold Inc. by Golder Associates: Vancouver, BC.

Golder. 2012c. *Report on Pre-feasibility Block Cave Mine Design—Mitchell Deposit*. Prepared for Seabridge Gold Inc. by Golder Associates: Vancouver, BC.

Gos, R. and A. Rubo. 2001. *The Relevance of Alternative Lixiviants with Regard to Technical Aspects, Work Safety and Environmental Safety*. Prepared for CyPlus and DeGussa AG: Hanau, Germany.

Hazardous Substances Data Bank. 2011. http://toxnet.nlm.nih.gov/cgi-bin/sis/htmlgen?HSDB (accessed April 2012).

Health Canada. 1987. *Water Quality - Sulphate*. http://www.hc-sc.gc.ca/ewh-semt/pubs/water-eau/sulphate-sulfates/index-eng.php (accessed January 2013).

Health Canada. 2008. *Environmental and Workplace Health, Workplace Hazardous Materials Information System : Material Data and Safety Sheets*. http://www.hc-sc.gc.ca/ewh-semt/occup-travail/whmis-simdut/msds-fs-eng.php (accessed January 2013).

Health Canada. 2010. *Environmental and Workplace Health, Workplace Hazardous Materials Information System Official Site*. http://www.hc-sc.gc.ca/ewh-semt/occup-travail/whmis-simdut/index-eng.php (accessed January 2013).

Hiskey, J. B. and F. W. DeVries. 1992. *Emerging process technologies for a cleaner environment. Proceedings of the Symposium on Emerging Process Technologies for a Cleaner Environment*. S. Chandler, ed. Phoenix, AZ.

ICMI. 2012. *The International Cyanide Management Code for the Manufacture, Transport and Use of Cyanide in the Production of Gold (Cyanide Code)*. Cyanide use: http://www.cyanidecode.org/cyanide_use.php; Environmental and health effects: http://www.cyanidecode.org/cyanide_environmental.php; Principles and standards of practice: http://www.cyanidecode.org/about_code.php (accessed April 2012).

iMapBC. n.d. *Welcome to iMapBC*. http://webmaps.gov.bc.ca/imfx/imf.jsp?site=imapbc (accessed January 2013).

International Water Power & Dam Construction. 2012. *Yearbook 2012*. Global Trade Media: Kent, UK.

KCB. 2009. *Site Investigation Report* Prepared for Seabridge Gold Inc. by Klohn Krippen Bergen: Toronto, Ontario.

KCB. 2011. *KSM Project 2010 Site Investigation Report*. Prepared for Seabridge Gold Inc. by Klohn Crippen Berger Ltd.: Vancouver, BC.

KCB. 2012. *2012 Geotechnical Design of Rock Storage Facilities and Design of Associated Water Management Facilities*. Prepared for Seabridge Gold Inc. by Klohn Krippen Berger: Toronto, Ontario.

Kuyucak, N., M. Lindvall, J. A. R. Serrano, and A. F. Oliva. 1999. *Implementation of a high density sludge "hds" treatment process at the Boliden Apirsa Mine Site*. Paper presented at Mine, Water & Environment

Sevilla, Spain:

la Brooy, S. R., H. G. Linge, and G. S. Walker. 1994. Review of gold extraction from ores. *Minerals Engineering*, 7 (10): 1213-41.

Laplante, A. and R. C. Dunne. 2002. *The gravity recoverable gold test and flash flotation*. Paper presented at The 34th Annual Meeting of the Canadian Mineral Processors, Ottawa, ON.

Li, A., B. Andruchow, I. Wislesky, and E. Olson. 2011. *Field testing of co-disposal techniques for acid generating tailings and waste rock at Cerro de Maimon Mine*. Paper presented at Tailings and Mine Waste 2011: Vancouver, BC.

Li, J., M. S. Sarafzadeh, M. S. Moats, J. D. Miller, L. K. M., M. Dietrich, and R. Y. Wan. 2012. Thiocyanate hydrometallurgy for the recovery of gold. Part V: Process alternatives for solution concentration and purification. *Hydrometallurgy*, V (113-114): 31-38.

Lyday, P. A. 2007. *Commodity Report 2007: Bromine*. United States Geological Survey. http://minerals.usgs.gov/minerals/pubs/commodity/bromine/mcs-2008-bromi.pdf (accessed May 2012).

Marsden, J. O. and C. I. House. 2006. *The Chemistry of Gold Extraction*. Second ed. Littleton, CO: Society for Mining, Metallurgy and Exploration, Inc.

Martin, A. J., S. Simpson, S. Fawcett, C. I. Wiramanaden, I. J. Pickering, N. Belzile, Y. W. Chen, J. London, and D. Wallschlager. 2011. Biogeochemical mechanisms of selenium exchange between water and sediments in two contrasting lentic environments. . *Environmental Science & Technology*, 45 (7): 2605-12.

MineSight. 2009. *Long-term Planning for Open Pit Mines (White Paper)*. http://madencilikrehberi.files.wordpress.com/2012/03/long-term-planning-for-open-pit-mines.pdf (accessed May 2012).

Muir, D. M. and M. G. Alymore. 2004. Thiosulphate as an alternative to cyanide for gold processing—issues and impediments. *Transactions of the American Institute of Mining and Metallurgical Engineers*, 113: C2-C12.

Paschka, M. G., R. S. Ghosh, and D. A. Dzombak. 1999. Potential water-quality effects from iron cyanide anticaking agents in road salt. *Water Environment Research*, 71 (6): 1235-39.

Piteau Associates. 1991. *Mine Rock and Overburden Piles Investigation and Design Manual: Interim Guidelines*. Prepared for the British Columbia Mines Waste Rock Pile Research Committee by Piteau Associates Engineering Ltd. http://www.empr.gov.bc.ca/Mining/Permitting-Reclamation/Geotech/Documents/MinedRock+OverburdenPiles/MinedRockOverburdenPile_Investigation+DesignManual.pdf (accessed January 2013).

Price, W. 1997. *Guidelines and Recommended Methods for the Prediction of Metal Leaching and Acid Rock Drainage at Minesites in British Columbia*. Prepared for the BC Ministry of Employment and Investment, Reclamation Section, Energy and Minerals Division: Smithers, BC.

Rescan. 2008. *Project Description for the Kerr-Sulphurets-Mitchell Project, Project Overview*. Prepared for Seabridge Gold Inc. by Rescan Environmental Services, and distributed by British Columbia Environmental Assessment Office: Vancouver, BC.

Rescan. 2012. *KSM Project: Highway 37 and 37A Traffic Effects Assessment*. Prepared for Seabridge Gold Inc. by Rescan Environmental Services Ltd.: Vancouver, BC.

Rezai, B., F. Peikary, Z. Mos'hefi, and A. Amini. 2003. Comparative Cyanide and Thiourea Extraction of Gold Based on Characterization Studies. *IJE Transactions B: Applications*, 17: http://www.sid.ir/en/VEWSSID/J_pdf/85620031B10.pdf (accessed

Sandy, T. and C. DiSante. 2010. *Review of Available Technologies for the Removal of Selinium from Water*. Prepared by CH2M Hill for the North American Metals Council: n.p.

Santa Cruz Biotechnology Inc. 2009. *Sodium Thiocyanate Material Safety Data Sheet*. http://datasheets.scbt.com/sc-203407.pdf (accessed January 2013).

Sciencelab.com Inc. 2012a. *Mercury MSDS*. http://www.sciencelab.com/msds.php?msdsId=9927224 (accessed August 2012).

Sciencelab.com Inc. 2012b. *Sodium Thiocyanate Material Safety Data Sheet*. http://www.sciencelab.com/msds.php?msdsId=9927605 (accessed January 2013).

Scousen, J., T. Hilton, and B. Faulkner. n.d. *Overview of Acid Mine Drainage Treatment with Chemicals*. West Virginia University. http://www.wvu.edu/~agexten/landrec/chemtrt.htm (accessed January 2013).

SGS. 2012. *Alternatives to cyanide in gold processing, thiosulphate*. http://www.sgs.com/en/Mining/Services-for-Industry-Challenges/Environmental-Issues/Thiosulphate-Leaching.aspx# (accessed August 2012).

SGS. 2013. *An Investigation in KSM Project HDS Pilot Plant*. Prepared by SGS Canada Inc. for Seabridge Gold: Burnaby, BC.

Shamas, J., C. Wagener, and T. Cooke. 2009. *Technologies and Strategies for the Treatment of Selenium as a Microconstituent in Industrial Wastewater*. Paper presented at WEF Microconstituents & Industrial Water Quality Specialty Conference: Baltimore, MD.

Sobolewski, A. 2005. *Evaluation of Treatment Options to Reduce Water-Borne Selenium at Coal Mines in West-Central Alberta*. Prepared for Alberta Environment Water Research Users Group: Edmonton, AB.

Statistics Canada. 2012. *British Columbia Census Profile*. http://www12.statcan.ca/census-recensement/2011/dp-pd/prof/index.cfm?Lang=E (accessed March 2012).

Transport Canada. 2011. *TDG Regulations Classification*. Government of Canada. http://www.tc.gc.ca/eng/tdg/clear-part2-339.htm#sec213 (accessed January 2013).

Transport Canada. 2012. *TDG Regulations-Schedule 1-Search* Government of Canada. http://wwwapps.tc.gc.ca/saf-sec-sur/3/sched1-ann1/schedule1form.aspx (accessed October 2012).

Tremblay, L., G. Deschenes, E. Ghali, J. MacMullen, and M. Lanouette. 1996. Gold recovery from a sulfide bearing gold ore by percolation leaching with thiourea. *International Journal of Mineral Processing*, 48: 225-44.

US DHHS. 2011. *Report on Carcinogens, Twelfth Edition*. National Toxicology Program, Department of Health and Human Services. http://ntp.niehs.nih.gov/ntp/roc/twelfth/profiles/Thiourea.pdf (accessed May 2012).

US EPA. 1994. *Treatment of Cyanide Heap Leaches and Tailings*. Prepared for the Office of Solid Waste Special Waste Branch, United States Environmental Protection Agency. https://fs.ogm.utah.gov/pub/mines/minerals_related/EPApublications/Cyanide.PDF (accessed April 2012).

US Geological Survey. 2000. *Mercury in the Environment Fact Sheet 146-00*. http://www.usgs.gov/themes/factsheet/146-00/ (accessed May 2012).

Wan, R. Y. 1997. *Importance of solution chemistry for thiosulfate leaching of gold*. Paper presented at World Gold '97, Singapore: Australasian Institute of Mining and Metallurgy.

Wardrop. 2011. *Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update 2011*. Prepared for Seabridge Gold Inc. by Wardrop Engineering: Vancouver, BC.

Wardrop. 2012a. *2012 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study*. Prepared for Seabridge Gold Inc. by Wardrop Engineering: Vancouver, BC.

Wardrop. 2012b. *Kerr-Sulphurets-Mitchell (KSM) Comminution Circuit Energy Comparison Between HPGR Circuit and SABC Circuit*. Prepared for Seabridge Gold Inc. by Wardrop Engineering: Vancouver, BC.

WHO. 1996. *WHO Guidelines for Drinking-Water Quality*. World Health Organization. http://www.who.int/water_sanitation_health/dwq/chloride.pdf (accessed January 2013).

WISER. 2012. US National Library of Medicine. http://wiser.nlm.nih.gov/ (Wireless Information System for Emergency Responders (WISER) Online Database accessed January 2013).

Woo, K., E. Eberhardt, and A. Van As. 2009. *Characterization and empirical analysis of block caving induced surface subsidence and macro deformations*. Paper presented at the 3rd CNUS Rock Mechanics Symposium, Toronto, ON.

Xia, C. 2008. Associated Sulfide Minerals in Thiosulphate Leaching of Gold: Problems and Solutions. PhD diss., Queens University.

Yannopoulos, J. C. 1991. *The Extractive Metallurgy of Gold*. New York: Van Nostrand-Reinhold.

Yarar, B. 2002. *Long term persistence of cyanide species in mine waste environments*. Paper presented at Tailings and Mine Waste 2002, Fort Collins, CO: Swets & Zeitlinger.

Ziegler-Syklakakis, K., J. Kielhorn, G. Könnecker, J. Koppenhöfe, and I. Mangelsdorf. 2003. *Thiourea - Concise International Chemical Assessment Document 49*. Prepared for the United Nations Environment Programme, the International Labour Organization, and the World Health Organization. http://www.inchem.org/documents/cicads/cicads/cicad49.htm (accessed May 2012).