Preliminary Economic Assessment for the Springpole Gold Project, Ontario, Canada

Report Prepared for

GOLD CANYON RESOURCES INC.

Report Prepared by

srk consulting

SRK Consulting (Canada) Inc.
2CG026.000
Effective Date: March 25, 2013
Amended: October 7, 2016

Preliminary Economic Assessment for the Springpole Gold Project, Ontario, Canada

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Cover: *Springpole camp March 2013 (source: The Northern Sun News)*

Disclaimer

This report was prepared as a National Instrument 43-101Technical Report for Gold Canyon Resources Inc. (Gold Canyon) by SRK Consulting (Canada) Inc. (SRK). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in SRK's services, based on: (i) information available at the time of preparation, (ii) data supplied by outside sources, and (iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Gold Canyon subject to the terms and conditions of its contract with SRK and relevant securities legislation. The contract permits Gold Canyon to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with Gold Canyon. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

Important Notice

This preliminary economic assessment (PEA) report is intended to provide an initial review of the Gold Canyon Resources Inc. Springpole Gold Project's potential and is preliminary in nature. The PEA includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA based on these mineral resources will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

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Executive Summary

SRK Consulting (Canada) Inc. was retained by Gold Canyon Resources Inc. of Vancouver, British Columbia, to prepare a technical report summarizing the mineral resources for the Springpole Gold Project. The mineral resource estimate forms the basis of this preliminary economic assessment (PEA) prepared by SRK.

Project Concept

The proposed Springpole Gold Project concept is to develop a low-grade, greenfield gold and silver bulk tonnage deposit with open pit mining and conventional milling methods. The production rate was assumed to be 20,000 tonnes per day (t/d) with a total of 72.4 million tonnes (Mt) of mineralized material mined and processed during the project life. The overall strip ratio (the ratio of waste rock to economic mineralized rock) of the mine is approximately 1:7 and the average grade of the plant feed is estimated at 1.2 grams per tonne (g/t) of gold and 6.0 g/t of silver.

Property Description and Ownership

The Springpole Gold Project is located 110 km northeast of Red Lake, Ontario, and is 100% controlled by Gold Canyon. The project's land position comprises 30 patented claims and 300 unpatented, contiguous mining claims and 6 leased unpatented mining claims totalling an area of approximately 32,448 hectares (80,181 acres).

During late spring, summer, and early fall, the project is accessible by float-plane direct to Springpole Lake or Birch Lake. During winter, an ice road approximately 85 km long is constructed from the South Bay landing point on Confederation Lake to a point about 1 km from the Springpole camp.

Geology and Mineralization

The Springpole area is underlain by a polyphase alkali, trachyte intrusive displaying autolithic breccia. The intrusive is comprised of a system of multiple phases of trachyte that is believed to be part of the roof zone of a larger syenite intrusive; fragments displaying phaneritic textures were observed from deeper drill cores in the southeast portion of the Portage zone. Early intrusive phases consist of megacrystic feldspar phenocrysts of albite and orthoclase feldspar in an aphanitic groundmass Successive phases show progressively finer grained porphyritic texture while the final intrusive phases are aphanitic. Within the country rocks to the north and east are trachyte and lamprophyre dikes and sills that source from the trachyte- or syenite-porphyry intrusive system.

The main intrusive complex appears to contain many of the characteristics of alkaline, porphyry style mineralization associated with diatreme breccias (e.g., Cripple Creek, Colorado). This style of mineralization is characterized by the Portage zone and portions of the East Extension zone where mineralization is hosted by diatreme breccia in aphanitic trachyte. It is suspected that the ductile shearing and brittle faulting have played a significant role in redistributing structurally controlled blocks of the mineralized rock. Diamond drilling in the winter of 2010 revealed a more complex alteration with broader, intense zones of potassic alteration replacing the original rock mass with biotite and pyrite. In the core area of the deposit where fine grained disseminated gold mineralization occurs with biotite, the primary potassic alteration mineral, gold displays a good correlation with potassium/rubidium.

Exploration Status

The initial geologic and engineering studies at the end of 2009 resulted in the establishment of systematic drill sections at 50 m intervals across the three identified prospect areas, namely Portage zone, East Extension zone, and Camp zone. The subsequently developed drill program lead to a multi-phase drill campaign starting in the summer of 2010 and ending in the summer of 2012, resulting in completion of 77,275 m of diamond core drilling in 196 drill holes. During the course of the 2010, 2011, and 2012 programs, drilling identified a precious metal deposit of significant strike, depth and width within the Portage zone.

Mineral Resource and Mineral Reserve Estimates

The mineral resource model prepared by SRK considers 512 core boreholes drilled by Gold Canyon and previous owners of the property during the period of 2003 to 2012. The resource estimation work was completed by Dr. Gilles Arseneau, PGeo (APEGBC #23474), an appropriate independent qualified person as this term is defined in NI 43-101. The effective date of the resource statement is October 17, 2012.

The revised mineral resource estimate (October 17, 2012) was based on a gold price of $1,400/oz and a silver price of $15/oz, both considered reasonable economic assumptions by SRK. To establish a reasonable prospect of economic extraction in an open pit context, the resources were defined within an optimized pit shell with pit walls set at 45°. Assumed recoveries of 80% for gold and 60% for silver were used. (Note: A silver recovery assumption of 85% was used for mine design and evaluation based on more recent data.) Mining costs were estimated at $2/tonne (t) of total material, processing costs estimated at $12/t and general and administrative (G&A) costs estimated at $2/t. A cut-off grade (COG) of 0.4 g/t gold was calculated, and is considered to be an economically reasonable value corresponding with breakeven mining costs. Approximately 90% of the revenue for the proposed project is derived from gold and 10% from silver.

Note: *For the mine development (Whittle™ optimization) and economic analysis in this PEA, updated input parameters were used.*

Mineral resources were estimated by ordinary kriging using Gemcom block modelling software in 10 m x 10 m x 6 m blocks. Grade estimates were based on capped, 3 m composited assay data. Capping levels were set at 25 g/t for gold and 200 g/t for silver. Blocks were classified as indicated mineral resources if at least two drill holes and six composites were found within a 60 m x 60 m x 40 m search ellipse. All other interpolated blocks were classified as inferred mineral resource. Mineral resources were then validated using Gemcom GEMS (6.4) software.

This resource model includes mineralized material in the Main, East Extension and Portage zones spanning from geologic sections 0-1, 500 m in the northwest to 0-250 m in the southeast. Along the axis of the Portage zone, resource modelling includes mineralized material generally ranging from the surface to a depth of 340-440 m below surface.

Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues. The quantity and grade of reported inferred resources in this estimation are uncertain in nature. There has been insufficient exploration to define these inferred resources as an indicated or measured mineral resource. It is uncertain if further exploration will result in upgrading them to an indicated or measured mineral resource category.

The mineral resources in this report were estimated using current Canadian Institute of Mining, Metallurgy and Petroleum (CIM) standards, definitions and guidelines. The updated resource estimate is summarized in Table i.

Table i: Mineral Resource Statement, Springpole Gold Project (October 17, 2012)

Category	Quantity	Grade		Contained Metal	
		Au	Ag	Au	Ag
	Mt	g/t	g/t	Moz	Moz
Open Pit**					
Indicated	128.2	1.07	5.7	4.41	23.8
Inferred	25.7	0.83	3.2	0.69	2.7

Source: Springpole Gold Project, Northwestern Ontario, SRK Consulting, October 17, 2012

Note: Mineral resources are reported in relation to a conceptual pit shell. Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate.

***Open pit mineral resources are reported at a COG of 0.4 g/t gold. COGs are based on a gold price of $1,400/oz and a gold processing recovery of 80% and a silver price of $15/oz and a silver processing recovery of 60%.*

Inferred resources were used in the life of mine (LOM) plan with inferred resources representing 10% of the material planned for processing. Mineral reserves can only be estimated as a result of an economic evaluation as part of a preliminary feasibility study or a feasibility study of a mineral project. Accordingly, at the present level of development, there are no mineral reserves at the Springpole Gold Project.

Mine Development and Operations

The mine development plan for Springpole contemplates open pit mining with a mine plan to produce a total of 72 Mt of processing plant feed and 121 Mt of waste (1.7:1 overall strip ratio) over an eleven year mine production life. The current LOM plan focuses on achieving steady plant feed production rates, and mining of higher grade material early in schedule, as well as balancing grade and strip ratios. Figure i illustrates the proposed overall site layout for the project, including the open pit, waste rock facilities, and proposed plant site locations.

The mine design process for the deposit commenced with the development of Whittle optimization input parameters. These parameters included estimates of metal price, mining dilution, process recovery, offsite costs, geotechnical constraints (slope angles) and royalties (Table ii).

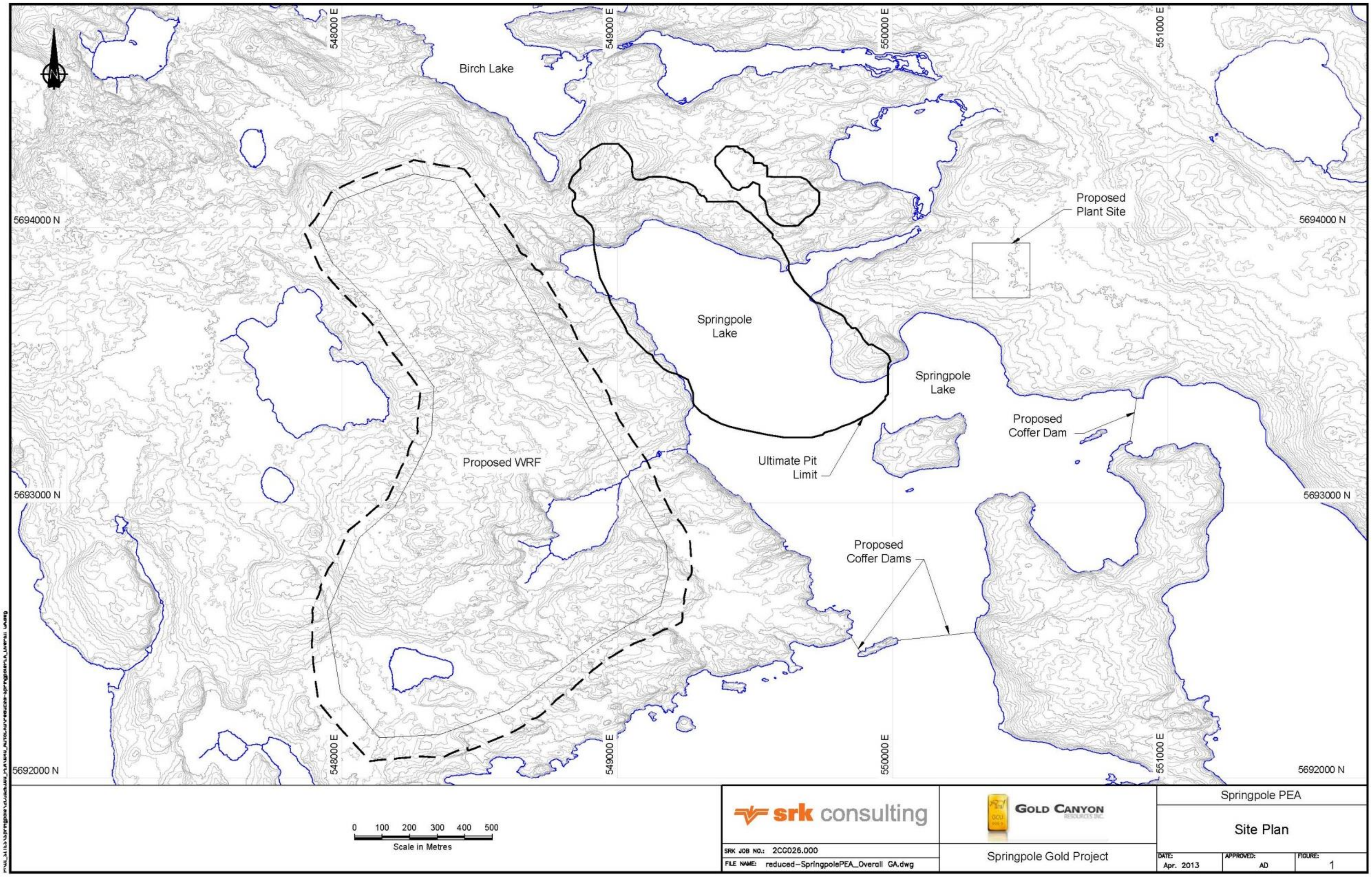

Figure i: Overall Springpole Site Plan

Table ii: Mine Planning Optimization Input Parameters

Item	Unit	Value
Metal Prices		
Au	$/oz	1,300
Ag	$/oz	25
Recovery to Doré		
Au	%	80
Ag	%	85
Smelter Payables		
Au in doré	%	99.5
Au deduction in doré	g/t	0
Ag in doré	%	98
Ag deduction in doré	g/t	0
Offsite Costs		
Au refining/transportation charge	$/oz pay Au	5
Other Parameters		
Royalties	%	3
Operating Costs		
Open Pit Waste mining Cost	$/t	2.4
OP Mineralized material Mining Cost	$/t	2.4
OP Processing and G&A Cost	$/t milled	13.02
Overall Pit Slope Angles	degrees	35 to 50
Mining Dilution	%	5
Mining Recovery	%	100
Strip ratio (est.)	t:t	1.7
Internal Net Smelter Return cut-off	$/t	13.67
Processing rate	t/day	20,000

**The open pit mineable resources are reported at an internal cut-off value of $13.67/t based on input parameters above.*

Whittle software was used to determine the optimal mining shells with the assumed overall slope angles based on a preliminary geotechnical assessment. Preliminary phases were selected and preliminary mine planning and scheduling was then conducted on these selected optimal shells. The mineable resources for the deposit are presented in Table iii. Indicated and inferred resources were used in the LOM plan of which indicated resources represent about 90% (~ 65 Mt) of the material planned to be processed. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. Mineral reserves can only be estimated as a result of an economic evaluation as part of a pre-feasibility study or a feasibility study of a mineral project. Accordingly, at the present level of development there are no mineral reserves at the project.

Table iii: Springpole PEA—Proposed Mining Plan

Description	Unit	Value
Mine Production Life	yr	11
Process Feed Material	Mt	72.4
Diluted Au Grade (mill head grade)	g/t	1.19
Contained Au	koz	2,777
Diluted Ag Grade (mill head grade)	g/t	6.01
Contained Ag	koz	13,991
Waste	Mt	120.8
Total Material	**Mt**	**193.2**
Strip Ratio	t:t	1.7

The mining sequence was divided into a number of stages designed to maximize grade, reduce pre-stripping requirements in the early years and, maintain the plant at full production capacity. The LOM production schedule is shown in Table iv. The open pit mining operation is planned as an owner-operated scenario. A total of 72.4 Mt of mineralized material is proposed to be processed.

Metallurgy and Mineral Processing

Three metallurgical testwork programs have been completed on Springpole material since 1989, with the majority of work undertaken in the past three years.

Based on the testwork results reported to date and the range of process flowsheet options considered in the 2012/2013 work, a likely flowsheet configuration is a moderately fine grind size followed by whole feed leaching. Removal of gravity gold prior to leaching appears to only benefit high grade (>5 g/t gold) feed and should be considered an option for the flowsheet.

This PEA is based on a design plant capacity of 20,000 t/d. Cyanide leach extractions of 80% for gold and 85% for silver are expected at this grind size. Product from the process plant will be doré bullion.

While considerable metallurgical testwork has been completed, additional testing is warranted to better define the plant design criteria and more confidently predict expected performance.

Future testwork can consider concentrating the sulphides via flotation or classification into a smaller mass so that it can be stored separately from the remainder of the tailings. For example, dry stack or subaqueous deposition in the tailings pond to minimise the potential for acid generation.

Table iv: Proposed LOM Production Schedule

Description	Unit	Total	Year										
			1	2	3	4	5	6	7	8	9	10	11
Mineralized Material Mined	Mt	72.4	3.9	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	2.9
Au Feed Grade	g/t	1.19	1.54	1.47	1.32	0.78	0.97	1.41	1.54	0.83	0.97	1.13	1.53
Contained Au	koz	2,777	191	345	309	184	227	330	362	194	227	266	141
Ag Feed Grade	g/t	6.01	3.64	8.78	6.29	2.99	5.21	7.52	5.78	5.95	5.91	6.25	7.72
Contained Ag	koz	13,991	452	2,060	1,476	703	1,222	1,765	1,356	1,395	1,386	1,466	709
Waste Mined	Mt	120.8	16.1	2.9	20.3	13.6	6.1	8.8	22.6	19.8	7.5	2.6	0.3
Strip Ratio	t:t	1.7	4.2	0.4	2.8	1.9	0.8	1.2	3.1	2.7	1.0	0.4	0.1
Total Material Mined	Mt	193.2	20.0	10.2	27.6	20.9	13.4	16.1	29.9	27.1	14.8	9.9	3.2

Waste Management

The waste rock facility is planned to be located immediately adjacent to the final pit limits. Given the deposit configuration and extraction sequence, no backfilling into previously mined out areas has been planned for Springpole.

The waste rock facility would be built in a series of lifts in a “bottom-up” approach, and the facility would be constructed by placing material at its natural angle of repose (approximately 1.5H:1V) with safety berms spaced at regular intervals giving an overall operational slope of 2:1. The total design capacity of the waste rock facility is 121 Mt.

Roughly 72 Mt of thickened tailings (about 60% solids by mass) will be centrally discharged at a site located about 5 km southeast of the proposed mill site. This results in an estimated 45 Mm^3 of tailings to be stored, based on conservative estimates for a tailings density of 1.6 t/m^3.

Due to the flat topographical relief of the project area, the tailings will be contained by a ring dam which will prevent migration of tailings, lake bed sediments, and any free water. The tailings facility is designed such that the dams will remain at the initial starter dam height of 2 m, except in the areas were a higher dam is required to contain lake bed sediments or prevent the tailings from encroaching on nearby lakes.

To minimize pre-production capital cost, a dam continuously raised over the LOM would be the preferred construction method. However, due to the need to contain lake bed sediments (from dike construction), the initial ring dam will be constructed to the final dam height in all but a few areas, and only minimal raises (less than 1 m) would be required in the remaining areas.

Three dewatering dikes with a total length of approximately 510 m will be constructed in Springpole Lake to allow a portion of the lake to be dewatered. The dikes will be constructed to elevation 391 m above mean sea level, which allows 3 m of freeboard above the lake level.The dikes will be constructed under wet conditions; therefore, two silt curtains will be deployed downstream of the dike locations to prevent high suspended solids in the remainder of the lake. Prior to the placement of fill material, the foundation of the dam will be dredged to remove any soft lakebed sediments. The rock fill material will be placed, and then the grout curtain and plastic concrete cut-off wall will be built through the completed dike.

An estimated 21.7 Mm^3 of water will have to be drained from the area of Springpole Lake within the dewatering dikes. Of this, ~80% (17.4 Mm^3) is estimated to be clean water which can be discharged directly over the dewatering dikes into Springpole Lake, inside the silt curtain. The remaining 20% (4.3 Mm^3) is assumed to be “murky” (i.e., have suspended solids higher than the allowable discharge limits). The murky water will be pumped to the tailings management facility (TMF) which will act as a sedimentation pond; no tailings will be in the TMF at this time. Clear water from the TMF will be pumped to Springpole Lake. Steps will be taken during the dewatering process to reduce the amount of sediments that become suspended in the water, including silt curtains around the water intake area.

Project Cost Estimates

Project costs were estimated from a combination of sources including first principles, reference projects, vendor's quotes, cost service publications and SRK experience.

The capital cost (CAPEX) estimate for the project is shown in Table v at a total of $544M. An overall, weighted-average contingency of 15% was used for this project. Property acquisition costs are not included in the capital estimate.

Table v: Capital Cost Estimates

Capital costs by timing	$M
Total Preconstruction Owners Costs	7
Initial Capital	431
Sustaining Capital	86
Mine Closure	20
***Total Capital Costs**	**544**

**Including weighted-average overall contingency of 15%*

A summary of the operating cost (OPEX) estimate by SRK is shown in Table vi. The OP mining OPEX assumes owner-operated mining including technical/supervisory support staff. Diesel fuel was estimated to cost $1.10/L and power was estimated to cost $0.08/kWh.

Table vi: Operating Cost Estimates

Activity	LOM ($M)	Per Tonne of Mill Feed	Per Ounce of AuEq*
Mining	511	7.10	209.9
Processing	760	10.50	312.3
Tailings Handling	16	0.20	6.4
G&A	167	2.30	68.4
Total Operating Cost	**1,454**	**20.10**	**597.0**
Royalty Per Ounce @3%			38.9
Total Cash Costs including Royalty		**20.10**	**635.9**

**Ounce of AuEq = total revenue from precious metals divided by gold price per ounce*

Economic Analysis

The Economic Analysis that forms part of this preliminary economic assessment (PEA) report is intended to provide an initial review of the Gold Canyon Resources Inc. Springpole Gold Project's potential and is preliminary in nature. The economic analysis incuded in this PEA includes consideration of inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA based on these mineral resources will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The base case economic analysis results indicate a post-tax NPV of $388M (pre-tax NPV: $579M) at a 5% discount rate with an IRR of 14% (pre-tax IRR: 25%). Payback (post-tax, non-discounted) will be in Year 3 of production in a projected 11 year LOM production period. The economics are based on a base case of $1,300/oz long-term gold price, $25/oz long-term silver price, and production rate of 20,000 t/d over 365 d/yr. Direct operating costs are estimated to be $636/oz of AuEq. Total capital costs are estimated at $544M, consisting of initial capital costs of $438M, and ongoing sustaining capital of $106M.

Sensitivity analysis indicates that, at base case metal price assumptions, the project may be able to absorb significant escalation in capital and operating costs and remain a potentially viable economic proposition.

Conclusions

Industry standard mining, process design, construction methods, and economic evaluation practices were used to assess the Springpole Gold Project. Based on current knowledge and assumptions, the results of this study indicate that the project has positive economics within the preliminary parameters of a PEA and should be advanced to the next level of study - either preliminary feasibility or feasibility.

The preliminary economic assessment is preliminary in nature; it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

While a significant amount of information is still required to do a complete assessment, at this point there do not appear to be any fatal flaws for the project. The study achieved its original objective of providing a preliminary review of the potential economic viability of the Springpole Gold Project.

Risks and Opportunities

As with almost all mining ventures, there are a large number of risks and opportunities that can affect the outcome of the project. Most of these risks and opportunities are based on uncertainty, such as lack of scientific information (test results, drill results, etc.) or the lack of control over external factors (metal price, exchange rates, etc.).

Subsequent higher-level engineering studies would be required to further refine these risks and opportunities, identify new risks and opportunities, and define strategies for risk mitigation or opportunity implementation.

The principal risks identified for the Springpole Gold Project are summarized as follows:

- Geological interpretation and mineral resource classification (10% of the resources used in the mine plan are Inferred);
- Due to a relatively small number of metallurgical samples tested, larger variations in mineralogy and metal recovery may exist than have been observed to date;
- Geotechnical and hydrogeological considerations;
- No information on baseline groundwater quality;
- No physical characterization of the tailings material has been done;

- Construction management and cost containment during development of the project;
- High exposure to potential escalation of costs associated with latent ground conditions due to need for dewatering dykes and large, shallow TSF;
- The permitting period associated with the project could be significantly longer than assumed in this study;
- Increased OPEX and/or CAPEX; and
- Reduced metal prices.

The following opportunities may improve the project economics:

- Metallurgical testwork has indicated that gold recoveries up to 90% are possible with a finer grind. Trade-off studies should be carried out to determine whether it's feasible to incorporate a finer grind process into the flowsheet;
- Pit optimization work with the Whittle software identified a number of potential pit shells (or phases) and the selected pit shell provides higher grades, lower strip ratio, and reduced capital and operating expense;
- Recently completed geotechnical drilling for pit slope stability analysis may increase pit slopes angles over those used in this PEA;
- There are other geophysical targets around the current resource, particularly to the southwest of the current resource. Additional drilling has the potential to add resources;
- Investigations may reveal that sufficient quantities of low permeability material for core construction may be available on-site and bedrock may be located at a shallower depth than assumed in the cost estimate;
- Lake dewatering could occur at a faster rate if the water was discharged into several different lakes;
- The potential to upgrade the mineral resource classification of the deposit; and
- Improved metal prices.

Recommendations

SRK believes the project should be taken to the next level of engineering study and economic assessment, typically a pre-feasibility study. It is estimated that a pre-feasibility, along with all of the accompanying engineering and field work would cost approximately $5M (excluding additional resource development drilling programs). Some of the activities involved to advance the project include:

- Additional metallurgical testwork;
- Initiate project permitting;
- Consummate agreements with First Nations groups; and
- Convert remaining inferred resources to indicated resources.

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1 Introduction and Terms of Reference

This report comprises a National Instrument (NI) 43-101 and Form 43-101F compliant technical report that summarizes the findings of an independent resource estimate and preliminary economic assessment of the Springpole Gold Project in northwest Ontario, Canada. SRK Consulting (Canada) Inc. prepared this report in collaboration with Gold Canyon Resources Inc. (TSX: GCU) at the request of Mr. Troy J. Fierro, CEO and director of the company, which currently owns a 100% interest in the Springpole Gold Project. SRK is not an insider, associate, or an affiliate of Gold Canyon and does not hold any interest in the Springpole Gold Project.

The contents of this report reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Accordingly, actual results may be significantly more or less favourable.

In addition, this report may include technical information that requires subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material. The estimate of mineral resources conforms to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves. These standards were prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM council on November 27, 2010. They are referred to in NI 43-101 Standards of Disclosure for Mineral Projects.

The economic analysis provides only a preliminary overview of the project economics based on broad, factored assumptions. As per CIM guidelines, reserves can only be declared with a preliminary feasibility-level study.

The mineral resources used in the life of mine (LOM) plan and economic analysis include inferred material. Inferred mineral resources are considered too speculative geologically to have economic considerations applied to them for categorization as mineral reserves, and there is no certainty the inferred resources will be upgraded to a higher resource category. Consequently, there is no certainty the results of this PEA will be realized.

This report is considered current as of March 25, 2013.

1.1 Source of Information

This report is based, in part, on internal company technical reports and maps, published government reports, company letters and memoranda, and public information (Section 27). Several sections from reports authored by other consultants are directly quoted in this report and are referenced accordingly. SRK has not conducted detailed land status evaluations and has relied upon previous qualified reports, public documents, and statements by Mr. Fierro regarding property status and legal title to the project.

In addition to the site visits described below, a qualified person carried out a study of all relevant parts of the available literature, documented results concerning the project, and held discussions regarding all pertinent aspects of the project with technical personnel from the company.

1.2 Site Visits

In accordance with NI 43-101 guidelines, Dr. Gilles Arseneau, PGeo, associate consultant with SRK, visited the project between February 10 and 12, 2012, for two days and again on August 8 and 9, 2012. An additional site visit to the Springpole property was conducted by Dino Pilotto, PEng; Dr. Maritz Rykaart, PEng; and Bruce Murphy, FSAIMM from November 27 to 29, 2012, accompanied by Jim Muntzert, senior project manager for Gold Canyon.

The purpose of these site visits was to review the digitalization of the exploration database and validation procedures, review exploration procedures, define geological modelling procedures, examine drill core, interview project personnel, and collect all relevant information needed for preparing the revised mineral resource estimate and preliminary economic assessment. The site visits also aimed at investigating the geological and structural controls on the distribution of the gold mineralization to aid the construction of three dimensional gold mineralization domains.

SRK was given full access to relevant data and conducted interviews of Gold Canyon personnel to obtain information on past exploration work and to understand procedures used to collect, record, store, and analyze historical and current exploration data. During the visits, particular attention was given to the treatment and validation of historical drilling data.

1.3 Springpole Gold Project

Abbreviations

A list of abbreviations and acronyms used throughout this report are provided in Section 26.

Units and Currency

Unless otherwise stated, all units used in this report are metric. Gold and silver assay values are reported in grams per tonne (g/t) unless some other unit is specifically stated. The currency used throughout this report is in U.S. dollars unless otherwise noted.

2 Reliance on Other Experts

The associated QP has not carried out an independent review of mineral titles, but has relied on information provided by Gold Canyon and on a legal title opinion provided by McMillan LLP, Toronto, Ontario, dated July 3, 2012.

Although copies of the tenure documents, operating licences, permits, and work contracts were reviewed, an independent verification of land title and tenure was not performed. SRK has not verified the legality of any underlying agreement(s) that may exist concerning the licences or other agreement(s) between third parties, but has relied on the clients solicitor to have conducted the proper legal due diligence. This reliance is limited to Section 3.3 of this report.

3 Property Description and Location

3.1 Project Location

The Springpole Gold Project lies approximately 110 km northeast of the Municipality of Red Lake in northwest Ontario, Canada (Figure 3.1). The property is centered on a temporary tent-based camp on a small land bridge between Springpole Lake and Birch Lake. The latitude and longitude coordinates are:

Latitude	N51° 23' 44.3"
Longitude	W92° 17' 37.4"

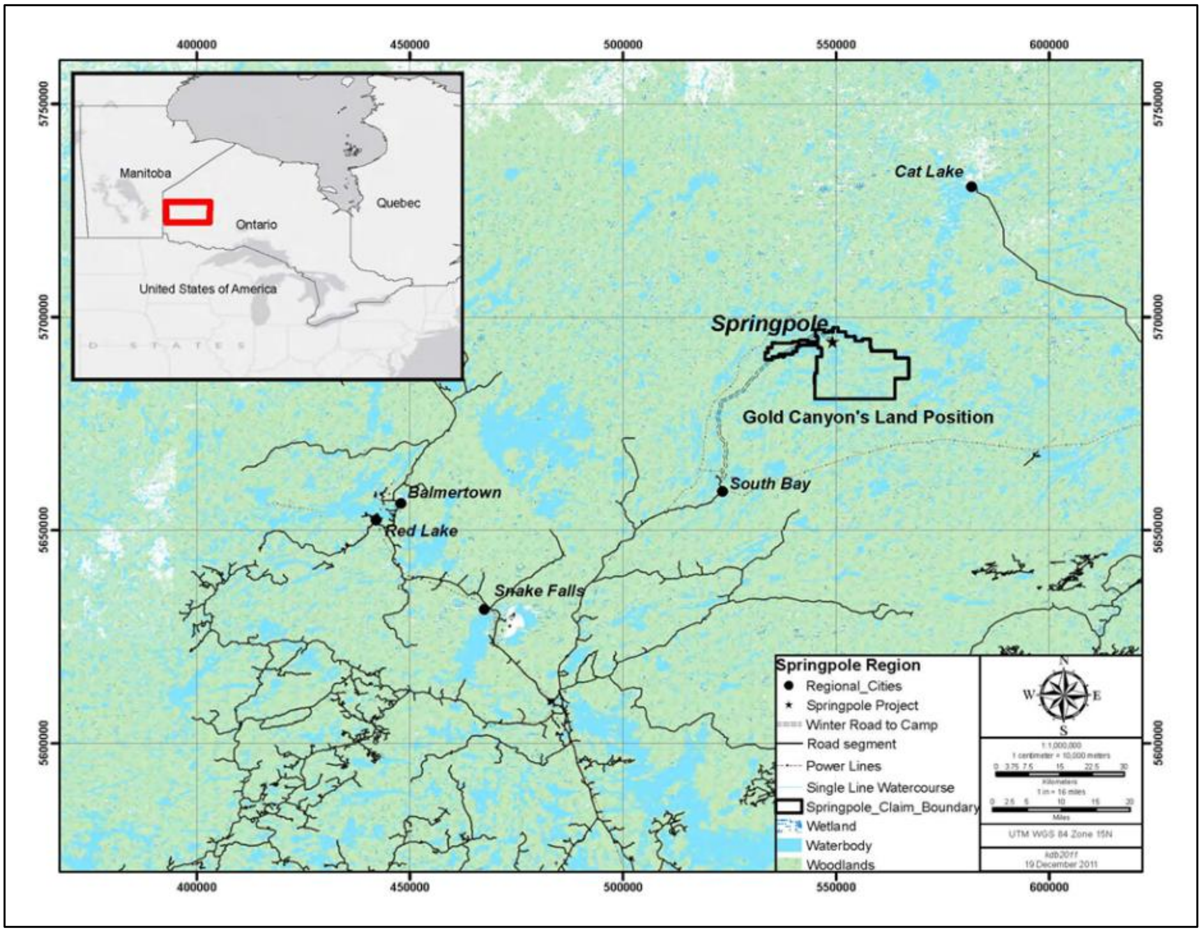
The Universal Transverse Mercator (UTM) map projection based on the World Geodetic System 1984 (WGS84) zone 15N is:

Easting	549,183
Northing	5,693,578
Average Elevation	395 m

3.2 Land Area

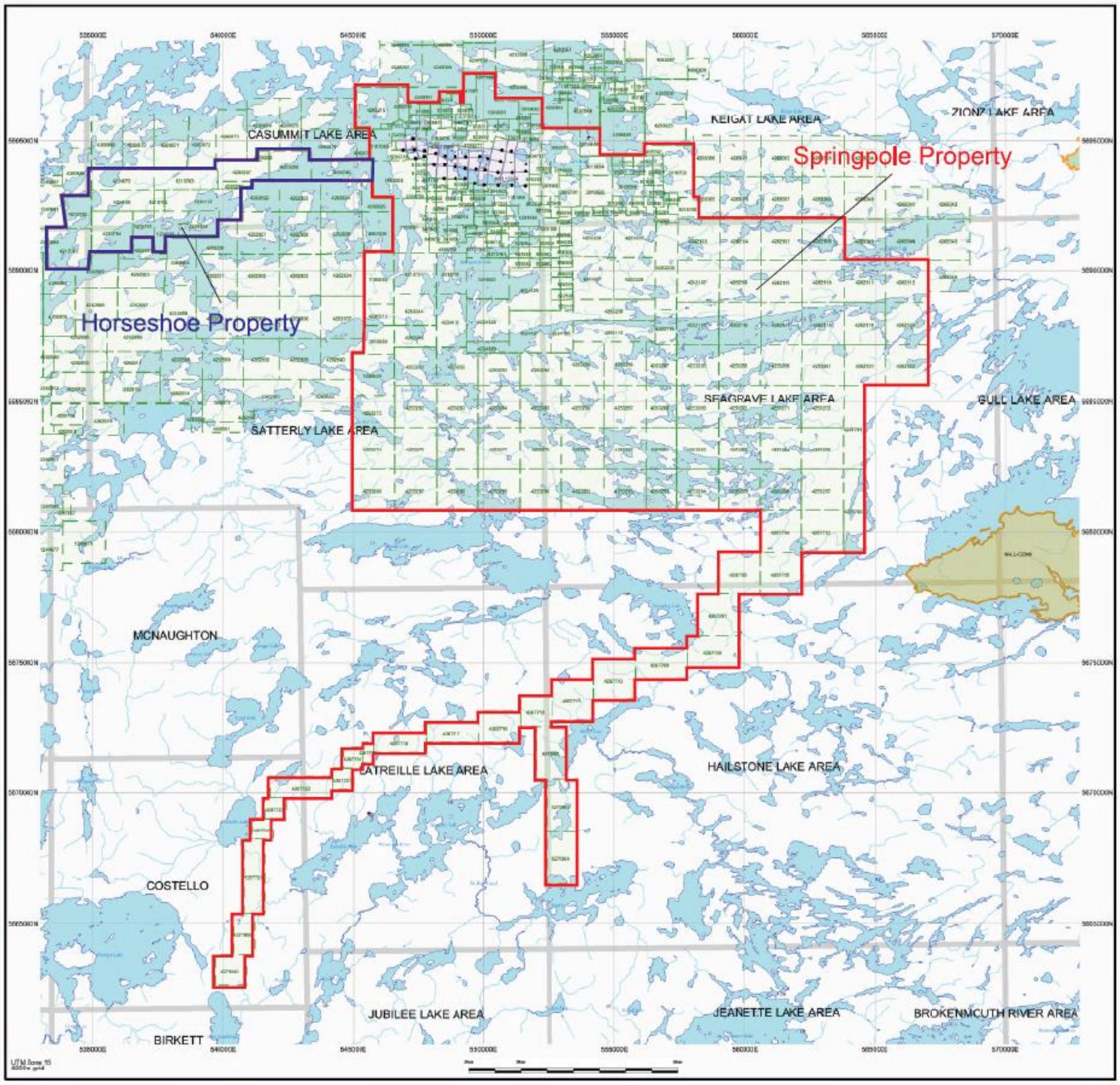
The Springpole Gold Project land area, wholly controlled by Gold Canyon, comprises 30 patented claims and 300 unpatented, contiguous mining claims and 6 leased unpatented mining claims totalling an area of approximately 32,448 hectares (80,181 acres). The overall Springpole Gold Project land area is represented in Figure 3.2.

The Ministry of Northern Development and Mines (MNDM) provides online data services that show claim status, though there is approximately a 30-day delay in reporting. As of July 3, 2012, independent legal counsel, McMillan LLP, Toronto, Ontario, confirmed that all unpatented claims controlled by Gold Canyon are in good standing and all fees are up to date, as is all required assessment work. Details can be found in Appendix A and by going to the claim information pages on the MNDM website.



Source Gold Canyon 2011

Figure 3.1: Springpole Gold Project Location Map



Source Gold Canyon 2011

Figure 3.2: Springpole Gold Project Land Tenure Map

3.3 Mineral Tenure

3.3.1 Claim Ownership

Gold Canyon acquired ownership of five patented claims in 1993 (11229, 11230, 11231, 12868, and 12869) covering a total area of 96.54 ha (238.55 acres) from Milestone Exploration Limited, a predecessor entity by way of amalgamation of Jubilee Gold Inc. These claims are subject to a 3% net smelter royalty (NSR) on all minerals mined, produced, and sold from these patented claims, provided the monthly average gold price is $700 or more. The NSR was increased to 5%, together with a NSR of 1 to 2.5% on other adjoining properties in which Gold Canyon conducted any mining operations.

In 2010, Gold Canyon renegotiated the applicable NSR on these patented claims with Jubilee This agreement terminated any applicable royalty on adjoining claims and set the applicable NSR rate payable upon commencement of commercial production at 3% with advance royalty payments of $70,000/yr, adjusted using the yearly Consumer Price Index.

Gold Canyon retained an option to acquire 1% of the NSR for $1,000,000 at any time. In consideration of the renegotiated royalties, the company agreed that previously paid advanced royalties would be forfeited and not credited to any NSR subsequently payable to Jubilee. The company paid Jubilee $50,000 and issued to it 100,000 common shares and agreed to issue a further 100,000 common shares on each anniversary date up to the fifth anniversary of TSX Venture Exchange approval of the new agreement.

Gold Canyon may terminate all royalty obligations by transfer of the patented claims back to Jubilee. Gold Canyon retains a right of first refusal on any sale of the remaining royalty interest on certain terms and conditions. The five patented claims identified above are fee simple parcels with mining and surface rights attached to all five claims registered with the Land Registry Office, Kenora, Ontario. The company has confirmed via independent legal counsel the five claims have been surveyed, are in good standing, and the property taxes are paid to date.

3.3.2 Leased Claims from Shirley Frahm

Gold Canyon leases 10 patented claims (11233-11235, 12896-12901, and 13043) covering a total area of 182.25 hectares (450.34 acres) from Shirley Frahm of Rochelle, Illinois, USA. These 10 patented claims are fee simple parcels with mining and surface rights attached to all 10 patented claims registered, together with the notices of lease, with the Land Registry Office in Kenora, Ontario. The lease is for a term of 21 years less one day and terminates on April 14, 2031. The lease stipulates Gold Canyon is to pay all applicable property taxes related to the 10 claims during the lease term together with advance royalty payments on a sliding scale of $50,000/yr (2011-2016), $60,000 (2016-2021), and $80,000 (2021-2031). These payments are to be credited to future NSR payables, if any.

A 3% NSR is payable upon commencement of commercial production. On the 10 patented claims, Gold Canyon retained an option to acquire up to 2% of the NSR for $1,000,000 per 1% at any time. Gold Canyon has the right to access the 10 claims to conduct mining operations and produce all ores, minerals, and metals that are or may be found therein or thereon— provided the company has reserved a small portion of aggregate surface area for the recreational use of Ms. Frahm.

Gold Canyon holds an option to acquire the 10 patented claims and would be required to do so upon the commencement of commercial production on these or certain adjoining patented claims, exercisable by the company within five years of date of the lease agreement. This option term is renewable for a further period of five years by providing notice and a $25,000 payment to Ms. Frahm. The consideration payable is, at the option of Gold Canyon on exercise or at the option of Ms. Frahm upon commencement of commercial production, either (a) a $5M with Ms. Frahm retaining a 1% NSR or (b) $4M with Ms. Frahm retaining a 2% NSR. Gold Canyon retains a right of first refusal on any sale of the remaining royalty interest on certain terms and conditions. The company has confirmed via independent legal counsel the 10 patented claims have been surveyed, are in good standing, and the property taxes are paid to date.

3.3.3 Lease Claims from Springpole Group

Gold Canyon has an option and lease to a further 15 patented claims (11236, 12867, 12871-12874, 12902-12909) covering a total area of 310.19 hectares (766.5 acres) from a group of individuals and/or companies collectively referred to as the "Springpole Group". These 15 patented claims are fee simple parcels with mining and surface rights attached to all 15 patented claims registered, together with the notice of option and lease, with the Land Registry Office, Kenora, Ontario. The term of the option is for five years with five renewal option periods of five years each. These options can be exercised by Gold Canyon before expiry of the earlier option period by confirmation of good standing of the agreement and payment of a $50,000 renewal fee.

Gold Canyon is required to make option payments in the aggregate amount of $35,000/yr and to expend an aggregate of C$300,000 on mining operations in each option term as a condition of any renewal and to pay all property taxes related to these patented claims. Gold Canyon is granted during the option term, the exclusive lease, the right and interest to enter upon the 15 patented claims, the right to conduct mining operations, and the right to have quiet possession thereof. The company also has the right, at its discretion to make any use or uses of the 15 patented claims consistent with the foregoing including the construction of roads, railways, conveyors, plants, buildings, and aircraft landing areas, as well as the alteration of the surface of the property subject to all applicable laws. Gold Canyon has reserved a small portion of aggregate surface area for the recreational use of a cabin by the members of the Springpole Group.

Gold Canyon holds an option to acquire the 15 claims and would be required to do so upon the commencement of commercial production at any time during the option period by payment of an aggregate of $2M. Upon exercise of the purchase option, Gold Canyon must also acquire the cabin on the property for the lesser of fair market value or $20,000. A 3% NSR is applicable during the option term upon commencement of commercial production or a 1% NSR if the purchase option is exercised prior to commercial production. Gold Canyon can acquire the remaining 1% NSR by a payment of $500,000. The company has confirmed via independent legal counsel that the 15 patented claims have been surveyed, are in good standing, and the property taxes are paid to date.

3.3.4 Claims Leased from the Crown

In Ontario, Crown Lands are available to licenced prospectors for the purposes of mineral exploration. A licenced prospector must first stake an unpatented mining claim to gain the exclusive right to prospect on Crown Land. Claims can also be staked in areas where surface rights are not owned by the Crown if the ground is open for staking and mineral rights can be obtained. Claim staking is governed by the Ontario Mining Act and is administered through the Provincial Mining Recorder and Mining Lands Consultant Office of the MNDM. A total of 300 contiguous unpatented mining claims covering approximately 31,776 ha (78,520 acres) make up the greater area of the Springpole Gold Project and have been staked directly by Gold Canyon. A list of these unpatented claims including township/area, claim number, recording date, claim due date, and status is included in Appendix A**.**

An additional six unpatented mining claims (KRL562895 to KRL562900) and related Crown leases for surface rights were acquired by Gold Canyon from an individual in July 2011 for an aggregate payment of $300,000. These claims are subject to a 3% NSR rate payable upon commencement of commercial production with advance royalty payments of $50,000/yr. Gold Canyon retained an option to acquire all or a portion of the applicable NSR at a rate of $500,000 per 1% of the NSR at any time. Gold Canyon has permitted the vendor to use a small portion of the property subject to the Crown leases, including a vacation home, for recreational purposes provided Gold Canyon was granted a 20 year option to purchase the vacation home for the price determined by an AACI valuator. The vacation home is required to be purchased upon commencement of commercial production.

Subsequent to the acquisition, the Crown leases were to expire. In consultation with the MNDM, Gold Canyon applied for the lease of these claims to be renewed for an additional 21 years, effective August 31, 2011. As of March 13, 2012, Gold Canyon has confirmed via independent legal counsel that it has complied with all the requirements for lease renewal and that payment has been received by the MNDM. Gold Canyon subsequently received the lease renewal from the Crown Lands Office.

3.3.5 Claim Maintenance

All unpatented claims are liable for inspection at any time by the MNDM and may be cancelled for irregularities or fraud in the staking process. Disputes of mining claims by third parties are accepted after one year of the recording date or after the first unit of assessment work is filed and approved. A claim remains valid as long as the claim holder properly completes and files the assessment work as required by the Mining Act and the Minister approves the assessment work.

To keep an unpatented mining claim current, the mining claim holder must perform $400 per mining claim unit worth of approved assessment work per year, immediately following the initial staking date. The claim holder has two years to file one year worth of assessment work.

Surface rights are separate from mining rights. Should any method of mining be appropriate, other than those claims for which Crown leases were issued, the surface rights would need to be secured.

3.4 Environmental Liabilities

PDAC’s Excellence in Environmental Stewardship e-toolkit (PDAC 2009) is used to ensure best practice methods are applied to mineral exploration at Springpole. Improvements to critical areas that affect the environment are underway at all times in an attempt to reduce the environmental footprint of exploration activities. No material environmental liabilities or public hazards associated with the Springpole Gold Project are known to exist on the property. A temporary camp (~0.5 ha) with wood frame tents was erected for ongoing drilling campaigns. There has been occasional surface clearing related to past drilling work.

3.5 Permits

GCU complies with permit, notice and consultation requirements as they relate to the on-going exploration work on the project. Legislation that requires material permits and notices include the provincial *Mining Act, Public Lands Act, Lakes and Rivers Improvement Act, Ontario Water Resources Act*, as well as the federal *Fisheries Act*.

Gold Canyon has initiated negotiations with surrounding First Nations communities, but to date no formal memorandum of understanding agreements have been signed.

4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

4.1 Accessibility

During late spring, summer, and early fall, the Springpole Gold Project is accessible by floatplane direct to Springpole Lake or Birch Lake. All fuel, food, and material supplies are flown in from Red Lake or Pickle Lake, Ontario, or from Winnipeg, Manitoba, with flight distances of 110 km, 167 km, and 370 km, respectively. The closest road access at present is the landing at the old South Bay Mine on Confederation Lake, approximately 50 km away by air.

During winter, an ice road approximately 85 km long is constructed from the South Bay landing point on Confederation Lake to a point about 1 km from Springpole Lake camp (Figure 3.1). During breakup in spring and freeze-up in fall, access to Springpole is by helicopter.

4.2 Local Resources and Infrastructure

There is no existing infrastructure within 50 km of the Springpole Gold Project area. Businesses in Red Lake, a long established mining community 110 km to the southwest, provide the majority of the camp's supply needs. The nearest emergency medical facilities are at the Margaret Cochenour Hospital in Red Lake.

The nearest major city is Winnipeg, Manitoba, which is approximately 370 km southwest of Springpole and about a 1.3 hour flight by Cessna Caravan.

4.3 Climate and Physiography

January temperatures range between -40°C and 0°C, and July temperatures range between 20°C and 40°C.

Springpole and Birch Lakes are part of the Albany River system, which flows eastward into Cat River and then northward into Hudson Bay. The property is underlain by glaciated terrain characteristic of a large part of the Canadian Shield. Land areas are generally of low relief with less than 30 m of local elevation and are separated by a series of interconnected, shallow lakes.

Tree cover consists of mature spruce, balsam, birch and poplar. Black spruce and muskeg swamps occupy low-lying areas. Glacial till is generally less than 1 m in thickness. Outcrops are limited and small and are generally covered by a thick layer of moss or muskeg.

Figure 4.1 displays the typical landscape of the Springpole Gold Project area. Note the drill rig working on Springpole Lake near the shore.

Figure 4.1: Typical Winter Landscape in the Project Area

5 History

The history of the Springpole Gold Project prior to 2006 is excerpted from the Technical Report and Resource Estimate on the Springpole Lake Gold Property (Armstrong et al 2006). Drill log compilation and assay data compilation have formed an important part of the work presented in this report.

Gold exploration on the property was carried out during two main periods, one during the 1920s to 1940s, and a second period from 1985 to the present.

In 1925, the discovery of gold at Red Lake brought prospectors into the Springpole Lake area. Visible gold in outcrop on the property was first discovered north of the Birch-Springpole Lake portage and prospected by Northern Aerial Mineral Exploration Ltd. in 1928 (Harding 1936). The showing was initially covered with eight claims around 1933 by prospector Tom Dunkin, who then completed the first stripping and shallow trenching in 1934.

Between 1933 and 1936, the Windigokan Sturgeon Mining Syndicate conducted extensive trenching and prospecting, including 10 short holes totalling 458.5 m (1,504 ft). The claims were then transferred to Springpole Mines Ltd. who carried out limited trenching and prospecting in 1945.

The Casey Summit Mine (later renamed the Casummit Mine), approximately 10 km to the north, started operation around this time. This mine ultimately produced 101,975 oz of gold and 9,788 oz of silver (Beakhouse 1990) and is the only significant past producer of precious metals in the Birch-Springpole Lake area.

This early prospecting activity and production from the Casummit Mine region prompted a more detailed geological investigation of the vicinity by the Ontario Department of Mines. The Birch Lake area was mapped at a scale of 1:63,360 by Harding (1936).

Reconnaissance-style mapping of the Birch-Springpole Lake area has since been repeated four times:

1. To study volcanic characteristics of selected Superior Province greenstone belts (Goodwin 1967),

2. To extend volcanic stratigraphy hosting the South Bay base metal mine into the Springpole area (Thurston et al. 1981),

3. To stimulate gold exploration in the area after closure of several mines near Red Lake (Good et al. 1988), and

4. To study the stratigraphy of epiclastic and volcaniclastic facies units, northern Birch-Uchi greenstone belt (Devaney 2001a).

The area remained dormant until 1985 when Goldfields Canadian Mining, Ltd. (GFCM) optioned the Frahm claims and, in 1986, the Milestone claims and Maple Leaf (now Springpole Group) claims. GFCM conducted an airborne (Aerodat) geophysical survey in 1985 over the entire claim group. On the 30 patented claims (Frahm, Milestone, and Springpole Group), line cutting was done at both 30.5 m (100ft) centers (Milestone claims) and 61 m (200ft) centers (Frahm and Springpole Group claimes). Subsequently, geological mapping, humus geochemistry, and ground geophysics (VLF, Mag, and IP) were conducted over the grids.

From 1986 through 1989, GFCM completed 118 diamond drill holes in seven drill phases totalling 38,349 m (125,816 ft). In addition, during 1986 and 1987, approximately 116,119 m^2 (1.25 ft^2) of mechanical stripping was carried out by the company, and four petrographic reports were produced. As a result of this work, GFCM identified several gold-bearing zones on the property that included:

- the Portage zone, entirely under the lake but the largest of the zones and, therefore, the main focus of the bulk of the exploration work;
- the Jasper zone, a deep narrow higher grade zone in a banded iron formation horizon; and
- several smaller but higher grade zones on the land portion of the property and close to surface, including the Main zone, Vein zone, Hillside zone, Camp zone, North Porphyry zone and East Extension zone.

Late in 1989, GFCM entered into a 50/50 joint venture with the combined interests of Noranda and Akiko-Lori Resources Ltd.

From 1989 through 1992 Noranda conducted an IP survey over the central portion of the Portage zone under Springpole Lake and tested the property with eighteen core holes totalling 6,195 m (20,323 ft). The majority of the drilling was conducted on the Portage zone.

At the same time, and under a separate option agreement with BP Resources Canada, Noranda completed a seven core hole drill program around the east margins of Springpole Lake on claims then owned by BP Resources. BP Resources in turn completed lake-bottom sediment sampling of Springpole Lake east of Johnson Island.

In 1992, Noranda dropped its interest in the property leaving Akiko-Lori to carry out further exploration while carrying its 50% partnership with GFCM. During 1993 and 1994, Akiko-Lori/Akiko Gold completed an additional 15 diamond drill holes on the Portage zone totalling 4,850 m (15,913 ft).

By 1995, Akiko Gold was reorganized into Gold Canyon Resources Inc. and GFCM's interest was acquired by Santa Fe Mining as part of an asset exchange with London based Hanson Plc., which controlled GFCM. During 1995, a joint venture between Gold Canyon and Santa Fe carried out an exploration program consisting of remapping of the main area, of some of the existing drill core, and a reinterpretation of the geology.

During the 1995 and 1996 programs, Santa Fe drilled an additional 69 holes totalling 15,085 m (49,492 ft) on the Springpole Gold Project proper and two drill holes on Johnson Island. By late 1996, the takeover of Santa Fe by Newmont Gold Company was nearing completion. Just prior to the merger with Newmont, Santa Fe exchanged their 50% interest in the property for a tax credit that left Gold Canyon with a 100% ownership. After Santa Fe’s departure, Gold Canyon continued exploration in 1997 and 1998 with another 51 core holes totalling 5,642 m (18,510 ft).

Paso Rico Resources Ltd. had an option to earn an interest in the Project and, in the summer of 1998, conducted with Gold Canyon a lake bottom sediment sampling program in several areas of Springpole. The results of this survey identified several follow-up targets that were tested in 1999 by Paso Rico with 12 core holes totalling 2,779 m (9,117 ft). In 2000, Paso Rico withdrew from the project leaving Gold Canyon with its current 100% interest.

During 2004, 2005, and 2006, diamond drilling programs were conducted on the property by Gold Canyon. Summaries of the drilling results are reported in Section 9 of the 2006 technical report (Armstrong et al.) and summarized in Table 5.1 below.

Table 5.1: Summary of Historic Drilling at Springpole 1986-2006

Diamond Drill Hole	Company	Period	Number of Holes	Metres drilled
BL-1 to BL124	Goldfields Canadian Mining Ltd	1986-1989	118	38,350
BL-125 to BL-141, OB-1 incl. ext 4 holes	Noranda / Akiko JV	1990-1991	18	6,167
SP-01 to SP-09	Akiko-Lori Gold Resources Ltd	1992	9	2,085
BL-142 to BL-147	Akiko Gold Resources Ltd	1993-1994	6	2,765
BL-148 to BL-216	Santa Fe Canadian / Gold Canyon Resources Inc. JV	1995-1996	69	15,085
BL-271 to BL-248 incl. 1 ext. hole	Gold Canyon Resources Inc.	1997	32	3,593
BL-249 to BL-268	Gold Canyon Resources Inc.	1998	19	2,050
BL-268 to BL-279	Paso Rico	1999	12	2,779
BL-280 to BL-304 Incl. 2 holes ext.	Gold Canyon Resources Inc.	2004	25	2,152
BL-304 to BL-320 incl 3 hole ext. BL-284D, -285D & 304D	Gold Canyon Resources Inc.	2005	19	2,983
BL06-321 to BL06-373	Gold Canyon Resources Inc.	2006	21	2,752

5.1 Fall 2007 Program

In the fall of 2007, Gold Canyon embarked on a limited exploration program to further investigate the Fluorite zone that was identified by Noranda during its trenching program in 1990. Noranda identified the potential for Ontario’s largest undeveloped fluorite deposit in the form of a Sovite (calcitic carbonatite) from four trenches and having over 850 m of strike with high grade values up to 35.6% CaF_2.

During the course of the program 46 1-m samples were collected from four “cuts” across a previously identified 23 m wide zone of fluorite mineralization at the western end of Long Skinny Pond—a thin narrow pond to the north of camp that channels water from Birch Lake to Round Pond and into Springpole Lake via a narrow stream channel.

Sampling results were inconclusive as fluorite content (CaF_2) was not analyzed. Additionally, the samples were tested for their rare earth element potential but these results also were inconclusive. Gold values were borderline anomalous and did not warrant any follow up.

5.2 Summer–Fall 2009 Program

From early August thru the end of October 2009, Gold Canyon embarked on a core re-logging and re-sampling program. Five geologists, under the supervision of Jeff Chambers, a senior consulting geologist, re-logged and re-sampled a portion of the historic drill core stored at Gold Canyon's project site and temporary tent camp.

A total of 417 diamond drill holes were completed on the Springpole Gold Project prior to 2009; drilling had begun in 1933 (Zabev 2004). This amounted to a total of approximately 98,262 m of core drilled. Unfortunately, not all the drill core is on-site. The 1933 thru 1936 drill holes 1 to 10 are missing. Also missing are drill holes BL-20 thru BL-53 completed by the GFCM exploration program from 1986to 1988. From drill log records, it appears the whole cores were sent for analysis. Drill hole BL-95A is missing—extension of BL-95 completed during the Noranda program in early 1990. In addition to missing holes, there are many intervals throughout the core inventory that are missing.

At the time the re-logging and re-sampling program was conducted, the full database of available historic core logs and historic assay data was not fully compiled and was not available to the geologists working in the field. The data used in the field were a compilation from the database that was compiled as a result of the work carried out for the previous technical report (Armstrong et al. 2006).

5.3 Core Re-Logging Program

A total of 115 drill holes were re-logged during the fall 2009 program, which equates to approximately 31% of the 374 drill holes that are believed to be on the property. Forty-nine drill holes are known to be missing, and the above count does not include the numerous mineralized intervals that are missing within drill holes that were and were not re-logged.

Core re-logging was carried out in a summary format designed to be easily incorporated into later modelling efforts. This meant drill holes were divided into broad units based upon average lithology, alteration, and mineralization. Quality of logging varied between geologists, as it was clear that a formal standard for logging was not adopted. Logging efforts were further hampered by core intervals that contained little, if any, useful material due to sampling of all or nearly all of the recovered core, as well as degradation and decay of core boxes and core racks.

The information obtained from the re-logging exercise was used to plan the phased drill program of 2010 to 2012. All re-logged core forms were scanned and now form a part of the digital database stored at the Gold Canyon's office in Vancouver, British Columbia.

At the end of the core re-logging program, several days were taken to examine drill core from critical areas. The top 6 to 12 m (20 to 40 ft) of core was examined briefly, and a simplified lithology was assigned. Overburden was excluded. The intent of the exercise was to apply the noted lithology to produce a crude geologic map. This could then be used to assess the outline geometry of the trachyte intrusive, and all the associated breccia phases.

A total of 2,580 samples were taken from the historic drill core. This included 132 standards, blanks, and duplicates, totalling approximately 5% of the number of samples collected. All samples were taken from drill core that was re-sampled by cutting the remaining drill core in half. This resulted in either a half or a quarter of the core remaining, depending on whether the interval had been sampled originally. Due to the small core diameter, core was not cut to less than one-quarter to preserve material for future reference. Table 5.2 represents significant intercepts from historic drilling combined with the re-sampling work outlined here.

At the end of the core re-sampling program, 14 samples for thin-section petrographic analysis and 3 samples for mineral petrographic analysis were collected. The samples collected were deemed representative of the principal lithologies occurring across the Springpole Gold Project.

Table 5.2: Historic Significant Intercepts from 2009 Re-Sampling Program

Main Zone					Portage Zone					East Extension & Main Zone				
Drill Hole	From	To	Interval	Au grade	Drill Hole	From	To	Interval	Au grade	Drill Hole	From	To	Interval	Au grade
	(m)	(m)	(m)	(g/t)		(m)	(m)	(m)	(g/t)		(m)	(m)	(m)	(g/t)
01	22.48	27.14	4.65	6.04	92-01	100.43	118.44	18.01	3.72	BL12	25.92	38.72	12.80	1.85
BL1	43.90	53.96	10.06	4.57	92-04	194.66	204.88	10.22	7.11	BL115	99.38	110.98	11.59	2.73
BL102	42.38	49.08	6.70	11.60	92-06	175.45	191.80	16.34	5.58	BL162	35.98	56.41	20.43	1.15
BL103	29.27	34.45	5.18	2.44	BL100	158.23	178.66	20.43	3.53	BL163	7.10	28.05	20.95	4.78
BL11	214.63	224.09	9.45	6.53	BL121	104.91	140.55	35.64	7.57	incl	16.16	28.05	11.89	7.92
BL11	295.42	317.38	21.96	1.75	BL122	163.41	241.16	77.75	1.57	BL163	88.20	102.44	14.23	2.07
BL157	60.68	62.20	1.52	206.74	incl	166.77	177.13	10.36	6.70	BL165	9.45	39.94	30.49	2.92
BL160	16.46	26.53	10.06	16.19	BL125	110.13	117.53	7.40	2.41	incl	17.98	33.23	15.24	4.38
BL161	4.48	25.61	21.13	3.61	BL125	150.74	158.74	8.00	5.68	BL166	10.36	24.39	14.02	1.42
BL183	105.80	119.82	14.02	1.33	BL126	104.33	120.13	15.80	2.60	BL168	72.26	87.50	15.24	1.60
BL190	110.06	111.28	1.22	15.70	BL127	123.53	131.73	8.20	7.07	BL172	18.14	39.63	21.49	10.44
BL197	40.30	54.60	14.30	1.54	BL128	174.04	211.65	37.61	2.13	incl	25.92	29.27	3.35	50.50
BL198	87.68	99.99	12.31	1.52	BL129	139.04	185.05	46.01	1.57	BL202	40.24	68.97	28.73	1.73
BL209	455.48	456.34	0.85	182.06	BL131	91.32	238.26	146.94	1.09	BL204	44.82	53.96	9.14	20.53
BL23	77.65	87.50	9.85	9.60	Incl	199.05	214.05	15.00	2.06	incl	45.73	47.16	1.43	136.58
incl	86.89	87.50	0.61	109.37	BL132	234.66	258.97	24.31	2.06	BL217	14.66	42.07	27.41	14.96
BL25	200.97	233.54	32.57	1.66	BL26	93.29	154.27	60.98	2.29	incl	14.66	15.24	0.58	46.18
BL264	5.18	45.73	40.55	4.56	BL308	154.76	179.27	24.51	1.29	incl	19.55	22.26	2.71	39.08
incl	5.18	13.72	8.54	7.04	BL308	214.45	321.34	106.89	2.35	incl	35.06	42.07	7.01	35.37
incl	34.39	42.56	8.17	8.96	Incl	225.06	241.49	16.43	5.81	BL220	16.25	54.52	38.26	3.54
BL280	15.85	23.14	7.28	4.59	BL310	98.08	118.54	20.46	1.77	incl	16.25	17.38	1.12	54.17
BL282D	95.70	97.41	1.70	17.84	BL310	136.62	151.53	14.91	2.91	BL221	2.13	17.34	15.21	2.92
BL285D	20.63	26.53	5.89	2.23	BL311	133.23	145.43	12.19	2.48	BL222	3.66	28.66	25.00	5.85
BL3	4.27	55.48	51.21	2.14	Incl	134.75	137.49	2.74	6.91	incl	17.38	18.76	1.38	73.03
incl	45.12	50.00	4.88	14.87	BL312	36.89	66.16	29.27	1.43	BL225	3.05	26.22	23.16	2.66
BL300	45.73	49.69	3.96	4.01	BL33	258.94	274.69	15.75	1.22	incl	21.95	26.22	4.26	12.13
BL302	27.44	31.53	4.09	3.73	BL41	110.37	134.63	24.27	2.70	227	87.95	92.98	5.03	5.25
BL303	44.82	63.94	19.12	5.00	BL41	164.63	282.92	118.29	1.64	BL228	43.29	67.84	24.55	18.63
BL305	67.56	68.90	1.34	23.87	Incl	233.84	263.11	29.27	2.92	incl	65.25	66.16	0.91	120.99
BL306	23.88	63.85	39.97	1.01	Incl	235.67	242.99	7.32	5.15	BL292	20.63	40.67	20.04	10.28
incl	33.84	38.29	4.45	4.76	BL42	101.52	127.44	25.92	1.05	incl	37.62	39.24	1.62	49.99
BL307	16.31	49.78	33.47	1.21	BL67	196.95	218.30	21.34	2.11	BL296	53.72	103.54	49.81	3.87
BL354	85.03	85.79	0.76	30.31	BL69	216.77	219.82	3.05	21.07	incl	59.40	59.84	0.45	102.00
BL356	36.89	40.91	4.02	31.67	BL79	248.78	253.35	4.57	6.09	incl	63.91	65.37	1.46	47.85
incl	39.94	40.91	0.97	127.13	BL80	448.47	460.67	12.20	2.52	incl	85.97	87.50	1.53	32.16
BL68	150.15	284.36	134.21	1.41	BL85	344.59	380.80	36.21	1.40	BL328	8.99	54.02	45.03	3.25
incl	150.15	181.16	31.01	1.88	BL88	297.52	350.91	53.38	1.97	incl	41.34	49.69	8.35	8.61
incl	217.98	243.90	25.92	2.30	BL90	65.86	76.04	10.18	3.92	BL330	33.84	34.45	0.61	16.59
BL7	50.61	68.90	18.28	1.57	BL93	169.20	178.66	9.45	2.25	BL336	173.72	174.60	0.88	14.02
BL9	25.00	37.20	12.20	4.23	BL94	264.03	271.65	7.62	2.09	BL340	25.46	39.97	14.51	15.54
incl	28.87	34.76	5.89	7.18	BL95	396.04	417.11	21.07	1.66	incl	35.34	39.97	4.63	43.64
BL96	39.63	58.71	19.08	2.89	BL99	198.47	314.33	115.86	1.53	BL343	25.70	56.13	30.43	4.33
incl	53.56	58.72	5.15	8.49						incl	25.70	29.64	3.94	27.38
BL98	39.94	73.48	33.54	1.16										

6 Geological Setting and Mineralization

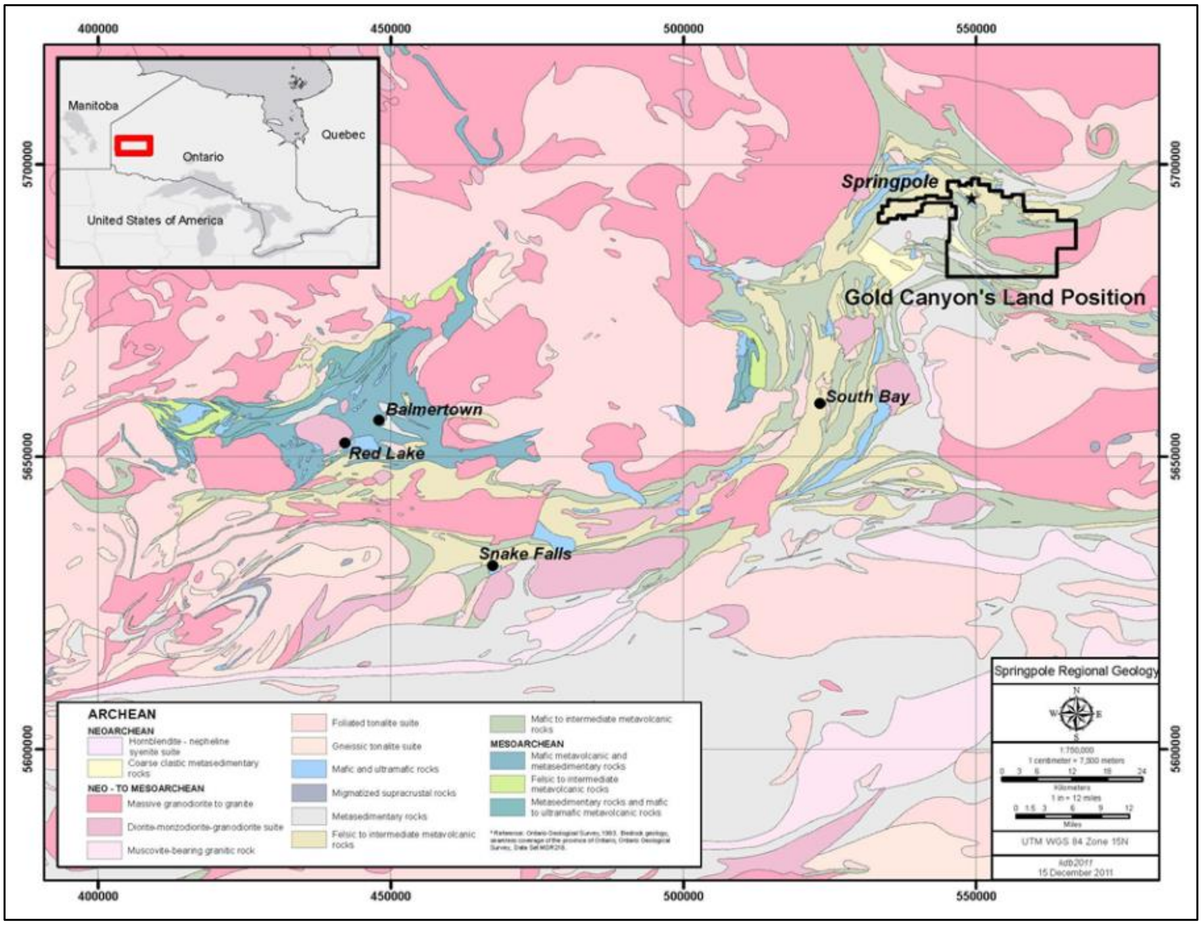
6.1 Regional Geology

The following excerpt is quoted from Devaney (2001b) and provides the most concise geologic description of the regional geology of the Springpole-Birch Lake area:

> *The Birch-Uchi Greenstone Belt (*Figure 6.1*) is the portion of the Uchi Sub-province with an arcuate, concave to the southeast, (i.e., a major oroclinal bend between the Red Lake and Meen-Dempster portions of the sub-province). Studies of the southern part of the Birch-Uchi greenstone belt as a rootless greenstone belt only a few kilometres thick, have revealed a long (ca. 3.0 to 2.7 Ga), multistage history of crustal development. Based on mapping, lithogeochemistry, and radiometric dating, the supracrustal rocks of the greenstone belt were subdivided into three stratigraphic group-scale units (listed in decreasing age): the Balmer, Woman and Confederation assemblages. This three-part subdivision was applied to most of the Uchi Subprovince. The Confederation assemblage is thought to be a continental margin (Andean-type) arc succession, versus the less certain tectono-stratigraphic context of the other assemblages. Workers performing recent and ongoing studies of the southern Birch-Uchi greenstone belt and the Red Lake greenstone belt (i.e., the Western Uchi Subprovince NATMAP Project) have proposed some modifications and additions to the Balmer-Woman-Confederation stratigraphic scheme. As discussed herein, some relatively small conglomeratic units likely form a synorogenic, discontinuously distributed, post-Confederation assemblage in the Birch-Uchi greenstone belt. Radiometrically dated plutons within the Birch-Uchi greenstone belt are of post-Confederation assemblage, ca. 2725-2700 Ma age.*
>
> *The northern margin of the Birch-Uchi greenstone belt forms a pattern of sub-regional scale cusps of supracrustal strata alternating with batholiths. Basaltic units are prominent around the periphery of the greenstone belt and may be part of the Woman assemblage but the accuracy of this stratigraphic assignment is unknown. Based on a ca. 2740 Ma age of Shabumeni Lake [intermediate to felsic fragmental] volcanic rocks at a site near the northern greenstone belt margin, suggested that Confederation assemblage age rocks make up the bulk of the greenstone belt.*

It is noteworthy that in many of the regional geology descriptions of the Birch-Uchi Greenstone Belt, especially those in the vicinity of Springpole and Birch Lakes, the structural geology is poorly understood. Many authors make relatively brief mention of the complexities that dominate the geology and geomorphology of the low lying areas. However, the Archean Orogenic gold deposit model developed by various authors has been applied to the mineral deposits of the Archean Superior Province. Recent concise summaries of these orogenic gold deposits can be found Groves et al. (1998), Hagemann and Cassidy (2000), Goldfarb et al. (2005), and Robert et al.(2005).

Orogenic gold deposits are epigenetic, structurally controlled gold deposits that are hosted in orogenic belts. They are generally accepted as having formed during late stages of continental collision. Most of the discovered orogenic gold deposits in the world occur in greenstone belts situated on the margins or within Archean cratons in North America, Australia, and southern Africa.



(Source: Ontario Geological Survey 2000)

Figure 6.1: Springpole Gold Project – Regional Geology

6.2 Property Geology

The Springpole prospect has been extensively studied during past programs and the findings of those studies will not be covered in detail here; however, they are adequately covered in the technical reports of Zabev (2004) and Armstrong et al. (2006).

The following subsections summarize the geology interpreted from field observations and petrographic analysis of drill core from the 2009 re-logging program and from drill core produced during the 2010 and 2011 programs. Simplified drill hole geology from a number of selected sections can be found in Appendix B.

6.2.1 Trachyte Porphyry Intrusive

A polyphase alkali, trachyte intrusive displaying autolithic breccia textures lies at the heart of the Springpole Gold Project. The intrusive is comprised of a system of multiple phases of trachyte believed to be part of the roof zone of a larger syenite intrusive, as fragments displaying phaneritic textures were observed from deeper drill cores in the southeast portion of the Portage zone. Early intrusive phases consist of megacrystic feldspar phenocrysts, up to 5 cm long, of albite and orthoclase feldspar in an aphanitic groundmass. Successive phases show progressively finer grained porphyritic texture while the final intrusive phases are aphanitic.

In 2009 and 2010, Gold Canyon carried out petrographic studies (Saunders and McIntosh 2009, 2010) of historic drill core and drill core from the drill holes SP10-001 through SP11-006. The study confirmed trachyte intrusive is the dominant lithology within the project area and is a host to mineralization. Interpretation of the intrusive complex is complicated by a mixture of overprinted regional and local metamorphic events related to burial and tectonism Pervasive alteration and metamorphism have reduced the original porphyry intrusive to a complex alteration assemblage dominated by sericite, biotite, pyrite, calcite/dolomite, and quartz. Primary igneous textures are remarkably well preserved in places and give indications to the possible genesis of the initial phase of gold mineralization. Within the country rocks to the north and east are trachyte and lamprophyre dikes and sills that source from the trachyte - or syenite-porphyry intrusive system.

6.2.2 Confederation Age Volcanic and Siliciclastic Rocks

The country rocks pre-date the alkali intrusive and are composed of a complex sequence of altered and metamorphosed intermediate andesitic volcanic rocks and associated volcaniclastics, siliciclastic sedimentary rocks, chemical sediments including banded iron formation (BIF), and coarse pebble conglomerates. Devaney (2001a) indicates that the sediments are likely of the Confederation assemblage dating at around 2,740 Ma, representing the proximal portions of a mixed volcanic-sedimentary basin.

6.2.3 “Timiskaming-type” Conglomerates

Barron (1996) states pebble conglomerate outcrops between Springpole Lake and Birch Lake contain clasts of the trachyte porphyry, suggesting that the “Timiskaming-type” conglomerates postdate intrusion. Devaney (2001a) suggests these arcuate form conglomerates represent late orogenic, deformed, dextral sense strike-slip (pull-apart) basins of “Timiskaming-type,” late Archean, post Confederation assemblage age rocks.

6.3 Structure

Deformation has added complexity to the apparent geometry of, and the potential of, the Springpole gold deposit. Gravity and magnetic surveys carried out across the Springpole Gold Project demonstrate that several phases of deformation are evident. Banded iron formations describe north-northwest facing tight to isoclinal antiforms and synforms, and are illustrated on the property geologic map produced during the Summer 2005 Mapping Program (Armstrong et al. 2006) and are evident as strong magnetic anomalies on the aeromagnetic surveys conducted by Fugro.

In 2011, SRK was contracted to carry out a preliminary study of the structural controls on mineralized deposit geometry. The study found the deposit was subjected to several deformational events including, but not limited to:

- Early folding resulting in tight to isoclinal fold geometries and development of associated shear zones,
- Intermediate large scale, potentially deep rooted shear zones, and
- Late stage brittle faulting.

Further study is required to definitively establish the relationship of the timing of deformational events with respect to economic mineralization.

6.4 Alteration

All rocks on the property exhibit pervasive alteration that consists of multiple overprinted phases. Distinguishing between the individual phases will take considerable study on a microscopic scale. The country rocks and alkali intrusive rocks exhibit pervasive green-schist facies metamorphism and alteration, probably the result of burial. This manifests as chlorite, calcite, and pyrite in the intermediate volcanic rocks, pyritization of the banded iron formation, and sericite-pyrite alteration within the alkali intrusive associated rocks.

Studies conducted as a part of the exploration work carried out from the fall of 2009 and the winter/spring of 2010 show there is evidence of early alteration phases. These probably resulted from magmatic hydrothermal fluids associated with porphyry gold mineralization and the associated epithermal/mesothermal style gold mineralization. This occurs as potassic and phyllic/sericitic alteration: K-feldspar, biotite, and muscovite (sericite), respectively, and is nearly pervasive in the alkali intrusive rocks and surrounding country rocks. Regional metamorphism has subsequently altered the primary hydrothermal mineral assemblages, but textures have been preserved with the exception of areas of high strain (e.g., northwest trending shear zones).

Advanced argillic alteration appears throughout the trachyte intrusive and occurs in some of the late stage lamprophyre dikes though on a small scale. It is difficult to assess at what stage argillic alteration occurs, but it appears to define an envelope around the Portage zone potassic-alteration/mineralization, suggesting an origin more in keeping with zoned alteration associated with epithermal-style porphyry intrusive hosted gold deposits.

6.5 Mineralization

6.5.1 Porphyry Style Mineralization

The main intrusive complex appears to contain many of the characteristics of alkaline, porphyry-style mineralization associated with diatreme breccias (e.g., Cripple Creek, Colorado). Direct comparison with drill core from the two sites shows a number of consistent textures and styles of mineralization. A recent observation made from drilling, combined with the airborne magnetic survey, shows the potentially economic gold mineralization is coincident with an unexplained geophysical anomaly. This style of mineralization is characterized by the Portage zone and portions of the East Extension zone where mineralization is hosted by diatreme breccia in aphanitic trachyte. It is suspected the ductile shearing and brittle faulting have played a significant role in redistributing structurally controlled blocks of the mineralized rock. Yet confirmed is a form of porphyry style alteration zoning consisting of an outer zone of phyllic (sericite) dominant alteration with narrow zones of advanced argillic alteration characterized by illite and kaolinite, and a core zone of intense potassic alteration characterized by biotite and K-feldspar.

Multi-element analysis conducted during the 1992 program on the Portage zone, combined with gold assays, gave the first indication of the style of mineralization at Springpole. Diamond drilling in the winter of 2010 revealed a more complex alteration with broader, intense zones of potassic alteration replacing the original rock mass with biotite and pyrite. The expected alteration zone envelopes or shells are very difficult to define due to complex sheared geometry and poorly defined contact zones of the deposit. In the core area of the deposit where fine grained disseminated gold mineralization occurs with biotite, the primary potassic alteration mineral, gold, displays a good correlation with potassium/rubidium.

6.5.2 Lode Gold Mineralization

The intrusion of the trachyte complex into the volcanic pile, as well as the chemical and siliciclastic sedimentary rocks in a near surface environment, produced mesothermal to epithermal style lode vein mineralization. The difference between mesothermal and epithermal mineralization regimes is the temperature and pressure of the mineralizing fluids.

Higher temperature (mesothermal) fluids would have existed within the emplaced intrusive, associated with the diatreme breccias, and in the immediately adjacent wall rock/country rocks. In the porphyry intrusive, and at the contact between intrusive and wall rock in the East Extension zone, and localized within the Main zone, mesothermal style quartz-biotite-calcite-sulfide veins with occasional tourmaline are observed with occasional coarse, visible gold.

Further from the intrusive complex and wall rock contact zones, where meteoric fluids have a greater influence, epithermal style vein textures and mineralization styles dominate. These consist of banded to sucrosic quartz-calcite veins with a lower temperature mineral assemblage including sericite, minor biotite, possible adularia, calcite, dolomite and ankerite; here gold-silver and tellurium alloys dominate including electrum and gold-silver tellurides.

6.5.3 Gold Remobilization during Metamorphism

As evidenced from the high degree of deformation, both ductile and brittle—in the form of isoclinal folding, ductile shear zones with protomylonite and blastomylonite textures, and brittle fault textures— the Springpole Prospect has been subjected to alteration and metamorphism. These processes alone have remobilized gold in epithermal quartz veins that were the principal motivation for exploring Springpole in the late 1980s and early 1990s, when shear zone hosted gold deposits were the targets of choice in the Red Lake area.

7 Deposit Types

Mineralization at the Springpole Gold Project is dominated by large tonnage, low grade disseminated porphyry-style or epithermal-style gold mineralization associated with the emplacement of the alkali trachyte intrusive. Textures observed in the extensive repository of drill core appear to confirm the disseminated gold-silver-sulfide mineralization, the mesothermal to epithermal lode vein gold mineralization, and branded iron-formation hosted gold mineralization are all the result of the emplacement of multiple phases of trachyte porphyry and associated diatreme breccias, hydrothermal breccias, dikes and sills.

The initial exploration on the property was conducted on the assumption the mineralization was a typical example of Archean mesothermal, sulfide-hosted lode gold type. While this model has not been completely ruled out, it has been replaced in favor of a high level emplacement porphyry model. Barron's thesis (1996) work presented strong evidence the gold and associated fluorite mineralization at Springpole are genetically related to the high level emplacement of a large, alkaline porphyry intrusive and breccia pipe complex.

Barron considered the Springpole Complex to be the end product of magmatic fractionation processes and of fluids that evolved from magmatic to hydrothermal in the high level, sub-volcanic porphyry environment. These processes produced a low grade gold-porphyry-epithermal type deposit and associated high-grade veins and breccia pipes.

Santa Fe geologists felt the nature of the mineralization at Springpole had many similarities with deposits of the Cripple Creek District, Colorado, including the Cresson Mine. Detailed mapping on the land based portions of the property by Santa Fe geologists showed that most, if not all, of the gold mineralization on the Springpole Gold Project is spatially associated with the feldspar porphyry diatreme dikes, veins, and diatreme breccia. The following is a brief description of this model in the Springpole area.

7.1 Depositional Environment

Based upon the abundance and size of epizonal trachyte porphyry intrusive masses and the widespread brecciation and alteration centered on the Portage zone, Barron (1996) considered this area to be the apex of a buried syenite stock. A high emplacement level for the Portage zone and surrounding porphyry is further supported by the lack of contact metamorphic effects in the enclosing country rocks. Trachyte clasts within the basal conglomerate overlying the intrusive complex indicate it was subjected to surface erosion.

The rarity of trachyte clasts and their restriction to the base of the conglomerate unit would seem to indicate erosion over a short time interval. The lack of voluminous trachyte flows suggests there was no markedly positive volcanic edifice. Barron (1996) concluded that collectively these features suggested that the Portage zone and surrounding Main and East Extension zones existed as a small island of maar craters of low relief in a rapidly deepening shallow basin.

This interpretation has its closest modern analogue in the Ladolam Gold Deposit, Lihir Island, Papua New Guinea. Mineralization at Lihir is believed to be less than 500,000 years old and is telescoped upon an earlier porphyry environment (Carman 2003) Deposition of gold is still an active process at

Ladolam as the hydrothermal system remains active. Host rocks at Ladolam can be divided into three groups (Carman 2003):

- Mafic lavas composed of alkali basalt, porphyritic trachybasalt, trachyandesite, and rare trachyte and phonolite;
- Alkali intrusions that are composed of multi-phase porphyry stocks with the most voluminous phase being biotite monzonite; and
- Ladolam Breccia Complex that is composed of porphyry breccias and volcanic breccias.

Porphyry breccias are dominantly monzonite composition and occur as poorly sorted, massive, matrix supported breccias with some rounding of clasts caused by magmatic milling; the clasts are supported by a cement of altered rock flour and anhydrite. The volcanic breccias are massive, moderately to poorly sorted, rock flour matrix supported breccias containing mafic clasts.

Mineralization/alteration at Ladolam can also be sub-divided into three broad phases:

- Biotite-orthoclase-anhydrite ± magnetite with minor copper-gold-molybdenum disseminated porphyry mineralization and veinlets;
- Refractory sulfide-gold mineralization associated with pervasive adularia-pyrite (leucoxene-illite) alteration near surfacethat comprises the bulk of the near surface bulk mineable mineralized material; and
- Quartz-calcite-adularia-pyrite-marcasite ±electrum stockwork veins.

If the Ladolam Gold deposit is accepted as a reasonable genetic analogue to the Springpole deposit, then the following genetic model can be applied. This model is adapted from Barron’s thesis (1996), Zabev’s genetic summary (2004), and the genetic model of Armstrong et al. (2006), as well as observations made during the 2009 through 2012 diamond drilling programs.

7.1.1 Springpole Genetic Model

The follow list summarizes the genetic model of Springpole Gold Project area:

- Intrusion into the lower crust of parental alkaline primitive and anhydrous magma slightly enriched in incompatible elements including fluorine.
- Fractionation at depth, precipitation of hornblende and apatite as early crystalline phases. The magma becomes increasingly anhydrous. Gold is retained in the melt.
- Diapiric uprise from 4 to 8 km levels into hydrous wall rock with the apex of the magma chamber at <2 km depth. Continued fractionation producing an increasingly fluorine-rich melt. Feldspar of extreme composition is precipitated and the lowered solidus allows emplacement of porphyry dykes and sills to very high crustal levels.
- High diffusivities and convection promotes water partitioning from wall rock into magma.

- The magma is quickly saturated and the sudden pressure is released (possibly from venting) prompting the immiscible separation of fluorine and carbon dioxide-rich phases, which escapes to high structural levels. Breccia pipes with rock fluorite and rounded clasts indicating turbulent fluidized and erosional vertical emplacement.

- Fluid pressures generate dyke offshoots.

- Fluorine escapes from brecciated wall-rock causing biotization or fluoritization of breccia and wall rock. Ultimately, the fluorine-water-carbon dioxide vapors condense, resulting in the precipitation of fluorite and calcite. Magmatic gold-rich fluids permeate the breccia and surrounding porphyry, depositing porphyry style, disseminated, pyritic mineralization. The fractures along the margins of breccia pipes acts as preferred sites for later deposition of quartz, electrum and tellurides.

- Intrusion of a series of lamprophyre and carbonatite dikes, sills and veinlets—due to the intensity of deformation.

- The complex is then buried by conglomerates derived from the complex and other areas (Devaney 2001b).

- Continued intense deformation and associated metamorphism manifesting as folding, strike-slip faulting and shearing, coupled with regional green schist metamorphism of the region obscures primary textures and likely leads to some (possibly minor) degree of precious metal remobilization.

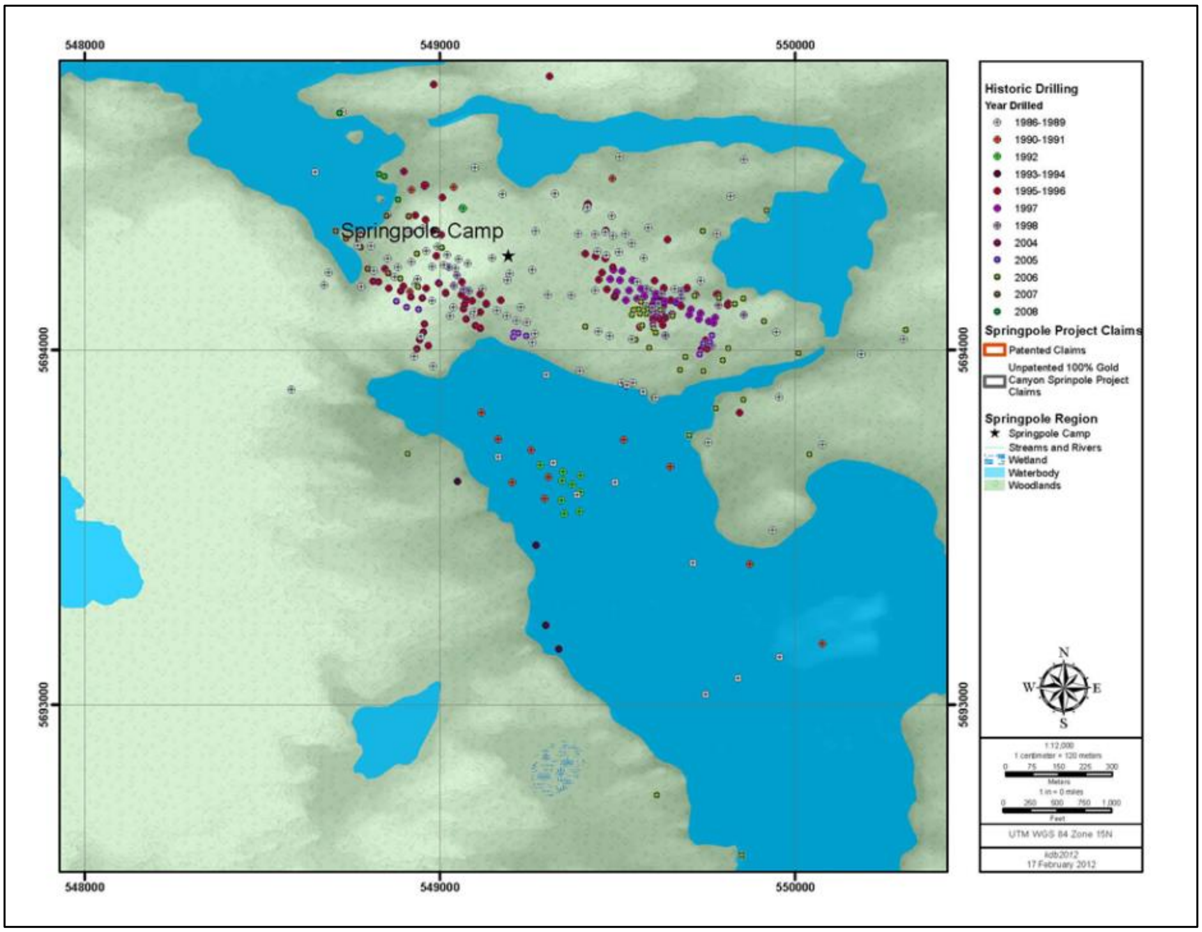
8 Exploration

Current exploration work on the property consists mainly of an ongoing drilling program carried out by Gold Canyon and discussed in detail in the following section of the report.

9 Drilling

9.1 Gold Canyon Drilling

During the winters of 2007 and 2008 Gold Canyon conducted drill programs that completed 21 holes totalling 3,159 m, 11 holes totalling 2,122 m, and 7 holes totalling 2,452 m of diamond core drilling, respectively (Figure 9.1). The details of the exploration work carried out are covered in Gold Canyon's internal Winter Drilling Report 2006-2007 (Smith 2008a) and Winter Drilling Report 2008 (Smith 2008b).



(Source Gold Canyon, 2011)

Figure 9.1: Springpole Gold Project Historical 2007 and 2008 Drill Hole Collar Location Map

9.2 2007 Diamond Drilling Program

During the winter of 2007 Gold Canyon conducted an 11 diamond drill hole program that totalled 2,122 m of drilling. Table 9.1 summarizes drill hole collar information and significant results of the 2007 diamond drill program are summarized in Table 9.2.

Table 9.1: Summary Data of 2007 Winter Diamond Drill Program

Hole ID	Azimuth	Dip	Length (m)	Easting* (m)	Northing* (m)	Elevation (m)
BL-07-374	180°	-45°	200.0	549,170	5,692,280	405.7
BL-07-375	180°	-45°	200.0	549,425	5,692,330	402.8
BL-07-376	180°	-45°	113.0	549,427	5,692,190	401.5
BL-07-377	180°	-45°	194.4	549,653	5,692,406	400.7
BL-07-378	230°	-45°	149.0	548,868	5,693,995	405.0
BL-07-379	230°	-45°	200.0	548,810	5,694,006	402.3
BL-07-380	230°	-45°	196.2	548,789	5,694,068	398.9
BL-07-381	230°	-45°	194.0	548,748	5,694,092	398.4
BL-07-382	240°	-45°	251.0	548,720	5,694,114	398.3
BL-07-383	240°	-45°	203.0	548,863	5,694,156	399.5
BL-07-384	230°	-45°	221.0	548,925	5,694,155	404.1
Total			2122			

* World Geodetic System 1984 (WGS84) converted from NAD27 original handheld GPS survey.

Table 9.2: Summary of Significant Drill Results from 2007 Drill Program

Hole ID	From (m)	To (m)	Interval (m)	Au (g/t)	Au (oz/t)
BL-07-374	93.33	95.00	1.07	0.41	0.012
	163.00	167.00	4.00	0.69	0.02
BL-07-375	110.55	111.24	0.69	2.32	0.068
BL-07-376	29.20	29.93	0.73	2.44	0.071
BL-07-377	105.45	105.95	0.50	3.16	0.092
	148.12	152.00	3.88	1.08	0.031
BL-07-378	89.62	90.16	0.54	19.32	0.564
	114.22	116.00	1.78	2.85	0.083
BL-07-379	56.89	57.26	0.37	14.07	0.410
	60.81	61.10	0.29	5.65	0.165
	107.00	107.51	0.51	2.13	0.062
	117.26	117.76	0.50	2.21	0.065
BL-07-380	116.05	116.61	0.56	1.05	0.031
	138.00	138.42	0.42	4.19	0.122
BL-07-383	42.00	47.26	5.26	9.79	0.286
BL-07-384	80.54	81.54	1.00	1.52	0.044
	149.36	149.91	0.55	2.85	0.083

9.3 Winter 2008 Drill Program

The winter 2008 program comprised seven core holes totalling 2,452 m and was designed to focus on step-out drilling to test the strike and down-dip potential of the new sedimentary hosted, semi-massive sulfide environment. The first 1 km of strike potential for the sedimentary hosted semi-massive sulfide environment has now been tested at a vertical depth of between 100 and 200 m. The results of the 2008 drilling program were inconclusive and did not return any gold intersections comparable to BL07-383. The sedimentary hosted gold target horizon is believed to continue for at least 7 additional km beyond the area tested.

Table 9.3 summarizes the 2008 drilling program and Table 9.4 summarizes the significant intersections from the drilling campaign.

Table 9.3: Winter 2008 Diamond Drill Hole Program Summary

Hole ID	Azimuth	Dip	Length (m)	Easting* (m)	Northing* (m)	Elevation (m)
BL08-385	240°	-45°	208.00	548,895	5,694,201	400.0
BL08-386	215°	-45°	272.00	548,856	5,694,267	400.0
BL08-387	215°	-45°	395.00	548,841	5,694,273	400.0
BL08-388	215°	-60°	356.00	548,841	5,694,273	400.0
BL08-389	258°	-45°	356.00	548,841	5,694,273	400.0
BL08-390	268°	-45°	446.00	548,841	5,694,273	400.0
BL08-391	240°	-45°	419.00	548,730	5,694,446	400.0

***Note:** *Universal Transverse Mercator (UTM) datum projection is North American Datum 1927 (NAD27)*

Table 9.4: Significant Drill Intersections from 2008 Drilling Program

Hole ID	From (m)	To (m)	Interval (m)	Au (g/t)	Au (oz/t)
BL08-385	74.00	75.56	1.56	3.28	0.10
	167.39	168.39	1.00	2.37	0.07
BL08-386	99.28	100.25	0.97	2.53	0.08
	222.44	223.24	0.80	13.17	0.38
BL08-387	193.06	194.00	0.94	1.59	0.05
	292.71	296.67	3.96	1.63	0.05
BL08-389	167.00	168.23	1.23	2.04	0.06
	207.00	207.93	0.93	1.78	0.06
	305.92	307.59	1.67	1.47	0.04
	345.50	346.52	1.02	5.98	0.17

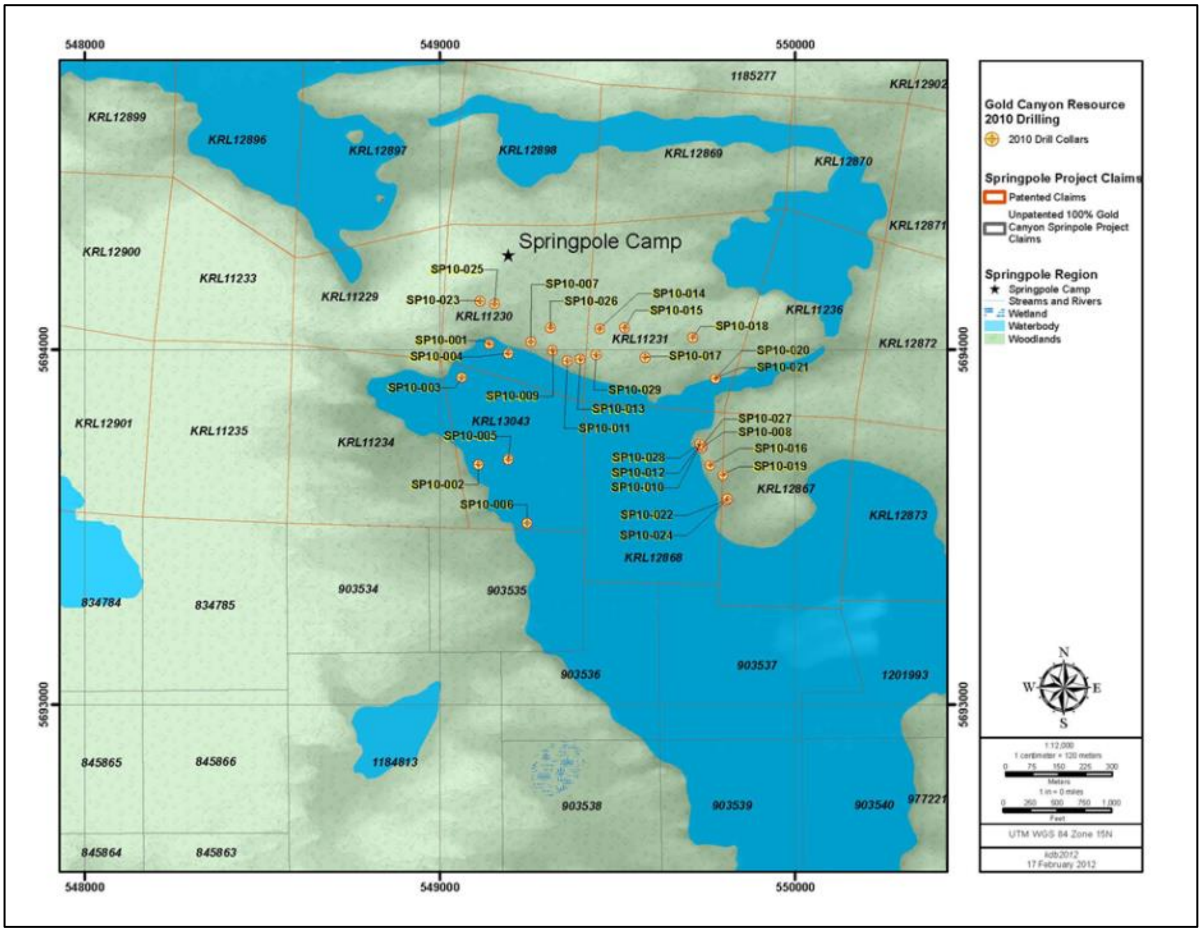
9.4 2010 Drill Program

Winter 2010 drilling operations began on February 17th with mobilization of two Longyear 38 drills from Boart-Longyear International's (BLI) base in Red Lake. Drilling commenced on February 23, 2010. A total of six diamond drill holes (SP10-001 thru SP10-006) were drilled for a total of 1,774.5 m of HQ drilling (Figure 9.2). A summary of the 2010 drilling can be found in Table 9.5.

BLI pulled out of the drill program and demobilized the drills on March 10, 2010, citing critical ice thickness problems with the access ice road to Springpole camp from the South Bay Mine landing. In doing so, BLI failed to complete drill holes SP10-005 and SP10-006, and both holes ended in altered and mineralized rock. Significant intercepts of the 2010 drill program are listed in

Table 9.6.

Drilling was suspended during the ice break-up on Springpole Lake and Birch Lake as the project has no land access route. Rodren Drilling Ltd of Winnipeg, Manitoba, was awarded the drilling contract in spring 2010 and mobilization of two Boyles 37 drills to the project site by helicopter began in June 2010. Drilling commenced on July 5, 2010, and ended on October 17, 2010. A total of 8,664.2 m of HQ core drilling was completed in 23 drill holes, averaging 44.23 m of drilling per 24-hour shift, including time for moving the drill between drill sites.



(Source Gold Canyon, 2011)

Figure 9.2: Springpole Gold Project - 2010 Drill Hole Collar Location Map

Table 9.5: 2010 Diamond Drill Program Summary Data

Hole ID	Azimuth	Dip	Length (m)	Easting* (m)	Northing* (m)	Elevation (m)
SP10-001	220	-45	252	549140.1	5694017	388.7
SP10-002	40	-45	392	549109.1	5693677	395
SP10-003	40	-45	225	549062.1	5693922	389.7
SP10-004	220	-45	274.5	549192.1	5693990	384.6
SP10-005	40	-59	268	549193.1	5693691	386
SP10-006	40	-45	363	549246.1	5693512	386
SP10-007	220	-45	252	549256.4	5694022	396.11
SP10-008	231	-45	451	549739.1	5693725	397
SP10-009	220	-45	322	549318.1	5693998	390
SP10-010	242	-45	317	549732.5	5693733	392.32
SP10-011	220	-45	328	549359.1	5693969	390
SP10-012	226	-45	431	549731.6	5693734	392.32
SP10-013	54	-45	313	549396.1	5693974	393
SP10-014	36	-45	262	549450.1	5694059	402
SP10-015	40	-45	272	549521.1	5694062	402
SP10-016	225	-45	511	549761.8	5693676	394.54
SP10-017	35	-45	298	549578.1	5693979	407
SP10-018	38	-50	226	549713.1	5694035	400
SP10-019	220	-45	490	549797.2	5693648	392.11
SP10-020	35	-45	349	549777.1	5693920	389
SP10-021	220	-45	502.2	549777.1	5693920	391
SP10-022	220	-45	396	549807.4	5693576	391.63
SP10-023	220	-45	454	549112.9	5694136	397.68
SP10-024	220	-45	505	549810.8	5693580	391.08
SP10-025	220	-45	430	549154.1	5694129	398.94
SP10-026	220	-45	466	549312.1	5694063	400
SP10-027	240	-45	115	549739.1	5693730	396.9
SP10-028	245	-45	475	549732	5693735	392.4
SP10-029	222	-45	499	549440.1	5693986	400

*Universal Transverse Mercator (UTM): World Geodetic System 1984 (WGS84) projection

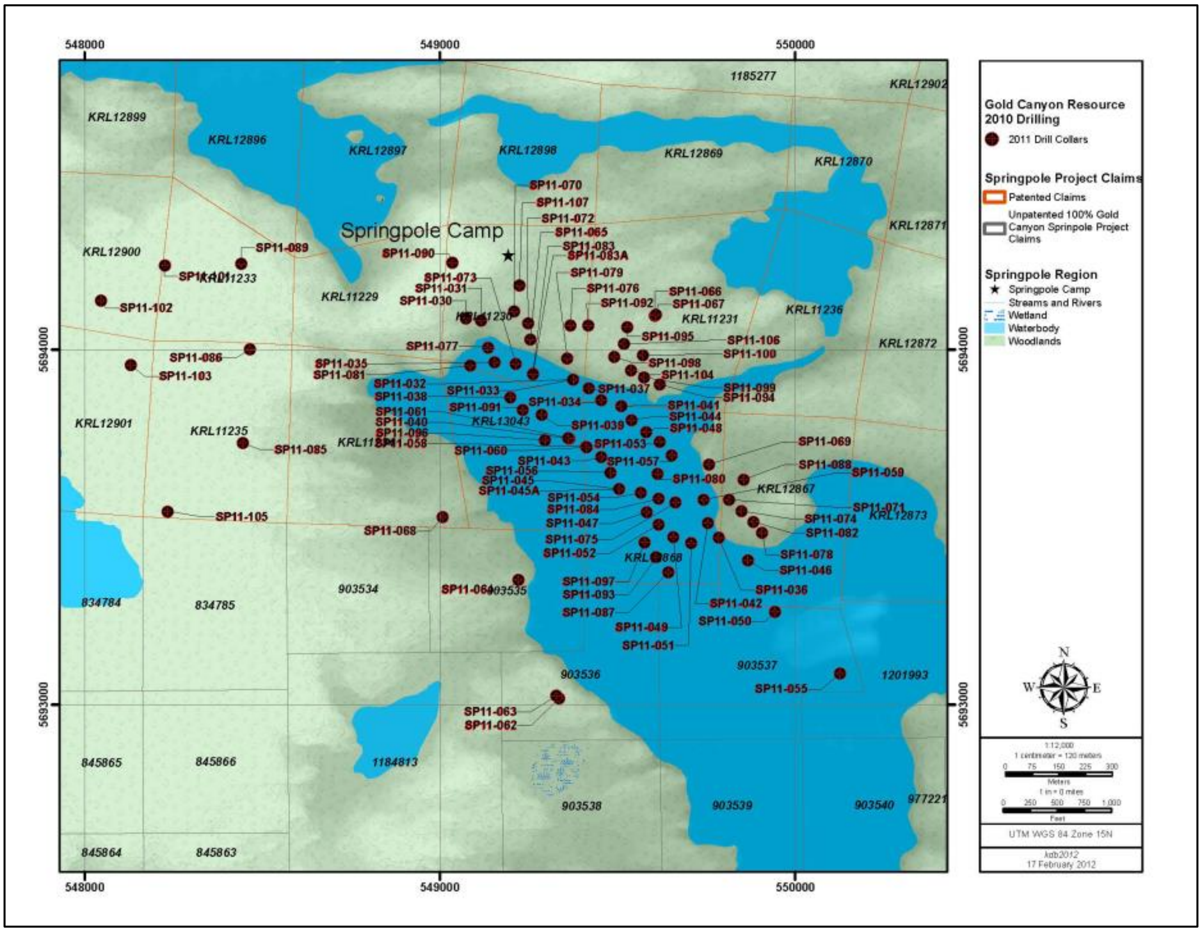
Table 9.6: Summary of Significant Gold and Silver Assays from 2010 Drill Holes

Hole ID	From (m)	To (m)	Interval (m)	Au (g/t)	Au (oz/t)
SP 10-001	12.5	64	51.5	0.93	0.027
SP 10-002	242	335	93	2.4	0.07
SP 10-004	31	182	151	0.72	0.021
SP 10-006	278	363	85	0.93	0.027
SP 10-007	33	250	217	1.57	0.046
SP 10-008	257	451	194	1.22	0.036
SP 10-009	3	167	164	2.68	0.030
SP 10-011	229	323	94	2.51	0.073
SP 10-012	275	408	133	-.79	0.023
SP 10-016	206	511	305	1.03	0.030
SP 10-019	182	489	307	1.44	0.042
SP 10-024	166	391	225	1.48	0.043
SP 10-026	54	407	353	1.17	0.034

9.5 2011 Drill Program

The 2011 drill program totaled 28,750 m in 80 diamond core holes and drill hole data are illustrated in Figure 9.3 and summarized in Table 9.7. Five of the diamond core holes were drilled for the purpose of metallurgical testing. All these holes (SP11-061, -065, -066, -069 and -090) were twins of previously drilled holes. The core obtained from SP11-061, -065 and -069 was not sampled in order to send the whole core for metallurgical testing. The drill core from SP11-066 and -090 was quartered and one-quarter was sent to SGS's Red Lake laboratory for assaying. The remaining three-quarters were sent to SGS's Lakefield metallurgical laboratory facility along with the whole cores. Results from the metallurgical testing are discussed in Section 12.

Table 9.8 summarizes the significant gold and silver intercepts from the 2011 diamond core drilling program.



(Source Gold Canyon, 2011)

Figure 9.3: Springpole Gold Project - 2011 Drill Hole Collar Location Map

Table 9.7: 2011 Diamond Drill Hole Program Summary Data

Hole ID	Azimuth	Dip	Length (m)	Easting* (m)	Northing* (m)	Elevation (m)
SP11-030	220	-45	238	5,694,088	549,074	396.73
SP11-031	220	-45	241	5,694,081	549,116	395.34
SP11-032	220	-45	70	5,693,915	549,376	391.06
SP11-033	220	-45	350.7	5,693,915	549,376	391.06
SP11-034	220	-55	379.5	5,693,857	549,454	390.32
SP11-035	0	-90	200.5	5,693,964	549,154	391.6
SP11-036	220	-45	396	5,693,470	549,785	390.06
SP11-037	220	-45	372	5,693,891	549,420	389.4
SP11-038	0	-90	202	5,693,865	549,199	390.25
SP11-039	220	-90	176	5,693,816	549,287	392.89
SP11-040	0	-90	151.5	5,693,749	549,364	390.95
SP11-041	220	-45	250.5	5,693,841	549,510	389.6
SP11-042	220	-45	411	5,693,511	549,755	390.49
SP11-043	0	-90	153	5,693,698	549,455	385.86
SP11-044	220	-45	351	5,693,802	549,540	389.79
SP11-045	0	-90	90	5,693,607	549,505	389.05
SP11-045A	0	-90	213	5,693,607	549,505	389.05
SP11-046	220	-45	395	5,693,406	549,867	389.41
SP11-047	0	-90	177	5,693,542	549,582	391.15
SP11-048	220	-45	360	5,693,768	549,581	389.85
SP11-049	0	-90	152	5,693,471	549,657	389.22
SP11-050	220	-45	402	5,693,262	549,944	389.06
SP11-051	0	-90	164	5,693,455	549,707	391.17
SP11-052	0	-90	158	5,693,508	549,616	389.49
SP11-053	220	-45	351	5,693,741	549,619	390.4
SP11-054	0	-90	165	5,693,597	549,565	390.46
SP11-055	220	-45	407.5	5,693,088	550,126	390.72
SP11-056	0	-90	228	5,693,653	549,481	391.9
SP11-057	220	-45	348	5,693,702	549,653	390.35
SP11-058	0	-90	159	5,693,727	549,411	389.73
SP11-059	220	-45	369	5,693,577	549,743	390.99
SP11-060	0	-90	255	5,693,725	549,413	391
SP11-061	0	-90	132	5,693,751	549,361	385.55
SP11-062	40	-45	462	5,693,018	549,335	401.5
SP11-063	40	-45	980	5,693,025	549,328	399.97
SP11-064	40	-45	980	5,693,351	549,221	395.9
SP11-065	220	-45	387.5	5,694,095	549,255	394.71
SP11-066	20	-45	301	5,694,095	549,606	403
SP11-067	40	-45	337	5,694,098	549,608	400.64
SP11-068	40	-50	902	5,693,529	549,009	398.72
SP11-069	225	-45	410	5,693,677	549,758	396.96
SP11-070	220	-55	491	5,694,107	549,209	396.94
SP11-071	220	-60	494	5,693,577	549,814	390.47
SP11-072	220	-55	492	5,694,073	549,248	397.07
SP11-073	0	-90	401	5,693,960	549,214	391.34
SP11-074	220	-45	498	5,693,546	549,849	393.83
SP11-075	0	-90	399	5,693,569	549,664	390.07
SP11-076	220	-45	409	5,694,068	549,368	400.69
SP11-077	0	-90	342	5,694,006	549,135	390.14
SP11-078	220	-45	494	5,693,485	549,908	391.9
SP11-079	220	-60	426.5	5,693,976	549,359	394.86
SP11-080	0	-90	420	5,693,651	549,614	389.2
SP11-081	0	-90	361	5,693,954	549,086	390.42
SP11-082	220	-45	481	5,693,515	549,883	397.69
SP11-083A	0	-90	144	5,693,930	549,262	389
SP11-083	0	-90	381	5,693,932	549,262	390.3
SP11-084	0	-90	349.5	5,693,580	549,616	390
SP11-085	220	-45	301	5,693,737	548,446	406.9
SP11-086	220	-45	302	5,694,001	548,465	409.6
SP11-087	0	-90	396	5,693,373	549,643	390.2
SP11-088	220	-60	598	5,693,634	549,856	392.64
SP11-089	220	-45	300	5,694,242	548,441	415.22
SP11-090	200	-45	206	5,694,244	549,035	410.8
SP11-091	0	-90	400.5	5,693,830	549,234	391.14
SP11-092	220	-55	424	5,694,068	549,418	400.51
SP11-093	0	-90	316.5	5,693,415	549,609	390.59
SP11-094	222	-50	570	5,693,902	549,619	395
SP11-095	220	-45	441	5,694,063	549,528	400.36
SP11-096	0	-90	327	5,693,745	549,295	391.38
SP11-097	0	-90	291	5,693,457	549,576	387.1
SP11-098	223	-45	401.5	5,693,979	549,491	398.29
SP11-099	223	-45	466	5,693,921	549,575	396.63
SP11-100	223	-50	521.5	5,693,984	549,572	392.84
SP11-101	220	-45	302	5,694,237	548,226	415.8
SP11-102	220	-45	302	5,694,138	548,047	417.33
SP11-103	220	-45	290	5,693,957	548,130	415.2
SP11-104	220	-45	458	5,693,942	549,538	395.14
SP11-105	220	-45	302	5,693,544	548,233	421
SP11-106	220	-45	508	5,694,016	549,519	400.12
SP11-107	220	-45	515	5,694,180	549,224	398.91

*Universal Transverse Mercator (UTM): World Geodetic System 1984 (WGS84) projection

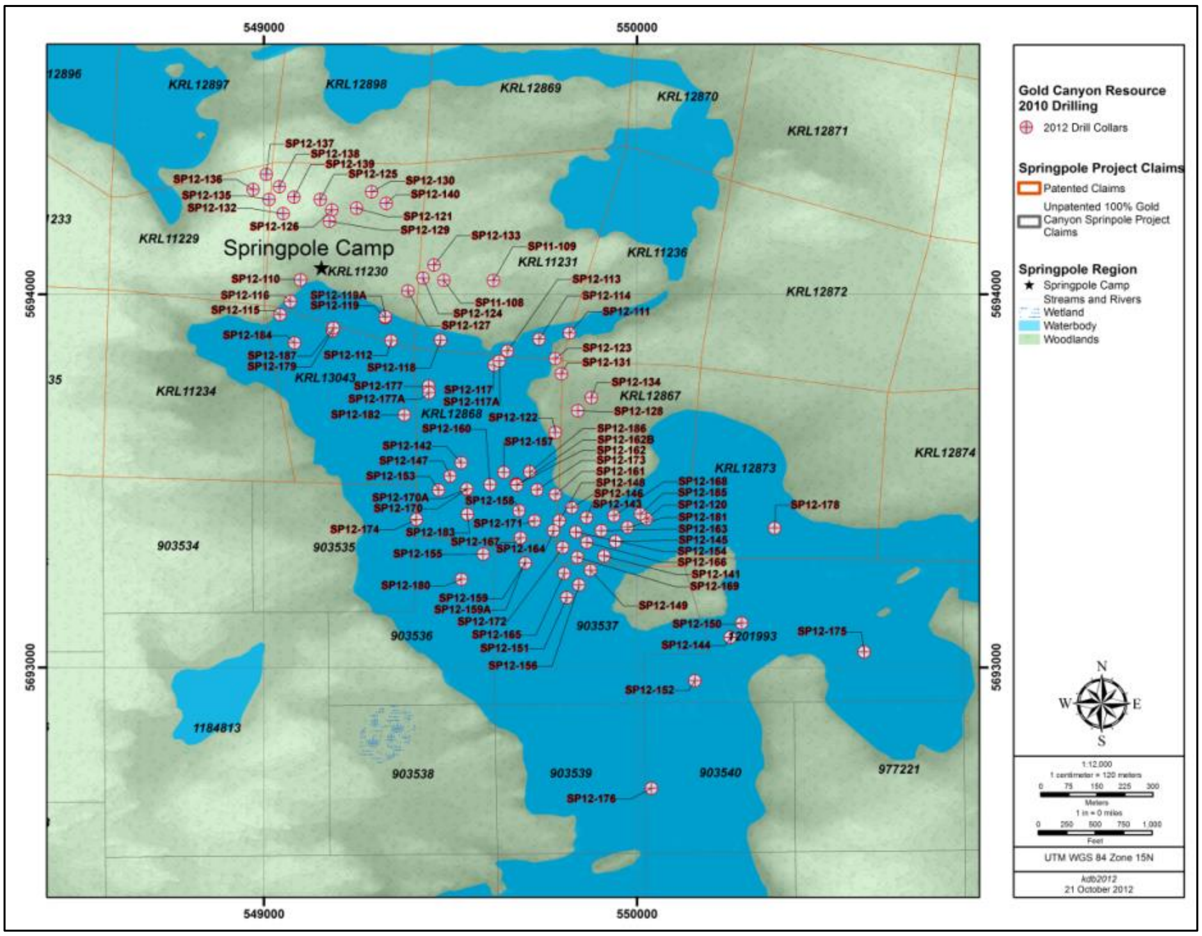
Table 9.8: Significant Intercepts from 2011 Diamond Core Drilling Program

Hole ID	From (m)	To (m)	Interval (m)	Au (g/t)	Ag (g/t)	Au (oz/t)
SP11-030	14.0	73.0	59.0	2.51	1.98	0.073
SP11-033	13.0	315.0	302.0	1.39	7.16	0.041
SP11-034	37.0	110.5	73.5	1.18	6.18	0.034
	162.0	331.0	169.0	1.08	6.29	0.032
SP11-035	37.0	68.0	31.0	1.01	3.60	0.029
	105.0	200.5	95.5	1.22	3.26	0.036
SP11-036	204.0	394.5	190.5	0.90	3.96	0.026
SP11-037	54.0	316.5	262.5	0.92	4.67	0.027
SP11-038	61.0	79.0	18.0	0.89	4.62	0.026
SP11-039	60.0	117.0	57.0	0.40	3.07	0.012
	132.0	165.0	33.0	0.53	4.72	0.015
SP11-040	51.0	151.5	100.5	7.23	8.83	0.211
SP11-041	161.0	237.0	76.0	1.50	5.60	0.044
SP11-042	9.0	411.0	402.0	0.76	2.88	0.022
SP11-043	42.0	153.0	111.0	2.03	7.00	0.059
SP11-044	132.0	351.0	219.0	0.71	11.80	0.021
SP11-045	36.0	90.0	54.0	2.15	19.13	0.063
SP11-045A	63.0	213.0	150.0	2.56	12.48	0.075
SP11-046	34.0	63.0	29.0	0.57	5.46	0.017
	238.0	306.5	68.5	0.82	6.74	0.024
SP11-047	22.7	177.0	154.3	0.99	8.69	0.029
SP11-048	121.0	315.0	194.0	1.11	13.79	0.032
SP11-049	20.0	152.0	132.0	1.37	7.59	0.040
SP11-050	139.0	247.0	108.0	0.54	3.30	0.016
	304.0	328.0	24.0	0.63	3.96	0.018
SP11-051	14.0	164.0	150.0	1.15	3.92	0.034
SP11-052	19.0	158.0	139.0	1.04	10.83	0.030
SP11-053	11.4	21.0	9.6	2.95	13.32	0.086
SP11-054	23.0	165.0	142.0	0.81	17.63	0.024
SP11-055	18.0	33.0	15.0	0.36	3.07	0.011
SP11-056	55.5	228.0	172.5	0.93	21.38	0.027
SP11-057	91.5	312.0	220.5	0.84	4.91	0.025
SP11-058	48.4	159.0	110.6	2.48	4.56	0.072
SP11-059	72.0	364.5	292.5	1.13	4.13	0.033
SP11-060	51.0	255.0	204.0	1.15	4.87	0.034
SP11-066	16.0	40.0	24.0	17.48	3.19	0.510
SP11-067	15.0	54.0	39.0	2.93	1.01	0.086
SP11-070	93.0	401.0	308.0	1.29	1.33	0.038
SP11-071	149.0	435.0	286.0	1.03	7.73	0.030
SP11-072	63.0	382.0	319.0	0.97	2.49	0.028
SP11-073	17.0	267.0	250.0	1.46	2.99	0.043
SP11-074	121.0	490.0	369.0	0.91	5.57	0.027
SP11-075	113.0	319.0	206.0	0.91	2.84	0.027
SP11-076	28.0	149.0	121.0	0.70	1.46	0.020
	295.0	387.0	92.0	0.60	2.15	0.018
SP11-077	10.0	87.0	77.0	0.73	0.43	0.021
	130.0	236.0	106.0	3.36	2.13	0.098
SP11-078	249.0	363.0	114.0	0.58	4.09	0.017
SP11-079	3.0	177.5	174.5	0.56	1.98	0.016
	312.0	416.0	104.0	0.59	2.12	0.017
SP11-080	48.0	124.0	76.0	0.62	1.90	0.018
SP11-081	92.0	321.0	229.0	0.82	2.39	0.024
SP11-082	85.0	171.0	86.0	1.07	17.95	0.031
	262.0	403.0	141.0	0.72	5.93	0.021
SP11-083	24.0	155.0	131.0	0.77	3.12	0.022
SP11-084	15.0	349.5	334.5	0.83	5.26	0.024
SP11-087	159.0	353.0	194.0	0.96	5.98	0.028
SP11-088	7.0	36.0	29.0	0.62	1.19	0.018
	300.0	346.0	46.0	0.58	7.17	0.017
	364.0	441.0	77.0	0.72	4.62	0.021
SP11-091	66.0	376.0	310.0	1.87	6.59	0.055
SP11-092	109.0	177.0	68.0	0.58	0.96	0.017
SP11-093	122.0	316.5	194.5	0.85	3.72	0.025
SP11-094	312.5	455.0	142.5	0.71	5.01	0.021
SP11-096	66.0	323.0	257.0	1.48	5.83	0.043
SP11-097	27.0	60.0	33.0	0.71	0.72	0.021
	200.0	291.0	91.0	0.79	4.62	0.023
SP11-098	3.0	124.0	121.0	1.67	3.61	0.049
	311.5	401.5	90.0	2.00	7.17	0.058
SP11-099	254.0	430.0	176.0	0.80	7.61	0.023
SP11-100	404.5	482.0	77.5	0.62	5.37	0.018
SP11-104	279.0	427.0	148.0	1.66	6.10	0.048
SP11-106	256.0	269.0	13.0	0.77	2.84	0.022
	344.5	472.0	127.5	3.51	10.70	0.102
SP11-107	247.0	377.0	130.0	0.72	2.39	0.021

9.6 2012 Drill Program

The 2012 drill program commenced on January 18, 2012, using the two Boyles 37 from Rodren and one discovery EF-50 drills from the 2011 program. Three Discovery LF-75 drills, mobilized to the project via the winter road, were also used. The drill program began in-filling the Portage zone based upon results of the 2011 drill program. The goal was to in-fill areas where inferred mineral resource had been defined in the February 2012 mineral resource update and to expand the mineral resource area to the southeast.

The 2012 drill program totaled 38,069 m in 87 diamond core holes. The drill hole data are illustrated in Figure 9.4 and summarized in Table 9.9. Significant drill intersections from the 2012 drilling program are summarized in Table 9.10.



(Source Gold Canyon, 2012)

Figure 9.4: Springpole Gold Project – 2012 Drill Hole Collar Location Map

Table 9.9: 2012 Diamond Drill Hole Program Summary Data

Hole ID	Azimuth	Dip	Length (m)	Easting* (m)	Northing* (m)	Elevation (m)
SP11-108	0	-45	540	549,483	5,694,037	400
SP11-109	0	-45	600	549,615	5,694,037	398
SP12-110	0	-90	480.5	549,098	5,694,038	392
SP12-111	220	-45	568	549,819	5,693,896	387
SP12-112	0	-90	824.2	549,341	5,693,876	390
SP12-113	221	-45	496	549,653	5,693,848	391
SP12-114	220	-45	569.6	549,738	5,693,880	388
SP12-115	0	-90	527	549,044	5,693,945	391
SP12-116	0	-90	449	549,071	5,693,981	389
SP12-117	220	-45	75.2	549,618	5,693,809	388
SP12-117A	220	-45	426	549,631	5,693,821	389
SP12-118	220	-45	413	549,474	5,693,877	389
SP12-119	0	-90	26	549,326	5,693,937	390
SP12-119A	0	-90	449	549,325	5,693,940	389
SP12-120	220	-45	332	550,026	5,693,397	390
SP12-121	220	-45	518	549,249	5,694,231	400
SP12-122	220	-45	587	549,781	5,693,629	392
SP12-123	221	-45	566	549,781	5,693,827	393
SP12-124	220	-45	491.5	549,427	5,694,043	401
SP12-125	221	-45	392	549,152	5,694,254	407
SP12-126	219	-45	509	549,183	5,694,226	402
SP12-127	221	-45	547	549,386	5,694,009	400
SP12-128	222	-45	654	549,841	5,693,688	394
SP12-129	221	-45	494	549,176	5,694,196	400
SP12-130	219	-45	614	549,289	5,694,275	401
SP12-131	222	-45	656	549,798	5,693,787	396
SP12-132	220	-45	287	549,052	5,694,216	411
SP12-133	220	-45	527	549,456	5,694,078	401
SP12-134	220	-45	701	549,878	5,693,723	394
SP12-135	220	-45	305	549,014	5,694,254	410
SP12-136	220	-45	251	548,972	5,694,281	408
SP12-137	220	-45	377	549,007	5,694,321	409
SP12-138	220	-45	404	549,042	5,694,288	410
SP12-139	220	-45	341	549,081	5,694,260	411
SP12-140	212	-55	618.5	549,328	5,694,243	401
SP12-141	0	-90	516	549,912	5,693,299	391
SP12-142	0	-90	361.5	549,529	5,693,549	391
SP12-143	0	-90	432	549,865	5,693,402	391
SP12-144	0	-90	473	550,250	5,693,081	391
SP12-145	0	-90	478	549,943	5,693,338	391
SP12-146	0	-90	455	549,825	5,693,428	391
SP12-147	0	-90	499.5	549,500	5,693,513	391
SP12-148	0	-90	534	549,792	5,693,394	392
SP12-149	0	-90	500	549,876	5,693,260	391
SP12-150	0	-90	602	550,280	5,693,119	391
SP12-151	0	-90	503	549,812	5,693,187	392
SP12-152	0	-90	671	550,155	5,692,964	392
SP12-153	0	-90	477	549,469	5,693,475	392
SP12-154	0	-90	525	549,836	5,693,363	392
SP12-155	0	-90	443	549,588	5,693,304	392
SP12-156	0	-90	435	549,844	5,693,221	393
SP12-157	0	-90	379.5	549,643	5,693,523	392
SP12-158	0	-90	395	549,684	5,693,420	392
SP12-159	0	-90	59	549,701	5,693,280	392
SP12-159A	0	-90	355.5	549,677	5,693,492	392
SP12-160	0	-90	420	549,606	5,693,490	391
SP12-161	0	-90	362	549,781	5,693,463	392
SP12-162	0	-90	29	549,678	5,693,489	391
SP12-162B	0	-90	468	549,678	5,693,488	391
SP12-163	0	-90	431	549,904	5,693,367	392
SP12-164	0	-90	464	549,776	5,693,366	392
SP12-165	0	-90	495.5	549,805	5,693,252	393
SP12-166	0	-90	354	549,866	5,693,335	392
SP12-167	0	-90	400	549,688	5,693,347	393
SP12-168	0	-90	473	549,939	5,693,406	391
SP12-169	0	-90	362	549,840	5,693,295	391
SP12-170	0	-90	257	549,544	5,693,477	391
SP12-170A	0	-90	458	549,544	5,693,478	391
SP12-171	0	-90	440	549,726	5,693,393	391
SP12-172	0	-90	405.1	549,801	5,693,322	391
SP12-173	0	-90	434	549,732	5,693,476	391
SP12-174	0	-90	506	549,409	5,693,397	392
SP12-175	0	-90	384.2	550,609	5,693,042	391
SP12-176	0	-90	296	550,039	5,692,676	392
SP12-177	0	-90	30	549,442	5,693,754	392
SP12-177A	0	-90	450	549,443	5,693,736	392
SP12-178	0	-90	395	550,369	5,693,373	392
SP12-179	0	-90	381	549,185	5,693,903	391
SP12-180	0	-90	440	549,529	5,693,236	392
SP12-181	0	-90	350	549,975	5,693,376	391
SP12-182	0	-90	395	549,377	5,693,676	392
SP12-183	0	-90	449	549,546	5,693,410	391
SP12-184	0	-90	398	549,082	5,693,870	391
SP12-185	0	-90	371	550,009	5,693,411	391
SP12-186	0	-90	468	549,713	5,693,524	391
SP12-187	0	-90	394	549,186	5,693,910	392
SP12-197	0	-90	400	549,511	5,693,612	392

*Universal Transverse Mercator (UTM): World Geodetic System 1984 (WGS84) projection

Table 9.10: Significant Intercepts from 2012 Diamond Core Drilling Program

Hole ID	From (m)	To (m)	Interval (m)	Au (g/t)	Au (oz/t)
SP12-127	251	398	147	1.14	0.03
SP12-128	230	549	319	1.02	0.03
SP12-131	301.3	546	244.7	0.80	0.023
SP12-146	77	91	14	5.03	0.147
SP12-158	16.7	60.2	43.5	1.81	0.053
SP12-160	23	384	361	1.08	0.032
SP12-163	130.9	265.0	134.1	0.91	0.027
SP12-181	157	225	68	0.72	0.021
SP12-183	202	385	183	0.61	0.018
SP12-186	114	240	126	1.17	0.034

9.7 Drill Collar Surveying

All historic holes drilled prior to 2010 were surveyed using various earth projections, either NAD27 (North American Datum 1927) Canada, WGS or NAD83 projections. In September 2006, W.J. Bowman Ltd of Dryden, Ontario, surveyed 275 historic drill hole collars from collar numbers BL-1 thru BL-373. For the purposes of inclusion in the data set for 3-D modelling, all the historic collar locations were converted to the UTM WGS84 projection.

For the 2007 and 2008 drill programs, the drill hole collars were located and surveyed using a handheld GPS and recorded in UTM NAD27 Canada projection. For the purposes of this report all the collar survey information has been converted to WGS84 and field checked against collar locations using handheld Trimble GeoXH DGPS.

The 2010 to 2012 drill hole collars were initially surveyed using handheld GPS devices. During the initial phases of the offshore 2010 drill program, drill hole collars on the lake ice were surveyed by handheld, real-time differential GPS with an average accuracy of 4 to 5 m and recorded in UTM NAD27 Canada projection. On-shore drill holes were initially located with handheld GPS and once the drill hole was complete, the hole location was temporarily marked; subsequently, the collars were surveyed using a Trimble GeoXH handheld DGPS device with an external antenna giving submetre (~10 cm) location accuracy.

For the offshore 2011 and 2012 drill program, with drills mounted on barges, the drill sites were marked by floating buoy and located using the Trimble GeoXH from a boat. All onshore drill collars were located and subsequently surveyed using the Trimble GeoXH. At the beginning of the winter 2011 drill program, the UTM WGS84 projection was adopted as the standard for surveying drill collars and others surface landmarks. All previously recorded UTM measurements were converted accordingly.

All drill site locations for inclined drill holes, onshore or offshore on the ice, were marked using two to four painted laths aligned along strike either side of the proposed drill hole location. These laths were used as fore- and back-sights for setting the drill location and orientation. Inclination of the drill hole was checked on the drill head, prior to commencing drilling, using either a Brunton compass or inclinometer accurate to half of one degree.

9.8 Oriented Core Surveying

Oriented core measurements were collected from a total of 44 drill holes. Oriented core is used to evaluate the structural geology by allowing the geologists to measure the real angular relationships, as opposed to apparent angles. The tool used was the ACT 2 from Reflex Technologies. This system is fully digital, using infra-red and digital technology to make measurements easier to record in the field by the drill crew. There were significant problems encountered during the winter 2011 drill program due to tool failures. Some oriented core information was collected, but too little to be of widespread use.

Where down-hole poor ground conditions were encountered, the oriented core tool proved to be of little value due to the incompetent nature of intensely altered and mineralized rock. Wherever competent rock was encountered, oriented core data were collected.

9.9 Down-Hole Surveying

All drill holes during the 2010 drill program were surveyed using a Reflex Technologies single EZ-Shot or EZ-Trax down-hole survey system. Drill holes were surveyed once completed – this procedure was used because of the chance that bad ground conditions encountered in the drill holes increased the risk of cave-in when pulling the drill string backwards to conduct a survey. Cave-in can result in increased cost due to time spent reaming the drill hole clean back to the bottom, or from the possibility of sticking the drill string, causing loss of drilling tools. The presence of magnetite in banded iron formation and relatively unaltered trachyte or greenstone caused problems with respect to azimuth readings and also the azimuth of the drill traces. This required many repetitions of the down-hole survey readings, which in some cases resulted in an inability to record consistent data.

For the 2011 and 2012 programs, the Reflex Down-Hole Gyro survey system was adopted with the EZ-Trax or EZ-Shot down-hole survey tools as back up. The Reflex Gyro is built around a digital micro-gyro, which consists of a silicon sensor chip and an integrated circuit assembled in a ceramic (non-magnetic) package. The gyro provides directional data (azimuth and dip) at any interval from inside the drill rods. This system is used to provide azimuth and inclination data in rocks with strong magnetic fields, because the gyros operate independently of the earth’s magnetic field. The system also records ambient temperature as well as collecting basic gravity measurements. The gyro system was successfully applied to the majority of the 2011 and 2012 drill programs.

Data recorded from the down-hole surveys were incorporated into 3-D planning and modelling.

9.10 Drilling Pattern and Density

The overall drill pattern approximate a 50 m grid along the long axis of the Portage zone and about 45 to 65 m spacing down the dip of the mineralized zone. SRK is of the opinion that the drill spacing and density is appropriate for this type of deposit and style of mineralization.

10 Sample Preparation, Analyses, and Security

10.1 Core Drilling Sampling

Detailed descriptions of the drill core were carried out under the supervision of a senior geologist, a member in good standing of the APGO (Association of Professional Geologists of Ontario) and AIPG (American Institute of Professional Geologists). The core logging was carried out on-site in a dedicated core logging facility. Drill log data were recorded onto paper logs that were later scanned and digitized.

Core was laid out 30 to 40 boxes at a time. First, the core was photographed in 15 m batches prior to logging or sampling. This is followed by a geotechnical log that records quantitative and qualitative engineering data including detailed recovery data and rock quality designation. Any discrepancies between marker blocks and measured core length were addressed and resolved at this stage. The core was then marked up for sampling.

For the 2010 and 2011 drill programs, all the drill core intervals were sampled using sample intervals of 1 m. During the 2012 drilling program, Gold Canyon changed its standard sample length from 1 to 2 m lengths. However, in zones of poor recovery, 1.5 or 3 m samples were sometimes collected. Samples over the standard sample length were typically half core samples and whole core was generally only taken in intervals of poor core recovery across the sampled interval. Sampling marks were made on the core and sample tickets were stapled into the core boxes at the beginning of each sample interval. Quality control samples were inserted into the sample stream.

Inserting quality control samples involved the addition of certified blanks, certified gold standards, and field and laboratory duplicates. Field duplicates were collected by quartering the core in the sampling facility on-site. Laboratory duplicates were collected by splitting the first coarse reject and crushing and then generating a second analytical pulp. Blank, standards and duplicates made up 10% of the total sample stream. Sample tickets were marked blank, field or laboratory duplicate, or standard, and a sample tag was stapled into the core box within the sample stream.

Geological descriptions were recorded for all core recovered. Separate columns in the log allow description of the lithology, alteration style, intensity of alteration, relative degree of alteration, sulphide percentage, rock colour, vein type, and veining density. A separate column was reserved for written notes on lithology, mineralization, structure, vein orientations/relations etc. The header page listed the hole number, collar coordinates, final depth, start/end dates, and the name of the core logging geologist.

10.1.1 Core Sampling, Handling and Chain-of-Custody

Following the logging and core marking procedures described above, the core was passed to the sampling facility. Core sampling was performed by experienced sampling technicians from Ackewance Exploration & Services of Red Lake, Ontario, and quality control was maintained through regular verification by on-site geologists. Core was broken, as necessary, into manageable lengths. Pieces were removed from the box without disturbing the sample tags, were cut in half lengthwise with a diamond saw, and then both halves were carefully repositioned in the box. When a complete hole was processed in this manner, one half was collected for assay while the other half remained in the core box as a witness. The remaining core in the boxes was then photographed at 51 cm (20 inch) intervals. All logs and photographs were then submitted to the senior geologist/project manager for review and were archived. Data were backed up.

The sampling technician packed one half of the split core sample intervals into transparent vinyl sample bags that were sequentially numbered to match the sample number sequences in the sample tag booklets used by the core-logging geologists. The numbered, blank portion of the triplicate sample tag was placed in the bag with the sample; the portion was marked with the sample interval remained stapled into the bottom of the core box at the point where the sample interval begins. Sample bags were then sealed with plastic tags. Sealed sample bags were packed into rice sacks five samples at a time. All sacks were individually labeled with the name of the company, number of samples contained therein, and the number sequence of the samples therein. Sacks were assigned sequential numbers on a per shipment basis. A project geologist then checked the sample shipment and creates a shipping manifest for the sample batch. A copy was given to the project manager and a copy was sent along with the sample shipment. A copy of the sample shipment form was also sent via e-mail to the analytical laboratory.

The project geologist prepared the sample submission form for the assay laboratory. This form identifies the number of sample sacks as well as the sequence of sample numbers to be submitted. Due to the remote location, the shipment was then loaded on to a plane or helicopter and flown direct to Red Lake where representatives of the commercial analytical laboratory met the incoming flight and took the samples to the laboratory by pickup truck.

Once at the laboratory, a manager checked the rice sacks and sample numbers on the submission form. The laboratory then split the received sample manifest into batches for analysis, assigned a work order to the batch, and sent a copy of the mineral analysis acknowledgement form to the project manager.

Aluminum tags embossed with the hole number, box number, and box interval (from/to) were prepared and stapled onto the ends of each core box. Core boxes were cross-stacked on pallets and then moved to on-site storage.

10.2 Sample Security

Core samples collected at the drill site were held in closed core boxes sealed with fiber tape; at various times of day, camp staff collected the core boxes that were then delivered to the core logging facility. All core logging, sampling and storage took place at the Springpole Gold Project site. Following the logging and marking of core (described in the preceding section), all core preparation and sampling was performed by technicians from Ackewance of Red Lake, Ontario, under the supervision of the project manager. All on-site sampling activities were directly supervised by the project manager.

10.3 Sample Preparation and Analytical Procedures

10.3.1 Analytical Laboratories

All primary assay work since the 2010 drill program has been performed by SGS Laboratories in Red Lake (gold), Ontario and Don Mills (silver and multi-element) in Toronto, Ontario. The SGS Red Lake and Don Mills facilities are certified and conform to requirements CAN-P-1579 and CAN-P-4E (ISO/IEC 17025:2005). Certification is accredited for precious metals including gold and silver and 52 element geochemical analyses.

10.3.2 Analytical Procedures

All samples received by SGS Red Lake were processed through a sample tracking system that is an integral part of the company's laboratory information management system. This system utilizes bar coding and scanning technology that provides complete chain of custody records for every stage in the sample preparation and analytical process.

Samples were dried, and then crushed to 70% of the sample passing 2 mm (-70 mesh). A 250 g sample was split off the crushed material, and pulverized to 85% passing 75 micron (-200 mesh). A 30 g split of the pulp was used for gold fire assay and a 2 g split was used for silver analysis. Crushing and pulverizing equipment was cleaned with barren wash material between sample preparation batches and, where necessary, between highly mineralized samples. Sample preparation stations were also equipped with dust extraction systems to reduce the risk of sample contamination. Once the gold assay was complete, a pulp was sent to the SGS Toronto facility for silver and possibly for multi-element geochemical analysis.

As part of the standard internal quality control procedures used by the laboratory, each batch of 75 Springpole core samples included four blanks, four internal standards, and eight duplicate samples. In the event that any reference material or duplicate result would fall outside the established control limits, the sample batches would be re-assayed.

Pulps and rejects of the samples were stored by SGS at its Red Lake facility at the request of Gold Canyon.

10.3.3 Gold, Silver and Multi-Element Analysis

Prepared samples were analyzed for gold by fire assay with atomic absorption finish. Samples returning assays in excess of 10g/t gold are re-analyzed with a gravimetric finish.

Prepared pulp samples shipped from SGS Red Lake to SGS Toronto are analyzed for silver by three-acid digestion with atomic absorption finish.

During the winter 2010 program, prepared samples were analyzed for 52 elements by acid digestion (3:1 HCl: HNO3). The list of elements is included in Table 10.1.

Table 10.1: SGS Multi-Element Analysis Method ICM14B

Elements	Limits	Element	Limits	Element	Limits
Ag	0.01 – 10 ppm	Hg	0.01 ppm - 1%	Se	1 ppm - 0.1%
Al	0.01 - 15%	In	0.02 ppm - 0.05%	Sn	0.3 ppm - 0.1%
As	1 ppm - 1%	K	0.01 - 25%	Sr	0.5 ppm - 1%
B	10 ppm - 1%	La	0.1 ppm - 1%	Ta	0.05 ppm - 1%
Ba	5 ppm - 1%	Li	1 ppm - 5%	Tb	0.02 ppm – 1%
Be	0.1 ppm - 0.01%	Lu	0.01 ppm - 0.1%	Te	0.05 ppm - 0.1%
Bi	0.02 ppm - 1%	Mg	0.01 - 15%	Th	0.1 ppm - 1%
Ca	0.01 - 15%	Mn	2 ppm - 1%	Ti	0.01 - 15%
Cd	0.01 ppm - 1%	Mo	0.05 ppm - 1%	Tl	0.02 ppm - 1%
Ce	0.05 ppm - 0.1%	Na	0.01 - 15%	U	0.05 ppm - 1%
Co	0.1 ppm - 1%	Nb	0.05 ppm - 0.1%	V	1 ppm - 1%
Cr	1 ppm - 1%	Ni	0.5 ppm - 1%	W	0.1 ppm - 1%
Cs	0.05 ppm - 0.1%	P	50 ppm - 1%	Y	0.05 ppm - 1%
Cu	0.5 ppm - 1%	Pb	0.2 ppm - 1%	Yb	0.1 ppm - 0.01%
Fe	0.01% - 15%	Rb	0.2 ppm - 1%	Zn	1 ppm - 1%

10.4 Bulk Density Data

Bulk density was obtained for select core samples using the paraffin wax method at SGS Lakefield Research Ltd. laboratory in Lakefield, Ontario. The bulk density of a sample is the weight of the sample divided by the volume of the sample including voids.

The procedure as applied by SGS metallurgical laboratory was as follows:

1) Oven-dry the samples and then cool to room temperature.
2) Label and weigh each sample in grams.
3) Coat the sample with paraffin wax heated in a container immersed in boiling water.
4) Repeatedly immerse the sample in the wax until completely sealed.
5) Avoid heating the sample.
6) Weigh the waxed sample and record.
7) Weigh the waxed samples (g)by suspending in water and recording the displaced volume (mL) and the water temperature (ºC).
8) Remove the wax by placing in boiling water, or freezing the core and chipping off if return of the sample is required.

Calculations:

1) Weight of wax = (weight of sample + wax) – (weight of sample)

2) Volume of wax = weight of wax /specific gravity (s.g.) of wax corrected for temperature.

3) Volume of sample = (volume of sample + wax) – (volume of wax)

4) Bulk density (t/m^3) = weight of sample (g) / volume of sample (mL)

5) Bulk Density (lb/ft^3) = (t/m^3) / 0.0160.

Results from selected analysis of bulk density are summarized in Table 10.2 and discussed in Section 13.14 of the report.

Table 10.2: Summary of Wax Bulk Density Measurements

ROCK SAMPLE BULK DENSITY

Project Number	13152-001	Date	13-Dec-11
Project Name	Springpole	Technician	ar
Sample Description	Core Sample Intervals		

Wax	SG	0.8913	g/cm^3
Water	Temp (C)	16	
	Density	0.9989	g/cm^3

Sample					Weight (g)				Volume (cm^3)			Rock Density	
No.	Description		Box No	m	Dry Rock	Rock Coated with wax	Weight in Water	Water Displacement	Rock Coated with wax	Wax	Rock	Density (g/cm^3)	Density (lbs/ft^3)
1	SP11-061	1	1	40.5 - 59	804.0	815.2	497.2	318.0	318	12.6	305.8	**2.63**	164.2
2		2	8	81.7 - 85.5	523.1	533.1	223.1	310.0	310	11.2	299.1	**1.75**	109.2
3		3	13	98.2 - 101	219.7	227.5	95.1	132.4	133	8.8	123.8	**1.77**	110.8
4		4	18	113.9 - 114.2	397.1	407.3	192.8	214.5	215	11.4	203.3	**1.95**	122.0
5		5	24	131.3 - 132	517.7	528.7	263.5	265.2	265	12.3	253.1	**2.05**	127.7
6	SP11-065	1	21	590.1 - 62.6	805.0	820.3	522.9	297.4	298	17.2	280.5	**2.87**	179.2
7		2	37	104.2 - 106.8	1042.4	1062.1	665.1	397.0	397	22.1	375.3	**2.78**	173.4
8		3	49	138.4 - 140.5	662.3	684.0	408.0	276.0	276	24.3	251.9	**2.63**	164.1
9		4	72	199.3 - 201.8	634.6	650.6	363.5	287.1	287	18.0	269.5	**2.36**	147.0
10		5	82	226.4 - 229.6	769.8	793.0	467.1	325.9	326	26.0	300.2	**2.56**	160.1
11	SP11-069	1	97	265.5 - 267.2	871.5	895.5	512.3	383.2	384	26.9	356.7	**2.44**	152.6
12		2	108	295.1 - 297.2	532.7	546.7	271.3	275.4	276	15.7	260.0	**2.05**	127.9
13		3	130	345.2 - 346.7	630.7	647.5	364.3	283.2	283	18.8	264.6	**2.38**	148.8
14		4	146	376.8 - 379.1	625.4	641.5	368.3	273.2	273	18.1	255.4	**2.45**	152.9
15		5	158	401.9 - 403.9	826.4	847.1	520.6	326.5	327	23.2	303.6	**2.72**	169.9

10.5 Quality Assurance and Quality Control Programs

10.5.1 Pre-2007 QA/QC Program

No documentation relating to sample handling and preparation or sample QA/QC documentation for the pre-2003 drilling were provided to SRK.

The QA/QC procedures for 2003 through 2006 drilling totalling 105 drill holes and comprising 12,956 assay intervals were summarily described by Armstrong et al. (2006). The reader is referred to this report for additional relevant descriptions.

P&E Mining Consultants checked a total of 1,725 entries in the database against the original certificates. According to the report, “A few data entry errors were observed and corrected.”; however, the total number of errors is not presented (Armstrong et al. 2006).

The QA/QC program for 2003to 2007 consisted of:

- Resubmission of approximately 10% of the sample pulps to a second laboratory (ALS Chemex);
- Insertion of two commercial standard reference materials (standards submitted every 30th sample); and
- Insertion of blanks.

There were no field or bulk reject duplicates submitted. Also, no pulp duplicates were submitted to the primary laboratory.

Due to the lack of detailed documentation, particularly for pre-2003 drilling, SRK elected to use the pre-2003 drilling only in estimating the proportionately minor East Extension and Camp zones. The Portage zone was estimated using only 2003 and later drill holes. The East Extension and Camp zones as now defined correspond to the deposits estimated by P&E in their 2006 study (Armstrong, 2006),

Also, because of the lack of documentation, the current estimates for the East Extension zone were restricted to the inferred resource category, although P&E classified these zones as measured, indicated, and inferred resources in their report (Armstrong, 2006), recommended that Gold Canyon continue the program of re-sampling and re-logging of the core for the pre-2007 drilling with focus on the mineralized intervals, to replace the missing duplicate field and pulp duplicate including appropriate insertion of blanks and standards that would demonstrate compliance with current NI 43-101 standards. The drill hole density in these areas was more than adequate for generating resource categories above inferred if appropriate sample methodologies and duplicates samples were included with the results.

10.5.2 2007/2008 QA/QC Program

A total of 18 drill holes were completed in 2007 and 2008 comprising a total of 1,374 assay intervals. These samples were assayed for gold only by the Accurassay Laboratories of Thunder Bay, Ontario. SRK checked a total of 137 samples representing 10% of the total against the original certificates. No errors were found.

No program was set up for duplicates, standards, or blanks for this drilling program. The laboratory ran their own set of duplicates for internal monitoring purposes; however, that data were not available to SRK.

10.5.3 2010 to 2012 QA/QC Program

A total of 196 drill holes, comprising 76,875 m, were completed and assayed in time for inclusion into this study. The vast majority of these drill holes targeted the Portage zone. The drill hole samples generated by the 2010 to 2012 drill programs were assayed by SGS Red Lake and SGS Mineral Services of Toronto, Ontario.

In 2010, Gold Canyon instituted a QA/QC program consisting of commercial standard reference materials for gold, and it instituted, consistent with current industry practice, blanks, field duplicates, and pulp duplicates. In addition, a “round robin” program was instituted in 2011 with ACT Labs of Red Lake, Ontario, that compared pulp re-assay results against the original SGS results for 469 samples.

SGS conducted their own program of internal duplicate analysis as well. The results of this program were also analyzed by SRK as a valuable comparison against the “blind” pulp duplicates submitted. Results are presented in Appendix C.

A summary of the blanks and standards submissions are presented below:

- A total of 1,336 field duplicates were submitted for gold.
- A total of 1,359 field duplicates were submitted for silver.
- A total of 1,303 lab or pulp duplicates were submitted for gold.
- A total of 1,302 pulp duplicates were submitted for silver.
- A total of 1,377 commercial gold standards were submitted from a set of 14 different commercial standards.
- No commercial standards were submitted for silver.
- A total of 1,371 blanks were submitted with the gold assays.
- A total of 1,006 blanks were submitted with the silver assays.

The total submissions for gold duplicates, standards and blanks was 5,387; 10.1% of the samples assayed for gold. The total submissions for silver duplicates, and blanks was 3,667 or 7% of the total samples assayed for silver.

10.6 SRK Comments

In the opinion of SRK, the sampling preparation, security and analytical procedures used by Gold Canyon for gold analyses are acceptable but not fully consistent with generally accepted industry best practices because of the lack of standard reference material for silver. However, because of the relative low economic value of silver, SRK concludes that the assay data are adequate for use in resource estimation. SRK recommends that Gold Canyon establishes a written QA/QC protocol for the acceptance of assay batches with respect to the performance of standard reference material, duplicates and blanks. SRK also recommends that Gold Canyon procure some standard reference material for silver before the beginning of the next drilling campaign.

11 Data Verification

Independent data verification was carried out by P&E and described in their technical report (Armstrong et. al 2006) for data collected from 2003 through 2006.

Of the 18 drill holes completed in 2007 and 2008, comprising a total of 1,374 assay intervals analyzed for gold, SRK checked a total of 137 samples representing 10% of the total against the original certificates. No errors were found.

A total of 3,135 assay values for gold and 3,161 assay values for silver in the database were compared against the original protected PDF assay certificates submitted by SGS Red Lake. These totals represent 10.1% and 10.4% of the total number of assays for gold and silver, respectively.

Of the original assay values checked against certificates, the focus was on values material to any resource estimate, either higher-grade intervals or very low grade intervals in proximity to higher-grade intervals. The average grade of gold samples verified was 2.05 g/t gold. The average grade of silver samples checked was 8.27 g/t silver.

Only two errors were found for gold:

- The gold value of sample interval SP10-028 from 433 m to 436 m (sample number 8287) found to have an entered value of 5.96 g/t gold against a value on the assay certificate of 9.00 g/t gold.
- The gold value of sample interval SP11-076 from 69 to 70 m (sample number 14583) having the value of 0.45 oz/t incorrectly placed in the parts per billion column.

No errors were found with respect to silver assays.

This represents an error rate of 0.064% in gold assays and an error rate of 0.0% in silver assays. This error rate is well within acceptable industry standards.

11.1 Verifications by SRK

11.1.1 Site Visit

SRK carried out visits to the Springpole site on February 10 and 11, 2012, and again on August 8 and 9, 2012. During the site visits, core logging procedures were reviewed. Several sections of core from the Portage, Camp, and East Extension zones were examined. Sampling procedures and handling were observed. The deposit geology, alteration, and core recovery data were observed for the Portage zone. SRK was fully assisted during the site visit by Gold Canyon personnel and was given full access to data during the site visit. Gold Canyon field personnel were very helpful and fully cooperative during both site visits.

During the site visit, SRK re-logged mineralized sections of drill core from the Springpole deposit and checked geological units against the recorded written logs. Down-hole survey data entered in the digital database were checked against data entered on paper logs at the site and no errors were noted. Drill site locations could not be verified as most drill sites are situated under Springpole Lake, but SRK did observe two drill platforms drilling on the lake during the visit.

11.1.2 Verifications of Analytical Quality Control Data

As part of the mineral resource estimation process, SRK reviewed the QA/QC data collected by Gold Canyon, reviewed the procedures in place to assure assay data quality, and verified the assay database against original assay certificates provided directly to SRK by SGS Red Lake, the assay laboratory. A total of 53,431 gold assays, 46% of the assay data, were checked against original assay certificates. No significant database errors were identified. About 143 minor rounding errors were observed. None of the rounding errors are deemed material or of any significance to the mineral resource estimate presented in this report.

11.1.3 Independent Verification Sampling

A total of three mineralized quarter core samples were collected during the February 2012 site visit. The intent of the sampling program was only to determine if gold did occur in concentrations similar to what had been reported by Gold Canyon. Assays from the samples collected by SRK are presented in Table 11.1. The re-sampling agrees with the original Gold canyon sampling.

Table 11.1: Assays from Duplicated Samples Collected During Site Visit

SRK Check Assay		Gold Canyon Original	
Sample	Au (g/t)	Sample	Au (g/t)
9135	8.64	9135	9.04
9136	7.49	9136	7.85
6152	2.37	6152	2.77

12 Mineral Processing and Metallurgical Testing

To date, three metallurgical testwork programs have been completed on Springpole material:

- Lakefield Research, Lakefield, Ontario (1989)—a preliminary program of whole feed leaching on two samples.
- SGS Mineral Services, Vancouver, British Columbia (2011)—a follow up program of leaching on eight samples.
- SGS Lakefield, Lakefield, Ontario (2012/2013)—comminution, whole feed leaching, flotation/leaching on six samples.

Reports were issued at the completion of all programs and are listed in the References (Section 27). This section discusses the testwork completed during these programs, as well as expected plant performance on the selected process flowsheet.

12.1 Lakefield Research 1989

In 1989, preliminary metallurgical testing on two types of material from the Portage zone was completed by Lakefield Research on behalf of Goldfields Canadian Mining Ltd.

The purpose of this testwork was to investigate gold extraction by direct, whole feed cyanidation at different feed sizes, as well as carbon in leach (CIL) extraction at fine grinds on two composite samples produced from core intersections received from Goldfields.

12.1.1 Sample Preparation

The two samples of core received were described as typical Portage zone (Sample 1) and intensely broken altered (Sample 2). Lakefield prepared composites for their test program from these two samples, labelled Composite A and Composite B, respectively.

The combined mass of sample received was 34 kilograms. For each composite, two 2.5 kg charges of “as received” material were riffled out and the remainder was crushed to -10 mesh or -2 mm. A head assay sample was riffled out and the remaining sample was made into test charges.

12.1.2 Head Analysis

Head analyses for the two composite samples are shown in Table 12.1. The precious metal grades were moderate to low and the sulphur levels relatively high, depending on gold association which may present challenges to cyanide leaching. Other elements analyzed did not raise major concerns.

Table 12.1: Sample Head Analysis (Lakefield 1989)

Sample	Au (g/t)	Ag (g/t)	Cu (%)	Fe (%)	As (%)	S (%)
Composite A	3.0	7.7	0.01	7.00	0.019	4.68
Composite B	1.8	9.3	0.01	4.47	0.017	4.19

12.1.3 Cyanidation Testwork

The leaching work was divided into “as-received” material, medium crush, fine grind, and fine grind with carbon present to identify any preg-robbing issues. Both composites were tested equally.

As Received Cyanidation Tests

Five kilogram splits of as-received Composite A and Composite B samples were pulped in a 10 L pail to 50% solids and leached with 2 g/L NaCN solution at a pH of 10.5 to 11, controlled with lime addition. The coarse samples were manually mixed periodically for 96 hours with intermediate solution samples taken at 2, 4, 8, 24, 48, and 72 hours with the results shown in Table 12.2.

Table 12.2: As Received Cyanidation Results (Lakefield 1989)

Test	Size	pH	Cyanide		Carbon	Au Extraction %			Residue	Head
			(g/L)	(kg/t)	(g/L)	(24 hr)	(48 hr)	(96 hr)	(g/t)	(g/t)
A-1	Core	11.0	2	0.83	nil	21.8	29.4	37.5	2.36	3.77
B-1	Core	11.1	2	0.89	nil	15.9	20.6	45.8	2.08	3.84*

**head assay much higher than composite*

The dimensions of the core samples were reported to be about 25 mm. Recovery on such material was relatively low, 35% and 45% after 96 hours. Sample B-1 showed a much higher head assay compared to the composite.

Medium Crush Cyanidation Tests

The sample splits were reduced to -10 mesh (-2 mm) and leached with 2 g/L NaCN solution at pH of 10.5 to 11 with controlled lime addition (Table 12.3). Duplicate tests were conducted at 50% solids in bottles rolled for 24 hours with intermediate solution samples taken at 2, 4, 8, and 16 hours.

Table 12.3: Medium Crush Cyanidation Results (Lakefield 1989)

Test	Size (mm)	pH	Cyanide		Carbon	Au Extraction %			Residue	Head
			(g/L)	(kg/t)	(g/L)	(4 hr)	(16 hr)	(24 hr)	(g/t)	(g/t)
A-4	-2	11.2	2	0.32	nil	59.9	62.4	65.5	1.15	3.33
A-5	-2	11.2	2	0.26	nil			65.7	1.05	3.06
B-4	-2	11.0	2	0.36	nil	68.5	71.4	64.0	0.57	1.58
B-5	-2	11.0	2	0.30	nil			62.6	0.67	1.79

Recovery on the medium (-2 mm) material improved to between 63% and 66% after 24 hours. Cyanide consumption was relatively low at 0.26 to 0.36 kg/t.

Fine Grind Cyanidation Tests

The sample splits were then ground to -200 mesh (75 µm) and leached with 2 g/L NaCN solution at a pH of 10.5 to 11 with controlled lime addition. Duplicate tests were conducted in bottles rolled for 24 hours with intermediate solution samples taken at 2, 4, 8, and 16 hours (Table 12.4).

Table 12.4: Fine Cyanidation Results (Lakefield 1989)

Test	Grind (% -75 µm)	pH	Cyanide		Carbon	Au Extraction %			Residue	Head
			(g/L)	(kg/t)	(g/L)	(4 hr)	(16 hr)	(24 hr)	(g/t)	(g/t)
A-2	93.5	11	2	0.99	nil	71.9	72.7	79.9	0.70	3.48
A-3	93.5	11	2	0.98	nil			82.6	0.59	3.39
B-2	85.8	11	2	0.82	nil	69.7	68	77.8	0.41	1.85
B-3	85.8	11	2	0.83	nil			76.7	0.41	1.76

Leach recovery on the A samples improved further to between 80% and 83% when ground to 93.5% passing 75 µm. This is estimated to be an 80% passing (P80) size of 50 to 65 µm. Leach recovery on the lower B samples was better at around 77%, but still lower than the A samples. This was due to the coarser grind of 86% passing 75 µm and lower head grade. Cyanide consumption was significantly higher in all instances at between 0.8 to 1.0 kg/t compared with the -2 mm tests.

Fine Grind Carbon-in-Leach Tests

The sample splits were ground to -75 µm and leached with 1 or 2 g/L NaCN solution and 10 g/L pre-attritioned carbon at a pH of 10.5 to 11 with controlled lime addition. The tests were conducted at 50% solids in bottles rolled for 48 hours with intermediate solution samples taken at 8 and 24 hours (Table 12.5).

Table 12.5: Fine Carbon in Leach Results (Lakefield 1989)

Test	Grind (% -75 µm)	pH	Cyanide		Carbon	Au Extraction %			Residue	Head
			(g/L)	(kg/t)	(g/L)	(8 hr)	(24 hr)	(48 hr)	(g/t)	(g/t)
A-6	93.5	11	1	0.86	10	76.4	80.6	83.3	0.61	3.66
A-7	93.5	11	2	1.62	10	79.3	82.2	83.1	0.59	3.50
B-6	85.8	11	1	0.84	10	71.0	74.0	75.6	0.43	1.76
B-7	85.8	11	2	1.35	10	72.0	74.2	75.5	0.43	1.74

The results of the CIL tests show no significant improvement over direct cyanidation on Composites A and B. Composite B extraction was lower than the A samples once again due to the coarser grind and lower head grade. Cyanide consumption was similar to the cyanidation tests at between 0.8 and 2 kg/t.

12.2 SGS Mineral Services 2011

During the first half of 2011, SGS Mineral Services in Vancouver, British Columbia, conducted a metallurgical test program on eight samples from the Springpole Gold Project. The primary objective was to conduct scoping level cyanide leaching tests on a range of material from the Springpole property.

12.2.1 Sample Preparation

Identification and inventory of the eight Springpole samples are shown in Table 12.6.

Table 12.6: Sample Identification and Inventory (SGS 2011)

Sample	Hole ID	From (m)	To (m)	Sample Weight (kg)	Zone/Material
Met 1	SP11-044	183	186	2.0	Portage Sulphide
Met 2	SP11-042	19	20	3.9	Portage Sulphide
Met 3	SP10-022	301	302	3.4	Portage Sulphide
Met 4	SP10-008	301	302	3.2	Portage Sulphide
Met 5	SP11-040	108	109	2.6	Oxide
Met 6	SP10-011	4	5	2.8	Portage Bridge
Met 7	SP10-026	290	291	3.8	Portage Sulphide
Met 8	SP11-031	31	32	4.3	Portage/Main

The samples were stage crushed to -2 mm, blended and riffled into 1 kilogram charges for cyanidation bottle leach tests. A 150 g sample was also split from each sample and submitted for gold, silver, arsenic, iron, sulfur, and multi-element inductively coupled plasma (ICP) scan head analyses. Due to the small amount of Gold Canyon Met 1 sample, only 900 g grind calibration and bottle cyanidation charges were prepared.

12.2.2 Head Analysis

Head analyses for the eight samples are shown in Table 12.7.

Table 12.7: Head Analysis (SGS 2011)

Sample	Au (g/t)	Ag (g/t)	As (g/t)	Sb (g/t)	Fe (%)	S (%)
Met 1	2.2	2.1	35	0.55	2.87	1.32
Met 2	1.3	1.7	42	0.82	5.59	2.33
Met 3	1.4	4.6	10	0.71	5.82	1.84
Met 4	1.3	3.9	33	0.56	5.61	4.10
Met 5	3.0	1.0	42	4.21	5.10	0.05
Met 6	1.5	1.9	74	1.70	2.57	1.40
Met 7	1.6	7.8	268	10.1	8.24	7.09
Met 8	1.7	1.5	194	1.69	6.99	4.16

Arsenic and antimony levels were low (except for Met 7 and 8) while the sulphur content of most samples was relatively high and, depending on gold association, may present challenges to cyanide leaching. Sample Met 5 was likely oxidized and clearly different in mineralogy.

12.2.3 Fine Grind Whole Feed Leach Tests

All samples were subjected to grind calibration to determine the time to achieve a P80 size of 65 microns. It was noted that the Met 1 sample took a long time to filter, suggesting the presence of clay material that could impact negatively liquid/solid separation processes.

The bottle roll leach tests conditions for the eight samples are outlined in Table 12.8.

Table 12.8: Bottle Roll Leach Test Conditions (SGS 2011)

Test Parameters	Condition
feed mass	1 kg
grind (80% passing)	65 microns
pulp density	40% solids
pH maintained	10.5 to 11 with hydrated lime
NaCN concentration	1 g/L
leach time	96 hours
solution samples	@ 4, 24, 48, 72, and 96 hr
residue assay	Au and Ag

The summarized results of the gold cyanidation tests are shown in Table 12.9. On average the 2010 samples yielded a leach extraction of 73% compared to 83% for samples generated during the 2011 drill program. The gold leach extraction after 24 hours for these eight samples averaged 79% and in the range 67 to 89%. Gold kinetic curves are shown in Figure 12.1 with extraction virtually complete at 24 hours with this grind size.

Table 12.9: Gold Leach Results (SGS 2011)

Sample	NaCN Cons (kg/t)	Au Assay			Au Extraction %				
		Residue, (g/t)	Calc Head (g/t)	Direct Head (g/t)	4 (hr)	24 (hr)	36 (hr)	48 (hr)	96 (hr)
Met 1	0.47	0.26	1.97	2.18	81	88	86	87	87
Met 2	0.45	0.54	1.72	1.32	58	70	70	70	69
Met 3	0.44	0.25	1.37	1.37	81	85	86	84	82
Met 4	1.51	0.38	1.32	1.26	39	67	70	73	71
Met 5	0.30	0.13	1.14	2.95	87	93	93	92	89
Met 6	0.37	0.5	1.51	1.54	66	69	70	69	67
Met 7	0.94	0.49	1.73	1.63	58	70	70	71	72
Met 8	0.37	0.23	1.83	1.65	80	90	89	89	88

SGS Mineral Services noted that all samples were amenable to cyanide leaching but appeared to fall into two groups. Samples Met 1, Met 3, Met 5 and Met 8 yielded gold leach extraction in excess of 80%, while samples Met 2, Met 4, Met 6 and Met 7 showed lower leach extractions around 70%. Further investigation would be required to understand the reasons behind certain samples being more refractory than others. It may be of relevance to note that, with the exception of Met 3, the samples that yielded higher leach extractions were from 2011 drill holes.

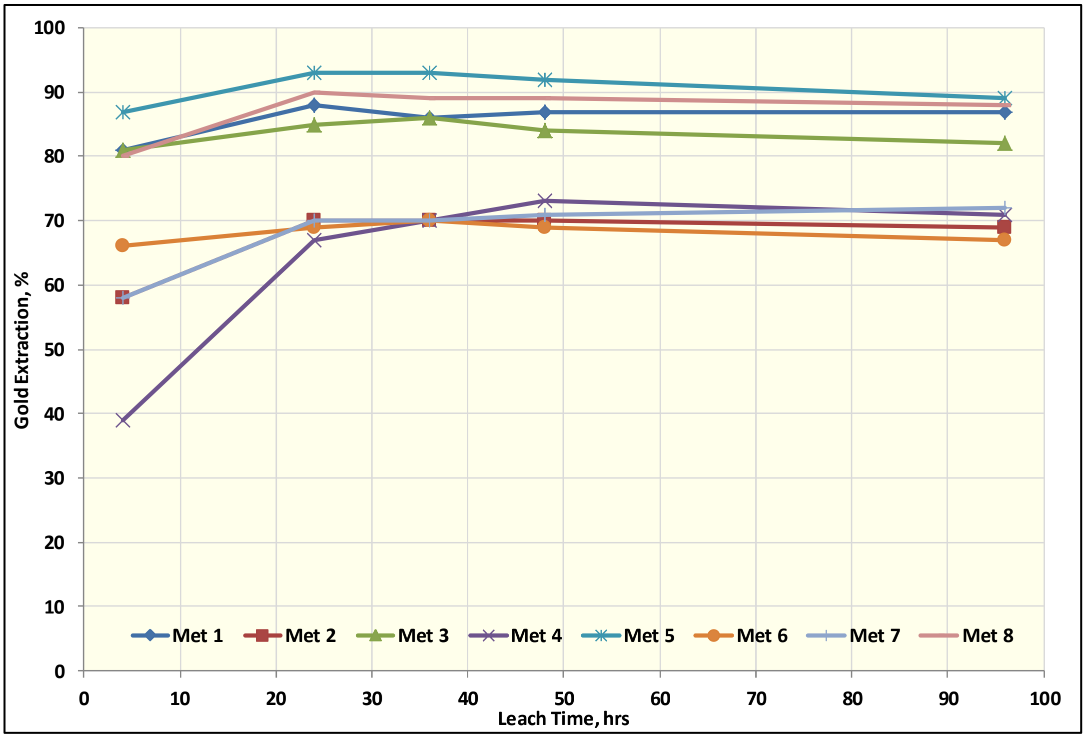


Figure 12.1: Gold Leach Kinetic Curves for SGS 2011 Samples

The summarized results of the silver cyanidation tests are shown in Table 12.10.

Table 12.10: Silver Leach Results (SGS 2011)

Sample	Ag Assay			Ag Extraction %				
	Residue (g/t)	Calc Head (g/t)	Direct Head (g/t)	4 (hr)	24 (hr)	36 (hr)	48 (hr)	96 (hr)
Met 1	0.60	2.27	2.12	66	73	72	73	74
Met 2	0.58	2.19	1.74	69	72	71	73	74
Met 3	0.78	5.66	4.63	85	87	85	88	86
Met 4	0.97	4.37	3.88	29	78	79	78	78
Met 5	0.76	1.49	1.02	47	50	49	52	49
Met 6	0.55	1.87	1.92	63	68	69	67	71
Met 7	1.23	8.89	7.80	82	84	85	87	86
Met 8	0.30	1.30	1.46	66	74	77	77	77

All samples were seen to be amenable to cyanide leaching except for Met 5. This sample had a low silver head grade, low sulphur analysis and relatively high antimony analysis. The sample was also observed to have a brownish color and likely oxidized. Silver leach extraction on the other seven samples ranged from 71 to 86%.

12.3 SGS Lakefield 2012

This testwork was conducted as part of the Springpole PEA project documented in this report and included comminution, flotation, leaching and environmental investigations.

12.3.1 Sample Preparation

Five samples were taken from individual holes, twinned to earlier drill holes from 2010 and 2011. Three were from the Portage zone (two sulphides and one oxide) and one from each of the East Extension, Camp, and Main zones. It was decided to separate SP11-61 into low grade (LG) and high grade (HG) samples, bringing the total to six. Table 12.11 shows the hole IDs and intervals sampled.

Table 12.11: Sample Identification and Inventory (SGS 2012)

Hole ID	From (m)	To (m)	Weight (kg)	Comments
SP11-61	50	150	150	Portage Zone (Oxide)
SP11-65	31	250	150	Portage Zone
SP11-66	16	40	58	East Pit Extension
SP11-69	206	511	150	Portage Zone
SP11-90	10	124	50	Camp/Main Zone

12.3.2 Head Analysis

Head assays showed the samples to be 3 to 4 g/t gold with 1 to 6 g/t silver and similar to the average grade of the resource. Sample 66 from the East Extension zone was much higher in gold at 12 g/t (Table 12.12). Sample 61 was much lower in sulphur but higher in arsenic and was later shown to be highly oxidized with iron sulphide minerals converted to oxides (similar to Met 5 from 2011).

Table 12.12: Head Analysis (SGS 2012)

Sample	Au (g/t)	Ag (g/t)	As (g/t)	Sb (g/t)	Fe (%)	S (%)
SP11-61 LG	2.3	2	129	12	6.6	0.05
SP11-61 HG	4.8	<0.5	143	10	9.2	0.05
SP11-65	1.9	4	63	<10	7.1	5.1
SP11-66	12	2	38	<10	4.8	3.2
SP11-69	1.2	6	<30	<10	7.0	4.0
SP11-90	1.2	1	36	<10	7.5	2.0

Duplicate screen metallic assays were performed at 150 mesh (106 µm) with the average results shown in Table 12.13. With the exception of the high grade 66 sample, most showed limited amounts of coarse gold, with only 1 to 5% present in the coarse fraction.

Full ICP analysis was completed on the sample heads with no deleterious elements noted.

A QEMSCAN Rapid Mineral Scan (RMS) was performed on the sample heads for the mineralogical assemblage. The main minerals are summarized in Table 12.14. Pyrite is the principal sulphide mineral in most of the samples with it converted to iron oxides in sample 61. Feldspar, mica and quartz are the most abundant host minerals.

Table 12.13: Screen Metallics Assay (SGS 2012)

	Head	+106 µm			-106 µm
Sample	Au (g/t)	Mass (%)	Au (%)	Au (g/t)	Au (g/t)
SP11-61 LG	2.3	2.45	0.73	0.70	2.36
SP11-61 HG	2.8	1.75	1.64	2.70	2.82
SP11-65	1.9	2.93	3.03	1.07	1.87
SP11-66	12.0	2.27	11.1	58.6	10.9
SP11-69	1.2	2.27	2.35	1.20	1.16
SP11-90	1.2	2.96	4.73	1.89	1.15

Table 12.14: Main Minerals by QEMSCAN RMS (SGS 2012)

Sample	Pyrite (%)	Fe Oxides (%)	K Feldspar (%)	Quartz (%)	Micas (%)	Others (%)	Main Other Minerals
SP11-61 LG	0.1	13.7	49.8	11.8	15.6	9.0	clays
SP11-61 HG	0.1	17.7	23.9	25.2	18.6	14.5	clays
SP11-65	9.5	1.2	35.8	12.7	21.5	19.3	plagioclase
SP11-66	5.4	0.4	47.0	13.7	11.7	21.8	calcite
SP11-69	10.5	0.8	37.2	8.3	33.3	9.9	plagioclase
SP11-90	3.7	0.9	5.0	22.8	24.9	42.7	ankerite

12.3.3 Material Hardness

Bond Work Index (BWI) tests were completed on five samples at a closing screen size of 150 µm. The results showed the oxide sample 61 was very soft at 7 kWh/t while the others were moderate at 12 to 17 kWh/t, with SP11-66 being the hardest.

With the primary grind size likely to be 80% passing 75 µm or finer, the specific energy requirements for both primary and secondary grinding need to be better defined with additional hardness testing on a number of samples from all areas of the Portage East Extension, Camp, and Main zones.

12.3.4 Gravity Recovery

Gravity concentration was performed prior to leaching or flotation to determine the expected level of gravity recoverable gold in the samples. Grind P80 sizes varied between 56 and 160 µm with 3 to 13% gravity recoverable gold for the lower grade samples (Table 12.15). The higher grade sample (SP11-66) achieved 48% gravity recoverable gold as was indicated in the screen metallics assay to contain a greater amount of coarse gold particles.

Initial leaching and flotation testwork were conducted on gravity recovery tailing.

A mineralogical study of the gold occurrence in the gravity concentrates was undertaken using the QEMSCAN Trace Mineral Search (TMS) mapping routine. The distribution of gold (and gold minerals) for each sample are shown in Figure 12.2.

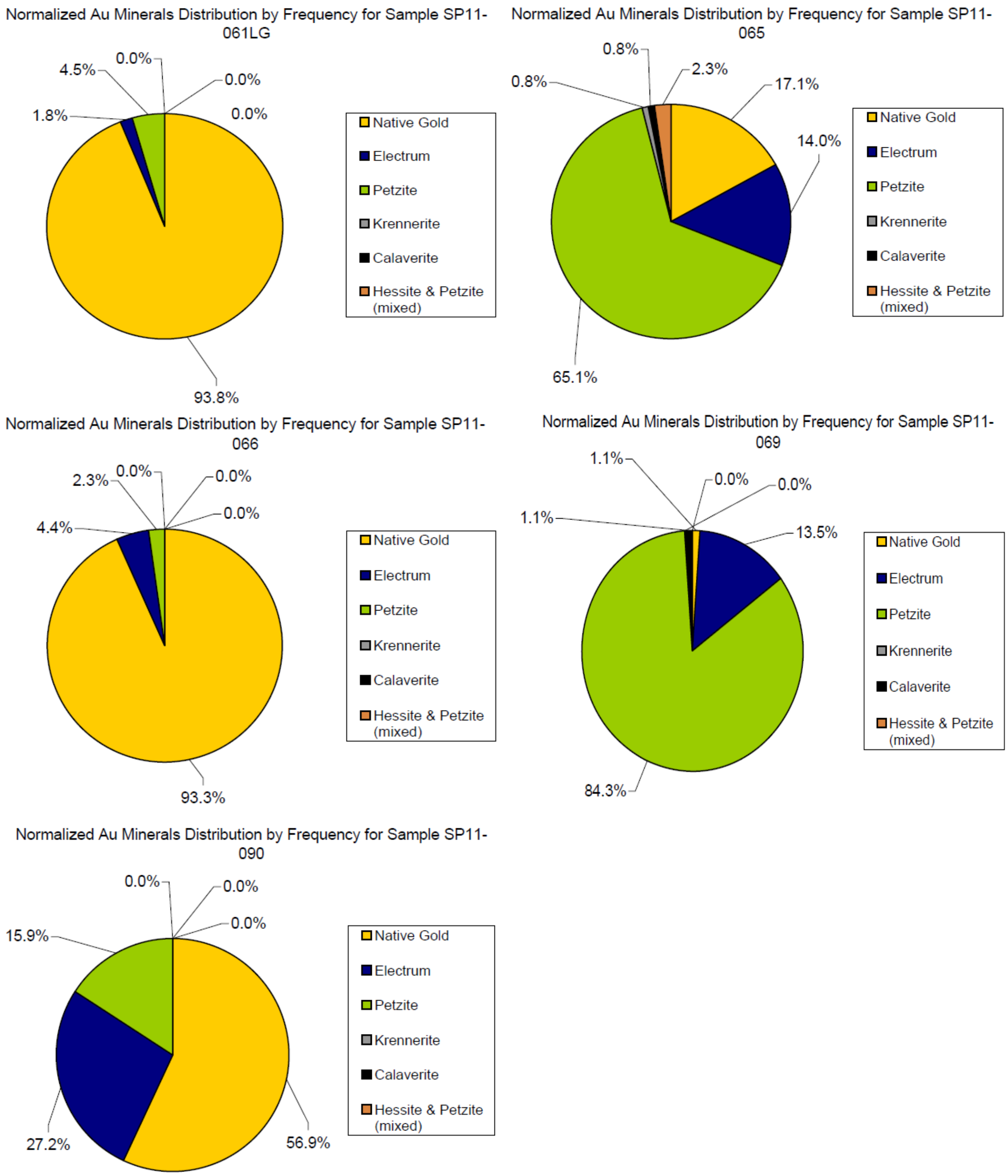


Figure 12.2: Gold Occurrence in Gravity Concentrates

Between 89 and 343 gold grains were identified during the QEMSCAN TMS with the high grade sample 66 producing the largest number of observed grains. The two Portage zone samples (65 and 69) showed gold was associated with petzite, a telluride mineral. Liberated gold grains were generally 20 µm in size while exposed gold grains were finer at <5 µm for the two Portage samples. Sample 69 showed 34% of the gold particles (only 89 observed) were locked in gangue.

The very fine gold grain size for the Portage samples is strong evidence as to why the leach extractions are sensitive to grind size.

12.3.5 Gravity Tailing Leaching

The gravity tailing from each sample was subjected to bottle roll, cyanide leaching at 1 and 2 g/L NaCN with/without the presence of pre-attritioned carbon over 48 hours. The results are summarized in Table 12.16. Similar to the results from earlier testwork, gold extractions varied from 70% to over 90% for the oxide sample 61. Carbon did not appear to improve the leach rates and the majority of the extraction was completed in 24 hours. The Portage samples (65 and 69) achieved around 70% gold extraction after 48 hours at the moderate grind size tested. Cyanide consumption was between 0.1 and 1.0 g/t, higher with carbon present and at higher cyanide concentrations.

Silver extraction was also variable, not correlated with gold and averaged around 70% for 48 hours.

12.3.6 Rougher Flotation

Rougher flotation tests were completed on each sample to recover a pyrite concentrate that would then be subjected to cyanide leaching. As indications were that the gold was both fine-grained and associated with pyrite, flotation concentration would allow the regrinding and leaching circuits to be smaller as they would treat only 20 to 30% of the mass. Table 12.15 summarizes the results.

Table 12.15: Rougher Flotation Test Results (SGS 2012)

Sample	Head Au	Feed P80 Size	Au Recovery		
			Gravity	Flotation	Grav + Float
	(g/t)	(µm)	(%)	(%)	(%)
Initial					
SP11-61 LG	2.3	128	6.5	75.3	76.9
SP11-61 HG	4.9	56	2.7	85.4	85.8
SP11-65	1.7	173	5.8	93.5	93.9
SP11-66	11.4	130	47.7	94.6	97.2
SP11-69	1.2	160	2.5	92.1	92.3
SP11-90	1.4	150	12.8	91.6	92.7
Oxide Flotation Conditions					
SP11-61 LG	2.3	101	7.5	74.0	76.0
	2.3	164	7.5	74.7	76.6

All samples except for the oxidized 61 sample recovered well in rougher flotation. Additional oxide flotation tests at different redox potential using sodium hydrosulphide were also completed, but the gold recovery remained at 75% after 20 minutes of rougher flotation.

Table 12.16: Gravity Tailing Leach Test Results (SGS 2012)

Sample	Head Au	Feed P80 Size	NaCN	NaCN	Carbon	Au Extraction			Residue Au	Ag Extraction
						24 hr	48 hr	Grav + Leach		48 hr
	(g/t)	(µm)	(g/L)	(kg/t)	(g/L)	(%)	(%)	(%)	(g/t)	(%)
SP11-61 LG	2.3	128	1	0.11		91.2	92.9	93.3	0.16	70.2
			2	0.20		92.7	93.4	93.8	0.14	70.3
			1	0.47	15		92.7	93.1	0.15	74.2
			2	0.71	15		93.5	94.0	0.14	72.1
SP11-61 HG	4.9	56	1	0.10		87.3	88.2	88.5	0.55	30.1
			2	0.24		92.5	92.2	92.4	0.35	28.5
			1	0.59	15		96.1	96.2	0.21	69.3
			2	0.84	15		96.1	96.2	0.21	58.9
SP11-65	1.7	173	1	0.11		69.0	70.4	72.1	0.47	76.9
			2	0.26		70.4	71.1	72.8	0.47	77.5
			1	0.31	15		70.6	72.3	0.48	77.9
			2	0.62	15		70.9	72.6	0.49	77.3
SP11-66	11.4	130	1	0.16		80.7	82.3	90.7	1.12	60.6
			2	0.63		81.9	81.9	90.5	1.09	56.9
			1	0.46	15		81.0	90.0	1.11	65.7
			2	1.06	15		80.9	90.0	1.10	66.5
SP11-69	1.2	160	1	0.08		66.4	69.3	70.0	0.36	73.2
			2	0.14		69.4	71.0	71.7	0.34	75.0
			1	0.37	15		70.3	71.1	0.34	79.2
			2	0.57	15		71.4	72.1	0.34	76.8
SP11-90	1.4	150	1	0.15		71.3	74.0	77.3	0.30	52.5
			2	0.52		71.4	73.0	76.4	0.30	53.7
			1	0.40	15		60.4	65.4	0.58	63.2
			2	0.85	15		73.6	77.0	0.31	66.2

Table 12.17: Rougher Concentrate Leach Test Results (SGS 2012)

Sample	Au Recovery			Rougher Conc			Au Extraction				Ag
	Gravity	Flotation	Grav + Float	Au	Regrind P80 Size	NaCN	24 hr	48 hr	Grav + Leach	Residue Au	Extraction 48 hr
	(%)	(%)	(%)	(g/t)	(µm)	(kg/t)	(%)	(%)	(%)	(g/t)	(%)
SP11-65	4.6	91.2	91.6	4.8	72	0.23	66.3	68.6	64.2	1.51	77.6
				4.8	66	0.26	64.4	67.2	63.0	1.58	77.4
				4.7	58	0.24	66.4	69.2	64.8	1.46	76.2
				4.7	54	0.24	66.9	69.7	65.3	1.44	78.2
SP11-69	1.5	88.6	88.8	4.5	60	0.30	39.3	50.6	45.7	2.24	76.9
				4.3	55	0.33	43.4	51.6	46.5	2.11	76.4
				4.4	48	0.38	43.0	53.6	48.3	2.05	78.5
SP11-90	15.3	89.7	91.3	5.0	66	0.54	77.8	73.6	71.2	1.34	58.9
				5.2	47	0.49	74.7	75.8	72.9	1.29	61.6
				5.3	48	0.52	75.0	76.3	73.2	1.27	36.0

NaCN maintained at 1 g/L

Table 12.18: Whole Feed Intensive Leach Test Results (SGS 2012)

Sample	Head Au	Feed P80 Size	NaCN	NaCN	Au Extraction			Ag
					24 hr	48 hr	Residue 96 hr Au	Extraction 96 hr
	(g/t)	(µm)	(g/L)	(kg/t)	(%)	(%)	(g/t)	(%)
SP11-65	1.7	149	5	0.56	75.8	76.7	0.39	82.7
	1.6	74	5	0.75	81.4	80.9	0.33	84.4
SP11-69	1.1	155	5	0.29	70.6	70.3	0.30	84.8
	1.1	61	5	0.89	75.8	75.1	0.24	86.9

Dissolved oxygen 20 to 29 mg/L and 40 to 45 deg C

12.3.7 Flotation Concentrate Leaching

As the proportion of oxide material (similar to sample 61) in the Portage zone is relatively small, work continued on samples 65, 69 and 90 as being more representative of the deposit.

Following gravity treatment, a rougher concentrate was recovered and reground to three different grind sizes (nominal P80 sizes of 65, 55, and 45 µm) prior to cyanide leaching. The results are summarized in Table 12.17. Unfortunately, the three regrind sizes tested did not differ significantly, and no clear relationship between extraction and concentrate regrind size was observed in the test results.

At 1 g/L cyanide concentration, leach extractions of gold did not exceed 80% after 48 hours, even at below 50 µm grinds. Cyanide consumptions were more stable at 0.23 to 0.54 g/t. Silver extraction was steady at almost 80%, except for sample 90.

12.3.8 Whole Feed Intensive Leaching

As an alternative to flotation concentration followed by leaching, whole feed leaching was tested without gravity pre-treatment. The two main Portage samples (65 and 69) were tested with the primary grind reduced from a nominal P80 size of 150 microns to below 70 µm with intensive leach conditions (high cyanide concentration, elevated temperature and dissolved oxygen) maintained for 96 hours (Table 12.18 for results).

The finer grind size achieved about 5% higher gold and 2% higher silver extractions. The higher cyanide, oxygen and temperature did not appear to improve leaching rates and test results indicated that most of the extraction was completed in 24 hours.

Diagnostic leach tests were completed on the two Portage samples to better understand the gold occurrence and association in the feed (not gravity concentrate). The results are shown in Figure 12.3 and indicate 72 to 75% of the gold was freely extractable by cyanide with a further 21% locked in sulphides (i.e., pyrite) at a grind size of 61 to 74 µm.

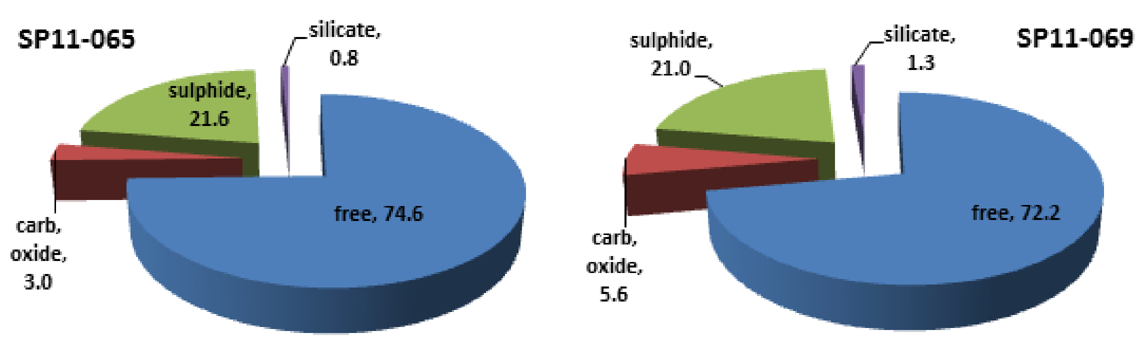


Figure 12.3: Diagnostic Leach Gold Distribution

These results confirm the whole feed leach extractions of around 70%. Finer grinding is needed to expose/liberate the gold particles from the sulphides and increase the extraction by up to 90%. The remaining gold was refractory in carbonates, oxides or silicates.

Due to the presence of telluride minerals identified in the gravity concentrate scan, thiosulphate leaching was performed on SP11-65 and SP11-69 as an alternative lixiviant. Initial tests showed 29 and 39% gold extraction for the two samples. Follow up tests which included resin showed 29 and 47% gold extraction after 24 hours with silver extractions of 58 and 52%, respectively.

Alternate leaching agents were not considered further as the relationship between cyanide leach extraction and particle size was well defined in the 2012 testwork results.

12.3.9 Geochemical Analysis

Whole rock, ICP and solid phase analysis were completed on a number of flotation tailings and leach residues and the content of the associated solutions was also analysed.

Acid Base Accounting (ABA) was conducted as well as ageing tests over 28 days on the rougher flotation tailings. In general, the flotation tailings samples were not net acid generating as the sulphide minerals had been recovered to concentrate. The whole feed leach residues contained up to 16% sulphur and were net acid generating, with net to acid producing ratios (NP:AP) under 1.

If the process flowsheet is whole feed leaching, it can be expected—based on analyses to date—that the leach residues will be net acid generating.

12.4 SGS Lakefield 2013

Test results showed gravity recovery was beneficial for higher grade material and whole feed leaching was sensitive to grind size as the gold particles were either fine (<5 µm for the two Portage samples) or locked with pyrite or gangue.

It was decided to re-test the two Portage samples without gravity recovery, at a very fine grind size and intensive leach conditions to see if gold extraction could be improved. As the Bond Work Index results suggested a relatively soft to moderate hardness, a primary grind of finer than 70 µm may prove to be economic if 85% or higher gold extraction was demonstrated.

Following the fine grind leach tests, additional rougher flotation tests followed by ultrafine regrinding was investigated to see how the flotation concentrate leaching was influenced by particle size.

12.4.1 Fine Grind Leach Tests

Both samples SP11-65 and SP11-69 were cyanide leached for 48 hours after being ground to a P80 size of around 38 µm and 30 µm. Cyanide levels were maintained at 5 g/L, oxygen levels were elevated along with the slurry temperature using a heat jacket. The results of the four tests are summarized in Table 12.19.

Compared to the earlier tests done on 61 to 74 micron feed, the 30 to 38 micron test results showed a 5 to 10% increase in 48 hour gold extraction. Grind size estimates were based on a laser-sizing method; therefore, subject to some error or variation due to the small sample size used.

Table 12.19: Whole Feed Intensive Leach Test Results (SGS 2013 vs. 2012)

	Sample	Head Au (g/t)	NaCN (kg/t)	Feed P80 Size (µm)	Au Extraction 48 hr (%)	Ag Extraction 48 hr (%)
2013	SP11-65	1.7	1.64	29	87.2	89.0
		1.7	1.94	38	85.5	88.7
	SP11-69	1.1	1.27	31	85.0	90.5
		1.2	2.16	39	82.8	90.3
2012	SP11-65	1.6	0.75	74	80.9	84.4
	SP11-69	1.1	0.89	61	75.1	86.9

NaCN maintained at 5 g/L; dissolved oxygen 20 to 29 mg/L and 40 to 45 deg C

The finer grinds resulted in higher cyanide consumptions with up to 2 kg/t for the -40 µm feed compared to 0.9 kg/t for -70 µm feed and 0.4 to 0.5 kg/t for similar grade samples tested in 2011 at lower cyanide concentrations.

These results confirmed the sensitivity of leach extraction to grind size. However, such a fine primary grind size, higher cyanide consumption as well as detoxifying and handling the very fine leach residues may result in higher costs (and issues) associated with the higher gold recovery. It is recommended that finer grinding be considered in future trade-off studies on the Springpole Gold Project.

12.4.2 Rougher Flotation with Fine Grind Leach Tests

With the higher extractions observed in the finely ground, whole feed leach results, additional tests were conducted using flotation to recover the sulphides and gold followed by fine regrinding and leaching of the concentrate. The earlier tests were not conclusive in the regrind size used and a regrind P80 size of 30 µm was targeted for these supplementary tests.

Following an extended rougher flotation period on a primary grind P80 size of around 100 µm, a concentrate of 2.8 to 5 g/t gold was generated which recovered 32 to 35% of the mass. This was reground to a P80 size of around 26 µm, and leached under intensive conditions for 48 hours. This included 5 g/L NaCN and dissolved oxygen of 20 to 29 mg/L at room temperature (Table 12.20).

Table 12.20: Rougher Concentrate Leach Test Results (SGS 2013)

Sample	Rougher Concentrate			Au Extraction*			Overall Recovery	
	Au (g/t)	Au Rec (%)	Ag Rec (%)	24 hr (%)	48 hr (%)	48 hr Residue, Au (g/t)	Au (%)	Ag (%)
SP11-65	5.1	90.8	89.7	81.8	86.4	0.69	78.5	79.8
SP11-69	2.8	89.2	92.1	73.3	83.2	0.55	74.2	80.8

**NaCN maintained at 5 g/L; P80 size of 26 µm, dissolved oxygen 20 to 29 mg/L at room temperature*

These results showed that, even with very fine regrinding, the overall gold recovery was 75 to 79% by concentrating the sulphides with flotation followed by leaching. Overall silver recovery was around 80% with fine regrinding.

The advantage of this flowsheet is the potentially acid-generating sulphides are concentrated in 25 to 35% of the feed and can be handled easier than leaching all of the feed. The cyanide detoxification circuit can also be considerably smaller.

However, whole feed leaching at a grind P80 size of 70 µm will achieve slightly better gold recovery compared to flotation plus fine regrinding ahead of leaching. The option of a finer primary grind size ahead of whole feed leaching will increase cyanide costs and make the leach residues more difficult to detox, thicken and possibly filter; however, should be investigated further in the future.

13 Mineral Resource Estimates

13.1 Introduction

The mineral resource statement presented herein represents the third mineral resource evaluation prepared for the Springpole Gold Project in accordance with the Canadian Securities Administrators' NI 43-101.

The mineral resource model prepared by SRK considers 512 core boreholes drilled by previous owners of the property and drilled by Gold Canyon during the period of 2003 to 2012. The resource estimation work was completed by Dr. Gilles Arseneau, PGeo (APEGBC #23474), an appropriate independent qualified person as this term is defined in NI 43-101. The effective date of the resource statement is October 17, 2012.

This section describes the resource estimation methodology and summarizes the key assumptions considered by SRK. In the opinion of SRK, the resource evaluation reported herein is a reasonable representation of the global gold and silver resources found in the Springpole Gold Project at the current level of sampling. The mineral resources were estimated in conformity with generally accepted CIM Estimation of Mineral Resource and Mineral Reserves Best Practices guidelines and are reported in accordance with the Canadian Securities Administrators' NI 43-101. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

The database used to estimate the Springpole Gold Project mineral resources was audited by SRK. SRK is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for porphyry gold mineralization and that the assay data are sufficiently reliable to support mineral resource estimation.

GEMS (6.4) was used to construct the geological solids, prepare assay data for geostatistical analysis, construct the block model, estimate metal grades and tabulate mineral resources. The Geostatistical Software SAGE2001 was used for variography.

13.2 Resource Estimation Procedures

The resource evaluation methodology involved the following procedures:

- Database compilation and verification,
- Construction of wireframe models for the boundaries of the Springpole gold mineralization,
- Definition of resource domains,
- Data compositing and capping for geostatistical analysis and variography,
- Block modelling and grade interpolation,
- Resource classification and validation,

- Assessment of "reasonable prospects for economic extraction" and selection of appropriate cut-off grades (COGs), and
- Preparation of the mineral resource statement.

13.3 Drill Hole Database

The Springpole Gold Project currently consists of three separate mineralized zones: East Extension, Camp or Main and Portage. The Portage zone is by far the largest of the three and represents more than 90% of the stated resource.

The entire Springpole database consists of 601 drill holes totalling 173,660 m. Of these, 89 drill holes totalling 27,808 m were discarded, 60 holes because of uncertainty relating to sampling methods and QA/QC, 27 holes because they were not drilled near the resource area, and 2 holes because assay results were not received at the time the resource was estimated

Of the 571 post-1986 drill holes, only those dating from 2003 through 2012 (331 drill holes) have documentation supporting a level of data verification and QA/QC sampling and analysis consistent with current NI 43-101 standards. Consequently, different restrictions were placed on which data could be used in which domain.

Due to the lack of detailed documentation—particularly for pre-2003 drilling and because of the apparent bias of the historical drilling in the Portage zone (Figure 13.1)—SRK decided to include the 1986–2003 drilling only in estimating the proportionately minor East Extension and Camp zones.

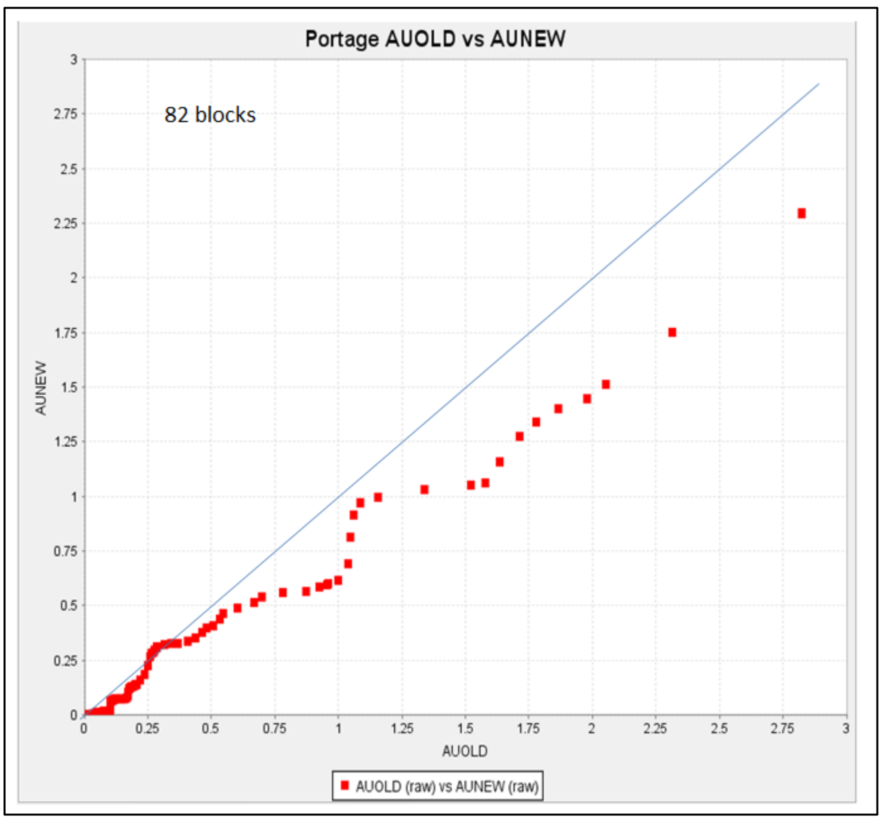


Figure 13.1: Comparison of Historic and Recent Drilling for the Portage Zone

Consequently, because of the good agreement between the recent and old drilling for the Camp and East Extension zones (Figure 13.2), it was decided that all historic drilling from 1986 to present would be included for estimation of these two zones. However, due to the lack of appropriate documentation, the estimates for the East Extension zone were restricted to the inferred classification. The Portage zone was estimated using only 2003 and later drill holes.

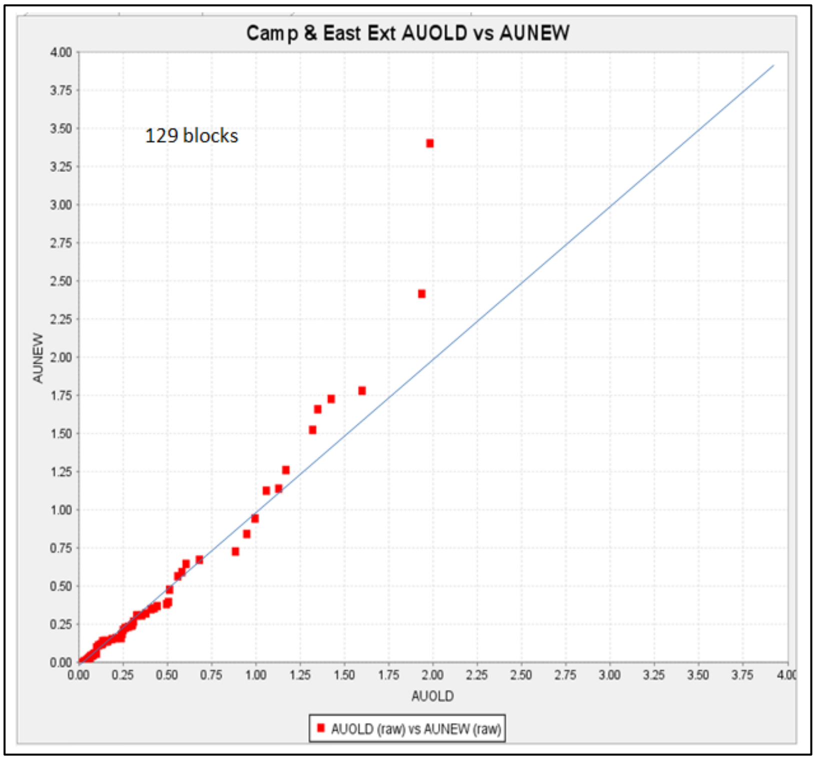


Figure 13.2 Comparison of Historic and Recent Drilling for the Camp and East Extension Zones

13.4 Core Recovery

Drill core recovery for both East Extension and Camp zones was generally very good with average recovery recorded as approximately 97%. For Portage, with areas of intense argillic and potassic alteration, core recovery was a much more significant issue, primarily affecting near surface intervals and intervals that appear to intersect a narrow zone of intense biotitic alteration.

SRK studied if there was any significant bias indicated, either high or low, as a function of core recovery. To a certain extent it was anticipated that more intense zones of alteration would also often reflect more intense mineralization.

Core recovery was generally recorded in 3 m intervals, with some data recorded in 1.5 m intervals. Consequently, for this analysis, it was decided to composite the core recovery values to the 3 m sample lengths and compare them with assay grades. The comparison indicates that the gold grade is generally lower with the increased recoveries (Figure 13.3). For this reason, SRK decided to model areas of low core recoveries and treat these areas as hard boundaries during grade interpolation.

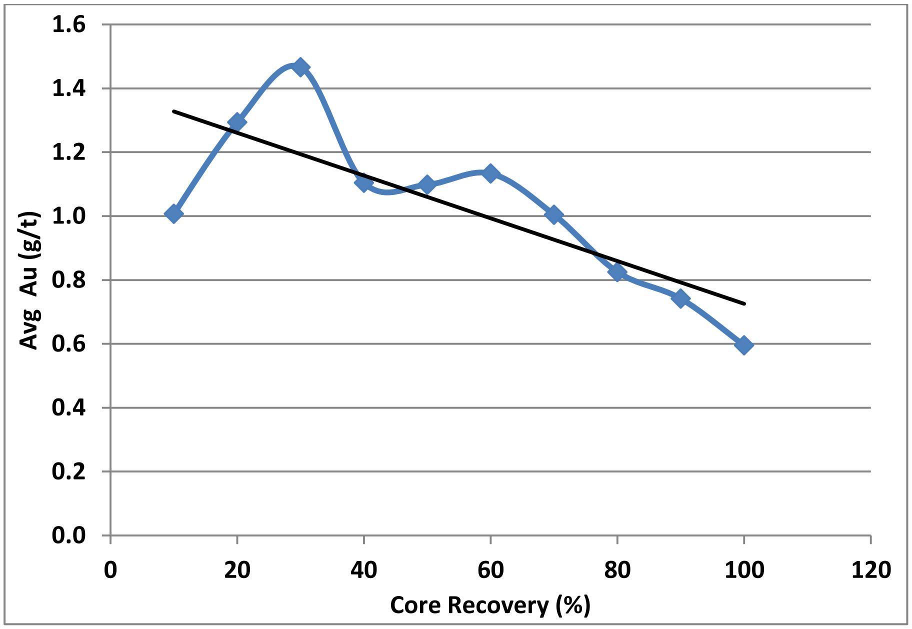


Figure 13.3: Gold Grade versus Core Recovery Relationship

13.5 Geological Domains

The Springpole Gold Project is comprised of three distinct domains: East Extension, Camp zone and Portage zones.

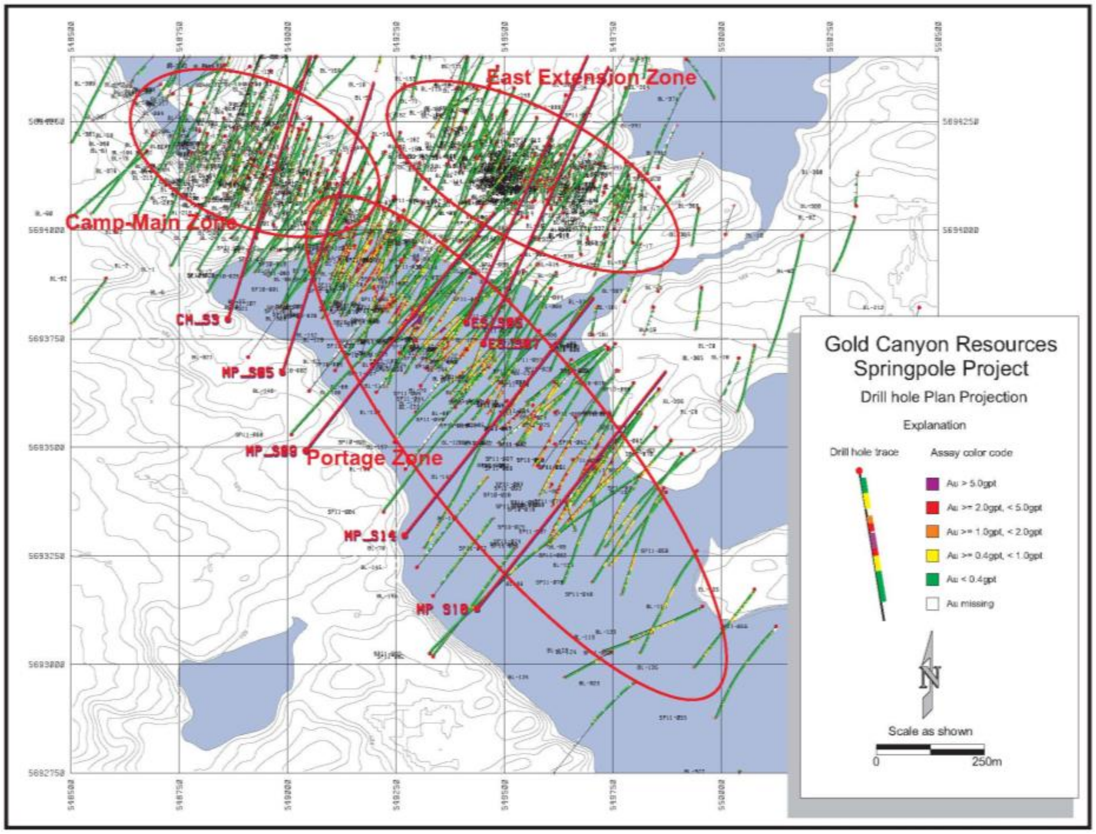
The East Extension zone lies to the east of Camp and Portage and is strike- oriented approximately 105° (N105°E). The zone exhibits erratic gold mineralization with slightly clustered "bonanza" grade drill hole intercepts intermixed with lower grade and barren intercepts.

The Camp zone lies to the north and, where the two domains overlap, above the Portage zone. The Camp zone strikes approximately 120° (N120°E) and part of the zone is very similar in character to East Extension with highly erratic grades showing very little spatial organization.

The Portage zone is by far the most significant domain, extending from beneath the southern extent of Camp zone for more than 1,500 m to the southeast. Other than location, the Portage zone exhibits few similarities with the other two domains. Also in contrast with East Extension and Camp, Portage has significant silver mineralization closely associated with gold. Drill-tested mineralization is extremely continuous with very little evidence of isolated erratic higher grade intervals. As drilled, Portage represents a zone of largely disseminated mineralization striking 135° (S45°E) and extending from the surface to a depth of over 400 m, on average approximately 150 m in width and over 1,500 m in length.

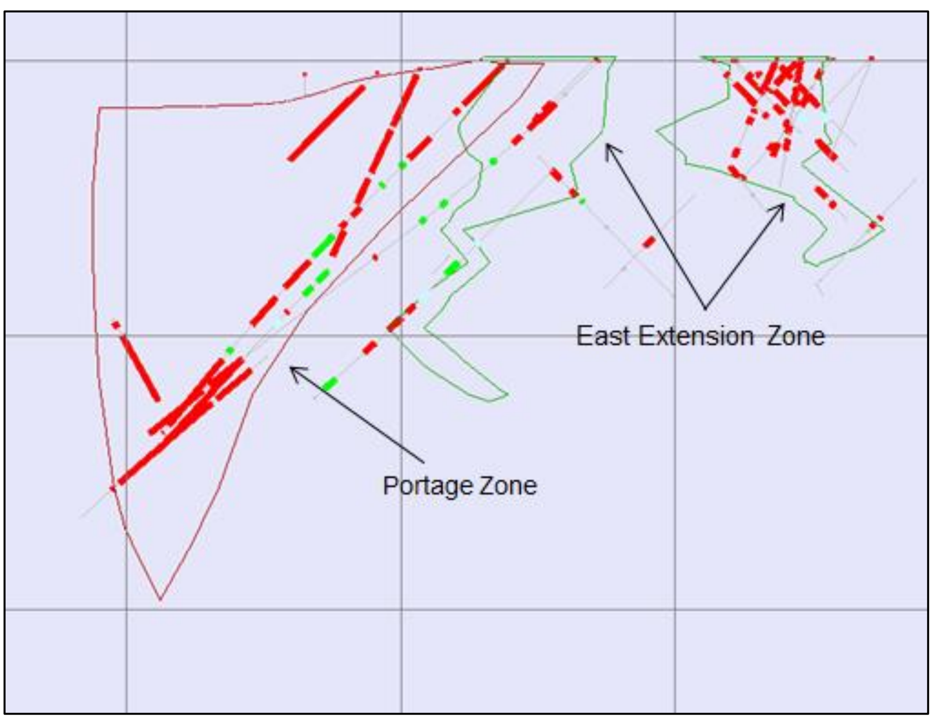
Geological domains were defined on sections spaced at 50 m intervals and a cut-off of 0.2 g/t was used to identify the geological domains on sections. Figure 13.4 shows the Springpole drill plan with the three geological domains and Note: Grid is 200 by 220 m. Green drill hole traces are > 0.2 g/t and red traces are > 0.3 g/t gold.

Figure 13.5 shows the domain boundaries on a typical section.



Source Gold Canyon 2011

Figure 13.4: Geological Domains for Springpole Gold Project



Note: Grid is 200 by 220 m. Green drill hole traces are > 0.2 g/t and red traces are > 0.3 g/t gold.

Figure 13.5: Cross Section 1100NW Looking NW Showing Portage and East Extension Domains

13.6 Surface Topography

Topography was provided in the form of a Drawing Interchange Format file containing data from a LIDAR survey with vertical precision of 1 m. The topographic surface beneath the portion of the lake overlying Portage was established by ground penetrating radar, Echo Sounder and sub-bottom profiling surveys conducted by Terrasond Ltd. of Palmer, Alaska, from the frozen surface (March 2011) and water lake surface (June 2011). These multiple surfaces were then merged to create a continuous surface to constrain the top of the block model. Overburden surface was modelled by extracting the base of the overburden from all available drill hole logs and generating a surface by simple triangulation of drill hole points.

13.7 Compositing

An analysis of the sample lengths within the mineralized domains shows that sample lengths are variable ranging from a low of 0.1 m to a maximum of 21 m; however, the majority of the samples are between 0.5 and 3 m in length with the largest proportion of the samples at 1 m in length (Figure 13.6). Most samples, 99%, are less than 3 m in length and for this reason SRK decided to composite all assays to a 3 m length within the mineralized envelopes. Compositing was generated from the drill collars and compositing was interrupted at domain boundaries. The compositing process generated 18,576 composites. A total of 274 composites with length less than 1.5 m were discarded from the database prior to resource estimation.

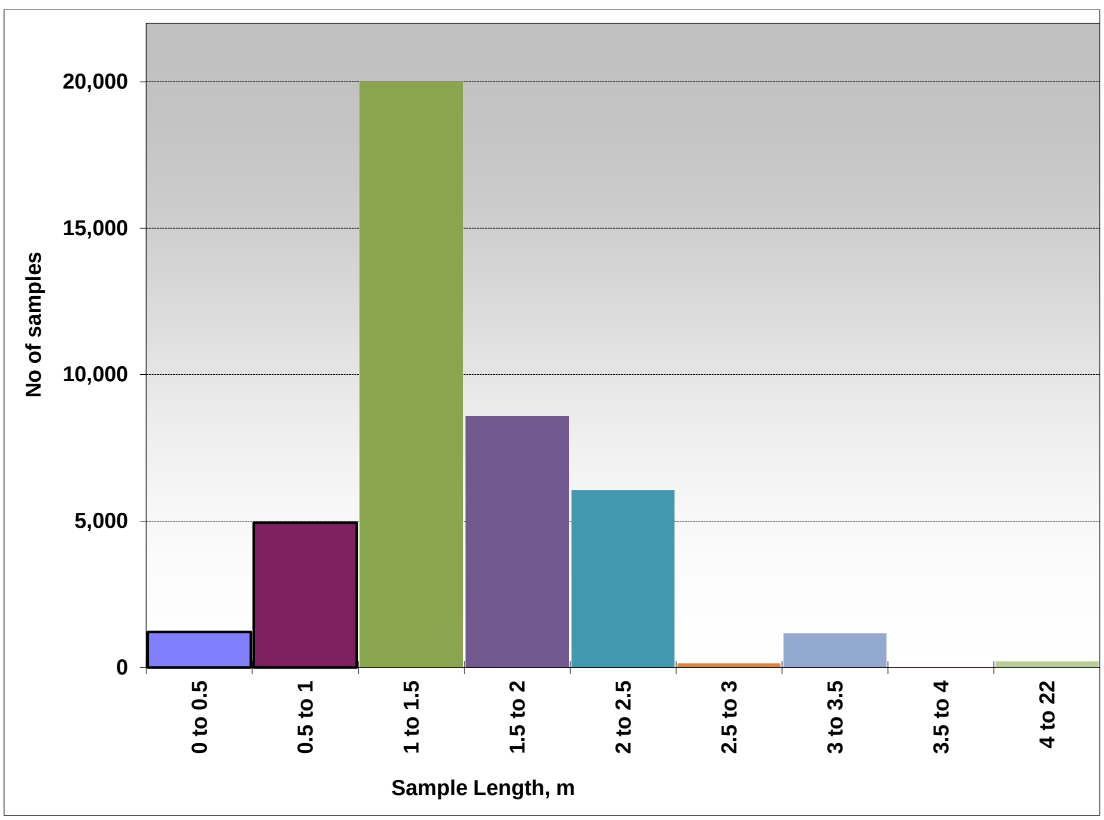


Figure 13.6: Histogram of Sample Lengths within Mineralized Domains

13.8 Grade Capping

The primary goal of grade capping is to identify and restrict the influence of suspected “outlier” grades in an estimate.

Grade capping for the Springpole Gold Project was carried out in two stages. First the assay data were investigated to determine if sample length could bias the average grade. An analysis of gold grade against sample length seems to indicate that sample length of less than 1 m has a significantly higher average grade than other sample lengths, indicating these samples were taken over a specific geological domain, perhaps quartz veins or narrow siliceous zones with visible gold (Figure 13.7). Most short sample lengths seem to have been taken from the Camp and East Extension zones; for this reason, SRK decided to treat these short sample lengths as a separate statistical population and caped these short assays prior to compositing. SRK capped all gold assays for sample lengths less than 1 m to 100 g/t gold prior to compositing.

All assays were then composited to 3 m lengths and all 3 m composites were evaluated for outliers by examining their distribution on cumulative probability plots and capped as outlined in Table 13.1.

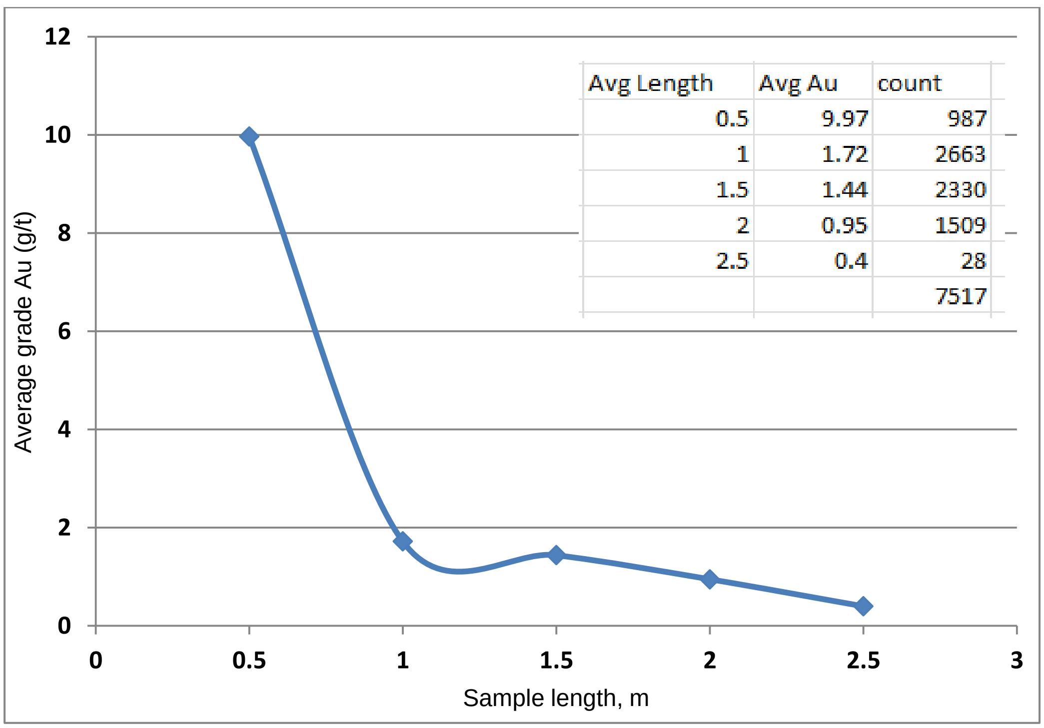


Figure 13.7: Comparison of Sample Length and Average Gold Grade

Table 13.1: Capping Levels for Springpole

Element	3 m Composite Capping Level
Au	25 g/t
Ag	200 g/t

13.9 Statistical Analysis and Variography

Statistical analyses were carried out on both the raw assay data and on the 3 m composited data. There are a total of 116,320 entries in the drill hole assay table for the Springpole Gold Project. Of these, 42,325 are within the interpreted wireframes representing the three mineralized domains. Some 8,191 historical assays within the mineralized domains were rejected because of uncertainties relating to quality control procedures. Of these, 61 samples were missing gold and silver assays because the results had not been received from the lab in time for the resource estimation, and 138 samples did not have gold assays because of missing core resulting from poor recovery. Data for these cores were omitted from the statistical analysis presented in Table 13.2. Statistical data for the 3 m composited data are presented in Table 13.3.

Table 13.2: Basic Univariate Statistical Information for Raw Assay Data

Zone	Max (g/t)	Min	Mean (g/t)	Std. Dev.	CoV	Count
East Extension Au	1568	0	3.75	41.22	10.98	3,583
Camp Au	341	0	1.27	8.12	6.40	2,899
Portage Au	168	0	0.79	2.11	2.67	27,453
Portage Ag	300	0	4.10	11.38	2.78	25,765[1]

[1]*Note: Silver assays only exist for the Portage zone and 1,688 samples from the Potage zone are missing silver assay data.*

Table 13.3: Basic Univariate Statistical Information for 3 m Composites

Zone	Max (g/t)	Min	Mean (g/t)	Std. Dev.	CoV	Count
East Extension Au	269.27	0	0.90	6.10	6.79	3,271
East Extension Capped Au	25.00	0	0.71	2.47	3.49	3,271
Camp Au	89.65	0	0.79	3.2	4.18	1,402
Camp Capped Au	25.00	0	0.73	2.18	2.99	1,402
Portage Au	95.30	0	0.82	1.65	2.06	12,964[1]
Portage Capped Au	25.00	0	0.79	1.34	1.70	12,964[1]
Portage Ag	280.51	0	4.50	10.35	2.30	12,174[2]
Portage Capped Ag	200	0	4.48	10.00	2.23	12,174[2]

[1]*Note: 665 composites have no gold values assigned to them; these were not used during grade interpolation.*
[2]*Note: 1,155 composites have no silver values assigned to them; these were not used during grade interpolation.*

Spatial continuity of gold and silver was evaluated with correlograms developed using SAGE 2001 version 1.08. The correlogram measures the correlation between data values as a function of their separation distance and direction. The distance at which the correlogram is close to zero is called the “range of correlation” or simply the range. The range of the correlogram corresponds roughly to the more qualitative notion of the “range of influence” of a sample or composite.

Variographic analysis was completed for gold in the Portage, Camp, and East Extension zones and for silver in the Portage zone. Directional correlograms were generated for composited data at 30° increments along horizontal azimuths. For each azimuth, correlograms were calculated at dips of 0, 30, and 60°.

A vertical correlogram was also calculated. Using information from these 37 correlograms, SAGE determines the best fit model using least square fit method. The correlogram model is described by the nugget (C_o) and two nested structure variance contributions (C_1, C_2) with ranges of the variance contributions and the model type (spherical or exponential). After fitting the variance parameters, the algorithm then fits an ellipsoid to the 37 ranges from the directional models for each structure. The final models of anisotropy are given by the lengths and orientations of the axes of the ellipsoids.

The experimental and modelled directional correlograms are presented in Appendix D. The correlogram models applied in the resource estimates in each domain are presented in Table 13.4.

Table 13.4: Gold and Silver Spherical Correlogram Parameters by Domain

Domain	Metal	Nugget C0	Sill C1, C2	Gemcom Rotations (RRR rule)			Ranges a1, a2		
				around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
Camp	Au	0.30	0.67	-27	57	52	26	8	5
			0.03	-27	57	52	61	57	180
East Extension	Au	0.30	0.48	-6	-67	-72	7	11	15
			0.22	-6	-67	-72	20	49	150
Portage	Au	0.19	0.56	31	8	34	20	40	20
			0.25	31	8	34	60	138	168
Portage	Ag	0.10	0.61	-48	30	27	22	9	18
			0.29	-48	30	27	100	76	174

13.10 Block Model and Grade Estimation

Block modelling was carried out in GEMS (6.4) software by Dr. Gilles Arseneau, PGeo, associate consultant with SRK. Block estimates were carried out in 10 by 10 by 6 m blocks using a percent model to weight partial blocks situated at zone boundaries. Block model parameters are defined in Table 13.5.

Table 13.5: Block Model Setup Parameters

	Model origin (WGS 84)	Block Size (m)	No. of blocks
Easting	548,500	10	220
Northing	5,692,400	10	210
Elevation	418	6	90

13.10.1 Grade Models

Grades were estimated by ordinary kriging with a minimum of 4 and a maximum of 15 composites with no more than three composites permitted from a single drill hole. Grade interpolations were carried out in three passes with each successive pass using a larger search radius than the preceding pass and only estimating the blocks that had not been interpolated by the previous pass. Table 13.6 summarizes the search parameters for each interpolation pass.

Table 13.6: Search Parameters by Zone and Metal

Metal	Zone	Pass	Rotation			Search Ellipse Size			Number of Composites		Max. Samples per DDH
			Z	Y	Z	X (m)	Y (m)	Z (m)	Min.	Max.	
Au	Camp	1	-84	7	-32	20	30	20	4	15	3
Au	Camp	2	-84	7	-32	40	60	60	4	15	3
Au	Camp	3	-84	7	-32	60	138	168	4	15	3
Au	East Ext	1	-84	7	-32	20	30	20	4	15	3
Au	East Ext	2	-84	7	-32	40	60	60	4	15	3
Au	East Ext	3	-84	7	-32	60	138	168	4	15	3
Au	Portage	1	-84	7	-32	20	30	20	4	15	3
Au	Portage	2	-84	7	-32	40	60	60	4	15	3
Au	Portage	3	-84	7	-32	60	138	168	4	15	3
Ag	Portage	1	-48	30	27	20	30	20	4	15	3
Ag	Portage	2	-48	30	27	40	60	60	4	15	3
Ag	Portage	3	-48	30	27	100	76	100	4	15	3

Uncapped gold was also estimated for all three domains for comparison against the capped results. The capped estimates were used for use in resource reporting and classification.

13.10.2 Bulk Density Model

There are 140 bulk density measurements in the Springpole database with an average of 2.89. SRK is of the opinion that while these are sufficient to estimate a mineral resource, the amount of bulk density data are very limited for a deposit of this size and additional data should be collected to develop a more robust density model. SRK recommends that Gold Canyon initiates an aggressive campaign of bulk density measurements for the next mineral resource update.

Gold Canyon collected samples for bulk density from 37 widely-spaced drill holes in the Portage zone. These samples attempted to represent the spectrum of alteration types and intensities, but are too few in number to derive volumetrically representative values for bulk density. The samples were tested by SGS Mineral Services in Toronto, Ontario using the waxed-immersion method to establish specific gravity values for each. The results ranged in value from a high of 3.08 to a low of 2.70 with an average of 2.89. The lowest values are generally representative of a narrow zone of intense argillic/biotitic alteration that will require additional drilling to define an accurate envelope.

In light of the paucity of specific gravity data, SRK decided to estimate the bulk density by inverse distance squared where data were nearby or assign an average density to unestimated blocks, as presented in Table 13.7.

Table 13.7: Bulk Density of Unestimated Blocks in the Model

Zone	Average S.G. of Un-Estimated Blocks
Camp	2.88
East Extension	2.88
Portage	2.65
Waste rock	2.88
Overburden	1.90

13.11 Model Validation

The Springpole resource block model was validated by completing a series of visual inspections. it was additionally validated by comparison of local “well-informed” block grades with composites contained within those blocks and by comparison of average assay grades with average block estimates along different directions – swath plots.

Figure 13.8 shows a comparison of estimated gold block grades with borehole composite assay data contained within those blocks within the mineralized domains and Figure 13.9 compares the silver grades. On average, the estimated blocks are similar to the composite data, although there is a large scatter of points around the x = y line. This scatter is typical of smoothed block estimates compared to the more variable assay data used to estimate those blocks. The thick white line that runs through the middle of the cloud is the result of a piece-wise linear regression smoother.

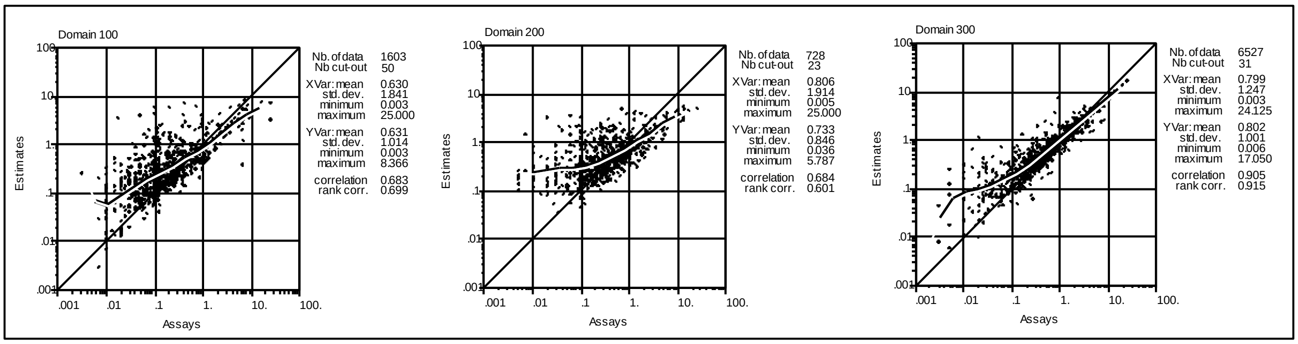


Figure 13.8: Comparison of gold grades for well-informed blocks

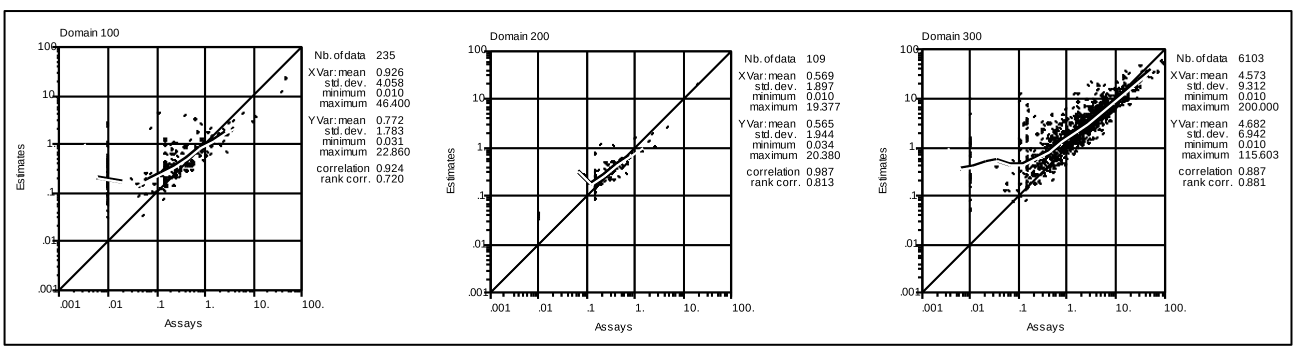
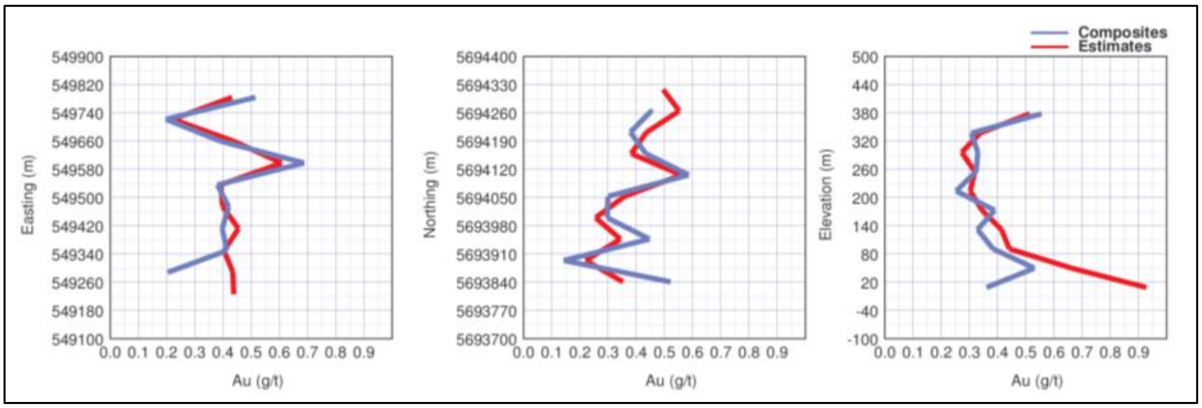


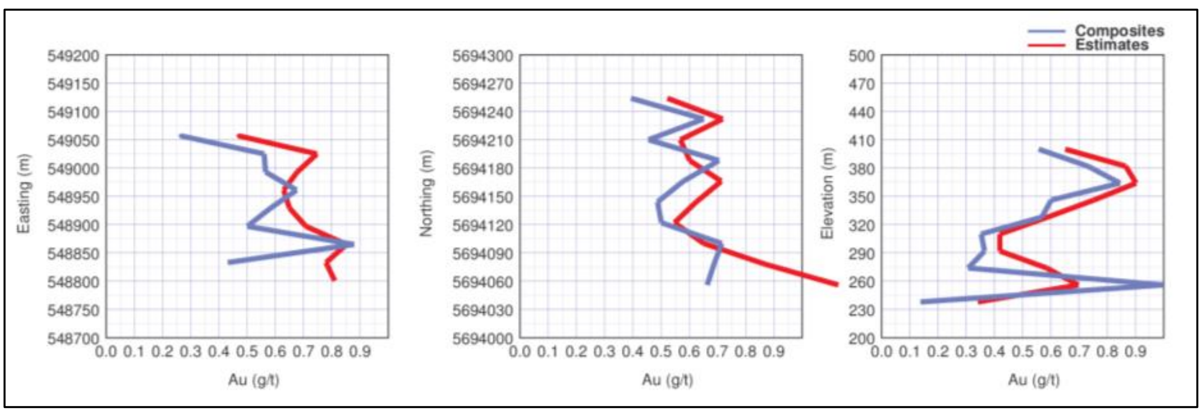
Figure 13.9: Comparison of Silver Grades for Well-Informed Blocks

Note that there are relatively few data for silver for the East Extension (domain 100) and Camp zone (domain 200). This is due to the fact that only the Gold Canyon drill holes had silver assay data for these two mineralized zones.

As a final check, average composite grades and average block estimates were compared along different directions. This involved calculating de-clustered average composite grades and comparison with average block estimates along east-west, north-south, and horizontal swaths. Figure 13.10 shows the swath plots in the mineralized zones, and Figure 13.11 shows the swatch plot for silver within the Portage zone. The average composite grades and the average estimated block grades are quite similar in all directions. Similar behaviour was documented for all other mineralized zones. Overall, the validation shows that current resource estimate is a good reflection of drill hole composited data.



(a)



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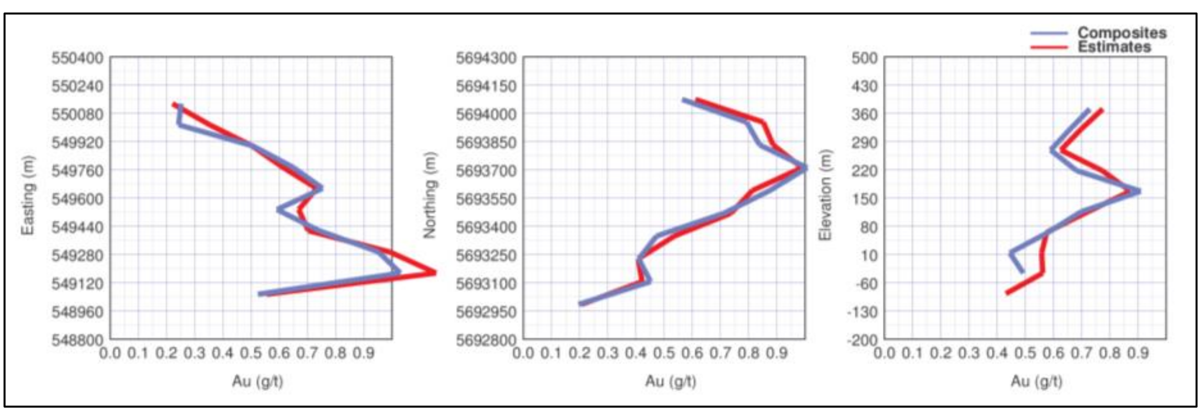

(c)

Figure 13.10: Swatch Plots for gold for (a) the East Extension, (b) the Camp, and (c) the Portage Zone

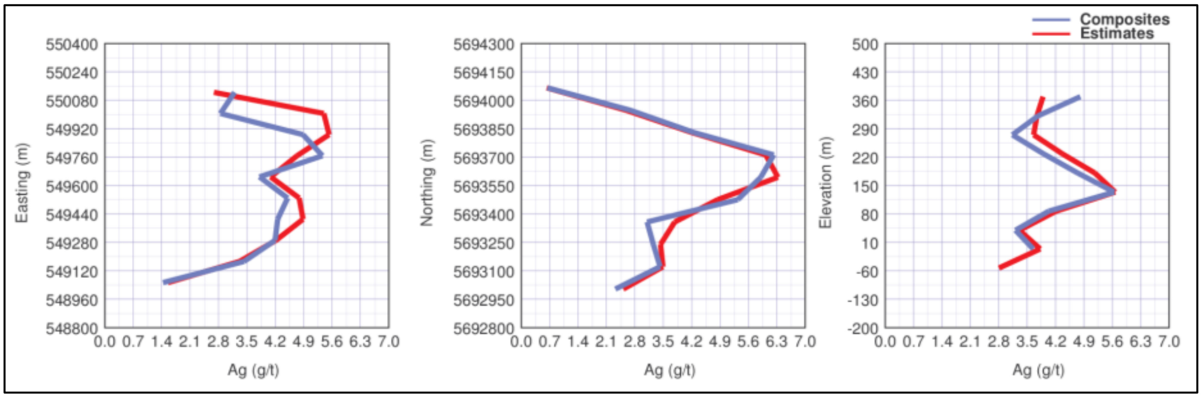

Figure 13.11: Swatch Plot for Silver within the Portage Zone

13.12 Mineral Resource Classification

Block model quantities and grade estimates for the Springpole Gold Project were classified by Dr. Gilles Arseneau, PGeo (APEGBC #23474), an appropriate independent qualified person for the purposes of NI 43-101. The classification was completed according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005).

Mineral resource classification is typically a subjective concept, industry best practices suggest that resource classification should consider the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate regular areas at similar resource classification.

SRK is satisfied the geological modelling honours the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource evaluation. The sampling information was acquired primarily by core drilling on sections spaced at 50 m.

The mineral resources were classified according to the following rules:

1) All blocks estimated for East Extension were assigned to an inferred category due to inclusion of drill hole data for which documentation of appropriate sample preparation, analysis, and QA/QC were lacking.

2) The Portage and Camp classification was based solely on the gold estimate. Silver, as a minor by-product, carries the classification associated with the gold. Any blocks that were estimated during Pass 1 or Pass 2 with at least two drill holes and six composites were classified as indicated mineral resources. All other interpolated blocks were classified as inferred mineral resources.

13.13 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a mineral resource as:

> *"(A) concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge"*

The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate COG taking into account extraction scenarios and processing recoveries. To meet this requirement, SRK considers that major portions of the Springpole Gold Project are amenable for open pit extraction.

To determine the quantities of material offering "reasonable prospects for economic extraction" by an open pit, SRK used a pit optimizer and reasonable mining assumptions to evaluate the proportions of the block model (Indicated and Inferred blocks) that could be "reasonably expected" to be mined from an open pit.

The optimization parameters were selected based on experience and benchmarking against similar projects (Table 13.8). The reader is cautioned that the results from the pit optimization are used solely for the purpose of testing the "reasonable prospects for economic extraction" by an open pit and do not represent an attempt to estimate mineral reserves. There are no mineral reserves on the Springpole Gold Project. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting COG.

Table 13.8: Assumptions Considered for Conceptual Open Pit Optimization

Parameter	Unit	Value
Au Price	$/oz	1,400
Ag Price	$/oz	15
Exchange Rate	$US/$CAD	1
Mining Cost	$/t mined	2
Processing	$/t of feed	12
General and Administrative	$/t of feed	2
Overall Pit Slope	degrees	45
Au Process Recovery	percent	80
Ag Process Recovery	percent	60
In Situ COG	g/t	0.4

SRK considers the blocks located within the conceptual pit envelope show “reasonable prospects for economic extraction” and can be reported as a mineral resource (Table 13.9).

Table 13.9: Mineral Resource Statement* (October 17, 2012)

Category	Quantity	Grade		Metal	
		Au	Ag	Au	Ag
	(Mt)	(g/t)	(g/t)	(Moz)	(Moz)
Open Pit**					
Indicated	128.2	1.07	5.7	4.41	23.8
Inferred	25.7	0.83	3.2	0.69	2.7

Source: Springpole Gold Project, Northwestern Ontario, SRK Consulting, October 17, 2012

**Mineral resources are reported in relation to a conceptual pit shell. Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate.*

***Open pit mineral resources are reported at a COG of 0.4 g/t gold. COGs are based on a gold price of $1,400/oz and a gold processing recovery of 80% and a silver price of $15/oz and a silver processing recovery of 60%.*

This resource model includes mineralized material in the Camp, East Extension and Portage zones spanning from geologic sections 0+1,500 m in the northwest to 0-250 m in the southeast. Along the axis of the Portage zone resource modelling includes mineralized material generally ranging from 340 to 440 m below surface.

Mineral resources that are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues. The quantity and grade of reported inferred resources in this estimation are uncertain in nature and there has been insufficient exploration to define these inferred resources as an indicated or measured mineral resource, and it is uncertain if further exploration will result in upgrading them to an indicated or measured mineral resource category. The mineral resources in this statement were estimated using current Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves.

13.14 Grade Sensitivity Analysis

The mineral resources of the Springpole Gold Project are variable depending upon the reported COG. To illustrate this sensitivity, the global block model quantities and grade estimates within the conceptual pit used to constrain the mineral resources are presented at different cut-off grades in Table 13.10 for the indicated mineral resource and in Table 13.11 for the inferred mineral resource. The reader is cautioned that the figures presented in this table should not be misconstrued with a mineral resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of COG. Figure 13.12 presents this sensitivity as grade tonnage curves for the indicated mineral resource and Figure 13.13 displays the inferred mineral resource.

Table 13.10: Indicated Block Model Quantities and Grade Estimates* at Cut-off Grades

COG Au (g/t)	Quantity (Mt)	Grade Au (g/t)	Grade Ag (g/t)
0.10	179.2	0.84	4.7
0.20	164.8	0.90	5.0
0.40	**128.2**	**1.07**	**5.7**
0.50	109.8	1.17	6.1
0.60	93.0	1.29	6.5
0.70	78.2	1.41	6.8
0.80	65.9	1.53	7.2
1.0	46.7	1.79	7.9
3.0	4.3	4.44	11.9

Source: Springpole Gold Project, Northwestern Ontario, SRK Consulting, October 17, 2012

* The reader is cautioned that the figures in this table should not be misconstrued with a mineral resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of COG.

Table 13.11: Inferred Block Model Quantities and Grade Estimates* at Cut-off Grades

COG Au (g/t)	Quantity (Mt)	Grade Au (g/t)	Grade Ag (g/t)
0.10	41.4	0.62	2.5
0.20	36.9	0.67	2.7
0.40	**25.7**	**0.83**	**3.2**
0.50	20.1	0.94	3.5
0.60	15.1	1.07	3.8
0.70	11.4	1.21	4.1
0.80	8.7	1.35	4.4
1.0	5.2	1.66	4.9
3.0	0.3	4.18	4.0

Source: Springpole Gold Project, Northwestern Ontario, SRK Consulting, October 17, 2012

* The reader is cautioned that the figures in this table should not be misconstrued with a mineral resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of COG.

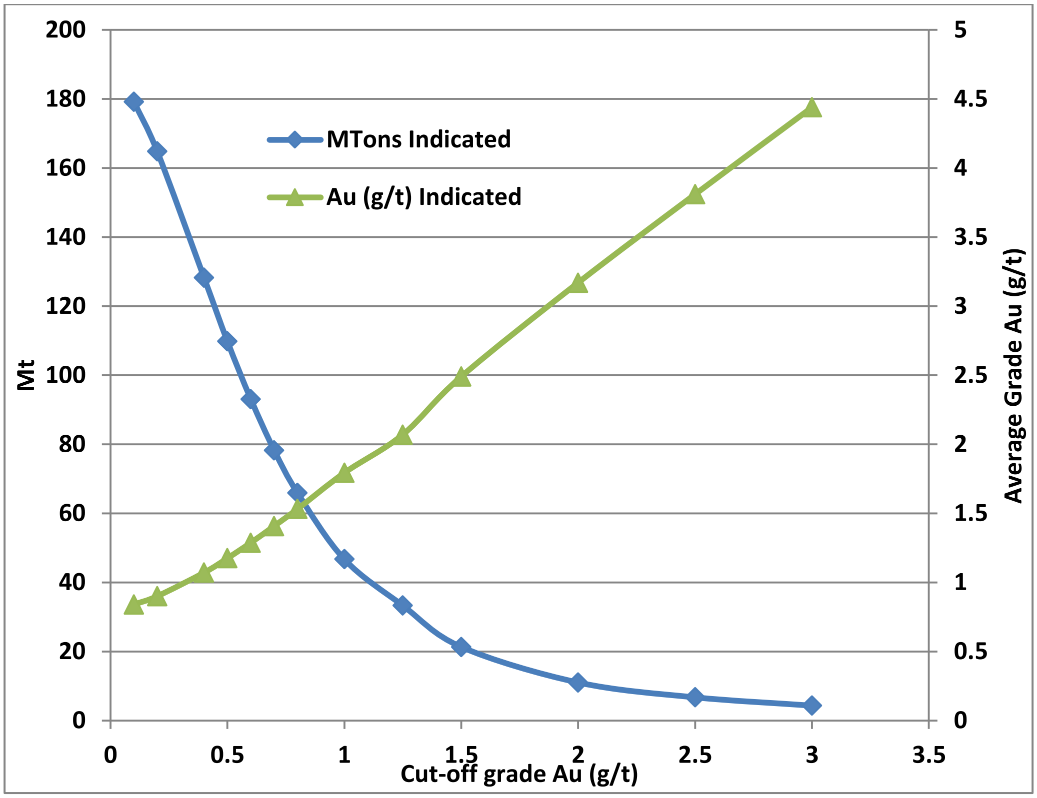

Figure 13.12: Grade Tonnage Curves for the Springpole Indicated Mineral Resource

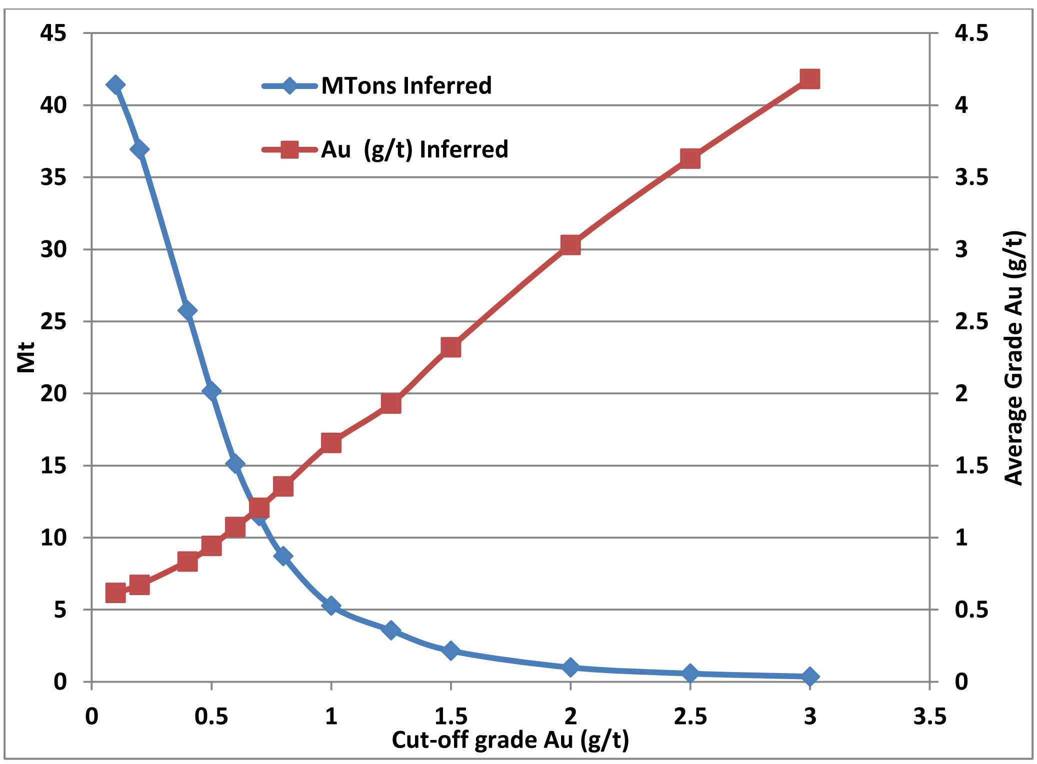

Figure 13.13: Grade Tonnage Curves for the Springpole Inferred Mineral Resource

13.15 Previous Mineral Resource Estimates

Mineral resources for the Springpole property were estimated and reported in a technical report filed on April 6, 2012 (Arseneau 2012). This resource model included mineralized material in the Main, East Extension, and Portage zones spanning from geologic sections 0+1,150 m in the northwest to 0-150 m the southeast. Along the axis of the Portage zone, resource modelling includes mineralized material generally ranging from the surface to a depth of 240 to 360 m below surface and included a total of 426 drill holes. Mineral resources were reported in accordance with NI 43-101 and are summarized in Table 13.12. These mineral resources are no longer current and are now replaced by the mineral resources presented in Table 13.9 of this report.

Table 13.12: Mineral Resource Statement of April 6, 2012

Classification	Tonnage (Mt)	Au (g/t)	Ag (g/t)	Au Contained (Moz)	Ag Contained (Moz)
Indicated	30	1.26	5.0	1.22	4.82
Inferred	60	1.27	6.0	2.45	11.58

14 Mineral Reserve Estimates

Inferred resources were used in the LOM plan with inferred resources representing 10% of the material planned for processing. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. Mineral reserves can only be estimated as a result of an economic evaluation as part of a preliminary feasibility study or a feasibility study of a mineral project. Accordingly, at the present level of development, there are no mineral reserves at the Springpole Gold Project.

15 Mining Methods

15.1 Open Pit Optimization

15.1.1 Input Parameters

The 3-D mineral resource block model, as developed by Dr. Gilles Arseneau, PGeo, Associate Consultant with SRK, was used as the basis for deriving the economic shell limits for the Springpole Gold Project. The block model dimensions were 10 m x 10 m x 6m.

Estimates were made for gold and silver price, mining dilution, process recovery, offsite costs and royalties. Mining, processing, and general administration OPEX were also calculated based on calculated processing throughput and, along with geotechnical parameters, formed the basis for open pit optimization (Table 15.1). The open pit mining costs assumed owner-operated mining.

Table 15.1: Mine Planning Optimization Input Parameters

Item	Unit	Value
Metal Prices		
Au	$/oz	1,300
Ag	$/oz	25
Recovery to Doré		
Au	%	80
Ag	%	85
Smelter Payables		
Au in doré	%	99.5
Au deduction in doré	g/t	0
Ag in doré	%	98
Ag deduction in doré	g/t	0
Offsite Costs		
Au refining/transportation charge	$/oz pay Au	5
Other Parameters		
Royalties	%	3
Operating Costs		
Waste Mining Cost	$/t	2.4
Mineralized Material Mining Cost	$/t	2.4
Processing and G&A Cost	$/t milled	13.02
Pit Slope Angles	overall degrees	35 to 50
Mining Dilution	%	5.0
Mining recovery	%	100
Strip ratio (est.)	t:t	1.7
Internal NSR cut-off	$/t	13.67
Processing rate	t/day	20,000

The mineral inventory block model for the Springpole deposit was then used with Whittle™ open pit optimization software to determine optimal mining shells. This evaluation included the aforementioned parameters.

The economic shell limits included indicated and inferred mineral resources. Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them to be categorized as mineral reserves. There is no certainty that the inferred resources would be upgraded to a higher resource category.

15.1.2 Cut-Off Grade

Table 15.2 summarizes the parameters used, along with incremental (or mill) COG calculations (based on NSR) and mining dilution. The incremental (or mill) COG incorporates all OPEX except mining. This incremental cut-off is applied to material contained within an economic pit shell where the decision to mine a given block was determined by the Whittle optimization. This mill cut-off was applied to all of the estimates that follow.

Table 15.2: Cut-Off Grade Calculations Used in Pit Optimization

Item	Unit	PEA	
		Mine COG	Mill COG
Revenue, Smelting & Refining			
Au Price	$/oz	$1,300	
Payable Metal	%Au	99.5%	
Refining/Transport	$/oz	$5	
Royalties @ 3% of NSR	$/oz	$38.66	
Net Au Price	$/oz	$1,249.85	
Net Au Price	US$/g	$40.18	
OPEX Estimates			
Waste Mining Cost	$/t mined	$2.4	
Mineralised Material Mining Cost	$/t mined	$2.4	
Strip Ratio (Estimated)	t:t	2.2	
Mining Cost	$/t milled	$7.68	
Processing Cost	$/t milled	$10.7	$10.7
G&A	$/t milled	$2.3	$2.3
Total Site Cost	$/t milled	$20.7	$13.0
Process and Mining Losses			
Process Recovery (Au)	%	80%	
Dilution	%	5%	
Open Pit COG			
In-Situ Au COG	g/t Au	0.68	0.43
Ag Contribution (Est.)	% of Au value	5%	
In-Situ Cut-Off Au Grade (Equivalent)	g/t Au	**0.64**	**0.41**
In-Situ Cut-Off NSR Value	$/t	**$21.74**	**$13.67**

Marginal=Mill=Incremental COG

Resource=Mine=External COG

It should be noted that the gold COGs are for comparative purposes only, as the NSR cut-off was used for Whittle optimization and mineral inventory estimates. The incremental NSR cut-off value of $13.67/t was applied to the mineral inventory estimates that follow. This equates to an incremental gold COG of approximately 0.41 g/t gold).

15.1.3 Optimization Results

A series of optimized shells were generated for the Springpole deposit based on varying revenue factors. The results were analyzed with shells chosen as the basis for ultimate limits and preliminary phase selection.

The results of the pit optimization evaluation on the deposit for varying revenue factors values are summarized in Table 15.3 and Figure 15.1 to Figure 15.3 for inferred and indicated resources. Note the NPV in this optimization summary does not take into account CAPEX and is used only as a guide in shell selection and determination of the mining shapes. The actual NPV of the project is summarized in the economics section of this report.

Whittle produces both “best case” (i.e., mine out shell 1, the smallest shell, and then mine out each subsequent shell from the top down, before starting the next shell) and “worst case” (mine each bench completely to final limits before starting next bench) scenarios. These two scenarios provide a bracket for the range of possible outcomes. The shells were produced based on varying revenue factors (0.3 through to 1.4 of base case) to produce the series of nested shells with the NPV results shown.

Table 15.3: Overall Optimization Results

Final	Revenue	Mine	Total Mineralized Diluted					Waste	Strip	Total	Total CF	NPV Best	NPV Worst
Pit	Factor	Life	(Mt)	Au (g/t)	Au (Moz)	Ag (g/t)	Ag (Moz)	(Mt)	Ratio	(Mt)	($M)	disc ($M)	disc ($M)
1	0.30	0.9	6	2.08	0	9.74	2	13	2.0	19	337	323	323
2	0.32	1.0	8	1.96	0.5	9.39	2.3	14	1.8	21	378	360	360
3	0.34	1.4	10	1.76	0.6	9.15	3.0	16	1.5	26	452	428	427
4	0.36	1.7	13	1.67	0.7	8.77	3.5	18	1.4	30	511	481	477
5	0.38	2.0	14	1.60	0.7	8.08	3.7	19	1.3	33	551	516	509
6	0.40	5.8	42	1.31	1.8	6.20	8.3	59	1.4	101	1,152	981	939
7	0.42	6.1	44	1.31	1.8	6.24	8.8	63	1.4	107	1,198	1,013	969
8	0.44	6.3	45	1.31	1.9	6.15	8.9	65	1.4	110	1,224	1,032	985
9	0.46	6.4	46	1.30	1.9	6.16	9.1	67	1.5	113	1,249	1,050	1,000
10	0.48	6.6	48	1.30	2.0	6.15	9.4	70	1.5	118	1,279	1,072	1,016
11	0.50	6.9	49	1.29	2.0	6.10	9.7	73	1.5	123	1,309	1,092	1,032
12	0.52	8.9	64	1.22	2.5	6.11	12.6	97	1.5	161	1,540	1,242	1,131
13	0.54	9.2	66	1.21	2.6	6.11	13.0	104	1.6	171	1,582	1,268	1,144
14	0.56	9.3	67	1.21	2.6	6.11	13.2	107	1.6	174	1,597	1,277	1,149
15	0.58	9.5	68	1.21	2.6	6.10	13.4	109	1.6	177	1,610	1,285	1,154
16	0.60	9.6	69	1.20	2.7	6.07	13.6	113	1.6	183	1,628	1,296	1,159
17	0.62	9.8	70	1.20	2.7	6.05	13.7	116	1.6	186	1,639	1,302	1,162
18	0.64	9.9	72	1.20	2.8	6.02	13.9	119	1.7	191	1,654	1,311	1,165
19	0.66	10.0	72	1.19	2.8	6.01	14.0	121	1.7	193	1,662	1,316	1,166
20	0.68	14.0	91	1.14	3.3	5.79	17.0	180	2.0	271	1,859	1,410	1,169
21	0.70	14.3	93	1.14	3.4	5.74	17.2	187	2.0	281	1,879	1,420	1,166
22	0.72	14.7	97	1.13	3.5	5.70	17.7	197	2.0	294	1,907	1,434	1,161
23	0.74	15.0	99	1.12	3.6	5.67	18.0	203	2.1	301	1,922	1,440	1,158
24	0.76	15.3	100	1.12	3.6	5.66	18.3	208	2.1	309	1,934	1,446	1,154
25	0.78	15.6	103	1.11	3.7	5.62	18.6	215	2.1	318	1,949	1,453	1,146
26	0.80	16.2	107	1.09	3.8	5.57	19.2	224	2.1	332	1,970	1,462	1,133
27	0.82	17.9	117	1.06	4.0	5.48	20.6	251	2.2	368	2,018	1,481	1,101
28	0.84	20.6	133	1.02	4.4	5.52	23.6	303	2.3	436	2,091	1,505	1,035
29	0.86	20.9	136	1.02	4.4	5.54	24.1	312	2.3	447	2,101	1,508	1,020
30	0.88	21.2	137	1.01	4.5	5.53	24.4	318	2.3	455	2,107	1,510	1,013
31	0.90	21.3	138	1.01	4.5	5.52	24.5	321	2.3	459	2,109	1,511	1,010
32	0.92	21.6	140	1.01	4.5	5.52	24.8	327	2.3	467	2,113	1,512	998
33	0.94	21.7	141	1.01	4.6	5.51	25.0	333	2.4	474	2,116	1,513	991
34	0.96	22.3	144	1.00	4.6	5.49	25.4	343	2.4	487	2,119	1,514	974
35	0.98	22.5	145	1.00	4.7	5.47	25.6	349	2.4	494	2,120	1,514	964
36	**1.00**	**22.5**	**146**	**1.00**	**4.7**	**5.47**	**25.6**	**350**	**2.4**	**496**	**2,120**	**1,514**	**961**
37	1.02	23.3	149	0.99	4.8	5.44	26.0	368	2.5	517	2,119	1,514	932
38	1.04	23.5	150	0.99	4.8	5.43	26.2	374	2.5	524	2,118	1,513	922
39	1.06	23.6	151	0.99	4.8	5.41	26.3	380	2.5	531	2,117	1,513	910
40	1.08	23.7	152	0.99	4.8	5.40	26.4	384	2.5	536	2,115	1,512	903

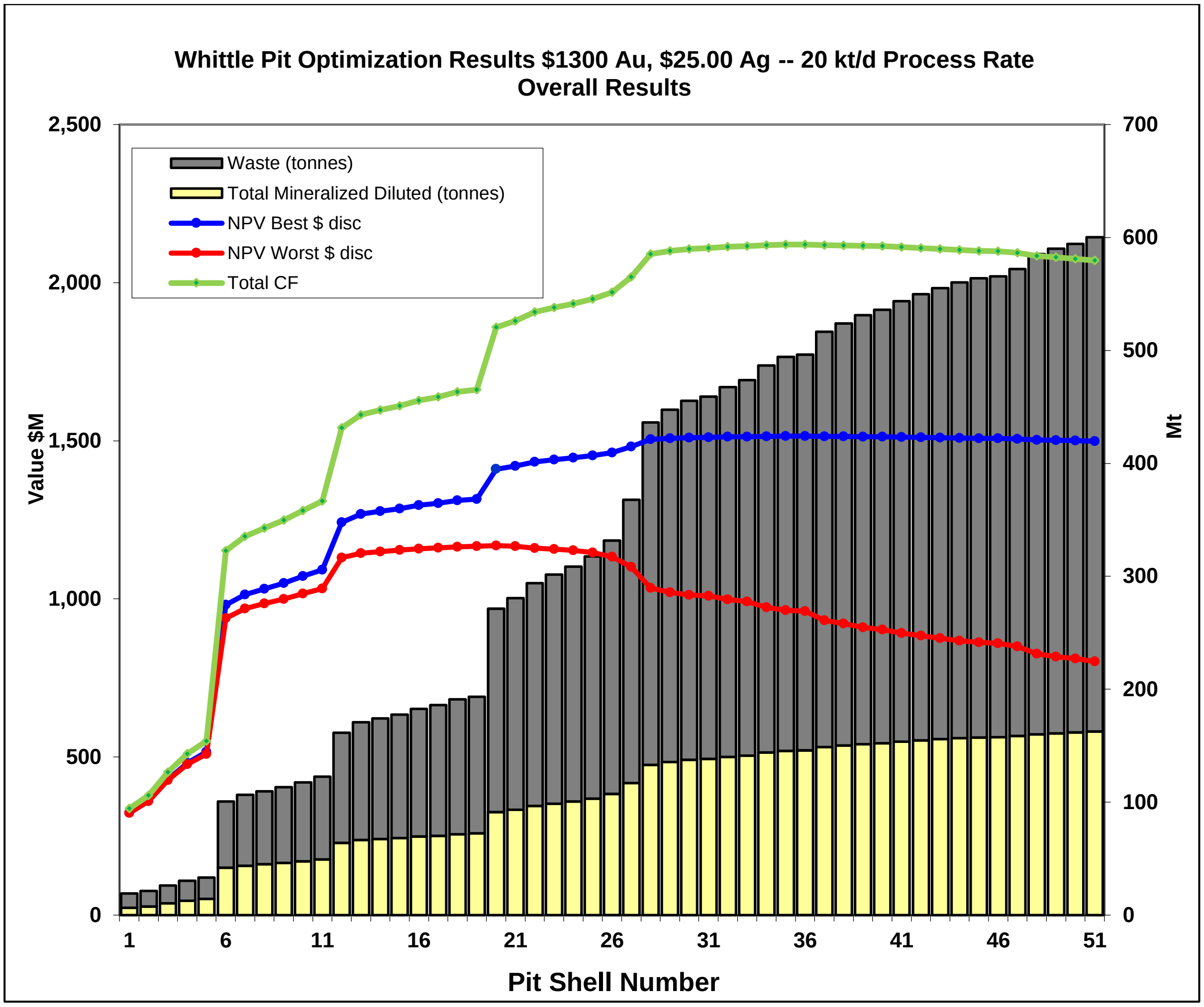

Figure 15.1: Open Pit Optimization Cumulative Results

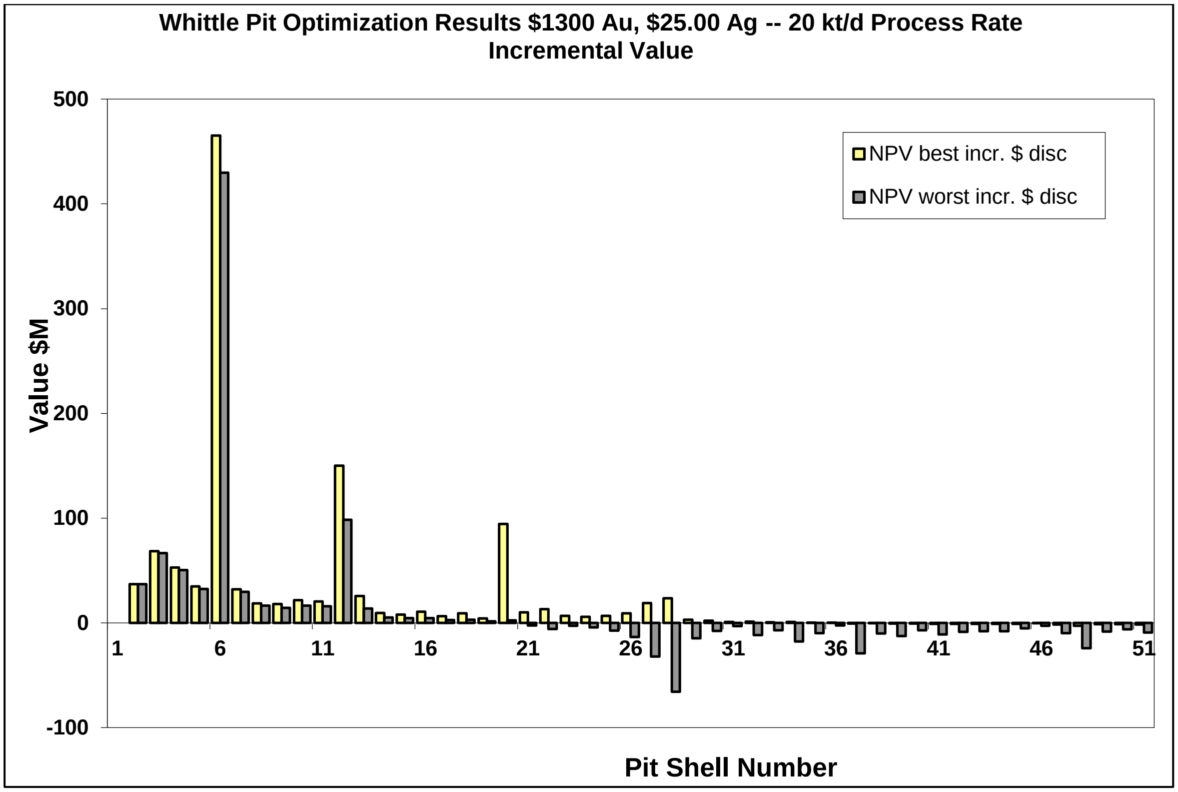

Figure 15.2: Open Pit Optimization Incremental Value Results

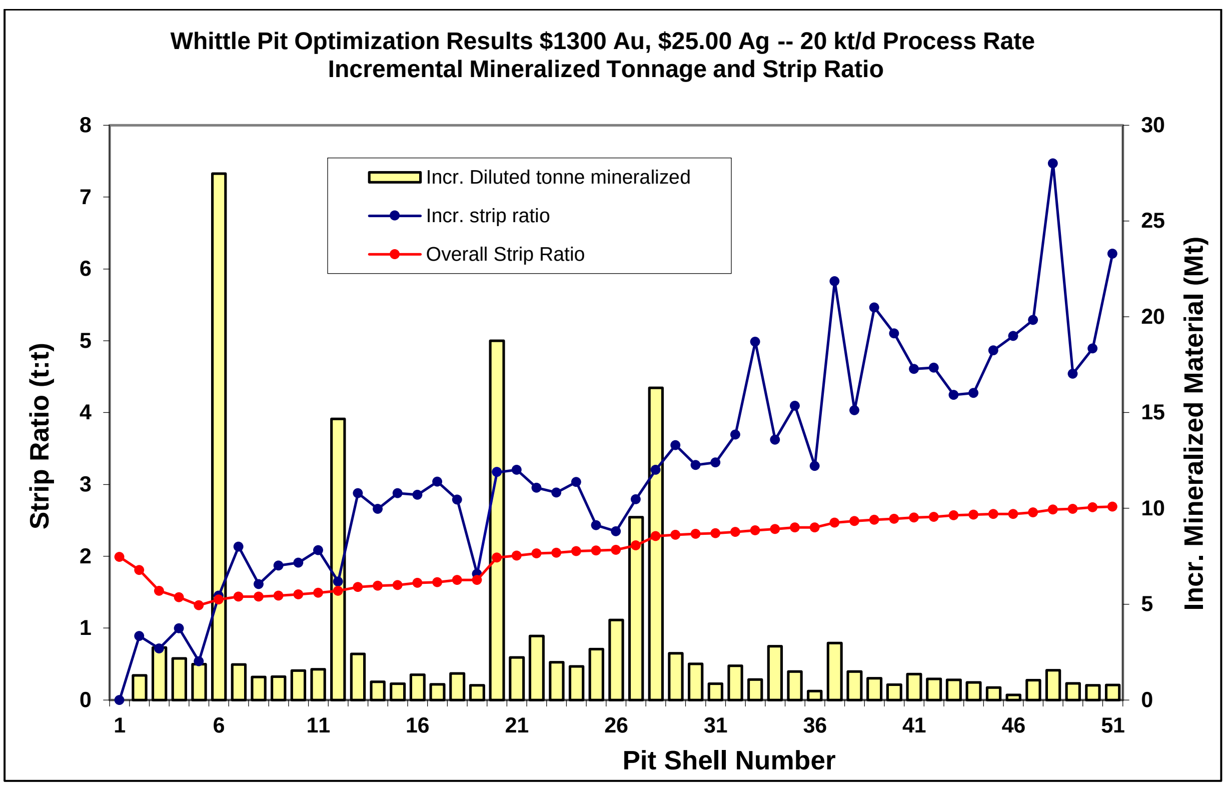

Figure 15.3: Open Pit Optimization Incremental Tonnage Results

For the Springpole deposit, shells beyond Pit Shell 19 add mineralized rock and waste tonnages to the overall pit, but have higher incremental strip ratios with minimal positive impact on the NPV. To better determine the optimum shell on which to base the phasing and scheduling and to gain a better understanding of the deposit, the shells were analyzed in a preliminary schedule. The schedule assumed a maximum processing rate of 7.3 Mt/yr. No stockpiles were used in the analysis and no CAPEX were added.

Based on the analysis of the shells and the preliminary schedule, Pit Shell 19 was chosen as the base case shell for further phasing and scheduling of the deposit. This shell contains 72.4 Mt of mineralized material above cut-off with an average diluted gold grade of 1.19 g/t and 2,777 koz contained gold along with a silver grade of 6.01 g/t and 13,991 koz of contained silver. The total waste tonnage in the shell is 121 Mt for a strip ratio of 1.7:1.

Both indicated and inferred resources were used in the LOM plan of which indicated resources represent 90% (65 Mt) of the material planned to be processed. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. Mineral reserves can only be estimated as a result of an economic evaluation as part of a Pre-Feasibility study (PFS) or a Feasibility (FS) of a mineral project. Accordingly, at the present level of development there are no mineral reserves at the project.

Table 15.4 summarizes the tonnages and grades contained within the shell limits, using the incremental cut-off value of $13.67/t. Table 15.5 further summarizes the resources by classification.

Table 15.4 Resources to Be Extracted in LOM Plan

Description	Unit	Value
Mine Production Life	yr	11
Process Feed Material	Mt	72.4
Diluted Au grade (mill head grade)	g/t	1.19
Contained Au	koz	2,777
Diluted Ag grade (mill head grade)	g/t	6.01
Contained Ag	koz	13,991
Waste	Mt	120.8
Total Material	**Mt**	**193.2**
Strip ratio	t:t	1.7

Table 15.5: Resources to Be Extracted in LOM Plan by Classification

Resource Category	PEA 2013 - Springpole Deposit				
	(Mt)	Au (g/t)	Contained Au (koz)	Ag (g/t)	Contained Ag (g/t)
Indicated	65.3	1.20	2,510	6.00	12,592
Inferred	7.2	1.16	267	6.07	1,399

15.2 Open Pit Mine Design

Mine planning for the Springpole deposit was conducted using a combination of software packages, including MINTEC Inc. MineSight™ and Gemcom GEMS™ (6.4) and Whittle™ . The base 3-D block model was analyzed using GEMS (6.4). The phase selection and production scheduling was undertaken with the use of MineSight and Whittle software.

For the Springpole deposit, the ultimate shell limits, along with the associated phasing (a total of three phases selected that correspond to Whittle shell 5, 7, and 19) were based on the shell analysis described in this report. Preliminary waste dumps were then designed to account for the material produced in each mining phase and shell.

Shell 19 was chosen as the mining shape limit for the deposit. Figure 15.4 and Figure 15.5 represent plan and section views of the ultimate pit shape.

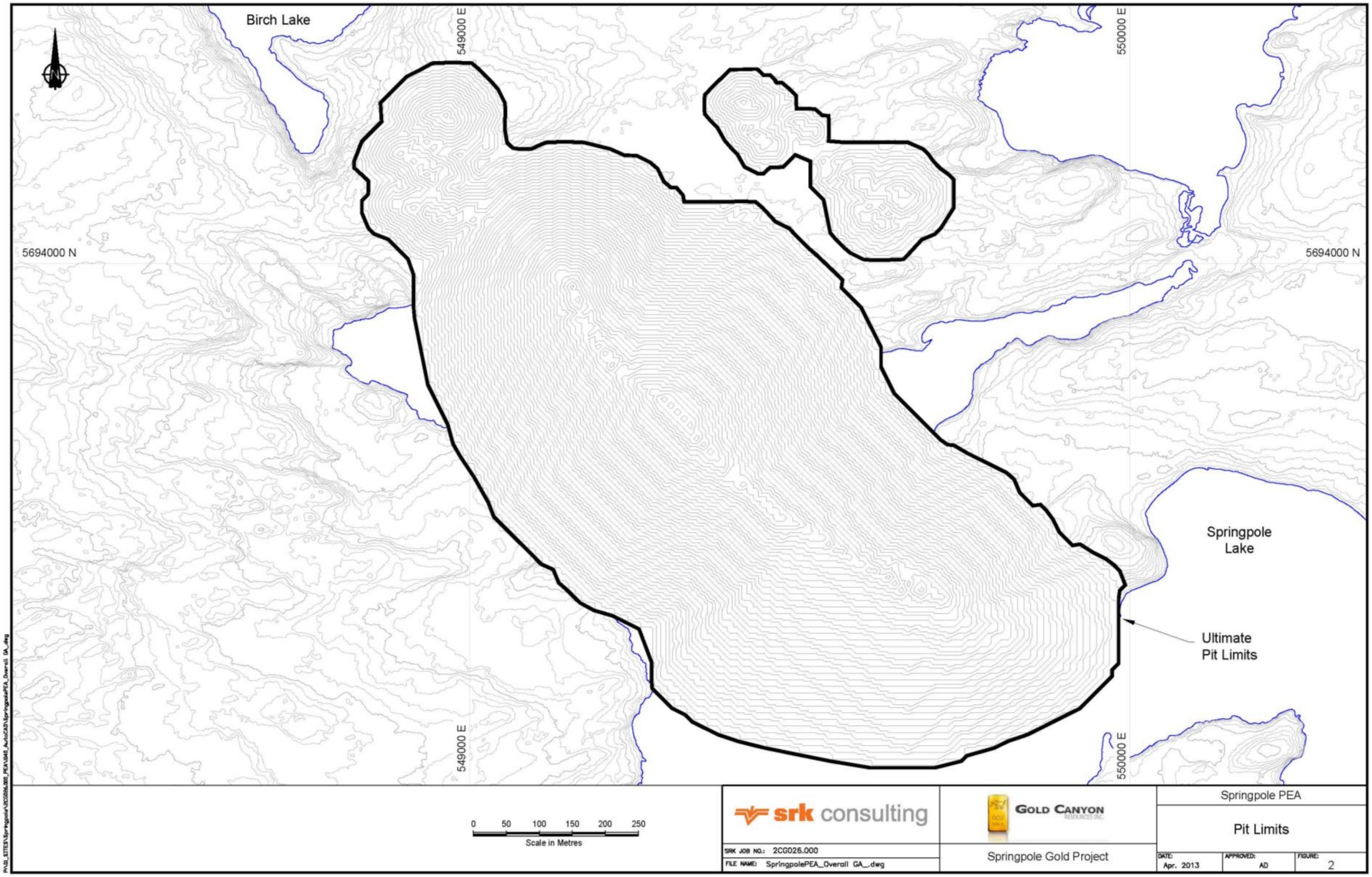

Figure 15.4: Plan View of Springpole Ultimate Pit Limits

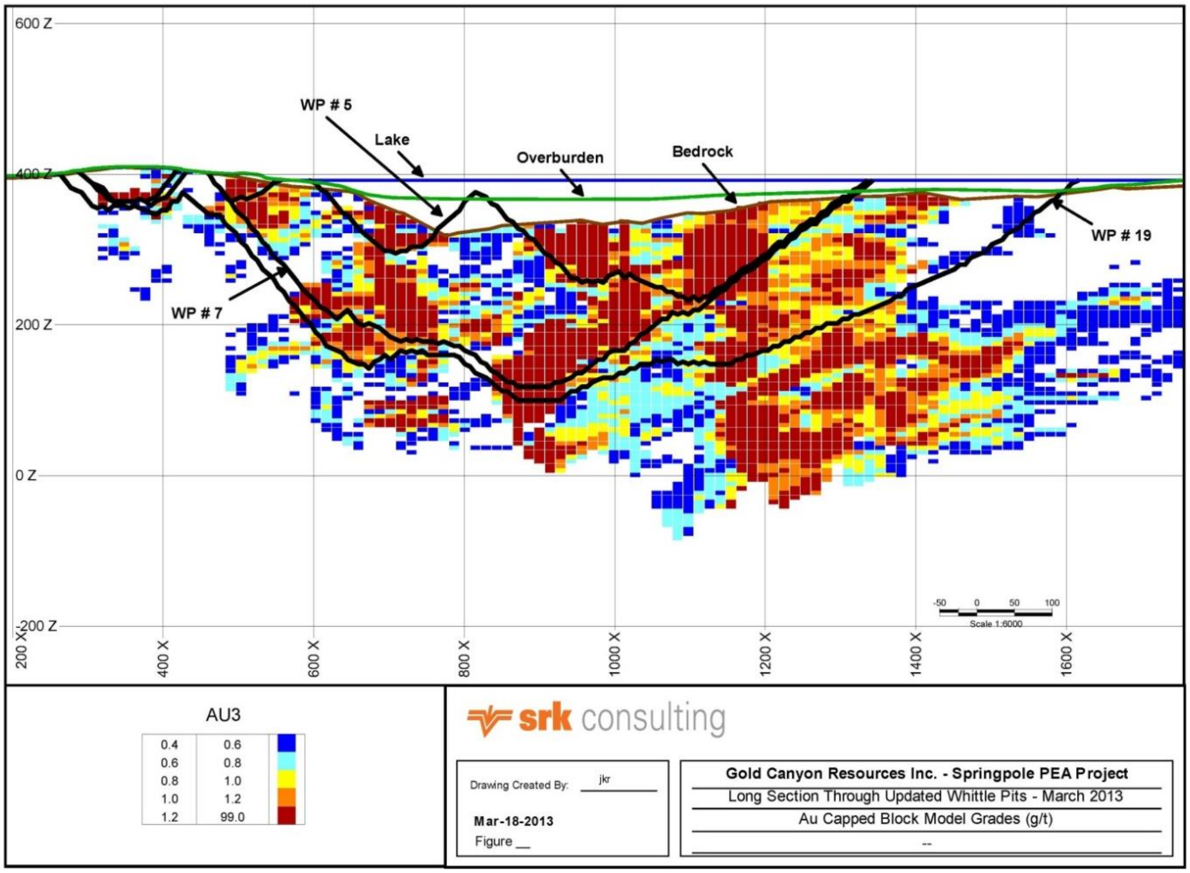

Figure 15.5: Typical Long Section through Springpole Pit Shells

15.3 Mine Sequence/Phasing

The preliminary shells for Springpole were further analyzed and optimizations were conducted to better define the possible stage designs within the ultimate shell limit. The Springpole pit was further divided into three phases for the mine plan development to maximize the grade in the early years, reduce the pre-stripping requirements, and to maintain the process facility at full production capacity. The shell tonnages, grades, and contained metal of the preliminary phases (stages) are summarized in Table 15.6.

Table 15.6: Springpole Pit/Phase Tonnages and Grades

Stage	Total Diluted Mill Feed					Waste	Strip	Total
	(Mt)	Au (g/t)	Au (koz)	Ag (g/t)	Ag (koz)	(Mt)	Ratio	(Mt)
Stage 1	14.4	1.60	740	8.08	3,739	18.9	1.3	33.3
Stage 2	29.3	1.17	1,100	5.33	5,028	43.9	1.5	73.2
Stage 3	28.7	1.02	936	5.66	5,223	58.0	2.0	86.7
Total	**72.4**	**1.19**	**2,777**	**6.01**	**13,991**	**120.8**	**1.7**	**193.2**

Figure 15.6 and Figure 15.7 further illustrate the phase designs for Springpole, with tonnes, grades, and contained metal shown.

The phases were based on the optimized shells summarized above. Shells selected provide reasonable pushback widths with mining starting in the higher grade zone and progressing southwards to ultimate limits.

During the active mining and processing of the deposit, the waste would be placed into a waste rock facility adjacent to the final shell limits. All mineralized material would be hauled to the process facility immediately to the east of the deposit.

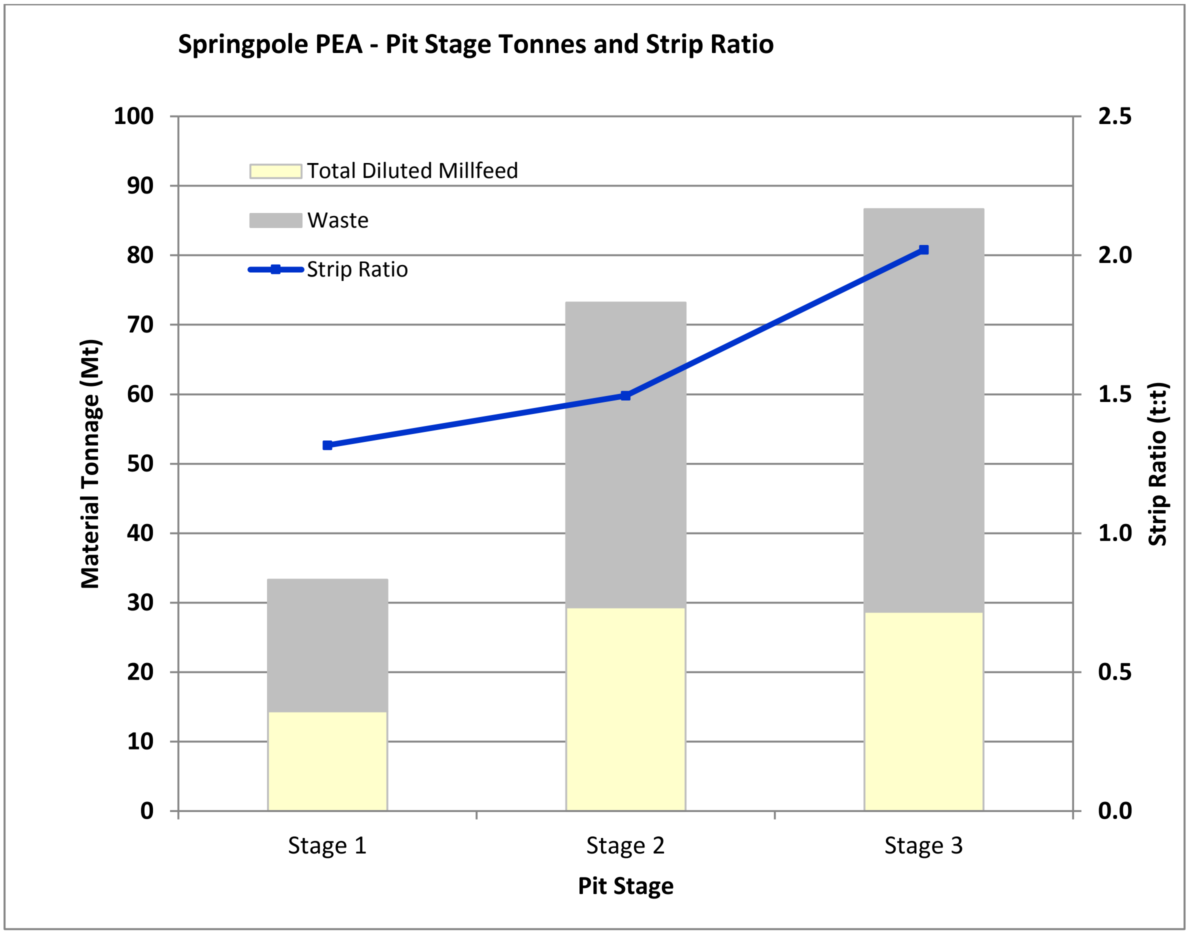

Figure 15.6: Springpole Gold Project – Pit Stage Summary

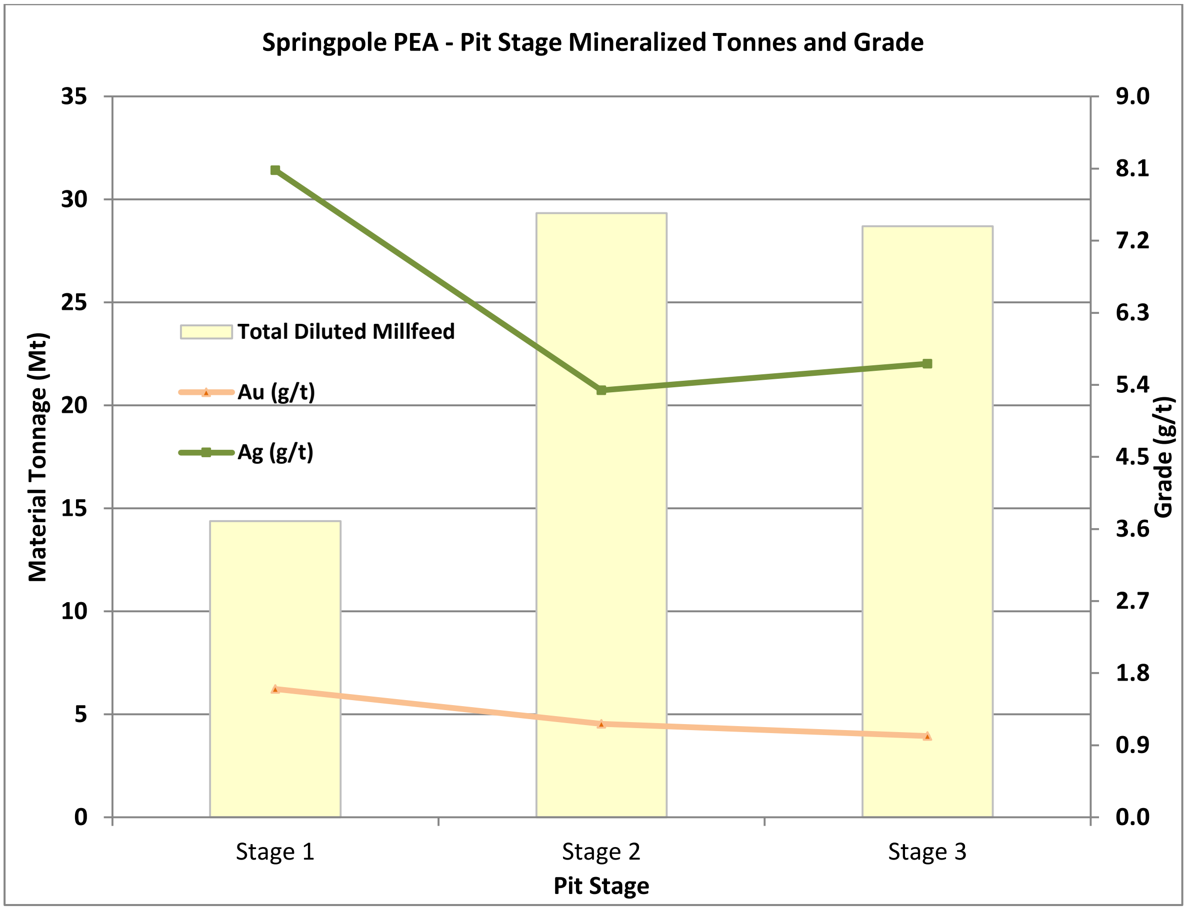

Figure 15.7: Pit Stage Mill Feed Summary

15.4 Open Pit Mine Operation

The open pit mining activities for the Springpole pit were assumed to be primarily undertaken by an owner-operated fleet as the basis for this preliminary economic assessment. The average unit mining costs used in the project economics was \$2.78/t of mineralized material mined and \$2.57/t waste mined, for pit and dump operations, road maintenance, mine supervision, and technical services. The cost estimate was built from first principles and based on experience of similar sized open pit operations and local conditions. The open pit mining costs for both mineralized material and waste mining take into account variations in haulage profiles and equipment selection.

Labour rates were estimated using local information.

15.4.1 Equipment

Table 15.7 summarizes the assumed, all diesel, major open pit equipment requirements used for the basis of this study and are based on similar sized open pit operations. The proposed processing rate of 7.3 Mt/yr was used, along with deposit and pit geometry constraints, to estimate the mining equipment fleet needed. The fleet has an estimated maximum capacity of 83,000 t/d total material, which would be sufficient for the LOM plan.

Table 15.7: Major Open Pit Equipment Requirements

Equipment Type	No. of Units
250 mm dia. Rotary, Crawler Drill	2
165 mm dia. Rotary, Crawler Drill	1
22 m^3 Front Shovel	2
20 m^3 Wheel Loader	1
136 t Haul Truck	17
16H-class Grader	3
D10-class Track Dozer	4
834H-class Wheel Dozer	1
115 mm dia. Rotary, Crawler Drill	1
136 t Water Truck	2

Unit Operations

The 250 mm diameter blast hole drills are planned to perform the bulk of the production drilling in the mine (both mineralized and waste rock), as well as in the mineralized zones to allow for better definition drilling. The 165 mm diameter rotary crawler drill bit is used for secondary blasting requirements and may be used on the tighter spaced patterns required for pit development blasts. The main loading and haulage fleet is planned to consist of 136t haul trucks, loaded primarily with the diesel powered 22 m^3 front shovels or the 20 m^3 wheel loader, depending on pit conditions.

As pit conditions dictate, the D10-class dozers are planned to rip and push material to the excavators and maintain the waste dumps and stockpiles.

The additional equipment listed inTable **15.7** is planned to be used to maintain and build access roads and to meet various site facility requirements, including stockpile maintenance and further exploration development.

15.5 Mine Schedule

The production schedule for the Springpole deposit was developed with the aid of Whittle and MineSight software, and incorporated the various pits and stages mentioned above.

With the mineralized material near surface at Springpole, Year 1 represents the commencement of pre-stripping as well as the processing of mineralized material (at approximately 53% of maximum mill throughput). The LOM maximum planned total material to be moved is approximately 82,000 t/d, while the average total mining rate was planned to be 49,000 t/d.

Indicated and inferred resources were used in the LOM plan, with indicated resources making up 90% of the total LOM tonnage processed. The resources calculated included an estimated external dilution factor of 5%. Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them to be categorized as mineral reserves. There is no certainty the inferred resources would be upgraded to a higher resource category.

Table 15.8 is a summary of total material movement by year for the LOM production schedule.

Table 15.8: Proposed LOM Open Pit Production Schedule

Description	Unit	Total	Year										
			1	2	3	4	5	6	7	8	9	10	11
Mineralized Material Mined	Mt	72.4	3.9	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	2.9
Au Feed Grade	g/t	1.19	1.54	1.47	1.32	0.78	0.97	1.41	1.54	0.83	0.97	1.13	1.53
Contained Au	koz	2,777	191	345	309	184	227	330	362	194	227	266	141
Ag Feed Grade	g/t	6.01	3.64	8.78	6.29	2.99	5.21	7.52	5.78	5.95	5.91	6.25	7.72
Contained Ag	koz	13,991	452	2,060	1,476	703	1,222	1,765	1,356	1,395	1,386	1,466	709
Waste Mined	Mt	120.8	16.1	2.9	20.3	13.6	6.1	8.8	22.6	19.8	7.5	2.6	0.3
Strip Ratio	t:t	1.7	4.2	0.4	2.8	1.9	0.8	1.2	3.1	2.7	1.0	0.4	0.1
Total Material Mined	Mt	193.2	20.0	10.2	27.6	20.9	13.4	16.1	29.9	27.1	14.8	9.9	3.2

The Springpole deposit is planned to produce a total of 72.4 Mt of plant process feed and 121 Mt of waste (1.7:1 overall strip ratio) over an eleven year mine operating life. The current LOM plan focuses on achieving consistent processing feed production rates, mining of higher grade material early in the schedule and balancing grade and strip ratios, while trying to maximize NPV. No blending of stockpiled material was included in this preliminary schedule. To achieve targets, up to two phases are active in any one year.

Figure 15.8 summarizes process tonnage, waste tonnages, and strip ratio by period. Figure 15.9 illustrates the feed tonnage by phase and period, as well as overall gold grades. During full production, the mine, on average, is estimated to produce total recovered gold of 217 koz/yr and recovered silver of approximately 1,200 koz/yr.

Figure 15.10 represents the proposed overall site layout for the Springpole Gold Project.

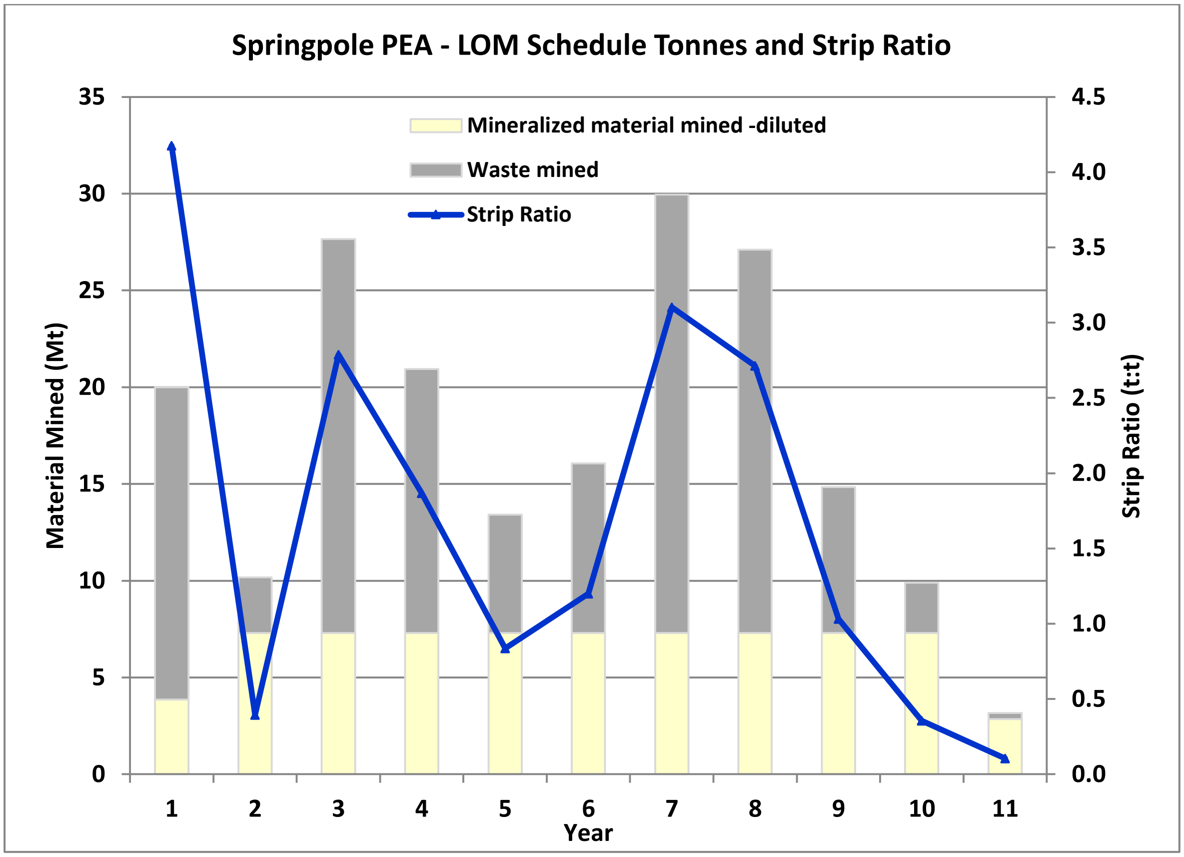

Figure 15.8: Process Tonnes, Waste Tonnes and Strip Ratio by Period

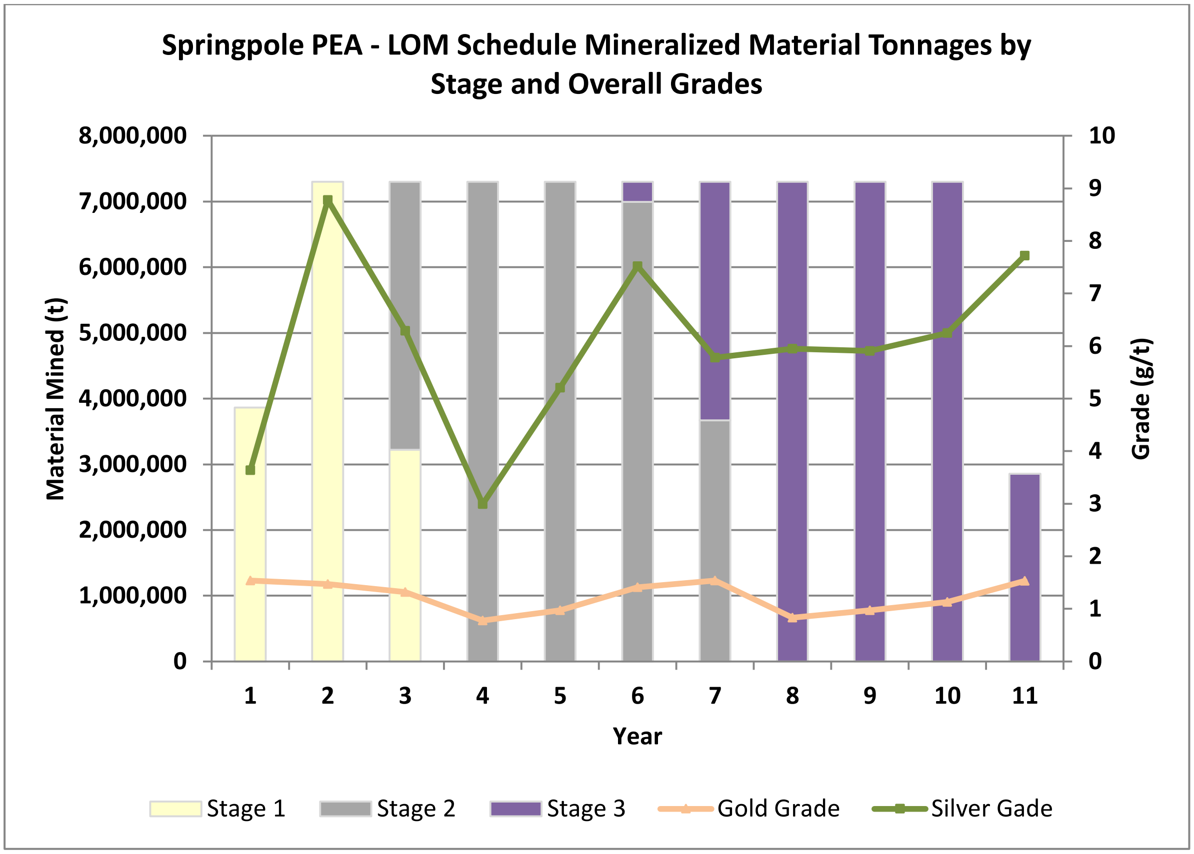

Figure 15.9: Mineralized Tonnes and Grade by Phase and Period

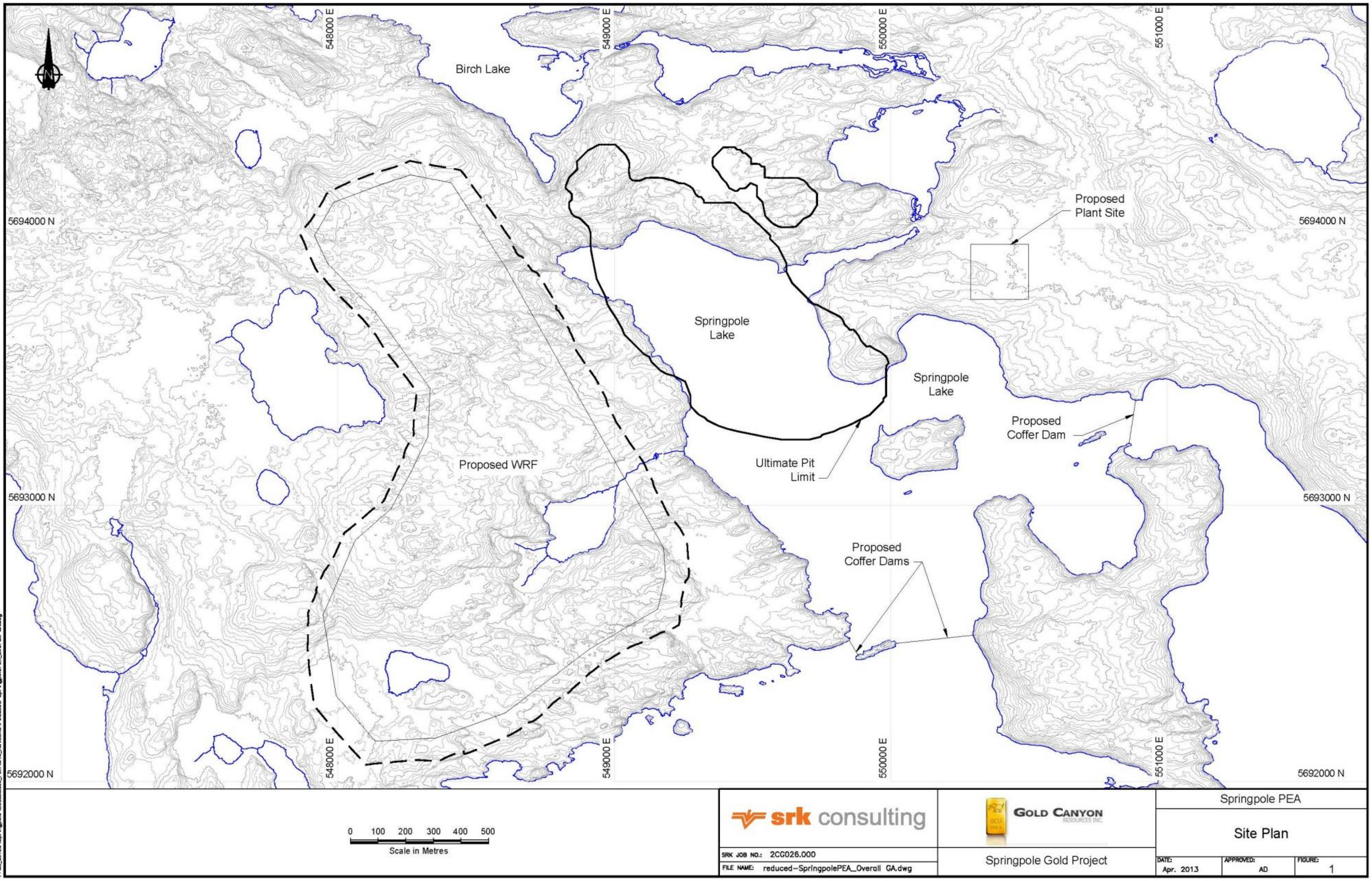

Figure 15.10: Overall Site Plan

To further illustrate the progression of mining of the Springpole deposit, Appendix E provides snapshots of the pit and waste rock facilities at the end of various periods.

The pit is mined out in a series of push-backs to achieve the required process feed, while trying to maximize the NPV of the project.

Mine Development Schedule

Year 1: Development of the Springpole Gold Project commences with pre-stripping and mine production of the pit. Plant processing is also scheduled to begin in Year 1 as a result of the location of mineralized material being near surface leading to minimal pre-stripping requirements. A total of 3.9 Mt of mineralized plant feed (~53% of full production target) and 16.1 Mt of waste are scheduled. The average diluted gold grade is estimated to be 1.54 g/t, and the silver grade is estimated to be 3.64 g/t with an overall strip ratio of 4.2:1 (waste: plant feed).

Year 2: The 7.3 Mt/yr target plant feed is envisioned to be attained with mining continuing in Stage 1. The average mine grade in Year 2 is estimated to 1.47 g/t gold and 8.78 g/t silver. A total of 2.9 Mt of waste rock is scheduled at a mined strip ratio of 0.4:1.

Year 3: Process plant feed production is scheduled to be maintained at the target of 7.3 Mt/yr (or 20 kt/d). Stage 1 is completed in the year and mining commences in Stage 2. Total waste planned to be mined from the two active phases is 20.3 Mt. The average gold grade is estimated to be 1.32 g/t, with a silver feed grade of 6.29 g/t. Stripping of push backs is planned to increase overall strip ratio to 2.8:1. Production rates are envisioned to increase to an average of 76,000 t/d total material for the year.

Year 4: Mining of Stage 2 continues. Process plant feed production is planned to be maintained at a steady state of 7.3 Mt total delivered. A total of 13.6 Mt of waste is planned to be mined at an overall strip ratio of 1.9:1. Production rates for the year are scheduled to near 57,000 t/d total material.

Year 5 to 8: Mining in Stage 2 and Stage 3, with an average of 21.6 Mt of total material mined and an overall strip ratio of 2.0:1. The average gold grade is expected to be 1.2 g/t with an average silver grade of 6.1 g/t.

Year 9 to 11: Mining occurs in Stage 3 over the final years of the LOM plan. The average annual strip ratio is expected to decrease to 0.5:1 with an average of 3.5 Mt of waste and 5.8 Mt of process plant feed planned to be mined in each period. Average mining rate is scheduled to be 30,000 t/d.

15.6 Rock Geotechnical Information

15.6.1 Slope Design Review

A scoping level review of available geotechnical and structural data for the purposes of open pit slope design was completed. This review was based on all available diamond drill core logs and included an on-site rock-core review. A core-photo review of ten drill holes was used to estimate the rock quality designation, solid core recovery, intact rock strength (IRS), and the geological strength index (GSI). These estimates were used to guide preliminary domain delineation used for the slope design guidelines. Additional details of the slope design review, and ancillary data, are presented in Appendix F.

Geotechnical data collection is on-going, with a 2013 drill-program designed and implemented to better understand the rock mass characteristics and enable slope design and stability analysis at an appropriate level of study.

15.6.2 Lithological and Rock Mass Information

A 3-D wireframe geological model was not available for review, and insufficient data were collected for the geotechnical model to be built at this stage. Individual drill logs have been recorded for rock drilling on the property, and they include lithological and alteration descriptions of major lithologies. Total core recovery and rock quality designation has been relatively consistently and correctly recorded for most of the drilling evaluated for the property. Empirical rock strength estimates are recorded for more recent (2011 onward) drill core recovered. No unconfined strength, point load test, or joint-condition data have been acquired to-date.

15.6.3 Structural Information

Faults within the proposed Springpole open pits and immediate vicinity have not been mapped or modelled. In the absence of these data (and to gain a preliminary understanding of the possible fault trends and their influence on the proposed pit) the topography and bathymetry in the form of a digital elevation model was interpreted for lineaments and zones of weakness possibly associated with rock mass damage in fault zones. This information assisted with the preliminary domain delineation (as described in Section 15.6.5).

15.6.4 Seismicity Potential

The Springpole property is located within in a low seismic hazard zone.

15.6.5 Drill Core Review and Rock Mass Characterization

Ten drill holes were selected to represent the rock mass likely to be encountered in the proposed pit walls. The core-box photographs, in conjunction with all available logs, were viewed and rated for solid screen recovery, GSI, and IRS. The solid screen recovery estimates were compared with the logged rock quality designation, and relatively good agreement was found between the two. The IRS estimates from core-box photographs was compared to the field strength estimates and some inconsistencies were found in the drill hole data acquired on-site. For this reason, the IRS estimates from core-box photographs (with field calibration from the site-visit) were used for rock mass assessment and design guidelines.

The data were interrogated for preliminary geotechnical domains. Below the lake-floor sediments and glacial overburden, there are at least three distinct rock mass domains within the pit +200 m envelope. A *Strong*-domain, which surrounds Springpole Lake, is considered the host or "country-rock". There is a relatively narrow transition into the *Intermediate*-domain and *Weak*-domain. The *Intermediate*-domain may be related to regional-scale faulting in some areas, but is also the transition towards the centre of mineralization. The *Weak*-domain appears to be directly associated, spatially, with the mineralization. *Strong* rocks dominate in the north, and on the south-western flank of the pit. The *Weak*-domain appears continuous through the centre of the pit, with the southern-most slopes likely being composed of these rocks. The three domains' intact rock strengths and geological strength indices are likely to be in the order of:

1. *Strong*-domain, IRS ≈ 150 ±50 MPa, GSI ≈ 60 ±30;
2. *Intermediate*-domain, IRS ≈ 75 ±35 MPa, GSI ≈ 40 ±25, and;
3. *Weak*-domain, IRS ≈ 30 ±20 MPa, GSI ≈ 25 ±20.

Down-hole rock data for the ten selected drill holes are represented in five vertical sections contained within Appendix F. A representative vertical section and the preliminary domain boundaries (in plan view) are presented in Figure 15.11. The overall trend is strong (Archaen-aged basement) rock in the north, strong to intermediate strength rock on the south-west and north-eastern flanks of the proposed pit, and weak mineralized (mafic) rock at depth, along the pit mid-line, and in the south. Alteration fluids introduced during the mineralizing events appear to have weakened the intact rocks in the immediate vicinity of higher-grade zones. The alteration "halo" is approximately 50 m wide.

15.6.6 Preliminary Slope Design and Recommendations

The pit was partitioned into preliminary design sectors, as illustrated in Figure 15.12. On-section rock mass domains were used to determine the slope composition and to estimate an overall rock mass rating (RMR) for each section. These estimated overall-slope rock mass rating values, taking into consideration the weaker rocks in the toe of most of the slopes, were compared to published design charts to estimate achievable Overall slope angles for a factor of safety of at least 1.2. The sectors and their maximum recommended overall slope angles (to the nearest five degrees) are contained in Table 15.9.

Table 15.9: Slope Angle Recommendations

Springpole PEA Slope Angle Recommendations			
Design Sector	Overall Slope		
	Rock Mass Rating (estimate)	Height (m)	Angle (˚)
N	GOOD	200	50
E	FAIR	300 - 375	45
SE-n	POOR to VERY POOR	375	35
SE-s	FAIR to POOR	375	40
SW	VERY POOR to POOR	275 - 375	35
W	FAIR to POOR	200 - 300	45

Note: based on estimated RMRs derived from core-box photograph review and the rock quality designation database.

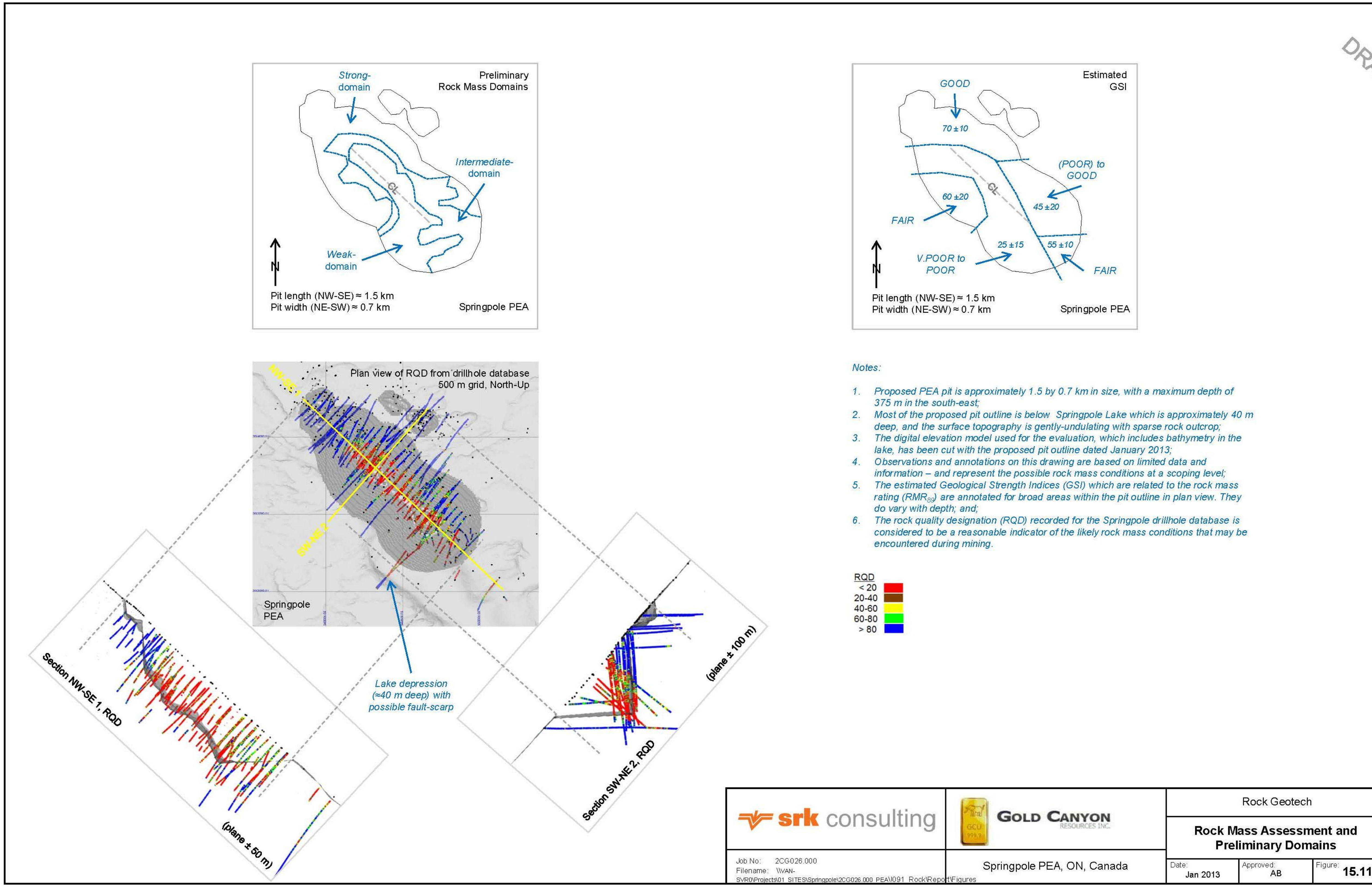

Figure 15.11: Rock Mass Assessment and Preliminary Domains

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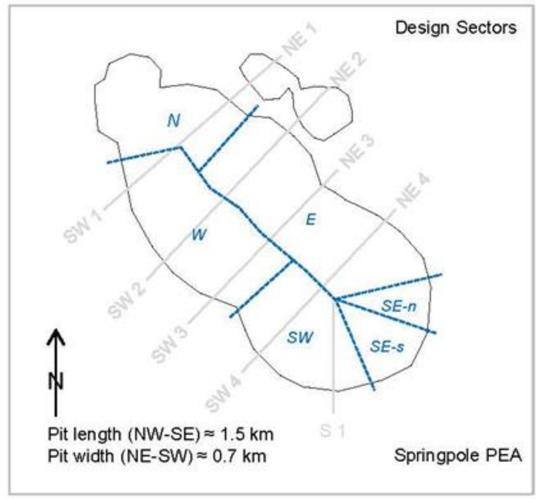
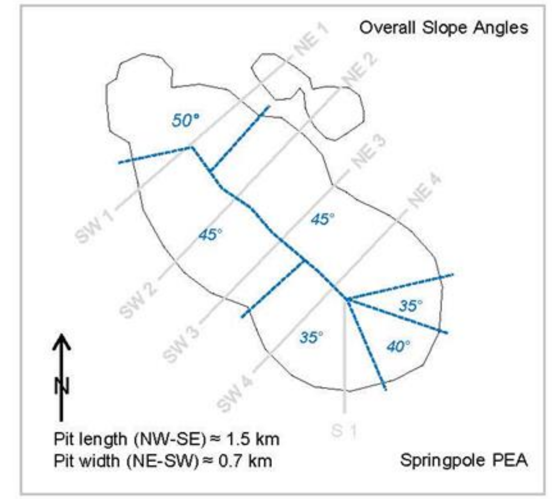
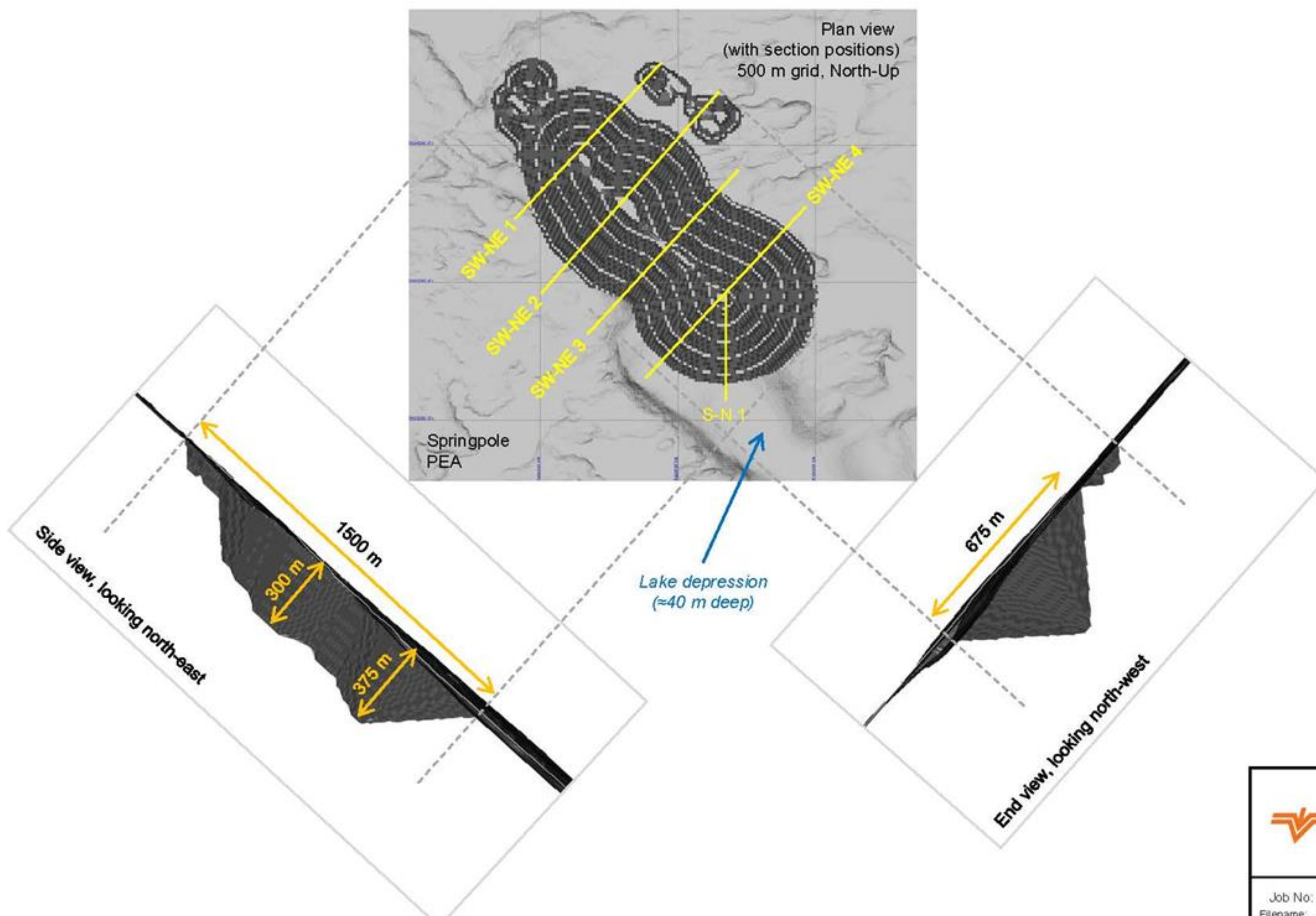






PEA Pit Design Guidelines:

1. *Bench height = 12 m;*
2. *Maximum stack height = 96 m;*
3. *Minimum width between stacks = 20 m (geotech berm), and;*
4. *Single benching in all sectors except possibly in sector-N.*

Sector	OSA (°)
N	50
E	45
SE-n	35
SE-s	40
SW	35
W	45

Notes:

1. *The pit design sectors are based on estimates from core-box photographs of rock mass parameters and rock mass conditions likely to be encountered within the proposed pit;*
2. *The maximum inter-ramp angles (IRAs), once determined from detailed analysis, may limit the achievable overall slope angles (OSA) – as determined from published charts for specific depths (Read and Stacey, 2010), and;*
3. *Accepted maximum OSAs for the design sectors are rounded to the nearest 5-degrees.*

srk consulting	GOLD CANYON RESOURCES INC.	Rock Geotech
		Slope Design Sectors and Recommendations
Job No: 2CG026.000 Filename: \\VAN-SVR0\Projects\01_SITES\Springpole\2CG026.000_PEA\091_Rock\Report\Figures	Springpole PEA, ON, Canada	Date: Jan 2013 · Approved: AB · Figure: 15.12

Figure 15.12: Slope Design Sectors and Recommendations

The pit design guidelines for these sectors (as detailed in Figure 15.12) are based on:

- Bench heights of 12 m, using a maximum stack height of 96 m,
- A minimum width of 20 m between stacks, and
- Single benching, with the possibility of double-benches in the stronger rocks of N-sector, if the fabric orientation is favorable.

There is possibly an up-side improvement in slope design-angles if a significant increase in the confidence of the geotechnical data (and model) can be achieved. What may ultimately control achievable slope angles (apart from hydrogeological constraints) is the *Weak* to *Intermediate*-domain spatial arrangement, and anisotropy in the host rock in the *Strong*-domain. To achieve a pre-feasibility level of confidence for the slope design input parameters:

- Design and implement an oriented-core geotechnical drill program to log the rock mass and acquire intact rock and joint samples.
- Do laboratory and field tests to adequately characterize the intact rock strength and joint properties of the main rock types.
- Map and describe all major faults, as viewed in drill core and rock outcrop within 200 m of the pit crest and integrate them with the regional structural interpretation.
- Produce robust 3-D digital wireframe models of lithology, alteration with intensity, and structures.
- Characterize the rock mass using an appropriate rock mass rating system (for example RMR_{89}), and map the geotechnical domains within a 3-D model.

15.7 Hydrogeology

This section presents a review of hydrogeological considerations for Gold Canyon’s Springpole Gold Project. The objectives of this review were to:

- Provide a general analysis and review of the project's hydrogeological characteristics as they relate to mining and infrastructure.
- Provide preliminary recommendations for water management.

15.7.1 Mine Plan

The proposed general layout for the Springpole mine is presented in Figure 15.13. SRK understands the deposit will be mined using a series of open pits with a maximum depth of approximately 300 m. The outline of these pits overlaps a small portion of Springpole Lake. For the PEA, it is proposed that a series of three coffer dams be constructed at the southern end of the pit area (Figure 15.12) and the lake water pumped out. Details of the lake pumping are provided in Section 15.8.

The mine waste facilities are also presented in Figure 15.13. The waste rock dumps will be located to the southwest of the pit. Tailings will be stored in uncemented paste (or thickened) form at the management facility located approximately 3 km to the southeast of the pit.

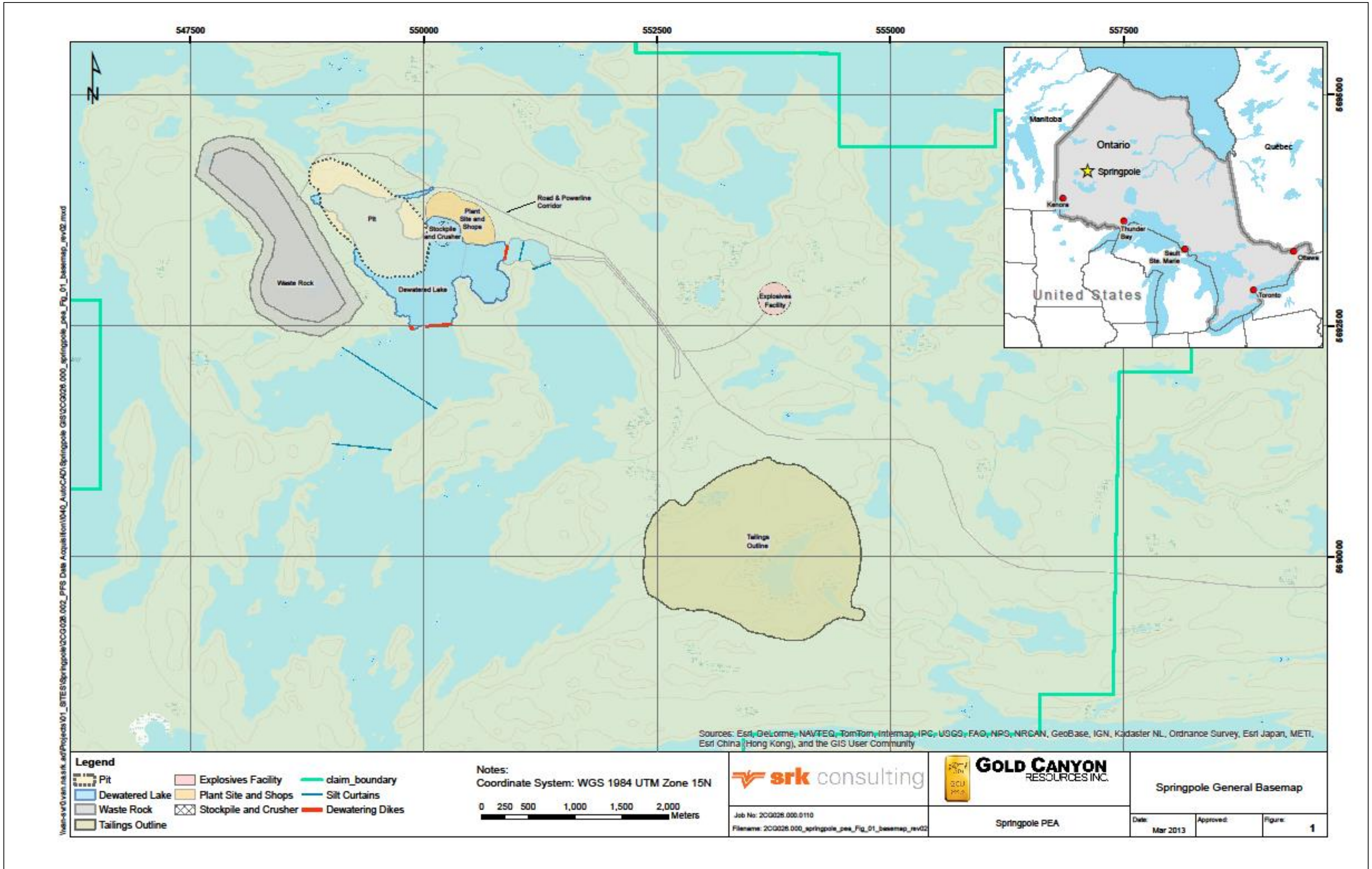


Figure 15.13: Springpole Mine Layout

15.7.2 Available Data

Prior to 2013, there was no collection of hydrogeological data from the Springpole site. In early 2013, SRK initiated a pre-feasibility level geotechnical data collection program for the open pit. This program concluded in March 2013 and included 20 packer tests in 7 geotechnical core holes drilled within the proposed pit footprint. A full interpretation of the data was not available for the PEA reporting; however, hydrogeological observations from site and preliminary testing data were used for this report.

15.7.3 Hydrogeological Conceptual Model

Based on regional observations and experience, a simplified hydrogeological conceptual model can be developed for the site. As there are little site-specific data, this conceptual model can only be used for broad assumptions, but can guide assessment of risk and areas for future data collection. The factors contributing to the conceptual hydrogeology are summarized below.

Topography and Soils

The property is underlain by glaciated terrain characteristics of a large part of the Canadian Shield. Land areas are generally of low relief with less than 30 m of local elevation. Tree cover consists of mature spruce, balsam, birch, and poplar. Black spruce and muskeg swamps occupy low-lying areas. Glacial till is generally less than 1 m in thickness. Outcrops are limited and small and are generally covered by a thick layer of moss or muskeg. Land areas are separated by a series of interconnected shallow ponds and lakes.

Climate

January temperatures range between -40°C and 0°C, and July temperatures range between 20°C and 40°C. Annual rainfall averages 704 mm, with evaporation estimated to be around 545 mm, indicating a net gain. The area receives approximately 200 cm of snow throughout the year.

Geology

Bedrock Lithology

The distribution of the lithological units is not well understood at the Springpole site. There has been extensive drilling over the past 40 years within the Springpole deposit, but this lithological information from drill core was not digitized into a geological model.

Summarized lithological units recorded in drill core within the pit area consist of the following:

- Andesite sequence containing massive and tuffaceous andesite;
- Clastic metasedimentary rocks, banded iron formation, and minor fragmental rocks cut by meter-scale megacrystic feldspar porphyry dykes and lamprophyre dykes; and
- Trachytic volcanic rocks and polymictic breccias cut by numerous meter-scale megacrystic feldspar porphyry dykes.

The gold and silver mineralization at the Springpole deposit occurs in association with the disseminated pyrite-rich argillic and biotite-rich alteration zone (that primarily affect the trachytic volcanic rocks and feldspar porphyry dykes) and stockworks of quartz-pyrite veins and veinlets. The physical degradation of this rock is considerable in places.

Structure

The geological structure on a concession scale is not currently well understood. The appearance of the altered rock, as described in the above section, makes identification of structures in drill core a challenge. In 2011, SRK undertook a preliminary study of the structural controls on the mineralized deposit geometry; however, this study was carried out on a small scale to investigate structural controls on mineralisation and was not projected out sufficiently to interpret larger features that could influence pit scale hydrogeology.

Quaternary Geology (Overburden)

There is no site specific data available for quaternary/overburden geology. From experience in other areas of the Canadian Shield, bedrock is typically scoured with deposits of glacial till varying in thickness from 1 m to up to 10 m in pockets. Soft lake sediments may vary in thickness between 1 m to 10 m.

Groundwater Levels and Groundwater Flow

There is currently no groundwater level data being collected from the project site. Given its proximity to the surrounding lakes, it is assumed that the water levels are similar to that of the lake level.

Experience in this environment suggests that groundwater flow will mostly follow surface water flow directions. It should be noted that there is a surface drainage divide between Birch Lake (to the north) and Springpole Lake (to the south). Groundwater monitoring plans for the project should be designed to verify and acknowledge these catchment divide so that the groundwater flow is adequately characterized.

Groundwater Inflows to the Pit

During operations, inflow rates will be a function of pit-development shape, volume and rate of excavation, as well as hydraulic conductivity of the bedrock and the hydraulic gradient between the mine and surrounding surface water sources (lakes). Estimates of potential inflow rates to the pit were made at an “order of magnitude” accuracy level using analytical methods (Dupuit 1863). Hydraulic conductivity (K) values were derived from preliminary testwork currently being undertaken by SRK in a separate geotechnical data acquisition program for the proposed open pit (reference the pit shells used here). Given the location of the pit relative to the surrounding lakes, and the limited information on geology and structure currently available, a conservative approach was taken in estimating the pit-inflow. Inflow rates were estimated to reach a maximum of approximately 10,500 m^3/d. The potential for geological features to connect to the lakes was flagged as a risk to the project in the form of unanticipated water management and environmental concerns.

Mine Water Supply

Final water requirements will be dictated by the plant design. Given the proximity of nearby surface water bodies, groundwater is unlikely to be required as a primary mine water source. Groundwater inflows collected in the pit will, however, be pumped out and incorporated into the mine site water balance.

Tailings Management Facility and Waste Rock Dumps

Both waste rock and tailings are considered to be potentially acid generating (Morin 2012). Although no site investigation work on the overburden has been undertaken to date, the depth of overburden is expected to be less than 0.5 m on average. Neither the waste rock piles nor the tailings management facility will have an under liner.

15.8 Hydrology

15.8.1 Topography and Watershed Delineation

Topographical information for the Springpole Gold Project site and surrounding area was obtained from Geobase (CTI 1997). The information utilized consisted of:

- Digital Elevation model files with 250 k resolution for areas 052a to 052p (CTI 1997)
- Digital Elevation model files with 50 k resolution for areas 052e13 to 052o12 (CTI 1997)

These digital elevation models were compiled, and the watershed delineation was computed with Esri® ArcMapTM 10.1 utilizing the hydraulics and hydrology analysis tools. The ArcMap software uses a finite element procedure to define watershed boundaries and drainage lines based on topographical elements as well as defined watercourses.

Regional flow watercourses were obtained from the USGS (2012) database HydroSHEDS (Hydrological data and maps based on Shuttle Elevation Derivatives at multiple Scales) including the definition of the most important watercourses, rivers and creek around the site area.

15.8.2 Annual Run-Off and Flow Rates

Historic daily flow rates were obtained from Environment Canada (2010) for five unregulated gauging stations surrounding the project site. The daily flow rates were utilized to determine the annual run-off, monthly flow rates and monthly distribution for each station. The average of the values from the five gauging stations was used for the site specific values. The average annual run-off was estimated to be 306 mm. The monthly run-off distribution can be seen in Table 15.10

15.8.3 Precipitation

Monthly mean precipitation data were collected for nine stations surrounding the Springpole site, using Canadian Climate Normals 1971-2000 (Environment Canada 2012). Only stations with more than 18 years of complete data and located within 200 km of the project site were utilized. Along with the precipitation data site coordinates and elevation were also obtained.

Mean annual precipitation was calculated for each station, and a graph of mean annual precipitation versus latitude was created. The mean annual precipitation for the Springpole site was determined to be 704 mm. Table 15.10 displays the rainfall distribution.

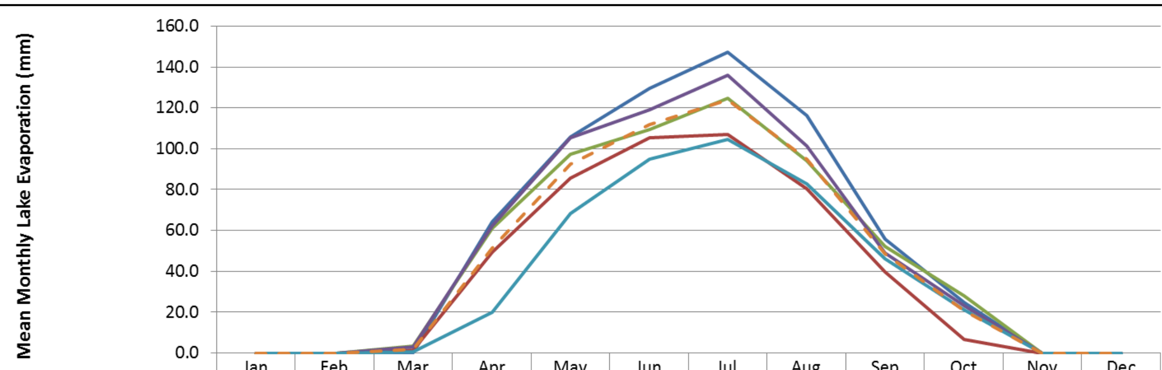
The 1:25 year, 24 hour storm, rainfall was estimated to be 80 mm based on Atlas of Canada extreme rainfall statistics (Hogg and Carr 1985).

Table 15.10: Average Monthly Run-Off and Precipitation Distribution

Month	Run-Off Distribution	Precipitation Distribution
Jan	4.3%	4.3%
Feb	3.4%	3.4%
Mar	3.5%	4.6%
Apr	8.9%	5.7%
May	15.1%	9.1%
Jun	14.3%	14.8%
Jul	12.8%	13.7%
Aug	9.8%	12.7%
Sep	8.8%	12.4%
Oct	7.9%	8.3%
Nov	6.3%	6.5%
Dec	4.9%	4.5%

15.8.4 Lake Evaporation

Lake evaporation was calculated using the WREVAP version 1.0 evaporation estimating software. Monthly mean calculated lake evaporation, mean monthly precipitation and daily bright sunshine hours was obtained for five stations surrounding the project site from Canadian Climate Normals 1951 to 1980 (Environment Canada 1982a, 1982c, 1982c). These data were inserted into the WREVAP software to obtain the calculated evaporation in mm and the monthly evaporation distribution for each of the surrounding stations. The average evaporation and evaporation distribution of the five stations will be utilized as the site evaporation. This is displayed in Figure 15.14. The average annual evaporation is 546 mm.



	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
WINNIPEG INT'L A	0.0	0.0	0.3	64.3	105.7	129.6	147.3	116.3	56.0	24.8	0.0	0.0
THOMPSON A	0.0	0.0	1.3	49.5	85.6	105.4	107.1	80.3	39.8	6.8	0.0	0.0
ATIKOKAN	0.0	0.0	3.5	60.9	97.5	109.4	124.7	94.0	52.1	28.0	0.0	0.0
BISSETT	0.0	0.0	2.9	62.2	105.2	119.0	136.0	101.5	48.8	23.1	0.0	0.0
MOOSONEE	0.0	0.0	0.8	20.1	68.5	94.7	104.4	82.7	46.2	21.0	0.0	0.0
AVERAGE	0.0	0.0	1.8	51.4	92.5	111.6	123.9	95.0	48.6	20.7	0.0	0.0

Figure 15.14: Mean Monthly Lake Evaporation for Five Stations Around the Springpole Site

15.8.5 Site Water Balance

Based on the estimated annual precipitation, run-off and lake evaporation the Springpole site has a net negative water balance. However, lake evaporation typically over estimates evaporation;

therefore, it is possible that the site may have a net zero water balance. A detailed water balance for site activities was not performed at this time.

15.9 Surface Water Management

15.9.1 Preliminary Design Assumptions

Diversion ditches, sumps, and ponds will be required to manage the surface water at the Springpole site. The following assumptions were used for the preliminary design of the water management plan.

- Lake dewatering is considered a part of the scope of the dewatering dikes and will not be included in the preliminary surface water management plan.
- Water management for project infrastructure such as roads and pads would be included in the infrastructure design.
- Surface water will be separated into contact and non-contact water.
- Non-contact water is assumed to be of discharge quality. Contact water is assumed not to be of discharge quality, based on preliminary geochemical testing. Contact water will be sent to the mill for treatment and reuse. Non-contact water will be diverted or pumped to the closest water body.
- Contact water will be considered all run-off from:
  - Waste rock piles (190 ha)
  - Ore stockpiles (5 ha)
  - Pits (70 ha)
  - Tailings Management Facility (365 ha)
  - Plant and Shop area (25 ha)
- Pollution control ponds are not required to contain run-off from the pit or tailings management facility as the management of this water is already accounted for elsewhere.
- Diversion structures, ditches, and ponds for non-contact water are assumed to be unlined.
- Diversion structures, ditches, and ponds for contact water are assumed to be lined.
- Sedimentation and pollution control ponds will be sized to contain the 1:25 year 24 hour storm event.

15.9.2 Design

The water management structures for the waste rock dump and tailings facility consist of 9,000 m of diversion ditches and several pollution control ponds to collect run-off from the waste rock. These can be seen in Figure 15.15. Pollution control ponds have a maximum depth of 2 m. Ponds have a freeboard of 0.5 m, 3 m wide crest and side slopes of 2H:1V on the upstream side and 1.5H:1V on the downstream side.

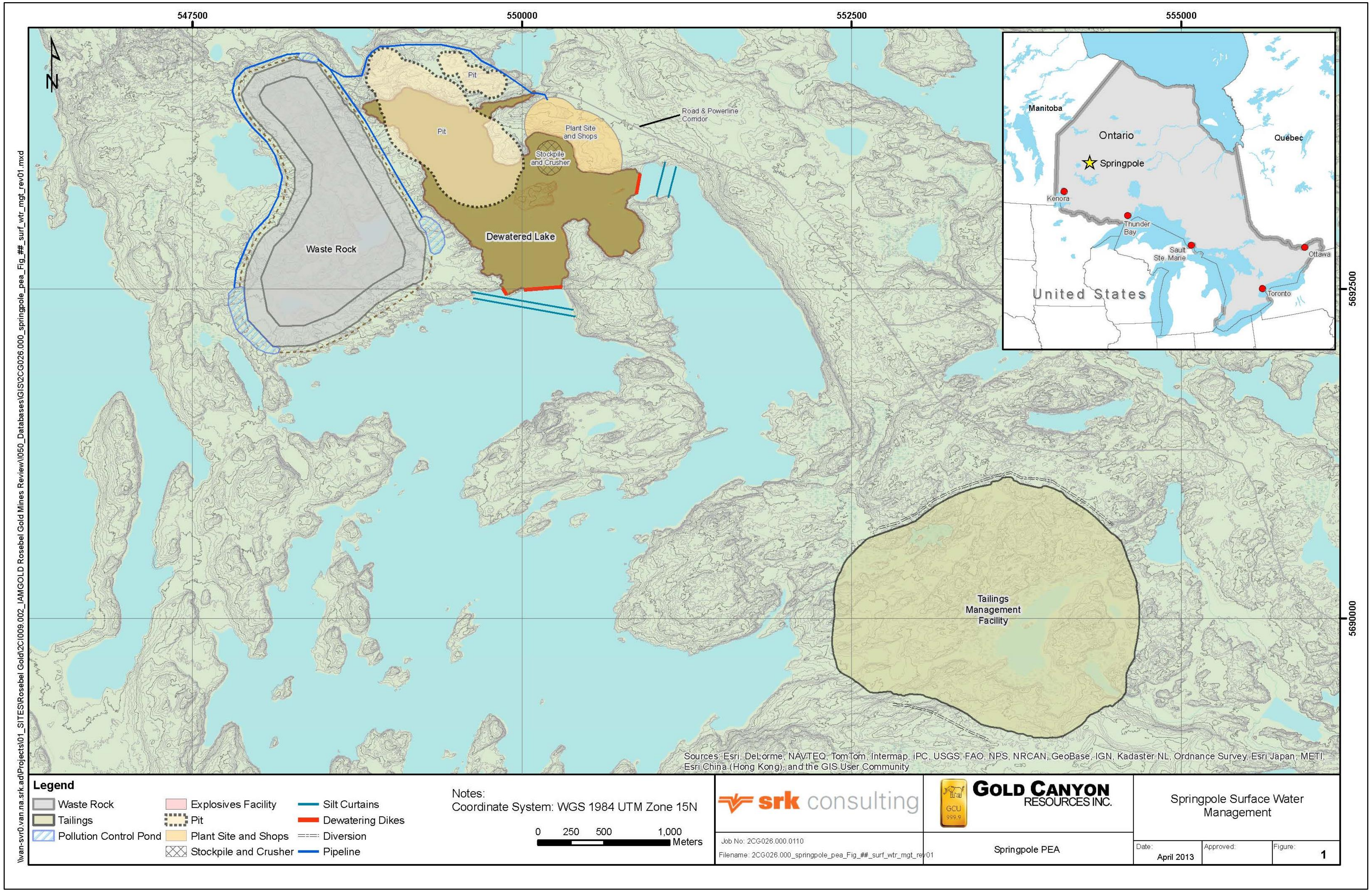


Figure 15.15: Springpole Surface Water Management

16 Processing Recovery Methods

The Springpole PEA is considering a 20,000 t/d process plant treating soft to moderate hardness (BWI of 7 to 14 kWh/mt) material averaging 1 g/t gold and 6 g/t silver. Testwork has determined that a moderate grind P80 size of 70 µm should achieve 80% gold extraction through cyanide leaching for at least 36 hours. Gravity recovery is considered optional, as only higher grade feed would benefit from including this circuit.

16.1 Process Flowsheet

The expected process flowsheet for Springpole is shown in Figure 16.1, based on testwork results to date.

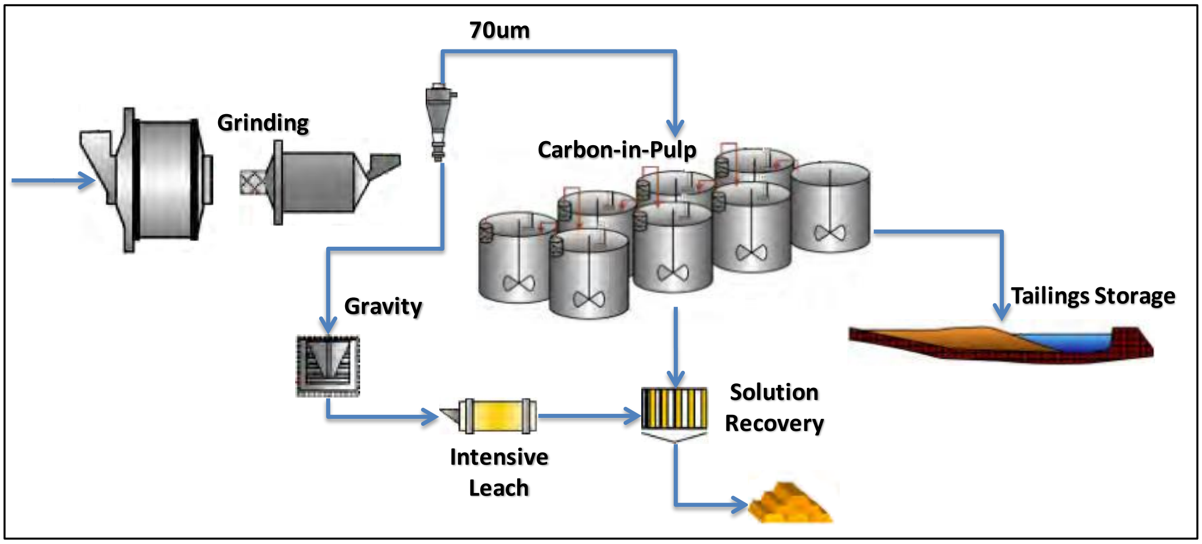


Figure 16.1: Springpole Process Flowsheet

Due to the relative hardness, consistency and poor integrity of the mineralized zones, it is likely that a conventional semi-autogenous grinding (SAG) mill followed by ball mill will be used to an 80% passing size of 70 µm. Multi-stage crushing could be considered, but handling the likely wet and incompetent feed will make slurry processing easier after only one stage of primary crushing.

A gravity recovery circuit should be included to treat the cyclone underflow stream when processing higher grade feed with possibly higher amounts of gravity-recoverable gold. It is a relatively inexpensive circuit to install and operate and, therefore, a cost effective way to maintain high gold recoveries. Gravity concentrate can be processed with an intensive leach reactor with the pregnant solution reporting to the electrowinning gold recovery circuit.

Grinding cyclone overflow will pass over a trash screen and be thickened prior to cyanide leaching. This circuit will likely comprise of a primary stage of aeration and leaching followed by secondary carbon in pulp (CIP) tanks. The target residence time for cyanidation should be 36 to 48 hours. Cyanide consumptions will be moderate at around 0.5 kg/t. Leach tailings will report to cyanide destruction prior to tailings impoundment in the storage facility.

Gold loaded carbon will go to stripping and reactivation prior to be returned to the CIP circuit. Pregnant strip solution will report to electrowinning where the gold sludge will be sent to the furnace for recovery as doré bullion.

Reagents to be used in the flowsheet include:

- Sodium cyanide for gold leaching and carbon stripping,
- Lime for pH control of cyanide leaching,
- Hydrochloric acid for carbon stripping,
- Sodium hydroxide for carbon stripping (after acid wash),
- Flocculant for thickening of feed and possibly tailings,
- Sulphur dioxide for cyanide destruction, and
- Copper sulphate for cyanide destruction.

16.2 Expected Plant Performance

Based on the testwork results reported to date and the range of process flowsheet options considered in the 2012/2013 work, a likely flowsheet is a moderately fine grind size followed by whole feed leaching. Removal of gravity gold prior to leaching appears to only benefit high grade (>5 g/t gold) feed and should be considered an option for the flowsheet.

The results from the 2012/2013 samples show a similar whole feed leaching response to the 2011 samples (Figure 16.2). Typical head grades of 1 to 2 g/t gold generated residues of 0.2 to 0.5 g/t gold after 48 hours of leaching. Higher grade samples showed better extractions with lower residue grades likely due to the nature of the gold and its association.

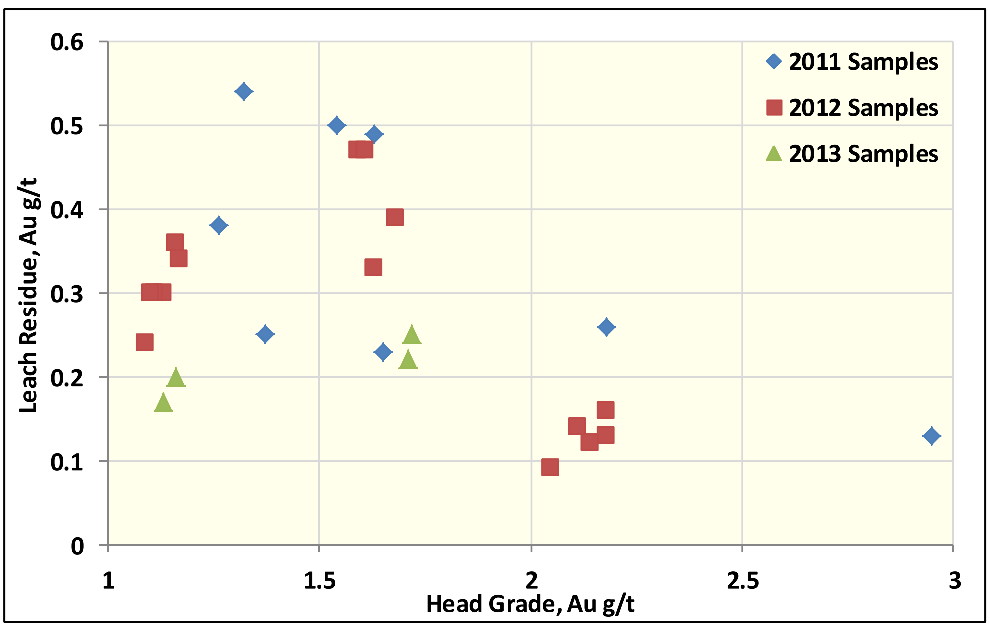


Figure 16.2: 48 Hour Whole Feed Leach Residue vs. Head Grade

For the different ranges of gold grade, a reasonable relationship between gold extraction and grind size was observed in the 2011 and 2012 whole feed leaching results (Figure 16.3). As the kinetic results indicated that extraction was almost complete at 24 hours, the plant design should be for 36 hours. In addition, a cyanide concentration of around 1 g/L seems reasonable as the intensive leach test (5 g/L CN, higher dissolved oxygen and elevated temperature over 96 hours, shown circled) did not increase overall gold extraction. Tests at 2 g/L NaCN or with carbon present did not improve leach extractions either. It seems most likely gold extraction is largely determined by particle size, with cyanide and oxygen levels contributing only a minor role in leaching rates.

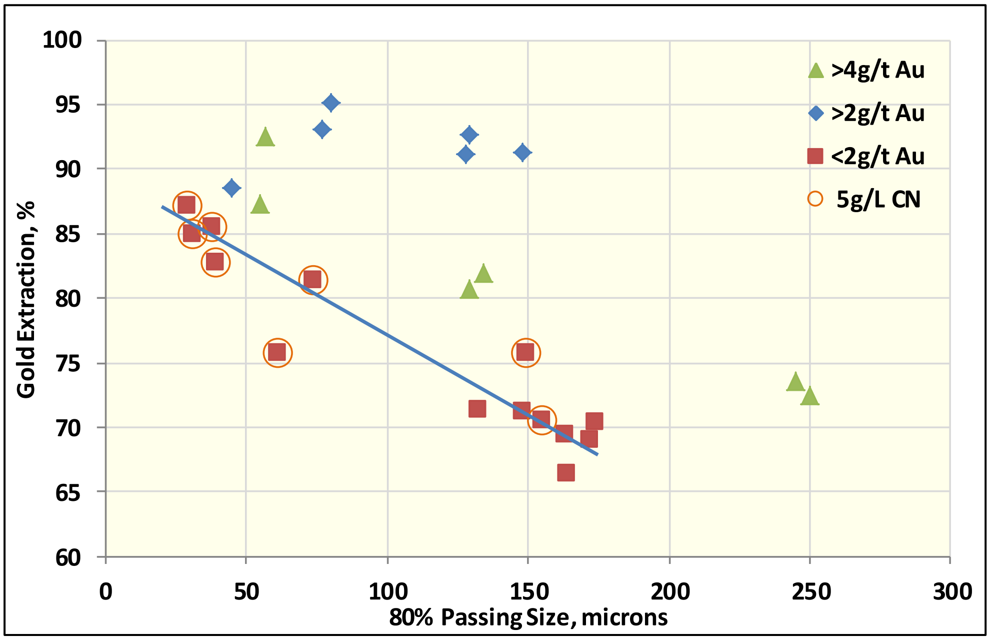


Figure 16.3: Whole Feed Gold Leach Extraction vs. Grind Size

For the <2 g/t gold samples (more typical of the deposit), 24 hour extraction increased with grind fineness as shown by the line in Figure 16.3. This relationship suggests 70% extraction at a P80 size of 150 µm, 80% at 70 µm, and 90% at <20 µm. The oxidized samples showed >90% extraction at any grind size while the higher grade (>4 g/t) samples followed a similar trend with 15% higher extraction for the same grind.

For the same ranges of gold grade, silver extraction appeared to be influenced by the intensive leach conditions. For samples of <2 g/t gold, >80% was achieved after 96 hours for a range of grind sizes (Figure 16.4).

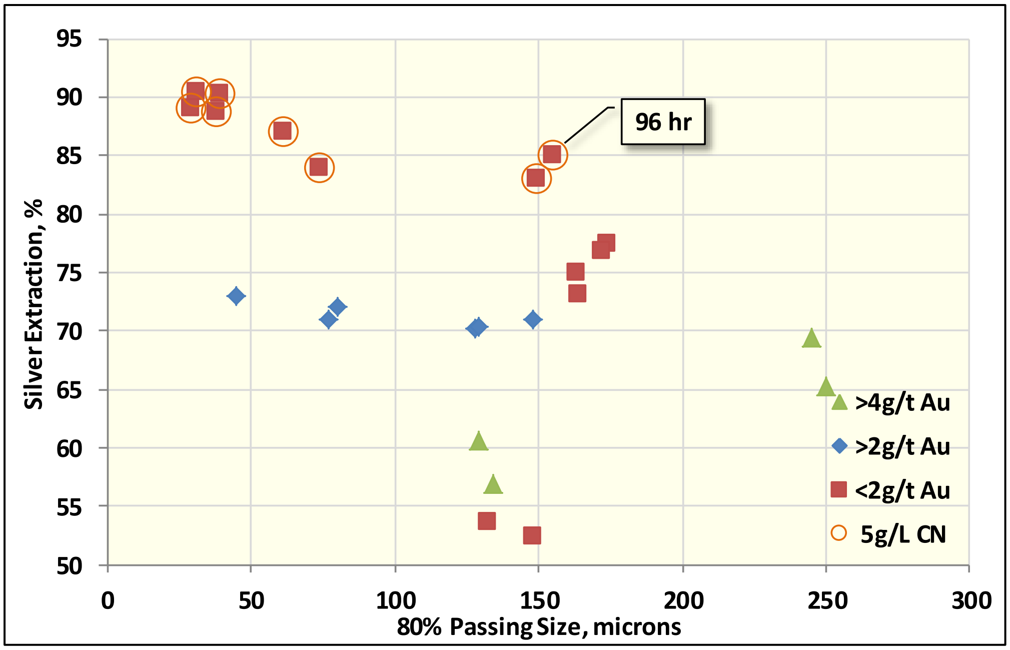


Figure 16.4: Whole Feed Silver Leach Extraction vs. Grind Size

Overall, a primary grind P80 size of 70 µm is required to achieve 80% gold extraction or better, depending on head grade. A trade-off study comparing the benefit of higher gold extraction at a finer grind size should be completed, considering the potential implications of very fine leach residues needing to be detoxified and handled by the waste management facility.

At a primary grind of 70 µm, whole feed leaching will achieve 80% gold extraction with silver extraction slightly higher at 85% as shown circled in red (Figure 16.5).

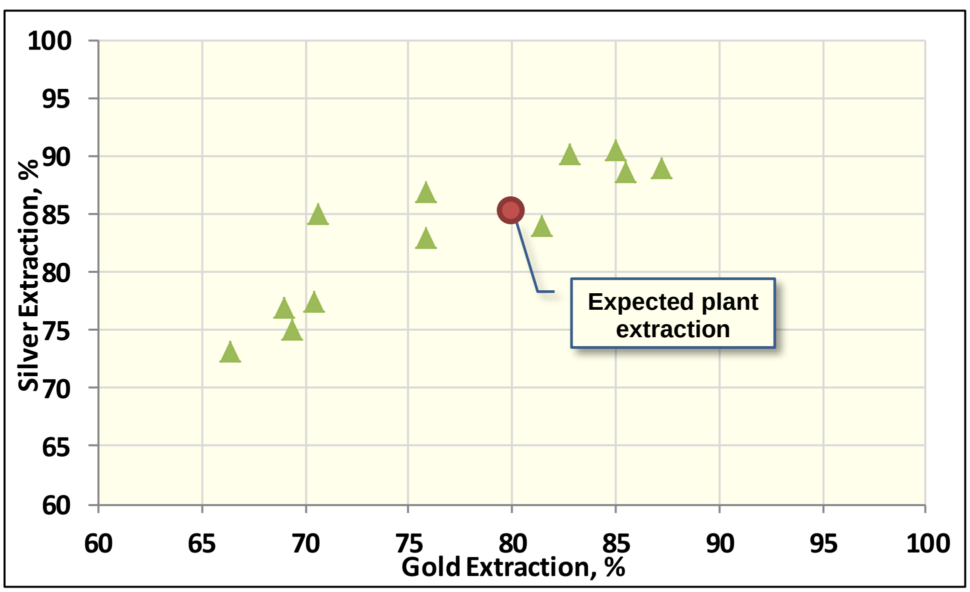


Figure 16.5: Whole Feed Silver Leach Extraction vs. Gold Extraction

17 Project Infrastructure

17.1 Waste Rock Facilities

The waste rock facility is planned to be located immediately adjacent to the final pit limits for the Springpole deposit. Given the deposit configuration and extraction sequence, no backfilling into previously mined out areas has been planned for Springpole.

The waste rock facility would be built in a series of lifts in a "bottom-up" approach, and the facility would be constructed by placing material at its natural angle of repose (approximately 1.5H:1V) with safety berms spaced at regular intervals giving an overall operational slope of 2:1. The total design capacity of the waste rock facility is 121Mt.

17.2 Tailings Management Facility

The following preliminary design assumptions and/or criteria were taken into consideration for the preliminary TMF design presented here:

- About 72 Mt of tailings (20,000 t/d) will be produced throughout the 11 year LOM.
- Tailings characterization has not been carried out, so for volumetric calculations, conservative estimates for tailings density were assumed.
- In addition to tailings, the tailings facility will contain soft lake bed sediments excavated from the dewatering dike foundations and pit area; an estimated 2.1 Mm^3 (assuming 4 m deep sediments in the pit area and 5 m deep sediments under the dikes).
- Minimum design criteria will be in accordance with Canadian Dam Safety Regulations (CDA 2007).
- The project is located in the low seismic hazard zone of Canada. Therefore, at this stage, there has been no consideration for specific design elements to address seismicity.
- No geotechnical and/or hydrogeological characterizations have been carried out to evaluate foundation conditions. The designs presented are based on engineering judgement considering visual observation of surficial conditions during the site visits by Megan Miller, EIT, and Maritz Rykaart, PEng.
- The site has a net negative water balance.
- The site has generally little natural relief offering limited opportunities for natural containment of tailings. The overall topography ranges from about 385 to 415 masl.
- There is competing interest in the available surface area for the placement of mine waste rock. As a rule, open pit project economics are more sensitive to waste rock location than tailings location; therefore, where conflicts occur, waste rock placement gets preference in siting decisions.

- Preliminary geotechnical testing carried out on the tailings suggests that the tailings will be acid generating, and therefore that complete environmental containment will be required for the tailings management facility. Since the containment dams are expected to be founded on competent bedrock a complete under liner is not included.

17.2.1 Options Analysis for the Tailings Management Facility

Alternate Deposition Methods

Alternate deposition methods considered for the project included:

- Conventional low solids content (typically less than 30%) slurry, deposited in an on-land facility;
- Conventional low solids content (typically less than 30%) slurry, deposited in a nearby lake;
- Thickened tailings slurry (solids content typically between 30 and 60%), but still pumpable with centrifugal pumps;
- Cycloned tailings, using the overflow to construct the embankments;
- Uncemented thickened tailings (solids content typically in excess of 60%) requiring positive displacement pumps; and
- Filtered (i.e., dry-stack tailings) tailings.

Due to the low topographical relief and lack of natural containment basins, ring-dams will be required for any on-land tailings deposition strategy. Therefore, tailings deposition methods that result in smaller containment dams are preferred. Conventional low solids content tailings would require the largest containment dams and were therefore not considered further. Thickened tailings slurry offers only a slight advantage in this regard and was therefore also dismissed. Cyclone tailings could be a cost effective method as additional borrow material is not needed to build the dams. However, this deposition strategy is operationally challenging in cold climates and borrow material would be required to construct the starter dams.

Sub-aqueous deposition of conventional low solids content slurry in a nearby lake could be done with little or no dam construction. Permitting of lake disposal would be environmentally and socio/economically challenging, and has, therefore, not been evaluated further.

Filtered tailings would require the smallest containment dams; however, the proposed production rate would require a significant capital investment in filter presses with a correspondingly high operational cost. Therefore, this option was not evaluated further.

Uncemented paste tailings (or more accurately, thickened tailings to the point where there is only a minor amount of bleed water) was, therefore, selected as the preferred tailings deposition strategy as it requires the least amount of containment dams to be constructed, while being more amenable to cold weather operation.

It should be noted that this is a high level evaluation, and considering the level of study, a more comprehensive evaluation of deposition strategies would be warranted during future stages of this project.

Alternate Retaining Structures

Hydraulically placed sub-aerial thickened tailings require construction of retaining structures. The alternative retaining structure technologies considered included:

- Earth fill dams,
- Lined rock fill dams, or
- In-pit deposition (mined out open pit).

In-pit disposal into a mined out open pit was considered but dismissed since the proposed mine development sequence does not provide for suitable locations.

The construction of a low permeability core earth fill dam requires the availability of suitable low permeability materials, within tight engineering tolerances. No borrow characterization study was carried out, and, based on the reconnaissance site visit; it is unlikely that large quantities of suitable materials would be readily available. Therefore, at this stage, the preferred dam design is a geosynthetically lined rock fill dam.

Alternate TMF Sites

Based on the lay of the land, the closest viable area to the mill that did not require a major stream or lake crossing was selected for the TMF. A high level reconnaissance desk top search for alternate sites was carried out, but ultimately only the currently earmarked location was given any real consideration. Within the confines of this general area, there is some room for minor optimization during future development stages.

Environmental and socio-economic criteria have only been considered in broad terms in this site evaluation, and it is therefore biased towards technical and economic criteria. As the project advances and more information become available about baseline conditions and potential environmental effects, the analysis should be revisited to confirm that the identified preferred option remains valid.

17.2.2 Tailings Management Facility Design

Roughly 72 Mt of thickened tailings (about 60% solids by mass) will be centrally discharged at a site located about 5 km southeast of the proposed mill site. This results in an estimated 45 Mm^3 of tailings to be stored, based on conservative estimates for a tailings density of 1.6 t/m^3. The centrally discharged tailings will have a positive beach angle of 4% from a central discharge point. This assumed beach angle should be confirmed with testwork during later studies on the project.

Due to the flat topographical relief of the project area, the tailings will be contained by a ring dam which will prevent migration of tailings, lake bed sediments, and any free water. The tailings facility is designed such that the dams will remain at the initial starter dam height of 2 m, except in the areas were a higher dam is required to contain lake bed sediments or prevent the tailings from encroaching on nearby lakes.

To minimize pre-production capital cost, a dam continuously raised over the LOM would be the preferred construction method. However, due to the need to contain lake bed sediments (from dike construction), the initial ring dam will be constructed to the final dam height in all but a few areas, and only minimal raises (less than 1 m) would be required in the remaining areas. The savings offered by deferring that construction is not significant and, as a result, all tailings dam construction is considered pre-production capital.

Dam

The tailings ring dam will be approximately 7,355 m in length, with a maximum height of 9 m. With the exception of the lake sediment storage area and areas where the tailings will encroach on surrounding lakes, the 2 m high ring dam has no defined crest elevation and will generally follow the original ground elevation. The minimum crest elevation in the region required for lake sediment storage is 401 masl, which includes a 2 m freeboard.

The tailings dam is assumed to be a run-of-quarry (Type A material) rock fill dam with a bituminous liner system on the upstream side. The liner system will extend into a trench excavated to bedrock where it will be tied into the bedrock with a concrete pony wall. Foundation preparation will consist of excavating all overburden material (Type C material) to expose bedrock.

The upstream side of the dam will have side slopes of 3H:1V to facilitate liner placement; the downstream slope will have 2H:1V slopes. The crest will be 5 m wide in areas where the dam is less than 3 m high and 10 m wide elsewhere. The dam crest will be utilized as an access road for maintenance. Figure 17.1 demonstrates a typical dam cross section.

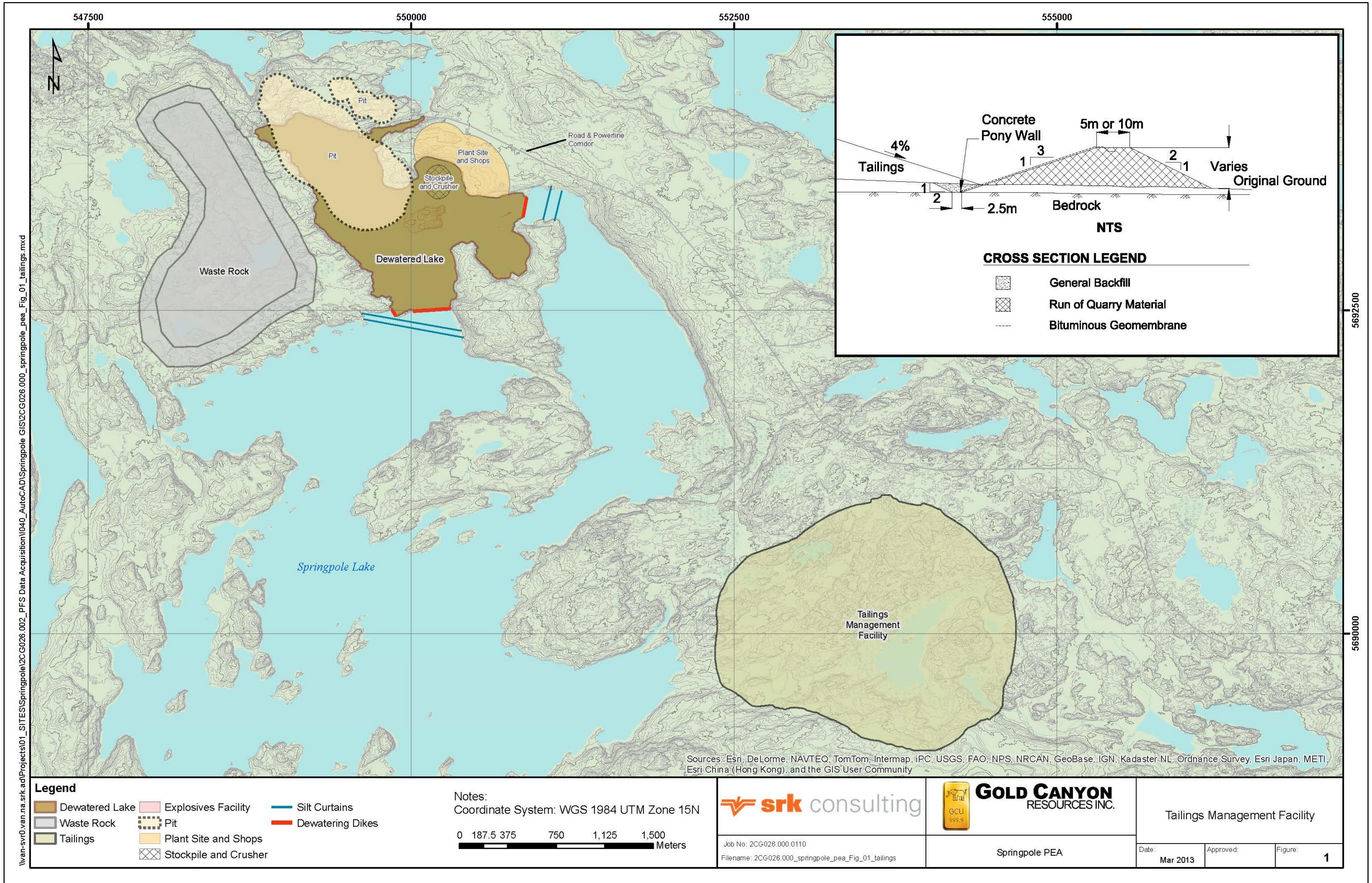


Figure 17.1: Springpole Gold Project Tailings Management Facility

Water Management

During lake dewatering (prior to the placement of tailings), the TMF dam will be used as a settlement pond for water with high total suspended solids and retention of dredged lakebed sediments.

During tailings operations, surface water draining towards the facility will be diverted away from the tailings facility with a diversion ditch. Surface water which collects within the TMF will be collected and pumped back to the mill for reuse/treatment.

Tailings Management

Tailings will be thickened and pumped via two 400 mm diameter heat traced and insulated HDPE pipelines to the tailings management facility a distance of 5.6 km. The tailings will be discharged via a single spigot located in a tower at the centre of the facility. This spigot point will continuously be raised and rotated as deposition advances. A 150 mm diameter heat traced and insulated HDPE return water pipeline will lead back to the plant.

Closure and Reclamation

Once mining operations cease, the dams surrounding the tailings facility will be flattened and contoured to allow for natural drainage. Assuming that the tailings composition does not pose an acid rock drainage risk or metal leaching risk, the tailings will be covered with a simple soil cover from a locally available borrow source. The covered tailings surface would be re-vegetated.

17.3 Dewatering Dikes

The following preliminary design assumptions and/or criteria were taken into consideration for the dewatering dike design:

- Minimum design criteria will be in accordance with Canadian Dam Safety Regulations (CDA 2007).
- The project is located in the low seismic hazard zone of Canada. Therefore, at this stage, there has been no consideration for specific design elements to address seismicity.
- A series of geotechnical drill holes were completed in 2012 at multiple candidate dike locations. These drill holes managed to measure depth to bedrock; however there was no recovery of any overburden soils or lake bed sediments. The designs presented are based on engineering judgement considering the available data supported by visual observation of surficial conditions during the site visit by Megan Miller, EIT and Maritz Rykaart, PEng.
- To allow sufficient time to construct the dikes and dewater the contained lake, the dikes will be constructed two years before mining starts.
- Mine sequencing does not allow mine waste rock to be available for dike construction. A dedicated rock quarry will therefore have to be developed..

Alternate Dike Technology

Alternate construction technologies considered for the dike include:

- Complete dewatering of Springpole Lake followed by conventional dike construction under dry conditions; and
- In-water construction which would not require dewatering of Springpole Lake.

Springpole Lake is a very large lake and dewatering of it to allow dry construction would not be practical or cost effective. In-water construction is, therefore, proposed. To ensure a watertight seal, sheet pile, slurry wall, and grout curtain construction technologies were considered.

Comparison of rock depths from preliminary drill holes in the general vicinity of the proposed dikes and lake bathymetry at the drill hole locations suggest lake sediment thickness between 0.6 m and 8 m, with an average thickness of 4 m. The lake sediments overly reasonably intact rock, although a transition zone of weathered rock is to be expected. Given this limited amount of data, the viability of sheet piles and a slurry wall to produce a good seal is limited due to constructability concerns. Therefore, at this time, a grout curtain has been assumed to be the preferred method.

Alternate Dike Locations

Several locations were proposed for the dewatering dikes as illustrated in Figure 17.2. The general intent was to find locations that minimized the area of lake dewatering, while ensuring that the dikes would be located outside of possible pit expansion footprints (at least 100 m from the final pit rim). In addition, locations with shallow water are preferred as it would allow for optimized dike construction. Based on these criteria dikes C1, C2, and D were selected as the preferred locations, requiring a total lake area of 162 ha to be dewatered, containing about 21.7 Mm^3 of water.

17.3.1 Dike Design

Design Concept

Three dewatering dikes with a total length of approximately 510 m will be constructed in Springpole Lake to allow a portion of the lake to be dewatered. The dikes will be constructed to elevation 391 m above mean sea level, which allows 3 m of freeboard above the lake level.

The dikes will be constructed under wet conditions; therefore, two silt curtains will be deployed downstream of the dike locations to prevent high suspended solids in the remainder of the lake. Prior to the placement of fill material, the foundation of the dam will be dredged to remove any soft lakebed sediments. The rock fill material will be placed, and then the grout curtain and plastic concrete cut-off wall will be built through the completed dike.

The dikes will be constructed with a 12 m wide crest which will act as an access road during construction in wet conditions and the sides of the dikes will slope at angle of repose. The dike will consist of an upstream rip-rap layer (Type D material) over a layer of run-of-quarry rock (Type A material), an inner layer of bedding material (Type B material), through which the plastic concrete slurry wall will be constructed, and finally a downstream layer of run-of-quarry (Type A material) rock. The typical cross section of the dewatering dikes can be seen in Figure 17.2.

A 1 m thick plastic concrete slurry wall will be constructed through the center of the dike and extend a minimum of 1 m into bedrock to provide a water retaining seal. A three row grout curtain cut-off wall will extend 5 m into the bedrock to prevent seepage.

Prior to dike construction, an assumed 5 m of soft lake bed sediments would be removed from within the footprint of the dewatering dikes. These lake bed sediments would be deposited in the TMF.

Lake bathymetry was available for all three of the dike locations selected. Average dike heights were determined to be 1.2, 2.3 and 5.1 m for dikes C1, C2, and D, respectively. The maximum overall dike height, above the lakebed surface, is 8.2 m for dike D.

Quantities associated with the dewatering dikes were calculated based on average lake depth and assumed geometry.

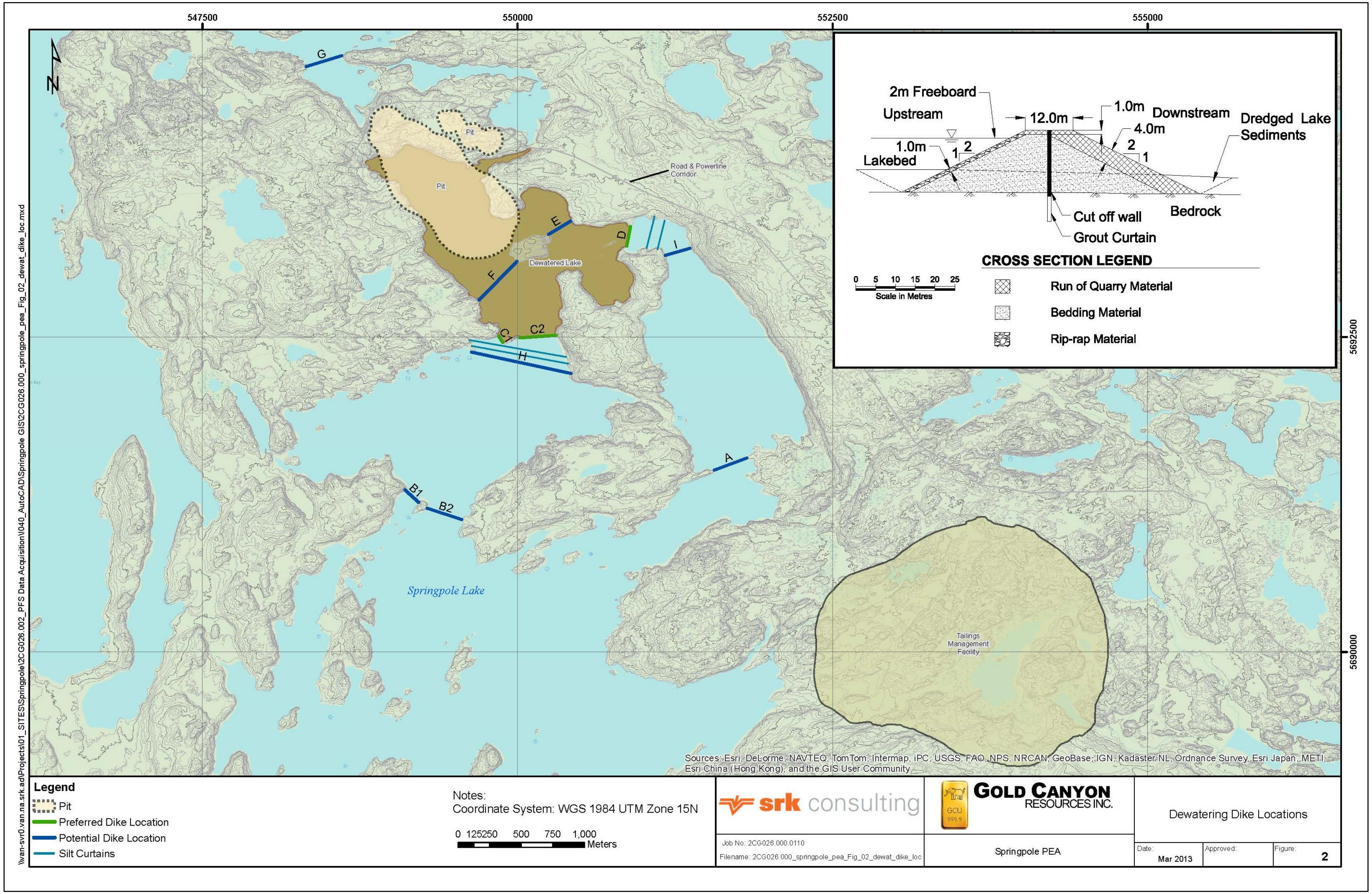


Figure 17.2: Springpole Gold Project Dewatering Dike Locations

Lake Dewatering

An estimated 21.7 Mm^3 of water will have to be drained from the area of Springpole Lake within the dewatering dikes. Of this, 80% (17.4 Mm^3) is estimated to be clean water which can be discharged directly over the dewatering dikes into Springpole Lake, inside the silt curtain. The remaining 4.3 Mm^3 (20%) is assumed to be 'murky' (i.e., have suspended solids higher than the allowable discharge limits). The murky water will be pumped to the TMF which will act as a sedimentation pond; no tailings will be in the TMF at this time. Clear water from TMF will be pumped to Springpole Lake. Steps will be taken during the dewatering process to reduce the amount of sediments that become suspended in the water, including silt curtains around the water intake area.

Due to project economics, it is preferred that the lake be drained within one year of dike completion. In order to dewater the lake in this time period, the dewatering flow rate will be 0.6 m^3/s (assuming dewatering 365 d/y, 24 hr/d). This dewatering rate is within the range of other planned and actual lake dewatering rates (BHP 1995, 1999, and 2001; Diavik 2003; Agnico-Eagle 2010; De Beers 2010). Additionally, literature from other lake dewatering projects indicates that dewatering during the winter months results in less suspended solids. Preliminary hydrological evaluations indicate that the 0.6 m^3/s dewatering rate is possible.

Water Management

During operations, surface water will be diverted from entering the drained portion of Springpole Lake. Management of direct precipitation and seepage into the pit will be included in the mine dewatering activities.

Closure and Reclamation

When mining ceases, pumping will cease and water will be allowed to re-enter the drained lake bed and pit. Once the lake has reached the natural lake level the dewatering dikes will be breached and the above water portions removed. Before breaching of the dikes, silt curtains will be installed. These silt curtains will remain in-place until water sediments reach acceptable levels.

17.4 Infrastructure

17.4.1 Roads

Two Lane Access Corridor Road

This 12 m wide, two-lane unpaved, 39 km access corridor road extends from the Springpole deposit along the Birch River before it connects up with the planned Wenasaga Road (Gold Canyon 2012). This will be the primary access road for the project. The primary normal design vehicles for the road are Super B-Train trucks for hauling of supplies and equipment. Heavy equipment and oversize vehicles will occasionally use the road, especially during the project construction phase.

Road construction will consist of clearing and grubbing of the right of way corridor, prior to placing of an approximately 0.5 m thick compacted sub-base layer sourced from locally developed and approved borrow sources (which has been assumed to be no more than 10 km apart, for a maximum haul distance of 5 km). Based on a cursory review of the alignment using low resolution topographical mapping, it is anticipated that only basic cut/fill techniques will be required to construct the road.

The unpaved road surface will require ongoing maintenance consisting of re-grading and topdressing the running surface to reduce the wear on the haul truck and heavy equipment tires. Topdressing will be sourced from the local borrow sources used during construction. For the purpose of the cost estimate, it has been assumed that annually on average at least 3 cm of new topdressing will be applied to the running surface of the road.

Single Lane Access Roads

There are four 7 m wide single lane access roads located throughout the project area:

- A 5 km section from the Mill to the South Dike,
- A 1 km section from the Access Corridor Road to the Explosives Facility,
- A 0.3 km section from the Access Corridor Road to the Landfill, and
- A 0.2 km section from the Access Corridor Road to the Overburden Dump.

The primary design vehicles for these roads are light trucks, although it is expected that heavy equipment will infrequently travel on these roads.

Construction and maintenance of these roads will be similar to the Access Corridor Road.

Stream Crossings

Two major stream crossings are required along the Access Corridor Road. A 5 m wide, 3 m high and 14 m long arched culvert will be constructed at the Deaddog Stream Crossing. The Birch River Crossing will require a 90 m long pre-fabricated bridge. Both crossings will be clear span bridges that are in accordance with requirements of the Ministry of Natural Resources and the Department of Fisheries and Oceans

Routine surface water management along all roads will be done through crowning of the roads. In 11 key areas along the Access Corridor Road, surface water will be allowed to cross the road via 0.45 m diameter corrugated steel culverts. No culverts have been identified for the single lane access roads.

17.4.2 Buildings

The buildings discussed below will be of modular design or consist of fully contained prefabricated components. They will be shipped to site either as complete units or will require minimal on-site construction, plumbing, and electrical work.

General Maintenance Shop

This is a pre-engineered, insulated sprung structure with a concrete slab on-grade measuring 30 m in length and 25 m in width. This facility will be used for general facility maintenance and upkeep.

Waste Management Building

This open floor plan structure measuring 13 m by 20 m will be used for sorting and handling all the waste produced from building construction and operation activities. If permitting allows, an incinerator associated with the Waste Management Building would be used to burn unpainted construction material and domestic waste, which would reduce the volume being transported to the landfill. Hazardous and recyclable waste will be transported off-site.

Emergency Response Building

This will be a framed structure measuring 10 m in length by 20 m in width and will house all the equipment and facilities to handle site emergencies

Mine Maintenance Shop

This is a framed structure measuring 30 m by 25 m and will be used for general service for mine site surface infrastructure, haul trucks and heavy equipment as well as light vehicles. The building will be equipped with overhead cranes, workbenches and equipment, as well as areas allocated to permit maintenance and fabrication activities.

Light Vehicle Maintenance Shop

This pre-engineered sprung structure with a concrete slab on-grade will be used for the maintenance of light vehicles. The shop will measure 30 m in length by 25 m in width. This building will also be equipped with overhead cranes, workbenches and equipment, as well as areas allocated to permit maintenance and fabrication activities.

Assay Laboratory

This building will be a framed structure measuring 20 m by 22 m. It will have areas allocated for sample preparation and analyses. The assay laboratory is a climate controlled, “clean” building.

Warehousing and Storage

This will be another pre-engineered insulated sprung structure with a concrete slab on-grade. It will measure 25 m in length by 30 m in width. This building will handle all inventories on-site and house stock that must be protected from the environment. It will be associated with a lay-down yard for larger pieces of equipment and materials.

Camp

This will be composed of modular structures that will be transported to site then connected together. The camp will accommodate 300 people in single rooms with dormitory style washrooms. This facility will also contain a kitchen, a mess hall, and recreation facilities.

Water Treatment Plant

This will be a packaged reverse osmosis treatment plant. The treatment plant will be self-contained within a couple of SeaCans which, when connected together, measures 13 by 9 m.

Sewage Treatment Plant

This will also be a packaged bio-reactor treatment plant with a sludge filter press. Just like the Water Treatment Plant, it has been assumed the Sewage Treatment Plant will be contained within a couple of SeaCans which, when connected together, measures 13 by 9 m.

Fuel Storage

Substantial storage of fuel will not be required on-site due to the easy access to the nearby highway. Some fuel storage will be required for the mine, haul, and light vehicle fleets, as well as for the heavy equipment and production of ammonium nitrate/fuel oil, a bulk explosive. For the cost estimate, it has been assumed a 5 ML fuel tank farm, within a bunded area is to be constructed at the mine site. This structure will be shipped as pre-formed panels and trussed which will require on-site erection and fabrication.

Office Complex

It is estimated the mine site will have an office staff of 200 people. Several administration buildings will be required. For the purposes of the cost estimate, four buildings measuring 20 by 20 m were assumed. These will be modular buildings with interior furnishings to allow for all the administrative and day-to-day activities needed for the mine to operate.

Mine Truck Shop

This will also be a framed structure measuring 55 m by 30 m used to service the mining fleet for mine specific operations. This building will also be equipped with overhead cranes, workbenches and equipment, as well as areas allocated to permit maintenance and fabrication activities.

Power Lines

A 60 km long by 23 m wide right-of-way will be cleared, grubbed and prepared for the installation of a 115 kV wood pole transmission line using 636,000 mils conductor. The right-of-way will start from Highway 105 near Ear Falls and travel a further 90 km alongside the existing Hydro One corridor overland where it will connect to and follow the access corridor road to the project site.

18 Market Studies, Pricing, and Contracts

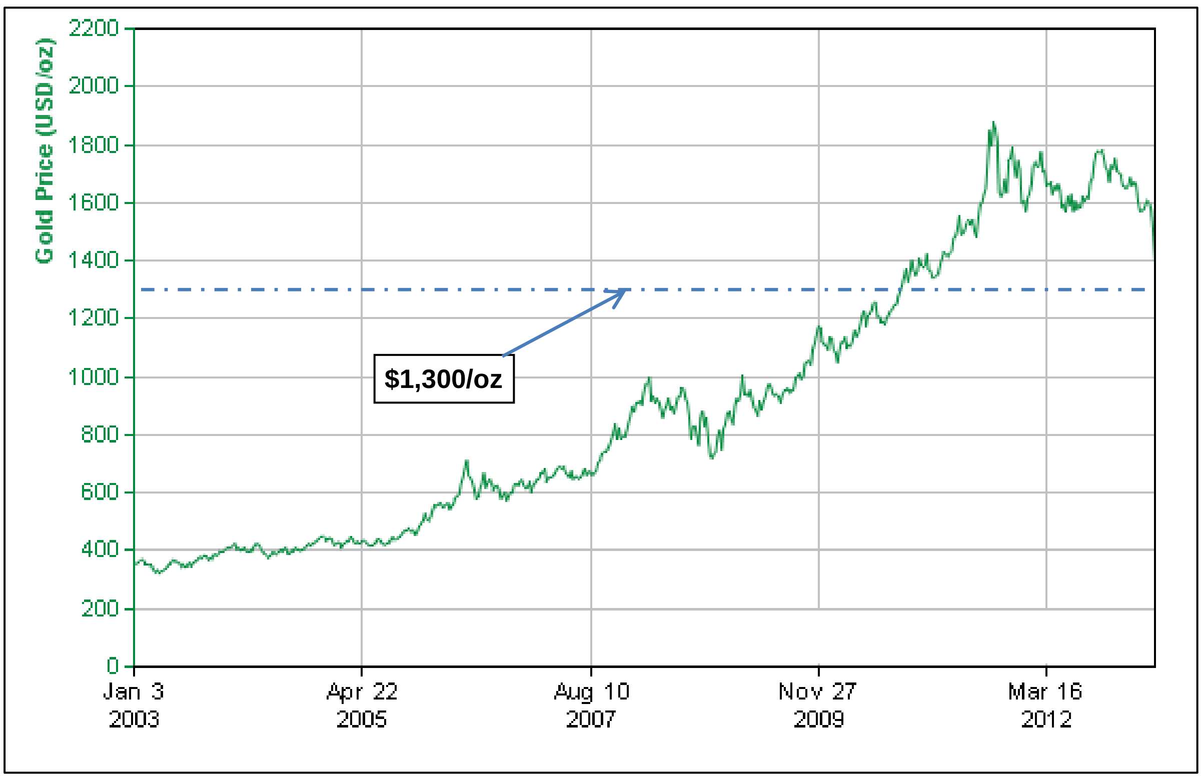
18.1 Market Studies

No project-specific marketing studies were undertaken for the PEA. The planned CIP processing will produce gold doré bullion that is a fungible commodity for which an efficient global market exists. It is of high value density meaning that the realised price of the contained gold is insensitive to the ultimate location of the customer and refinery as freight costs are negligible in comparison to contained value.

Refinery terms of 99.5% payable gold in doré bullion and a refining charge of $5/oz that were used are typical of current terms being offered for CIP produced gold doré bullion.

18.2 Pricing

Based on SRK's review, long-term gold pricing forecasts used for the design of the mining project at $1,300/oz is consistent with gold prices being used in similar publicly released studies (Figure 18.1).



(Source: http://www.infomine.com/ChartsAndData/ChartBuilder.aspx?g=127681&df=20030101&dt=20130423&dr=10y)

Figure 18.1: Historical Gold Prices

18.3 Contracts

No contracts for the sale of the production have been entered into.

19 Environmental Studies, Permitting and Social Setting

19.1 General

The Springpole Gold Project is located in an area of Northwest Ontario which has hosted mineral exploration and mining projects for most of the previous century. The Springpole Gold Project property has a history of gold exploration being carried out during two main periods, one during the 1920s to 1940s, and a second period from 1985 to the present. Previous exploration activities on the property have comprised surface drilling, geophysical surveys, geological mapping, and exploration trail development. The exploration activities have resulted in a network of exploration trails and minor disturbances to the environment due to line cutting, trenching, and surface stripping.

19.2 Environmental Regulatory Setting

The environmental assessment (EA) and permitting framework for metal mining in Canada is well established. The federal and provincial EA processes provide a mechanism for reviewing major projects to assess potential impacts. Following a successful EA, the operation undergoes a licensing and permitting phase to allow operations to proceed. The project is then regulated through all phases (construction, operation, closure, and post-closure) by both federal and provincial departments and agencies.

19.2.1 Federal Environmental Assessment Process

In the spring of 2012, the1992 Canadian Environmental Assessment Act was amended and replaced (CEAA 2012). Two significant results of the updated act were the redefinition of what “triggers” a federal EA and the introduction of legislated time periods within a federal EA, if it is required.

Under CEAA 2012, an EA focuses on potential adverse environmental effects that are within federal jurisdiction including:

- Fish and fish habitat,
- Other aquatic species,
- Migratory birds,
- Federal lands,
- Effects that cross provincial or international boundaries,
- Effects that impact on Aboriginal peoples such as their use of lands and resources for traditional purposes, and
- Environmental changes that are directly linked or necessarily incidental to any federal decisions about a project.

With respect to the Springpole Gold Project, there are two main methods in which a federal EA could be required under CEAA 2012:

1. A proposed project will require an EA if the project is described in the Regulations Designating Physical Activities (CEAA 2012).

2. Section 14(2) of CEAA 2012 allows the Minister of Environment to (by order) designate a physical activity that is not prescribed by regulation if, in the Minister's opinion, either the carrying out of that physical activity may cause adverse environmental effects or public concerns related to those effects may warrant the designation.

With respect to item one above, Section 15 of the Regulations Designating Physical Activities (2012) states:

15. The construction, operation, decommissioning and abandonment of

(a) a metal mine, other than a gold mine, with an ore production capacity of 3000 t/d or more;

(b) a metal mine with ore input capacity of 4,000 t/d or more;

(c) a gold mine, other than a placer mine, with an ore production capacity of 600 t/d or more.

If the proposed project is captured under Section 15 of the Regulations Designating Physical Activities (2012), which is the case for the Springpole Gold Project, the proponent is required to submit a project description to the Canadian Environmental Assessment Agency for screening. The agency will then screen the project to determine if a federal EA is required. If a federal assessment is required, the minister then determines what type of EA the project will require. There are two types of EAs conducted under CEAA 2012: an environmental assessment by "responsible authority" (standard EA) and an environmental assessment by a review panel. Both types of assessments can be conducted by the federal government alone or in conjunction with another jurisdiction. The responsible authority in the case of base and precious metal mining is the CEAA.

Under CEAA 2012, the federal government may also delegate any part of an environmental assessment to the province. At the province's request, the agency may also substitute the provincial process for a federal EA if the provincial EA process meets the requirements of CEAA 2012. Both processes have the potential to streamline the EA process.

In addition to the federally legislated requirements defining the need for an environmental assessment, the federal government introduced the Major Projects Management Office (MPMO) in 2007. The MPMO role is to provide a management and coordinating role for major resource development projects in Canada. The authority and mandate of the office is provided through a committee comprised of deputy ministers from federal departments typically identified as "responsible authorities" in the conduct of a federal environmental assessment. The MPMO has no legislative authority. The MPMO would self-determine their level of involvement in the assessment as part of the original screening process.

19.2.2 Provincial Environmental Assessment Process

In the Province of Ontario, the Environmental Assessment Act (EAA) is administered by the Ministry of Environment (MOE). The Act promotes responsible environmental decision making and ensures that interested persons have an opportunity to comment on projects that may affect them. Under the act, the environment is broadly defined and includes the natural, social, cultural, and economic environment.

Mining projects in Ontario are not usually subject to the EAA because the act does not apply to private companies unless designated by regulation or the proponent has volunteered to be subject to the requirements of the EAA. However, some of the activities associated with the development of a mining project may be subject to the requirements of the EAA through existing class EAs (Class EA) or regulations. Such activities include:

- Granting permits on Crown land, disposition of Crown resources;
- Constructing power generation or transmission facilities;
- Constructing infrastructure related to provincial transportation facilities; and
- Establishing a waste management facility.

19.3 Environmental Assessment Project Requirements

The proposed project will need to be screened under the CEAA 2012. The requirement of a federal EA will become clearer once consultations with CEAA for the development of a project description are completed; however, it is expected that a federal assessment of the proposed project will be required given the project's potential impacts on fish, fish habitat, and other aquatic species.

It is anticipated the project will require multiple class EAs or individual EAs to develop the mining project. Gold Canyon may decide to enter into a Voluntary Agreement with the MOE to subject the Springpole Gold Project to one EA instead of multiple EAs.

The Springpole Gold Project is likely to require a federal and/or provincial EA before it can proceed. Completion of an individual EA, following its initiation, would require approximately 12 to 24 months. Based on current CEAA 2012 guidance documents and the act's new legislated timelines, a standard EA would require 12 to 24 months from the commencement of the federal EA. In the event the final design of the project dictates an amendment to Schedule II of the Metal Mining Effluent Regulations, the time necessary to complete the environmental assessment and subsequent licensing phase would be increased.

Ontario and Canada honour an EA cooperation agreement that harmonizes the two assessment processes to run concurrently under a single administrative process. This process is typically administered jointly by Ontario's Assessment Branch and the CEAA regional office located in Toronto, Ontario. Combining the assessment requirements of both jurisdictions under this cooperation agreement would make it possible to streamline the assessment process.

19.4 Environmental Licensing Process

Following a successful EA, the project will be required to obtain a number of provincial and federal licences/permits. This process can generally be initiated, in part, during the final stages of the EA if one is required. The following sections contain lists of both federal and provincial licences and permits the project will require.

19.4.1 Federal Licences and Approvals

The following federal licences/permits/authorizations are typically required of mining projects of this nature (Table 19.1).

Table 19.1: Required Federal Approvals

Statute	Authorization	Agency	Purpose
Explosives Act	Licence No. 682 – the main Magazine storage of explosives and detonators No. 1168 – Magazine storage for avalanche explosives and detonators	Natural Resources Canada	Authority to manufacture and store explosives
Species at Risk Act	Authorization	Environment Canada	Protect species at risk or near risk
Fisheries Act	Authorization of work affecting fish habitat	Fisheries and Oceans	Any work that has the potential to impact waters defined as fish habitat
Fisheries Act	Fish Habitat Compensation Agreement	Fisheries and Oceans	Habitat compensation agreement to offset fish habitat altered or destroyed as a result of project activities
Navigable Waters Protection Act	Authorization of work affecting navigable waters.	Transport Canada	Authorization for bridge and power lines crossing over navigable waters
Nuclear Safety Control Act	Radioisotope Licence 09-12586-99	Canadian Nuclear Safety Commission	Authorization for Nuclear Density Gauges / X-ray analyzer

19.4.2 Provincial Licences and Approvals

The following provincial licences/permits/authorizations are typically required (Table 19.2).

Table 19.2: Required Provincial Approvals

Statute	Authorization	Agency	Purpose
Environmental Protection Act	Environmental Compliance Approval	MOE	Approval to discharge air emissions and noise
Ontario Water Resources Act	Environmental Compliance Approval	MOE	Approval to treat and discharge effluent
Environmental Protection Act	Environmental Compliance Approval	MOE	Operate a landfill or waste transfer site
Ontario Water Resources Act	Permit to take water	MOE	Use of surface or groundwater
Public Lands Act/ Lakes and Rivers Improvement Act	Work Permit	MNR	Work Permit on crown land
Public Lands Act	Land Use Permit	MNR	Construction of permanent facilities on crown land
Aggregate Resource Act	Aggregate Permits	MNR	Approval to develop and operate aggregate pits
Crown Forest Sustainability Act	Forest Resource Licence	MNR	To harvest crown timber
Endangered Species Act	Overall Benefit Permits	MNR	
Mining Act	Closure Plan	MNDMF	To allow construction and production
Lakes and Rivers Improvement Act	Approval	MNR	Construction of dams and dykes

19.5 Social Setting

Gold Canyon's Springpole Gold Project is located in northwestern Ontario, approximately 110 km northeast of Red Lake. The project is located in an unorganized township, Red Lake Mining District, Casummit Lake Area within the Trout Lake Forest Management Plan. The Red Lake area has been a historic mining camp since the gold rush of the 1920s, and it currently has five active mines and numerous decommissioned or abandoned mines situated within the Municipality of Red Lake. Mineral exploration, mining, mining spin-offs and wilderness tourism (hunting, fishing) comprise the majority of economic activity in the area.

Groups that maybe impacted by the project are the Aboriginal communities in the area including Cat Lake, Slate Falls, Lac Seul, and Wabauskang First Nations; the Métis Nation of Ontario (MNO); remote tourism outfitters and local land owners.

19.5.1 Aboriginal Consultation

Gold Canyon has made it a priority to identify and protect the Aboriginal values and sensitive sites and is committed to carry out meaningful and good faith consultation with the aboriginal communities that may be affected by the project. They have maintained an open-door policy and have provided regular notices and updates regarding their activities on the project. During the archaeological and biological assessment work that was completed in 2012, Gold Canyon hired technicians from the Cat

Lake, Slate Falls, and Lac Seul First Nations to help complete the assessment work and to be liaisons to their communities and participate in the open-house information sessions.

In the spring of 2012, the Chiefs from the First Nation communities of Cat Lake, Slate Falls, and Lac Seul signed an internal protocol agreement to work together for the purpose of negotiations with Gold Canyon. Gold Canyon is also engaged in regular meetings with a working group that is comprised of members from each of the partnership First Nations.

Gold Canyon has had introductory meetings with the Wabauskang First Nation. It is anticipated that there will be future meetings to discuss various parts of the project as they develop and progress.

Gold Canyon has had an introductory meeting with the MNO to provide information about the Springpole Gold Project and has provided notification of the project to the Lands and Resources Branch of the MNO. Additional meetings are anticipated as the various parts of the project develop and progress.

19.5.2 Public Consultation

Gold Canyon has identified many relevant stakeholders in the region including Domtar, the Red Lake Local Citizen’s Committee, the Township of Ear Falls, local tourist operators, outfitters, commercial bait fisherman, bear licence holders and private landowners. Introductory presentations and updates about the project have been delivered by Gold Canyon. Gold Canyon is committed to advancing the consultation process with the affected stakeholders in the region to seek feedback and to help identify concerns so that the appropriate mitigation measures may be developed.

19.6 Preliminary Reclamation Plan

The final closure plan for the proposed project will be developed for the entire project as part of the assessment and licensing process in accordance with Ontario legislation, a financial bond will also be required in accordance with the Ontario Mining Act.

Conceptually, the closure of the proposed project will consist of the following main components:

- decontamination,
- asset removal,
- demolition and disposal, and
- reclamation of all impacted areas.

All project components will be decontaminated as necessary. Surplus chemicals and other hazardous materials will be removed and stored in designated temporary storage facilities within the facility footprint until such time that they can be resold or permanently stored in a licenced facility.

All salvageable or recyclable components will be dismantled and stored in a designated lay down area to allow for secondary decontamination and eventual shipment off-site. All infrastructure that cannot be salvaged and re-used will be demolished and disposed of in an approved facility.

Following any required re-grading, an appropriate cover for the tailings management facility, as well as any remaining waste rock storage piles, will be developed and constructed.

The open pit will be allowed to flood, and, once the water quality is acceptable the coffer dams will be breached, the pit area will again form part of Springpole Lake.

The impacted areas including the tailings and waste rock covers will be vegetated with an appropriate seed mixture designed to enhance natural re-vegetation of the site.

20 Capital and Operating Costs

20.1 Capital Costs

20.1.1 Mine Capital Cost

The CAPEX estimate for the open pit operation is based on the scheduled plant throughput rates, as well as comparing to similar sized open pit gold operations (throughput of 7.3 Mt/a process plant feed). The open pit mining activities for the Springpole pit was assumed to be undertaken by an owner-operated fleet as the basis for this preliminary study with the fleet having an estimated maximum capacity of 85,000 t/d total material, which would be sufficient for the proposed LOM plan.

The open pit equipment CAPEX (including sustaining and replacement costs) required to achieve the target processing rate is summarized in Table 20.1 below. Mining cost service information, as well as factors based on experience, was taken into consideration in determining the open pit CAPEX estimate. No equipment was considered as lease.

The ancillary equipment includes light trucks and service vehicles, backhoes, and fuel trucks, along with a number of other required open pit mining support equipment.

Table 20.1: Open Pit Equipment CAPEX Summary

Item	Unit Cost (M$)	Initial Units	Replace Units	Total Units	Total Cost (M$)
Crawler-Mounted, Rotary Tri-Cone, 250mm Dia. Diesel	2.5	2	1	3	**7.5**
Crawler-Mounted, Rotary Tri-Cone, 165mm Dia. Diesel	1.7	1		1	**1.7**
Crawler-Mounted, Rotary Tri-Cone, 115mm Dia. Diesel	0.7	1		1	**0.7**
Diesel, 22 m^2 Front Shovel	6.7	2		2	**13.4**
Diesel, 20 m^2 Wheel Loader	4.6	1		1	**4.6**
136 t class Haul Truck	2.5	13	12	25	**62.5**
D10-class Track Dozer	1.4	4	3	7	**9.7**
834H-class Rubber tire Dozer	1.1	1	1	2	**2.1**
16H-class Grader	1.0	3	2	5	**5.0**
136 t-class Water truck	1.8	2	2	4	**7.1**
Subtotal Primary					**114**
Subtotal Ancillary					**9.4**
Subtotal Miscellaneous					**4.4**
Total Equipment & Misc.					**128**
Spares Inventory @ 5%					6.4
TOTAL MINE CAPITAL					**135**

Given that mineralized material outcrops at surface, no pre-stripping requirements were allocated for the Springpole pit. The first year of mine productions predicts that 3.9 Mt of mineralized material will be fed to the process plant. As such, no pre-stripping CAPEX have been estimated, other than those described below in relation to lake drainage.

20.1.2 Process Capital Costs

SRK prepared an estimate of capital costs for a 20,000 t/d process plant to treat Springpole material based on testwork results to date. As part of this PEA, such estimates should be considered accurate to ± 40%.

The process flowsheet includes crushing, grinding, gravity recovery, CIP leach as well as gold recovery via activated carbon to produce doré bullion.

Table 20.2 summarizes the process plant capital cost estimate of $148M ± 40%. It is based largely on comparative methods with similar leach plants and adjusted for local conditions and material hardness.

Table 20.2: Springpole Process Plant Capital Cost Estimate (20,000 t/d)

CAPEX Item	$M
Comminution	$27
Leaching	$70
Thicken/Filter	$5
General & Admin	$19
EPCM	$16
Working Capital	$11
Total	**$148**

Notes: cost estimates are considered accurate to ± 40% for this PEA
EPCM = Engineering, Procurement, Construction & Management

This estimate includes working capital of $11M (or 7% of the total). It does NOT include a tailings management facility or contingencies. Figure 20.1 shows a breakdown of the major process capital costs.

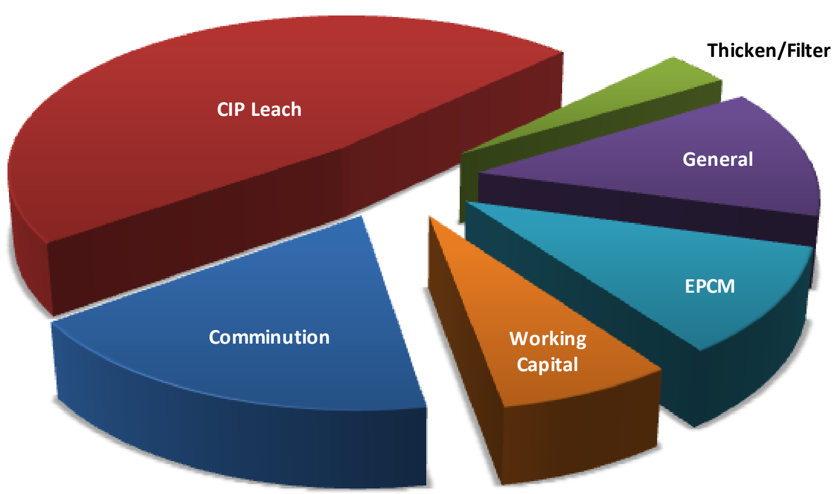


Figure 20.1: Breakdown of Major Process Capital Costs

20.1.3 Infrastructure Costs

A PEA level cost estimate (accurate to ± 40%) has been developed for the infrastructure. Material take-off for earthworks (e.g., roads) is based on first principles calculations using averaged sections. Building costs are based on typical building sizes for similar operations using an in-house SRK database and engineering judgement. Site specific vendor quotes have not been obtained. Project indirect costs are assumed to be 30% and a contingency of 40% applies to the direct costs.

A summary of these costs are presented in Table 20.3.

Table 20.3: Summary of Infrastructure CAPEX and Sustaining Capital

<table>
<tr><th>Cost Type</th><th>Year -3</th><th>Year -2</th><th>Year -1</th><th>Year 1+</th></tr>
<tr><td>Two Lane Access Road (including Stream Crossings)</td><td>$9,679,205</td><td></td><td></td><td></td></tr>
<tr><td>Single Lane Access Roads</td><td></td><td>$794,200</td><td>$218,462</td><td></td></tr>
<tr><td>Maintenance (All Roads)</td><td></td><td>$345,786</td><td>$373,456</td><td>$380,565</td></tr>
<tr><td>General Maintenance Shop[1]</td><td></td><td colspan="2">$1,705,867</td><td></td></tr>
<tr><td>Waste Management Facility[1]</td><td></td><td colspan="2">$1,358,603</td><td></td></tr>
<tr><td>Emergency Response Building[1]</td><td></td><td colspan="2">$384,017</td><td></td></tr>
<tr><td>Mine Maintenance Shop[1]</td><td></td><td colspan="2">$1,074,062</td><td></td></tr>
<tr><td>Light Vehicle Maintenance Shop[1]</td><td></td><td colspan="2">$1,705,867</td><td></td></tr>
<tr><td>Assay Laboratory[1]</td><td></td><td colspan="2">$2,548,734</td><td></td></tr>
<tr><td>Warehouse – Storage[1]</td><td></td><td colspan="2">$1,240,731</td><td></td></tr>
<tr><td>300 – Person Camp[1]</td><td></td><td colspan="2">$20,140,099</td><td></td></tr>
<tr><td>Water Treatment Plant[1]</td><td></td><td colspan="2">$639,498</td><td></td></tr>
<tr><td>Sewage Treatment Plant[1]</td><td></td><td colspan="2">$798,766</td><td></td></tr>
<tr><td>Fuel Storage[1]</td><td></td><td colspan="2">$1,157,838</td><td></td></tr>
<tr><td>Office Complex[1]</td><td></td><td colspan="2">$1,825,011</td><td></td></tr>
<tr><td>Mine Truck Shop[1]</td><td></td><td colspan="2">$8,793,457</td><td></td></tr>
<tr><td>Power Line[1]</td><td></td><td colspan="2">$10,831,739</td><td></td></tr>
<tr><td>Sub-Total DIRECTS</td><td>$9,679,205</td><td><b>$ 28,242,130</td><td>$28,242,130</td><td>$380,565</td></tr>
<tr><td>Indirects (30%)</td><td>$2,903,761</td><td>$8,472,639</td><td>$8,472,639</td><td>$114,169</td></tr>
<tr><td>Contingency (40%)</td><td>$3,871,682</td><td>$11,296,852</td><td>$11,296,852</td><td>$152,226</td></tr>
<tr><td>TOTAL</td><td>$16,454,648</td><td>$48,011,622</td><td>$48,011,622</td><td>$646,960</td></tr>
</table>

Notes: 1. These costs were split equally between Year -2 and Year -1 in the economic model.

20.1.4 Tailings Management, Lake Dewatering and Dike Costs

A PEA level cost estimate (accurate to ± 40%) has been developed for the TMF and dewatering dikes. Primary material take-offs based on the designs presented in Section 17.1 and 17.2 were calculated using Global Mapper. Total quantities for each structure are presented in Table 20.4 and Table 20.5.

Table 20.4: Summary of Key TMF Construction Quantities

Element	Unit	Year -2
Type A fill material (ROQ)	m^3	408,500
Footprint of Dam	m^2	437,600
Excavated Material (Type C material)	m^3	36,800
Bituminous Geomembrane	m^2	110,400
Concrete	m^3	60
Tailings Pipeline (2-400mm HDPE)	m	11,200
Water Reclaim Pipeline (1-150mm HDPE)	m	5,600
Heat Trace (pipelines)		16,700

Table 20.5: Summary of Key Dike and Lake Dewatering Construction Quantities

Element	Unit	Year -2	Year -1
Type A fill material (ROQ)	m^3	52,000	0
Type B fill material (Bedding)	m^3	129,800	0
Type D fill material (Rip-Rap)	m^3	12,600	0
Lake bed sediments	m^3	130,700	0
Silt Curtain (Primary and Secondary)	m^2	28,000	0
Slurry Wall (Plastic Concrete)	m^2	4,800	0
Grout Curtain	m	260	0
Water (to dewater)	m^3	0	21,684,000

Assumptions have been made about haul distances, road grades, material properties, productivity, labour rates, fleet type and equipment and materials rates. These assumptions were based on engineering judgement, past project experience and supplemented with conventional costing databases. Key assumptions are summarized as follows:

- Construction will be done by a dedicated specialist contractor using their own fleet of equipment.
- A fleet of five 40 t (19.85 m^3) CAT 740 haul trucks will be used.
- A CAT 980 loader and a CAT 345 excavator will be used to load the haul trucks at the rock quarry (borrow source).
- Due to mine sequencing, waste rock will not be available for construction of the dewatering dikes or TMF dams. Local rock quarries will be developed and are assumed to be within 5 km of their intended use. Costs include quarry development, drilling, blasting, crushing and screening as appropriate.

- The overburden thickness is unknown, but expected to be limited. Local till within the confines of the TMF is therefore assumed to not be a viable construction source.
- Barges already located at the project site will be utilized for dredging activities.
- In the absence of geotechnical investigations for the dikes, it has been assumed that a grout curtain will be installed to create a suitable seal.
- Costs do not include construction of access roads from the mill or haul roads from potential quarries (these costs are included in the infrastructure cost estimate).
- Costs for the tailings thickeners and both tailings and reclaim water pumps are excluded here. Those costs are included in the processing CAPEX.
- Lake dewatering costs are considered to be CAPEX.
- Closure and reclamation costs are not included in this cost estimate.
- Costs are estimated in 2013 USD.
- Indirect costs have been assumed to be 30% of the direct costs
- A contingency of 40% of direct costs was applied.

The dewatering dikes must be constructed at least two years before mine production starts to provide enough time for lake dewatering. Since the TMF will be used as the settling basin during this time, it will be constructed at the same time. Table 20.6 and Table 20.7 summarize these costs.

Table 20.6: Summary of TMF CAPEX and Sustaining Capital

Cost Type	Year -2
Direct Costs	$22,557,100
Indirect Costs	$9,022,900
Contingency	$6,767,100
Total Costs	**$38,347,100**

Table 20.7: Summary of Dike and Lake Dewatering CAPEX and Sustaining Capital

Cost Type	Year -2	Year -1
Direct Costs	$17,336,400	$1,502,400
Indirect Costs	$5,200,900	$450,700
Contingency	$6,934,600	$600,900
Total Costs	**$29,471,900**	**$2,554,000**

20.1.5 Surface Water Control

Costs were built up based on the proposed waste rock facility design configurations and existing topography. Water diversion and catchment requirements were estimated.

PEA level capital costs were estimated for the preliminary design and presented in Table 20.8.

The following assumptions were utilized in the cost estimate.

- The construction fleet will consist of five CAT 740 haul trucks, a CAT 345 excavator, and a CAT 980 loader.
- No blasting is required to excavate channels for diversion ditches.
- Sedimentation and pollution control ponds will be lined with bituminous geomembrane liner.
- Construction material (quarried rock, bedding material and rip-rap) is assumed to be available within 5 km.
- Indirect costs at 30% of direct costs.
- Contingency at 40% of direct costs.

Table 20.8: Preliminary Cost Estimate for Surface Water Management

Item	Cost (2013 Dollars)
Direct Costs	*4,186,000*
Excavation	30,000
Pond Berms	895,000
Liner Deployment	2,990,000
Pumping	271,000
Indirect Costs	*2,930,000*
Indirect costs	1,256,000
Contingency	1,675,000
Total Costs	*7,116,000*

Note: Costs are estimated in 2013 USD.

20.2 Operating Costs

20.2.1 Mining Operating Costs

The open pit mining activities for the Springpole deposit were assumed to be primarily undertaken by the owner as the basis for the PEA. The cost estimate was built from first principles, input from Gold Canyon, as well as SRK experience of similar sized open pit operations. Equipment efficiency was estimated based on-site conditions (e.g., estimated haul routes for each phase). Local labor rates (for operating, maintenance, and supervision/technical personnel) and estimates on diesel fuel pricing ($1.10/L) were taken into consideration for the mining cost estimate.

The Open Pit mining costs were calculated for both mineralized material and waste mining, where variations in haulage profiles and equipment selection were taken into account in the cost estimate. Open pit mining costs for this preliminary assessment were estimated to be $2.57/t waste mined and $2.78/t mineralized material mined.

The mining cost estimates encompass open pit and dump operations, road maintenance, mine supervision and technical services.

20.2.2 Process Operating Costs

A comparative estimate was made by SRK for the process OPEX of $10.5/t ± 40% (excluding tailings and general and administrative (G&A) costs). Table 20.9 summarizes the breakdown of operating expenses by area.

Table 20.9: Springpole Process Plant Operating Cost Estimate (20,000 t/d)

OPEX Item	$/t
Comminution	$4.9
Leaching	$4.5
Thicken/Filter	$0.3
General	$1.1
EPCM	$1.2
Sustaining Capital	$1.0
Total	**$13.0**
Tailings	$(0.2)
G&A	$(2.3)
Net Processing	**$10.5**

Notes: cost estimates are considered accurate to ± 40% for this PEA
EPCM = Engineering, Procurement, Construction & Management

An electrical power unit cost of $0.08/kWh was assumed for this project which was factored in with the material hardness and relatively fine primary grind requirements. The net processing cost after tailings and G&A costs are removed is $10.5/t.

Grinding media and liner wear were considered typical and the labour cost is for 95 process employees (staff + hourly) with a 42% burden applied. The OPEX includes sustaining capital which has been estimated at $1/t.

20.2.3 Tailings, Dewatering Dike and Surface Water Management Operating Costs

OPEX for the TMF consist of operating the secondary thickener, and subsequently pumping tailings from the secondary thickener to the central TMF discharge point. This point has to be raised over the LOM. This cost equates to $0.83/t. The cost of pumping tailings slurry to the secondary thickener, and subsequently pumping recycle water back to the mill, is included in the mill operating cost.

No operating costs were included for the dewatering dikes. It is assumed that any water seeping through the dikes will be included in the mining water management costs. Yearly operating costs for surface water management are estimated to be $33,000, assuming pumps are operating 50% of the time.

21 Economic Analysis

21.1 Important Notice

The Economic Analysis that forms part of this preliminary economic assessment (PEA) report is intended to provide an initial review of the Gold Canyon Resources Inc. Springpole Gold Project's potential and is preliminary in nature. The economic analysis incuded in this PEA includes consideration of inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA based on these mineral resources will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

21.2 Introduction

A PEA-level technical economic model has been developed for the Springpole Gold Project.

The PEA contemplates mining and processing material at 20,000 t/d and at an average head grade of 1.2 g/t gold and 6.0g/t silver. Summary parameters and economic results are presented in Table 21.1.

Table 21.1: Summary of Economic Parameters and Results

Parameter	Units	11 year LOM
Mill Feed Mined	kt	72,421
Payable Au	koz	2,211
Payable Ag	koz	11,658
Au Price	$/oz	1,300
Ag Price	$/oz	25
Gross Revenue	$M	3,166
Treatment and Refining Costs	$M	11
Royalty	$M	95
Operating Costs	$M	1,454
Operating Surplus	$M	1,606
Capital Costs	$M	544
Economic Results		
Pre-tax $NPV_{5\%}$	$M	579
Pre-tax IRR	%	25.4%
Non-discounted Pre-tax Payback[1]	months	21
Post-tax NPV5%	$M	**388**
Post-tax IRR	%	**13.8%**
Non-discounted Post-tax Payback[2]	**months**	**35**

21.3 Key Assumptions

The following production related assumptions have been applied to the technical economic model:

[1] From commencement of production
[2] From commencement of production

- Production rate at maximum of 20,000 t/d over 365 d/yr.
- Pre-production period of five years.

In addition, the following general assumptions have been applied for mine design and economic evaluation:

- A base case discount factor of 5% has been applied for NPV calculations. SRK considers this to be typical for gold projects of this type and in this location. .
- An average LOM sales price of $1,300/oz gold.
- An average LOM sales price of $25/oz silver.
- Net smelter royalty of 3% on revenue.
- Working capital days have been assumed at 45 days for creditors and 45 debtors.

21.3.1 Modelling Practice

The project was evaluated using an Excel® based discounted cashflow model. The periods used were annual. The model used real, un-escalated Q1 2013 USD.

The asset-level model assumes a simple all-equity project ownership and financing. No consideration of equipment leasing, project financing, bonding, metal strips, royalty sales (except for existing government and private royalties) forward sales, hedging, or any other financial arrangements was undertaken. No consideration was given to the structure of the ownership company.

The valuation was undertaken on a pre-tax and post-tax basis.

21.3.2 Construction Schedule

For the purposes of economic evaluation, five years of pre-production were assumed, comprising three years for permitting and two years for construction.

Delays to commencement of construction do not materially alter the economic potential of the underlying project, but it must be recognised that costs associated with permitting, studies and management activities will accrue during the pre-construction phase. It should be noted that the impact of costs associated with a delayed construction schedule on the economics of the overall project have not been considered in modelled scenarios.

21.3.3 Production Schedule

The mining production schedule evaluated was generated by SRK as described in Section 15.5 and is reproduced in Table 21.2.

Table 21.2: Base Case Production Schedule

Production	Units	Total	Year										
			1	2	3	4	5	6	7	8	9	10	11
Waste	kt	120,752	16,136	2,881	20,346	13,632	6,111	8,765	22,644	19,809	7,534	2,594	300
Mill Feed	kt	72,421	3,864	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	2,857
ROM Au Grade*	g/t Au	1.19	1.54	1.47	1.32	0.78	0.97	1.41	1.54	0.83	0.97	1.13	1.53
Contained Metal	koz Au	2,777	191	345	310	183	228	331	361	195	228	265	141
ROM Ag Grade*	g/t Ag	6.01	3.64	8.78	6.29	2.99	5.21	7.52	5.78	5.95	5.91	6.25	7.72
Contained Metal	koz Ag	13,995	452	2,061	1,476	702	1,223	1,765	1,357	1,396	1,387	1,467	709
Total Material	kt	193,173	20,000	10,181	27,646	20,932	13,411	16,065	29,944	27,109	14,834	9,894	3,157
Strip Ratio	t:t	1.67	4.18	0.39	2.79	1.87	0.84	1.20	3.10	2.71	1.03	0.36	0.11

**includes 5% mining dilution*

21.3.4 Mine Life

The LOM is 11 production years. The first year of production includes near-surface material that does not require pre-stripping in year 1. Both mineralized material and waste mining ends in Year 11.

21.3.5 Commodity Pricing

The base case economic evaluation uses long-term commodity prices of $1,300/oz for gold and $25/oz for silver. Sales prices have been applied to all LOM production without escalation or hedging.

SRK considers the use of these prices for design and evaluation to be reasonable and to lie within the range of published price assumptions recently used for studies of this type.

21.3.6 Revenue Calculations

Revenue is determined by applying selected metal prices to the annual payable metal contained in doré, minus refining and royalty payments. Detailed calculation of revenue is presented in Table 21.3.

Table 21.3: Base Case Revenue Calculation

Parameter	Units	11 year LOM
Milled Feed	**kt**	**72,421**
Contained Au	koz Au	2,777
Contained Ag	koz Ag	13,995
Metallurgical Recovery		
Au	%	80
Ag	%	85
Recovered Metals		
Au	koz	2,222
Ag	koz	11,896
Refinery Payables		
Au	%	99.5
Ag	%	98.0
Payable Metals in Doré		
Au	koz	2,211
Ag	koz	11,658
Revenues		
Commodity Sales Prices		
Au	$/oz	1,300
Ag	$/oz	25
Value of Metal in Doré Before Deductions		
Au	$M	2,889
Ag	$M	297
Gross Revenue from Doré Before Deductions	**$M**	**3,186**
Revenue from Doré after Payable Deductions		
Au	$M	2,874
Ag	$M	291
Revenue from Doré after Payable Deductions	**$M**	**3,166**
Treatment and Refining Costs		
Au	koz	2,211
Treatment Charge		
Au Refining Costs	$/oz Au	5
Treatment and Refining Costs	$M	11
Net Smelter Return	**$M**	**3,154**
NSR based Royalty		
Rate	%	3
NSR Royalty	$M	95
Net Revenue from Doré after Deductions	**$M**	**3,060**

21.4 Mining

21.4.1 Capital Costs

Mining capital costs were estimated by SRK on the basis of a detailed equipment schedule matched to the mining production schedule. Total mining capital was estimated at $140.9M for the life of the project inclusive of 5% contingency and 5% spares allowance.

An amount of $20M is estimated for mine closure costs.

21.4.2 Operating Costs

Mine operating costs were estimated by SRK based on the mine design, production schedule, equipment rates and other input costs.

Table 21.4 shows a high-level summary of the LOM costs expressed per tonne of material moved and per unit of processed material. No contingency has been assumed for mine operating cost.

Table 21.4: Mine Operating Unit Costs (USD)

Mine Operating Unit Cost	Per Tonne of Material	Per Tonne of Mill Feed
Drilling	0.17	0.44
Blasting	0.31	0.83
Loading	0.32	0.86
Hauling	0.98	2.60
Roads/Dumps/Support Equipment	0.50	1.33
General Mine/Maintenance	0.15	0.40
Supervision & Technical	0.22	0.59
Total Mine Operating Unit Cost	**2.65**	**7.06**

21.5 Processing

21.5.1 Capital Costs

Processing capital costs for mill and plant infrastructure were estimated by SRK to be $168M inclusive of $20M contingency.

21.5.2 Operating Costs

Operating costs were estimated by SRK to be $10.5/t of processed material based on 20,000 t/d plant capacity. The base case assumes a power cost of 0.08 $/kWh and no contingency has been assumed for processing operating cost.

21.6 Overall Capital Costs

Overall Capital costs are summarized in Table 21.5 and detailed capital costs for the LOM are presented in Table 21.6.

Table 21.5: Capital Cost Summary

Capital costs by timing	$M
Total Preconstruction Owners Costs	7
Initial Capital	431
Sustaining Capital	86
Closure	20
***Total Capital Costs**	**544**

Including weighted-average overall contingency of 15%

Table 21.6: Capital Cost Breakdown

Pre-production owners costs	$M
Permitting	4
Fisheries Compensation	3
Total pre-production capital costs	7
Mine Capital Costs	
Primary Mine Equipment	114
Ancillary Mine Equipment	9
Miscellaneous	4
Spares Inventory @ 5%	6
Contingency @ 5%	6
Total Mine Equipment Capital	**141**
Tailings and Water Capital	77
Indirect Costs	6
Contingency	9
Total Other Mine Capital	**91**
Total Mine Capital (excl. closure)	**232**
Processing, Roads and Infrastructure Capital Costs	
Processing	
Comminution	22
CIP Leach	60
Thickening/Filtering	4
Indirect Costs	62
Contingency	20
Total Processing Initial Capital	**168**
Roads and Infrastructure	
Roads	26
Surface Infrastructure Buildings	43
Indirect Costs	21
Contingency	28
Total Roads and Infrastructure	**117**
Total Processing and Infrastructure	**285**
Mine Closure Costs	**20**

21.7 Overall Operating Costs

Operating costs for LOM and operating unit costs are presented in Table 21.7 and Table 21.8, respectively.

Table 21.7: Life of Mine Operating Cost

Activity	$M
Mining	511
Processing	760
Tailings Handling	16
General & Administrative	167
Total Operating Cost	**1,454**

Table 21.8: Operating Unit Costs

Cash Costs	Per Tonne of Mill Feed	Per Ounce of AuEq*
Mine Unit Costs	7.1	209.9
Processing Unit Costs	10.5	312.3
Tailings and Water Unit Costs	0.2	6.4
G&A Unit Costs	2.3	68.4
Total Operating Cost	**20.1**	**597.0**
Royalty Per Ounce		38.9
Total Cash Costs including Royalty	**20.1**	**635.9**

**AuEq = total revenue divided by gold price*

21.8 Taxes and Royalties

21.8.1 Royalty

For project evaluation purposes in the PEA, an average royalty of 3% was applied to the net smelter return based on the Mineral Resource Update Section 13 of this report. The net smelter return is $3,154M calculated from: (payable metal) x (payable %) x (price minus refining charges). Total royalty payment over the LOM is estimated at $95M.

21.8.2 Taxes

Tax was estimated on a simplified basis, appropriate for a PEA-level evaluation. Taxes modelled were Ontario Mining Tax (OMT), Ontario Provincial Corporate Tax and Federal Corporate Tax.

Depreciation

Simplified depreciation schedules were estimated based on capital investment. A depletion method was applied as a proxy for more complex depreciation schedules that are likely to be used in practice. In SRK's experience, depletion methods tend to be slightly conservative when compared to more complex methods and result in earlier tax payments modelled earlier than is typically acievable. Losses associated with post closure expenditure were carried back for a maximum of three years.

Pre-esisting depreciation allowances associated with prior expenditure of $45m were modelled on advice from the client. These amounts are not material in the context of a PEA-level evaluation and were not audited by SRK.

Ontario Mining Tax

The assumption was made that the mine was “remote” for the purposes of OMT caluations and accordingly a rate of 5% was applied. Royalties were included in the basis of calculation for this tax.

Ontario Provincial Corporate Tax

A rate of 10% was applied to taxable income.

Federal Corporate Tax

A rate of 15% was applied to taxable income. Note the provicnical and federal taxes are applied to the same basis. Neither is deductible for estimation of the other.

21.9 Working Capital

A high level estimation of working capital has been incorporated into the cashflow. This comprises estimates of accounts receivable terms, accounts payable terms and stores stock levels.

21.10 Base Case Valuations

21.10.1 Important Notice

The Economic Analysis that forms part of this preliminary economic assessment (PEA) report is intended to provide an initial review of the Gold Canyon Resources Inc. Springpole Gold Project’s potential and is preliminary in nature. The economic analysis incuded in this PEA includes consideration of inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA based on these mineral resources will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

21.10.2 Results

The primary economic evaluation measures used were total LOM cashflow, NPV of this cashflow at a 5% discount rate, the internal rate of return of the project cashflows, and the payback period on a non-discounted basis. Table 21.9 summarizes the high level economic outputs from the modelling. Note that payback is quoted from the commencement of production.

Table 21.9: Summary Economics

Parameter	Units	Base Case
Off-Site Cost	$/oz AuEq	5
Royalty @ 3% of NSR	$/oz AuEq	39
Mill Feed (LOM)	Mt	72
Average ROM Au Grade	g/t Au	1.2
Average ROM Ag Grade	g/t Ag	6.0
Au Process Recovery	%	80%
Ag Process Recovery	%	85%
Payable Au Produced	koz	2,211
Payable Ag Produced	koz	11,658
Unit Operating Cost per Tonne Milled	$/t	20
Unit Operating Cost	$/oz AuEq	636
Pre-Production Capital Cost	$M	438
Capital Cost (LOM)	$M	544
Pre-Tax $NPV_{0\%}$	$M	1,058
Pre-Tax $NPV_{5\%}$	$M	579
Pre-Tax IRR	%	25.4%
Pre-Tax Payback Period	Months from start prod.	21
Post-Tax $NPV_{0\%}$	**$M**	**760**
Post-tax NPV5%	**$M**	**388**
Post-tax IRR	**%**	**13.8%**
Non-discounted Post-tax Payback[3]	**Months from start prod.**	**35**

Note: At $1,300/oz gold and $25/oz silver

A summary of annual cashflows produced from the technical economic model at gold prices of $1,300/oz and silver price of $25/oz are presented in Table 21.10.

[3] From commencement of production

Table 21.10: Annual Cashflow Summary

	Units	Total	NPV	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Production Schedule																					
Au Price	$/oz	1,300		1,690	1,600	1,490	1,340	1,330	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Ag Price	$/oz	25		32	28	26	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25
Mineralised Material	kt	72,421		0	0	0	0	0	3,864	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	2,857	0	0
Run-of-mine Au Grade	g/t	1.19		0.00	0.00	0.00	0.00	0.00	1.54	1.47	1.32	0.78	0.97	1.41	1.54	0.83	0.97	1.13	1.53	0.00	0.00
Run-of-mine Ag Grade	g/t	6.01		0.00	0.00	0.00	0.00	0.00	3.64	8.78	6.29	2.99	5.21	7.52	5.78	5.95	5.91	6.25	7.72	0.00	0.00
Au Recoverable Grade	g/t	0.95		0.00	0.00	0.00	0.00	0.00	1.23	1.18	1.06	0.62	0.78	1.13	1.23	0.66	0.78	0.90	1.22	0.00	0.00
Ag Recoverable Grade	g/t	5.11		0.00	0.00	0.00	0.00	0.00	3.09	7.46	5.35	2.54	4.43	6.39	4.91	5.06	5.02	5.31	6.56	0.00	0.00
Au Payables	%	100%		0%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%
Ag Payables	%	98%		0%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%
Au Payable Metal	koz	2,211		0	0	0	0	0	152	275	247	146	181	263	288	155	181	211	112	0	0
Ag Payable Metal	koz	11,658		0	0	0	0	0	377	1,717	1,230	585	1,019	1,470	1,130	1,163	1,155	1,222	591	0	0
Revenue	**$M**	**3,166**	**2,034**	**0**	**0**	**0**	**0**	**0**	**207**	**400**	**351**	**204**	**261**	**379**	**402**	**231**	**264**	**305**	**160**	**0**	**0**
Treatment and Refining Costs	$M	11	7	0	0	0	0	0	1	1	1	1	1	1	1	1	1	1	1	0	0
Net Smelter Return																					
NSR Royalty	$M	95	61	0	0	0	0	0	6	12	11	6	8	11	12	7	8	9	5	0	0
Net Revenue after TCRC and Royalties	**$M**	**3,060**	**1,966**	**0**	**0**	**0**	**0**	**0**	**200**	**387**	**340**	**197**	**252**	**367**	**389**	**223**	**256**	**295**	**155**	**0**	**0**
Operating Costs																					
Mining	$M	511	329	0	0	0	0	0	45	33	61	54	43	46	68	67	47	34	13	0	0
Processing	$M	760	485	0	0	0	0	0	41	77	77	77	77	77	77	77	77	77	30	0	0
Tailings and Water	$M	16	10	0	0	0	0	0	1	2	2	2	2	2	2	2	2	2	1	0	0
G&A	$M	167	106	0	0	0	0	0	9	17	17	17	17	17	17	17	17	17	7	0	0
Total Operating Costs	**$M**	**1,454**	**930**	**0**	**0**	**0**	**0**	**0**	**95**	**128**	**156**	**149**	**138**	**141**	**163**	**162**	**142**	**129**	**50**	**0**	**0**
Unit Operating Costs per Ounce																					
Mining Unit Costs	$/oz$_{AuEq}$		210	0	0	0	0	0	283	108	224	342	214	159	220	378	232	146	103	0	0
Processing Unit costs	$/oz$_{AuEq}$		312	0	0	0	0	0	254	249	284	488	382	263	248	432	377	327	243	0	0
Tailings and Water Unit Costs	$/oz$_{AuEq}$		6	0	0	0	0	0	6	5	6	10	8	5	5	9	8	7	6	0	0
G&A Unit costs	$/oz$_{AuEq}$		68	0	0	0	0	0	56	55	62	107	84	58	54	95	83	72	53	0	0
Total Unit Operating Costs	**$/oz$_{AuEq}$**		**597**	**0**	**0**	**0**	**0**	**0**	**598**	**417**	**576**	**947**	**687**	**484**	**527**	**914**	**699**	**551**	**406**	**0**	**0**
Royalty per ounce	$/oz$_{AuEq}$		39	0	0	0	0	0	39	39	39	39	39	39	39	39	39	39	39	0	0
Total Unit Operating Costs Including Royalty	**$/oz$_{AuEq}$**		**636**	**0**	**0**	**0**	**0**	**0**	**637**	**456**	**614**	**986**	**726**	**523**	**566**	**952**	**738**	**590**	**445**	**0**	**0**
Unit Operating Costs per Tonne																					
Mine Unit Costs	$/t		7.1	0	0	0	0	0	12	5	8	7	6	6	9	9	6	5	4	0	0
Processing Unit costs	$/t		10.5	0	0	0	0	0	11	11	11	11	11	11	11	11	11	11	11	0	0
Tailings and Water Unit Costs	$/t		0.2	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
G&A Unit costs	$/t		2.3	0	0	0	0	0	2	2	2	2	2	2	2	2	2	2	2	0	0
Total Unit Operating Costs	**$/t**		**20.1**	**0**	**0**	**0**	**0**	**0**	**25**	**18**	**21**	**20**	**19**	**19**	**22**	**22**	**19**	**18**	**18**	**0**	**0**
Operating Surplus	**$M**	**1,606**	**1,036**	**0**	**0**	**0**	**0**	**0**	**105**	**258**	**184**	**49**	**114**	**225**	**226**	**61**	**113**	**166**	**105**	**0**	**0**
Capital Costs (High Level)																					
Preconstruction Owners Costs	$M	7	7	1	2	2	1	1	0	0	0	0	0	0	0	0	0	0	0	0	0

	Units	Total	NPV	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Initial Capital	$M	431	373	0	0	17	200	214	0	0	0	0	0	0	0	0	0	0	0	0	0
Sustaining Capital	$M	86	56	0	0	0	0	0	7	15	1	1	1	23	31	6	1	0	0	0	0
Closure	$M	20	9	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	10	10
Total Capital Costs	**$M**	**544**	**445**	**1**	**2**	**19**	**201**	**215**	**7**	**15**	**1**	**1**	**1**	**23**	**31**	**6**	**1**	**0**	**0**	**10**	**10**
Working Capital	$M	4	11	0	0	0	0	0	21	22	-7	-17	7	14	2	-20	5	6	-12	-16	0
Pre-tax Cash Flow	**US$m**	**1,058**	**579**	**-1**	**-2**	**-19**	**-201**	**-215**	**78**	**222**	**190**	**64**	**106**	**189**	**192**	**75**	**107**	**160**	**117**	**6**	**-10**
Ontario Mining Tax	US$m	56	36	0	0	0	0	0	2	10	8	1	3	8	9	1	3	5	5	1	0
Ontario Provincial Corporate Tax	US$m	147	94	0	0	0	0	0	6	27	20	2	8	22	23	3	6	14	13	2	0
Federal Corporate Tax	US$m	98	63	0	0	0	0	0	4	18	13	2	5	15	15	2	4	9	9	2	0
Post-tax Cash Flow	**US$m**	**760**	**388**	**-1**	**-2**	**-19**	**-201**	**-215**	**69**	**167**	**149**	**60**	**89**	**143**	**145**	**70**	**94**	**132**	**90**	**1**	**-10**

Note: At $1,300/oz gold and $25/oz silver.

21.11 Sensitivities

21.11.1 Important Notice

The Economic Analysis that forms part of this preliminary economic assessment (PEA) report is intended to provide an initial review of the Gold Canyon Resources Inc. Springpole Gold Project's potential and is preliminary in nature. The economic analysis incuded in this PEA includes consideration of inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA based on these mineral resources will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

21.11.2 Results of Sensitivity Analysis

SRK has performed a sensitivity analysis on the base case settings by applying sensitivity to changes in commodity prices, OPEX, and CAPEX. The results of this analysis are presented in Table 21.11, Table 21.12, and Table 21.13. The values presented are Post-tax Net Present Value at a 5% discount rate

An optimized mining and processing plan was not developed for each case.

Table 21.11: Effect of Variation in Commodity Prices and Operating Costs on Post Tax NPV_5

Net Present Value			Metal Prices ($/oz)								
		Au>>>	900	1000	1100	1200	1300	1400	1500	1600	1700
		Ag>>	17	19	21	23	25	27	29	31	33
		388	-31%	-23%	-15%	-8%	0%	8%	15%	23%	31%
Operating Cost ($/t)	$16.06	30%	(240)	(133)	(26)	82	189	297	404	511	619
	$20.07	20%	(174)	(67)	41	148	256	363	470	578	685
	$22.08	10%	(108)	(1)	107	215	322	429	537	644	751
	$24.09	0%	(42)	66	174	281	**388**	495	603	710	817
	$26.10	-10%	25	132	240	347	454	562	669	776	884
	$28.10	-20%	91	199	306	413	521	628	735	843	950
	$30.11	-30%	158	265	372	480	587	694	802	909	1016

Table 21.12: Effect of Variation in Commodity Prices and Capital Costs on Post Tax NPV_5

Net Present Value			Metal Prices ($/oz)								
		Au>>>	900	1000	1100	1200	1300	1400	1500	1600	1700
		Ag>>	17	19	21	23	25	27	29	31	33
		388	-31%	-23%	-15%	-8%	0%	8%	15%	23%	31%
LOM Total Capital ($M)	490	-10%	(2)	105	213	320	427	535	642	749	857
	544	0%	(42)	66	174	281	**388**	495	603	710	817
	599	10%	(80)	28	136	243	350	457	565	672	779
	653	20%	(116)	(9)	99	206	313	421	528	635	743
	708	30%	(152)	(45)	*63*	170	278	385	492	600	707
	762	40%	(186)	(79)	29	136	243	351	458	565	673
	816	50%	(220)	(112)	(4)	103	210	317	425	532	639

Table 21.13: Effect of Variation in Operating and Capital Costs on Post Tax NPV$_5$

Net Present Value			Operating Cost ($/t feed)								
			$12.04	$14.05	$16.06	$18.07	$20.07	$22.08	$24.09	$26.10	$28.10
		388	-40%	-30%	-20%	-10%	0%	10%	20%	30%	40%
LOM Total Capital ($M)	381	-30%	774	708	642	576	509	443	377	310	244
	435	-20%	733	666	600	534	468	401	335	269	202
	490	-10%	692	626	560	494	427	361	295	229	162
	544	0%	653	587	521	454	**388**	322	256	189	123
	599	10%	615	549	483	416	350	284	218	151	85
	653	20%	578	512	446	380	313	247	181	115	48
	708	30%	543	477	410	344	278	212	145	79	12
	762	40%	508	442	376	310	243	177	111	45	(22)
	816	50%	475	409	343	276	210	144	78	11	(55)

Note: At $1,300/oz gold and $25/oz silver

A graphical illustration of sensitivity analysis is presented in Figure 21.1.

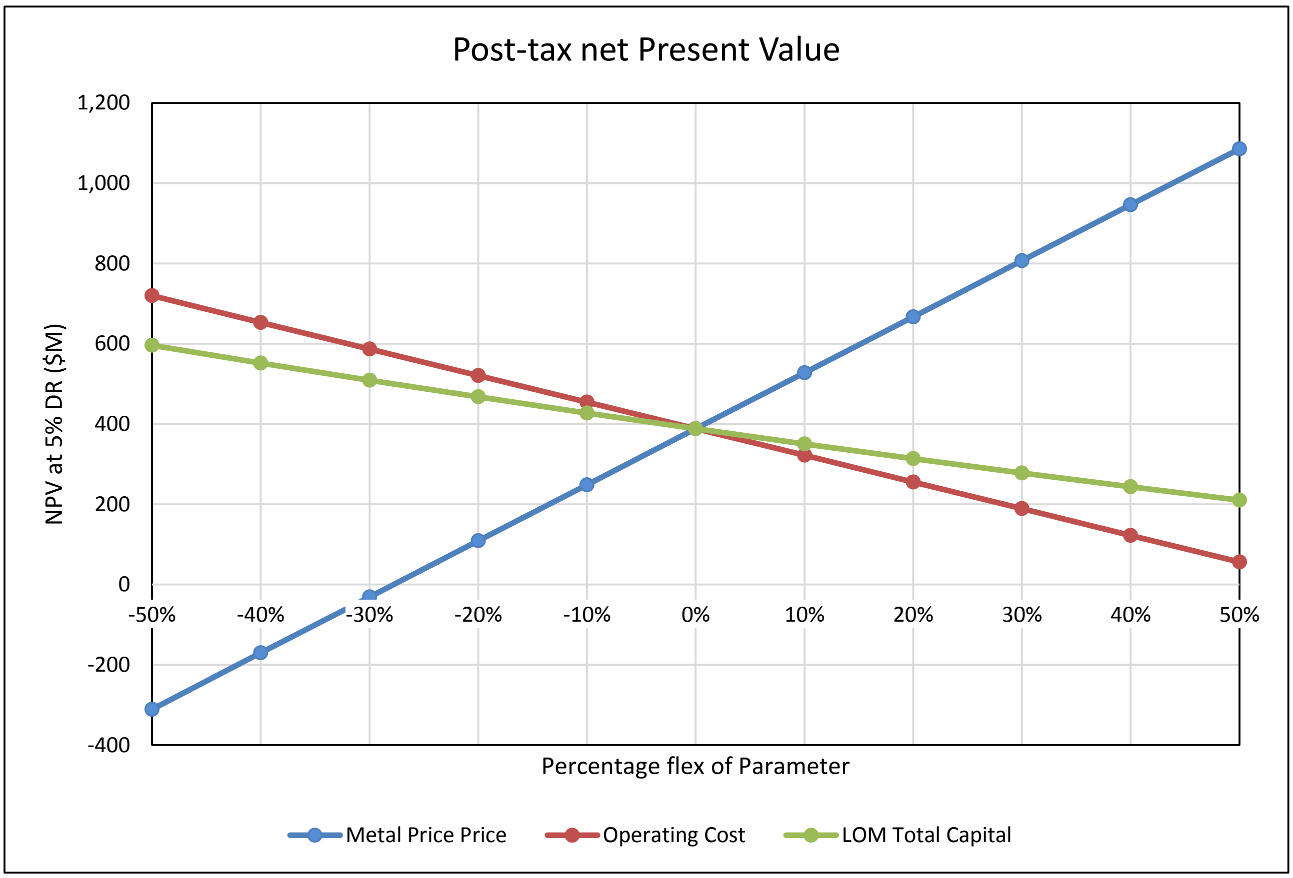


Figure 21.1: Sensitivity Analysis

As a result of sensitivity analysis, to achieve a breakeven NPV—assuming there is no change in the current operating and capital cost assumptions—the price of gold would have to drop by approximately 30% to 916 $/oz or the price of silver to 17.6 $/oz.

21.12 Conclusions and Recommendations

21.12.1 Important Notice

The Economic Analysis that forms part of this preliminary economic assessment (PEA) report is intended to provide an initial review of the Gold Canyon Resources Inc. Springpole Gold Project's potential and is preliminary in nature. The economic analysis incuded in this PEA includes consideration of inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA based on these mineral resources will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

21.12.2 Conclusions and Recommendations

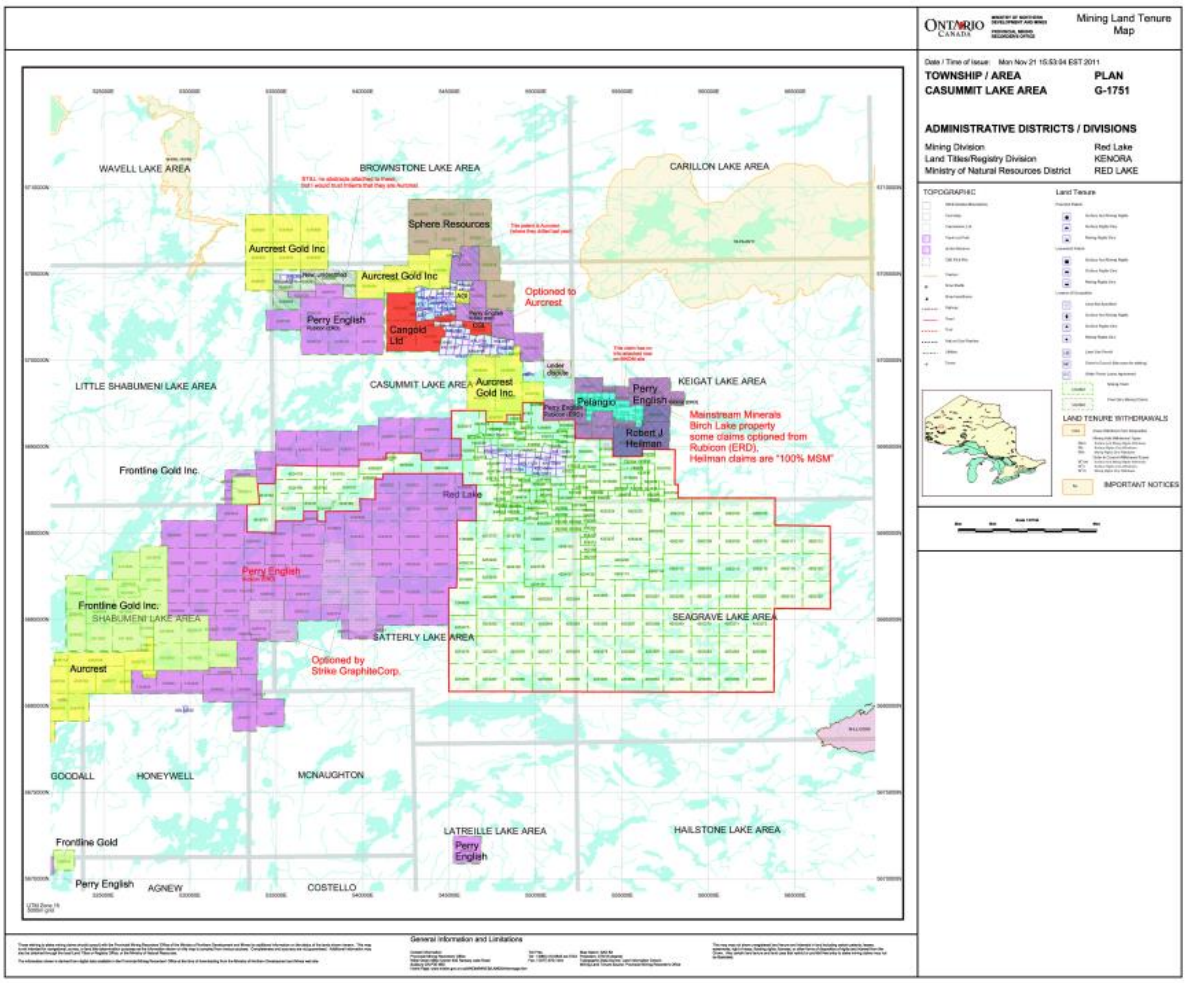
The economic analysis of the PEA indicates that a conventional open-pit mining and milling operation has reasonable prospect of being an economic exploitation strategy. On the basis of the assumptions used in the this PEA, the a base case post-tax $NPV_{5\%}$ of $388M (pre-tax NPV: $579M) and an IRR of 14% (pre-tax IRR: 25%) on an initial investment of $438M is estimated.

The need for lake drainage is a significant cost to the project and is subject to a higher level of uncertainty in terms of cost and schedule. Nevertheless, sensitivity analysis inidcates that, at base case metal price assumptions, the project may be able to absorb significant escalation in capital and operating costs and potentially remain a viable economic proposition.

22 Adjacent Properties

SRK has not done the necessary work to verify the information presented in this section of the report. The information presented in this section of the report is not necessarily indicative of the mineralization on the Springpole property.

The largest adjacent property holder is Perry English with a large property position to the west of the Springpole Gold Project claim block and surrounding Gold Canyon's Horseshoe Island claim block (Figure 22.1). Perry English has an agreement with Rubicon Minerals Corporation (Rubicon) in which the claims held are developed thru the English Royalty Division (ERD) of Rubicon. The ERD has a program of acquiring mineral properties and then optioning them to mineral exploration companies. The claims are listed as being owned by Perry English and the MNDM records do not show the details of the joint venture agreements. Information contained here was obtained from Rubicon's website (www.rubiconminerals.com/projects/English-Royalty-Division).



Source: Gold Canyon 2011

Figure 22.1: Springpole Gold Project—Location of Adjacent Properties

Strike Graphite Corporation has optioned a claim package from the ERD to the east of the Springpole Gold Project totalling 1,600 ha. The prospects have a history of gold exploration dominated by trenching and drilling in the 1980s. Mineralization is hosted within sedimentary rock packages associated with felsic and intermediate intrusive. The website describes gold mineralization hosted by various sulphide minerals including pyrite, pyrrhotite and arsenopyrite. Information contained here was obtained from Strike's website (www.strikegraphite.com/satterly-lake.html).

Mainstream Minerals Corporation has a package of 13 claims covering 2,080 ha staked on the east shore of Birch Lake. Historic exploration work has shown significant gold showings hosted within banded iron formations. There is no indication of any recent exploration work on the prospect. Information contained here was obtained from Mainstream's website (www.mainstreamminerals.com/properties/birch-lake).

Pelangio Exploration Inc. holds a claim package consisting of twenty eight contiguous unpatented claims totalling 453 ha and covers a series of small islands in Birch Lake. The Birch Lake property is subject to an option agreement with Trade Winds Ventures Inc. Trade Winds last completed exploration on the property in 2004 and 2005. This comprised drilling seven diamond drill holes that intersected gold mineralization in several drill holes including 115.89 g/t gold over 2.90 m in drill hole TWBL-096. Information contained here was obtained from Pelangio's website

(www.pelangio.com/Projects/Canada).

AurCrest Gold Inc. holds 31 claims immediately north of the Springpole claim block called the Richardson Lake prospect. The prospect comprises four separate claims blocks totalling 5,876 ha including an option from Rubicon's ERD. At the time of writing this report AurCrest had released results from five diamond drill holes for a total of 802 m from its winter 2011/2012 drill program. Highlights include 3 m averaging 3.96 g/t gold including 0.5 m of 7.88 g/t gold from drill hole RL12-03. Information contained here was obtained from AurCrest's website (www.aurcrestgold.com).

Cangold Limited holds claims that surround the Argosy Gold Mine and comprise forty-four patented and fifty-seven unpatented claims totalling 1,616 ha. The last exploration work on the property consisted of nine diamond drill holes totalling 1,814.8 m of drilling in 2004. Highlights of the drilling include mineralized intercepts of 52.73 g/t gold over 0.3 m from drill hole AM04-01 which tested the No. 5 vein. Cangold continues to be involved with the prospect and believes that the Argosy Mine hosts the potential for up to 1M ounces of gold. Information contained here was obtained from Cangold's website (www.cangold.ca/s/Argosy.asp).

Sphere Resources Inc. has eight unpatented claims north of the Springpole Gold Project. No mention is made of any exploration activity in respect of these claims on the company website. Information contained here was obtained from Sphere's website (www.sphereresources.com).

23 Other Relevant Data and Information

There are no other relevant data available about the Springpole Gold Project that has not been included or discussed in this report.

24 Interpretation and Conclusions

Industry standard mining, process design, construction methods, and economic evaluation practices were used to assess the Springpole Gold Project. Based on current knowledge and assumptions, the results of this study indicate that the project may have positive economics within the very preliminary parameters of a PEA. and should be advanced to the next level of study - either preliminary feasibility or feasibility.

The preliminary economic assessment is preliminary in nature; it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

While a significant amount of information is still required to do a complete assessment, at this point there do not appear to be any fatal flaws for the project. The study achieved its original objective of providing a preliminary review of the potential economic viability of the Springpole Gold Project.

24.1 Geology

The quality assurance/quality control program instituted by Gold Canyon and conducted by SGS is of a standard generally consistent with current industry practice. SRK acknowledges that the QA/QC procedures have evolved rather recently and much of what is presented above is “catch up” work. In that respect Gold Canyon have done well to bring the database, at least from 2007 onward up to an acceptable industry standard. The principal exceptions lie with:

- Lack of documentation on QA/QC procedures for drilling prior to 2003;
- Blank analyses suggest intermittent contamination introduced at some stage of material storage or processing; and
- The lack of standard reference materials for silver.

The analysis for gold and silver confirms an acceptable degree of reproducibility of samples for gold and a very good reproducibility for silver.

There is no evidence of bias in either gold or silver as a function of grade but the company needs to implement written QA/QC procedures for deciding which assay batches are acceptable or not and which samples need to be re-assayed because of failed QA.

The drill hole database from 2003 through 2012 is of a standard acceptable for public reporting of resources according to NI 43-101 guidelines.

24.2 Mineral Processing and Metallurgy

The investigations to date on Springpole mineralized material have allowed the following conclusions to be reached:

- The presence of coarse gold in higher grade portions of the East Extension and Camp zones suggests that gravity concentration should be included in the comminution circuit.

- Cyanidation of a finely milled product looks promising.
- Flotation is an option that requires further investigation.
- Heap leaching does not appear attractive.

To further develop the likely process routes, further mineralogical, and metallurgical investigations are recommended.

This PEA is based on a design plant capacity of 20,000 t/d.

Based on the testwork results to date, a recommended flowsheet of whole feed leaching after grinding to a P80 size of 70 µm is proposed. A gravity circuit can be included for higher grade feed material. Cyanide leach extractions of 80% for gold and 85% for silver are expected at this grind size. Product from the process plant will be doré bullion.

24.2.1 Risks

While considerable metallurgical testwork has been completed to date in three programs on different samples, additional testwork is warranted to better define the plant design criteria and more confidently predict expected performance.

To date, a relatively small number of composite samples have been used for flowsheet development. Additional samples should be collected for variability testing. These samples should each come from a small area of the deposit (e.g., a single drill hole interval or series of intervals from nearby holes). Each sample should be tested on the expected flowsheet to quantify variability in performance for different parts of the deposit. These samples also should be hardness tested using a number of methods to determine their resistance to SAG milling, ball milling and possibly fine grinding.

Additional testwork needs to be done to confirm cyanide detoxification can be completed successfully and within normal reagent cost levels. Thickening and filtering characteristics should be confirmed so that unexpected dewatering costs are not needed. For whole feed leaching, the plant tailings will likely be acid generating and the associated costs with treatment/handling of this material need to be estimated.

Future testwork can consider concentrating the sulphides via flotation or classification into a smaller mass so that it can be stored separately from the remainder of the tailings. For example, dry stack or subaqueous deposition in the tailings pond to minimise the potential for acid generation.

24.2.2 Opportunities

The testwork conducted to date has identified a number of options that should be considered in trade-off studies; namely the effect of primary grind size on whole feed leach extraction. With a finer grind, it appears that gold recoveries up to 90% are achievable, but at the expense of higher grinding power, media wear and in particular, use of specialised grinding equipment for target P80 grind sizes below 40 µm. In addition, with a whole feed leaching flowsheet, grinding all the feed finer will lead to issues of detoxifying and handling the very fine leach residues.

The option of concentrating the gold and sulphides with flotation should continue to be investigated in future testwork. However, it is recommended that a more thorough mineralogical investigation of the flotation concentrate be conducted prior to any leach testing. The grain size and association of gold particles should reveal why the flotation concentrate is somewhat resistant to cyanidation at the same grind sizes. Such information was not possible through the diagnostic leach tests previously conducted.

24.3 Mineral Resource Estimate

Review of the pre-2003 data lead to some drill hole data for Portage zone, East Extension, and Camp zones being excluded from the mineral resource estimate. A systematic re-sampling of the available drill core stored on-site at the Springpole Gold Project would enable the reclassification of the East Extension zone into indicated resource category without the need to carry out an additional, extensive drilling campaign. This re-sampling exercise would involve ground survey of drill collar locations in respect of historic records as well as the inclusion of a systematic program of certified blanks, certified gold and silver standards, and field and pulp duplicates where sufficient drill core remains.

The same is not true of pre-2003 drill cores from Portage zone—the inability to accurately verify the original drill collar locations by any means in respect of surveyed UTM or mining grid locations combined with the paucity of material due to initially very low core recovery excludes this data set for resource estimation.

The current mineral resources for the Springpole Gold Project prepared by SRK consider 512 core boreholes drilled by Gold Canyon and previous owners of the property during the period of 2003 to 2012. The resource estimation work was completed by Dr. Gilles Arseneau, PGeo (APEGBC #23474), an appropriate independent qualified person as this term is defined in NI 43-101. The effective date of the resource statement is October 17, 2012.

In the opinion of SRK, the resource evaluation reported herein is a reasonable representation of the global gold and silver resources found in the Springpole Gold Project at the current level of sampling. The mineral resources have been estimated in conformity with generally accepted CIM Estimation of Mineral Resource and Mineral Reserves Best Practices guidelines and are reported in accordance with the Canadian Securities Administrators' NI 43-101 and at a 0.4 g/t gold cut-off include 128.2 Mt grading 1.07 g/t gold classified as indicated mineral resource and 25.7 Mt grading 0.83 g/t gold classified as inferred mineral resource.

The revised mineral resource estimate (October 17, 2012) was based on a gold price of $1,400/oz and a silver price of $15/oz. These are considered reasonable economic assumptions by SRK. To establish a reasonable prospect of economic extraction in an open pit context, the resources were defined within an optimized pit shell with pit walls set at 45° . Assumed processing recoveries of 80% for gold and 60% for silver were used. Mining costs were estimated at $2/t of total material, processing costs estimated at $12/t and G&A costs estimated at $2/t. A COG of 0.4 g/t gold was calculated and is considered to be an economically reasonable value corresponding with breakeven mining costs. Approximately 90% of the revenue for the proposed project is derived from gold and 10% from silver.

Note: *For the mine development (Whittle optimization) and economic analysis in this PEA, updated input parameters were used.*

Industry standard mining, process design, construction methods and economic evaluation practices were used to assess the Springpole Gold Project. In SRK's opinion, there is adequate geological and other pertinent data available to generate a PEA.

Based on current knowledge and assumptions, the results of this study show that the project has the potential for positive economics (within the very preliminary parameters of a PEA) and should be advanced to the next level of study; a preliminary feasibility study.

As with almost all mining ventures, there are a large number of risks and opportunities that can influence the outcome of the Springpole Gold Project. Most of these risks and opportunities are based on a lack of scientific information (test results, drill results, etc.) or the lack of control over external drivers (metal price, exchange rates, etc.). The following section identifies the most significant potential risks and opportunities currently identified for the project, almost all of which are common to mining projects at this early stage of project development.

Subsequent higher-level engineering studies would need to further refine these risks and opportunities, identify new ones, and define mitigation or opportunity implementation plans.

While a significant amount of information is still required to do a complete assessment, at this point there do not appear to be any fatal flaws for the project.

The study achieved its original objective of providing a preliminary review of the potential economic viability of the Springpole Gold Project.

24.4 Mining

It is proposed that the Springpole deposit is amenable to be developed as an open pit mine. Mining of the deposit is planned to produce a total of 72 Mt of processing plant feed and 121 Mt of waste (1.7:1 overall strip ratio) over an eleven year mine production life.

24.4.1 Risks

The current understanding of the project is based upon limited and time sensitive information. Changes in the understanding of the project, the ability to convert resources to reserves and market conditions could affect the project's economic viability.

24.4.2 Opportunities

With additional data, the project should be subjected to a series of strategic option reviews aimed at determining the most valuable strategy for exploiting the resource including the LOM schedule. Planning and executing the project at the correct scale, with the optimum mine design and processing systems will result in the maximum possible value to shareholders and other economic stakeholders.

Pit optimization work has identified larger pit shells which are economic with current input parameters with the Whittle software identified a number of potential pit shells (or phases) and the selected pit shell provides higher grades, lower strip ratio, and reduced capital and operating expense.

If conditions change such that a larger pit shell is selected to recover more resources, infrastructure such as the proposed cofferdam should not be impacted. Consideration in the location and design of the cofferdam took this possibility into consideration.

24.5 Geotechnical

There is possibly an up-side improvement in slope design-angles if a significant increase in the confidence of the geotechnical data (and model) can be achieved. What may ultimately control achievable slope angles (apart from hydrogeological constraints) is the *Weak* to *Intermediate*-domain spatial arrangement, and anisotropy in the host rock in the *Strong*-domain. To achieve a pre-feasibility level of confidence for the slope design input parameters:

- Design and implement an oriented-core geotechnical drill program to log the rock mass and acquire intact rock and joint samples.
- Do laboratory and field tests to adequately characterize the intact rock strength and joint properties of the main rock types.
- Map and describe all major faults, as viewed in drill core and rock outcrop within 200 m of the pit crest and integrate them with the regional structural interpretation.
- Produce robust 3-D digital wireframe models of lithology, alteration with intensity, and structures.

Characterize the rock mass using an appropriate rock mass rating system (for example RMR_{89}), and map the geotechnical domains within a 3-D model.

24.6 Hydrogeology

Hydrogeological data are limited for the Springpole site. Based on experience from other northern mining operations in Canada, the following conclusions have been developed for the Springpole Gold Project:

- Management of groundwater within the pit may be required through conventional methods such as dewatering wells and sumps to reduce mine trafficability issues, although this should be within reasonable mining costs.
- Elevated pore pressures may develop during excavation of the pit in both overburden and bedrock. Without adequate hydrogeological data collection, the assumption for the PEA is that water and potential excess pressures can be managed and will not impact the slope design.
- Groundwater quality is unknown. Groundwater will likely be used in the mine water supply and excess is typically sent to the mill pond if its quality is below groundwater discharge guideline limits.

Recommendations for pre-feasibility level studies are provided in the following section.

24.6.1 Risks

In general, hydrogeological uncertainties for the Springpole property include:

- The mine will be located within the existing (dewatered) lake footprint. The hydrogeology of the project area has not been fully characterised with respect to groundwater flow directions, transmissive features, and the potential for hydraulic connection to the surrounding lake. Seepage may be anticipated through and beneath the coffer dams.
- Limited understanding of the geological and structural models will result in low confidence of the distribution of hydraulic conductivity within—and in the vicinity of—the proposed pit, resulting in uncertainty relating to the magnitude of possible groundwater inflows to the pit.
- No information on baseline groundwater quality.

These uncertainties correspond to the following potential risks:

- **Higher than anticipated groundwater flow.** Resulting from highly transmissive features such as structures or highly permeable horizons, or weathered/altered zones, there is a risk of higher than anticipated inflows to the pit, resulting in high pumping requirements / management costs. Geological and structural models to be developed further to increase confidence in the conceptual hydrogeological model. Seepage through coffer dams has not been assessed and may result in unanticipated seepage rates leading to high water management/ treatment costs.
- **Elevated pore pressures.** In low hydraulic conductivity bedrock, drainage of groundwater may not be able to keep up with the excavation of the pit, resulting in a buildup of pore pressures in the pit walls that may lead to geotechnical instability if not accounted for or mitigated. Hydrogeological investigations of the lithological units will identify areas within the vicinity of the proposed pits that may require management, with respect to depressurisation.
- **High inflow rates and groundwater compartmentalization.** Uncertainty in the structural model may result in potentially high hydraulic conductivity zones (faults) in connection with lakes that may result in significantly high pit inflows. If not anticipated, such features can cause instability in lower slopes or lead to problematic inflows that require management. Low permeability structural features (relative to the surrounding bedrock) can result in compartmentalization of remnant pressures (in overburden and bedrock) within the excavated slopes that may also create unstable conditions. Structures on a concession scale require study to understand the degree of groundwater flow anisotropy.
- **Trafficability issues.** The mineralized zone consists of highly altered rock that may require focussed water management in high seepage areas to avoid poor working conditions.
- **Groundwater may have to be treated prior to discharge.** Quality of groundwater into the open pit may not be suitable for discharge without treatment. Water treatment may be required.

24.7 Hydrology

Stream flows, annual average precipitation and evaporation were estimated for the Springpole site based on nearby gauging and weather stations. This information was used in the preliminary design of the surface water management.

24.7.1 Risks

No borrow source investigations have been performed therefore it is recommended that an investigation be performed to confirm the availability of suitable construction materials within the assumed haul distances. If borrow material is not available that could increase the capital costs associated with surface water management.

24.7.2 Opportunities

Additional geochemical testing of waste rock may indicate that lined pollution control ponds are not required; this could reduce the water management infrastructure costs.

24.8 Tailings Management Facility

The TMF is designed to contain centrally discharged paste tailings, and soft lakebed sediments from lake dewatering. The paste tailings will slope from the center at 4% and be contained by a ring dam. Due to the requirement to store soft lake bed sediments the dam will be constructed to final height before the start of mining.

24.8.1 Risks

The following should be considered as potential risks to the project outcome given the current state of understanding:

- It has not been confirmed whether lining of the TMF would be required given subsurface conditions, tailings geochemistry or environmental regulations. Should lining be required, it would result in an increase in both initial and sustaining capital.
- No geotechnical or hydrogeological investigations have been carried out at the TMF; therefore, it cannot be confirmed whether the allowances for foundation seepage control are adequate. Complex foundation conditions would result in increased CAPEX.
- No borrow source investigations have been carried out to confirm availability of suitable construction materials within the assumed haul distances. Should the assumptions prove to not be valid, CAPEX and sustaining capital will increase.
- No physical characterization of the tailings has been done to confirm whether the material would be amenable to thickened tailings as proposed, specifically attaining of a 4% beach angle. Should this not be possible, the TMF design concept would not be viable and the costs as presented would increase due to increased containment requirements.
- Several small ponds exist within the tailings footprint. These may result in permitting challenges.

24.8.2 Opportunities

Potential opportunities that have been identified at this time include:

- Borrow source investigations may reveal that sufficient quantities of low permeability material for core construction may be available on-site. This would result in a reduced CAPEX.

- Geotechnical investigations may indicate that bedrock is located at a shallower depth than assumed in the cost estimate. This would result in a reduced CAPEX.
- Alternative sediment storage options outside the TMF should be evaluated to conserve capacity and to make the sediment available for rehabilitation at closure.

24.9 Dewatering Dikes

Three dewatering dikes will be required to allow for the dewatering of a portion of Springpole lake to allow for open pit mining. These dikes will be constructed under wet conditions with a rockfill shell and concrete slurry cut-off wall. Dewatering of the lake is projected to take 1 year assuming continuous pumping at a rate of 0.6 m^3/s.

24.9.1 Risks

Potential risks associated with the preliminary dewatering dike designs include:

- Limited geotechnical and no hydrogeological investigations have been carried out at the dewatering dike foundation; therefore, it cannot be confirmed whether the allowances for foundation seepage control are adequate. Complex foundation conditions would result in increased CAPEX.
- The assessment of the downstream effect of discharging 0.6 m^3/s of water during lake dewatering into Springpole Lake was a very high level assessment. A more detailed assessment may indicate that a lower discharge rate, and consequently a longer discharge period, may be required to dewater the lake.

24.9.2 Opportunities

Potential opportunities that have been identified at this time include:

- Lake dewatering could occur at a faster rate if the water was discharged into several different lakes. Faster dewatering could improve the overall project economics.
- A sheet pile cut-off wall through the dewatering dike may reduce CAPEX; however, it could result in additional pumping costs. This should be assessed in more detail in future work.
- Geotechnical investigations may indicate that bedrock is located at a shallower depth than assumed in the preliminary design and cost estimate. This would result in a reduced CAPEX.

24.10 Project Infrastructure

The two lane Access Corridor Road and all single lane access roads will be constructed using conventional cut and fill techniques prior to placing of an approximately 0.5 m thick compacted sub-base layer sourced from locally developed and approved borrow sources. Routine surface water management along all roads will be achieved by ensuring the roads are graded with a crown. Eleven locations along the Access Corridor Road will have corrugate steel culverts installed to allow surface water to pass while no culverts have been identified for the single lane access roads.

Two major stream crossings will be required along the Access Corridor Road. An arched culvert will be constructed at the Deaddog Stream Crossing while a pre-fabricated bridge will be constructed at the Birch River Crossing.

Surface infrastructure earthworks will use also conventional cut and fill techniques to provide suitably graded areas to place the buildings and allow for surface drainage. The buildings will be of modular design or consist of fully contained prefabricated components. These structures will require minimal on-site construction, plumbing, and electrical work.

The Fuel Tank Farm should be located on a blasted bedrock foundation. Compacted engineered backfill will be used to bring up to the appropriate grades and provide suitable bedding material for the lined containment facility as well as be used for pedestal supports for the fuel tanks.

24.10.1 Risks

The following should be considered as potential risks to the project outcome given the current state of understanding:

- No geotechnical or hydrogeological investigations have been carried out along the road alignments or within the surface infrastructure footprint; therefore, it cannot be confirmed whether complex foundation conditions are required which would result in increased CAPEX.
- No borrow source investigations have been carried out to confirm availability of suitable construction materials within the assumed haul distances. Should the assumptions prove to not be valid, CAPEX will increase.
- The surface infrastructure component sizes have not been finalized. Increasing building sizes or the number of buildings will increase CAPEX costs.

24.10.2 Opportunities

Potential opportunities that have been identified at this time include:

- Given the location and climate, it is possible that road maintenance may be less than currently allowed for in the cost estimate.

24.11 Environmental Studies and Permitting

The potential impacts the project may have on Springpole and/or Birch Lake are considered to be the more environmentally and socially sensitive components of the project. Gold Canyon is cognizant of these sensitivities and has taken steps to design the project with these sensitivities in mind. To that end, the project is designed to avoid direct interaction with the Birch Lake watershed, and all baseline studies carried out to date are structured to identify areas of risk so they can be protected to minimize impact during the development of the project or totally avoided.

The development of the open pit into Springpole Lake will require the isolation of a portion of the lake throughout the 11 year mine life of the project and for a number of years following depletion of all mineralized material before the coffer dams can be breached or removed. Fish habitat compensation will be required as a result of this advancement into the lake. However, following decommissioning and closure of the mine the coffer dams would be breached and the pit would be reunited to the lake. The potential exists for Gold Canyon to incorporate significant aquatic habitat enhancement to Springpole Lake through the re-introduction of the pit area to the lake proper.

The management of the mine waste (tailings and waste rock) also represents a longer term environmental concern. The tailings management facility and waste rock repository will likely assimilate fish bearing ponds and doing so will likely involve additional fish habitat compensation as well as an amendment to Schedule II of the metal mining effluent regulations. The next phase of engineering for the project will further evaluate alternative mine waste management areas to avoid impacting water bodies and, therefore, the need for a Schedule II amendment. Tailings and waste rock management are covered in more detail under separate sections of this study. The environmental risks associated with these facilities following operations will be addressed as part of the project's detailed closure plan.

All potential environmental impacts associated with the project can be mitigated through the implementation of accepted engineering practices currently employed throughout Canada's mining industry. A detailed monitoring plan will also be developed to ensure environmental compliance of all components of the mine throughout its construction, operation, closure, and post-closure activities.

24.12 Economic Analysis

The Economic Analysis that forms part of this preliminary economic assessment (PEA) report is intended to provide an initial review of the Gold Canyon Resources Inc. Springpole Gold Project's potential and is preliminary in nature. The economic analysis incuded in this PEA includes consideration of inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA based on these mineral resources will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The base case economic analysis results indicate a post-tax NPV of $388M at a 5% discount rate (pre-tax NPV: $579M) with an IRR of 14% (pre-tax IRR: 25%). Payback (post-tax, non-discounted) will be in Year 3 of production in a projected 11 year LOM production period. The economics are based on a base case of $1,300/oz long-term gold price, $25/oz long-term silver price, and production rate of 20,000 TPD over 365 d/yr. Direct operating costs are estimated at $636/oz of AuEq. Total capital costs are estimated at $544M, consisting of initial capital costs of $438M, and ongoing sustaining capital of $106M.

Sensitivity analysis inidcates that, at base case metal price assumptions, the project can absorb significant escalation in capital and operating costs and remain a potentially economic proposition.

24.13 Summary of Risks and Opportunities

24.13.1 Project Risks

As with almost all mining ventures, there are a large number of risks and opportunities that can affect the outcome of the project. Most of these risks and opportunities are based on uncertainty, such as lack of scientific information (test results, drill results, etc.) or the lack of control over external factors (metal price, exchange rates, etc.).

Subsequent higher-level engineering studies would be required to further refine these risks and opportunities, identify new risks and opportunities, and define strategies for risk mitigation or opportunity implementation.

The principal risks identified for the Springpole Gold Project are summarized as follows:

- Geological interpretation and mineral resource classification (10% of the resources used in the mine plan are Inferred);
- Due to a relatively small number of metallurgical samples tested, larger variations in mineralogy and metal recovery may exist than have been observed to date;
- Geotechnical and hydrogeological considerations;
- No information on baseline groundwater quality;
- No physical characterization of the tailings material has been done;
- Construction management and cost containment during development of the project;
- The permitting period associated with the project could be significantly longer than assumed in this study;
- Increased OPEX and/or CAPEX; and
- Reduced metal prices.

24.13.2 Project Opportunities

The following opportunities may improve the project economics:

- Metallurgical testwork has indicated that gold recoveries up to 90% are possible with a finer grind. Trade-off studies should be carried out to determine whether it's feasible to incorporate a finer grind process into the flowsheet;
- Pit optimization work with the Whittle software identified a number of larger potential pit shells (or phases) and the selected pit shell provides higher grades, lower strip ratio, and reduced capital and operating expense;
- Recently completed geotechnical drilling for pit slope stability analysis may increase pit slopes angles over those used in this PEA;
- There are other geophysical targets around the current resource, particularly to the southwest of the current resource. Additional drilling has the potential to add resources;
- Investigations may reveal that sufficient quantities of low permeability material for core construction may be available on-site and bedrock may be located at a shallower depth than assumed in the cost estimate;
- Lake dewatering could occur at a faster rate if the water was discharged into several different lakes;
- The potential to upgrade the mineral resource classification of the deposit; and
- Improved metal prices.

25 Recommendations

SRK recommends the following next phase work program for the Springpole Gold Project.

25.1 Quality Assurance and Quality Control Program

SRK recommends a remedial program of re-sampling of the core for the pre-2007 drilling with focus on the mineralized intervals, to replace the missing field and pulp duplicate information and including appropriate insertion of blanks and standards that would demonstrate compliance with current NI 43-101 standards. The drill hole density in these areas is more than adequate for generating resource categories above inferred; all that is missing is adequate demonstration of reproducibility of results.

SRK recommends that a silver standard be introduced as a regular routine with all new assay batches sent to the laboratory for analysis. SRK recommends that Gold Canyon consider re-assaying some of the available pulps with a silver standard to assure the robustness of the silver data in the Springpole database.

SRK recommends that Gold Canyon implements a written protocol for QA/QC data review so that quick action can be taken if sample batches fall outside of the acceptable QA/QC acceptance guidelines.

25.2 Mineral Resource

SRK recommends dedicated program of s.g. measurement on core sufficient to establish volumetrically representative values for s.g.

25.3 Resource Development Program

SRK recommends a work program for the Springpole Gold Project including incremental step-out and infill drilling adjacent to the Portage zone, especially to the southeast and southwest to expand the resource and better define the extent of mineralization. This drilling will use the established drill section spacing of 50 m with infill between sections where deemed necessary. Assuming a total of 38 holes with an average hole length of 400 m, this comes to 15,200 m. This drilling could be accomplished within a 12-month period. Drilling can be undertaken from the ice during the winter and utilizing Gold Canyon's four drill barges during spring, summer and fall. In addition to drilling in and around the Portage zone, an additional 5,000 m of drilling should be allocated to testing new exploration targets, especially ones proximal to the existing deposits.

The recommended work program is expected to cost approximately $11.5M as outlined in Table 25.1.

Table 25.1: Proposed Budget for the Resource Development Program

Recommendation	Estimated Cost
Drilling 20,200 m including materials and fuel	$4,646,000
Assays	$323,000
Bulk Density Sampling and Analysis	$100,000
Operating 65-man Camp	$1,520,000
Transportation	$2,050,000
Equipment Rentals/Leases	$480,000
Salaries	$1,050,000
Contract Services	$890,000
Winter Road	N/A
Sustainability Management	$140,000
Environmental/Permitting	$280,000
Total Budget	**$11,479,000**

25.4 Metallurgical Testwork Program

Additional testwork needs to be done to confirm cyanide detoxification can be completed successfully and within normal reagent cost levels. Thickening and filtering characteristics should be confirmed so that unexpected dewatering costs are not needed. For whole feed leaching, the plant tailings will likely be acid generating and the associated costs with treatment/handling of this material need to be estimated.

Future testwork can consider concentrating the sulphides via flotation or classification into a smaller mass so that it can be stored separately from the remainder of the tailings. For example, dry stack or subaqueous deposition in the tailings pond to minimise the potential for acid generation.

A well-defined metallurgical testwork program can address these items and determine if there are any issues that may impact the overall economics. It is expected that such a program will cost between $250,000 and $500,000, depending on the number of samples included.

25.5 Hydrogeological Characterization

The following site investigation recommendations are standard scope items for a pre-feasibility study:

- Hydraulic testing is recommended in the area between the lakes and the proposed pit, with the main objective to identify and focus testwork on larger scale geological features with the potential to act as hydraulically transmissive conduits for groundwater inflows. This work should be done in conjunction with a concession scale structural assessment and any geophysical surveys that are available.

- Secondary objectives would be to characterise the hydrogeological properties of the bedrock, which will be used to constrain estimates of groundwater inflow. This work should continue the PFS data collection for the geotechnical program, which consisted of packer testing within the pit area. Drill holes for testing should be located during site investigation programs for the TMF and waste rock dumps. The testing program should consider:
  - Short-duration tests within all geotechnical drill holes (e.g., packer testing).
  - Short-duration tests within at least two hydrogeological drill holes targeting areas outside of open pit area.
  - Contingency should be included for longer-term testing (e.g., airlift testing) with observation wells if preliminary analyses indicate that there are areas with relatively high permeability.

Final design of this program will not be possible until the structural review is completed.

SRK estimates the cost of this hydrogeological characterization program to be $60,000 assuming that a drilling rig is on site and a five hole program will take six weeks to complete. This cost does not include long-term test pumping.

- **Groundwater monitoring.** Groundwater monitoring wells to be sited and installed with well screens to isolate key aquifer horizons. The construction of the wells should take into consideration any overburden and geology that is encountered. Multi-depth wells should be considered in key areas to monitor water levels in significant overburden and bedrock lithology. Locations of the wells should be up-gradient and down-gradient of any mine infrastructure. Wells should be constructed to best practise guidelines to ensure that sampling, hydraulic testing and groundwater measurements are representative of the zone they are in. A water level monitoring database should be established with regular measurements taken on a weekly basis. This can either be done manually with a water level meter, or automatically by installing pressure transducers with integrated dataloggers into selected boreholes. A combination of these methods is recommended to ensure calibration against logger readings.
- **Groundwater quality** Groundwater quality should be characterized to establish baseline conditions using the monitoring wells described in the preceding paragraph, and should consider shallow (within overburden) and deep (base of pit) groundwater regimes. This work is typically undertaken as part of the environmental baseline study. Water quality data will be used in the load balance, and in cost estimates for potential treatment options. This work should be initiated as soon as possible to ensure seasonal groundwater data are captured.

A site water balance is recommended to be developed at an early stage so that all aspects of groundwater, surface water and process water around the project site are quantified.

25.6 Hydrological Monitoring

It is recommended that the following be installed at the Springpole site to obtain a better understanding of the site hydrology and to help define the specific site water management requirements in the future phases of the project:

- Gauging stations at the outflows from Springpole Lake
- Meteorological station

This monitoring will also be required for baseline studies to support the environmental assessment. SRK's estimate for the cost of these two automated level sensors is approximately $100,000.

25.7 Tailings Management Facility

It is recommended that the following studies be undertaken to confirm the tailings management plan design:

- Complete a detailed geotechnical/geohydrological investigation of the tailings management facility including boreholes and test pits. Packer testing, and in situ characterization is required, as well as sampling and submitting of samples to geotechnical testing laboratories for engineering property classification. It is estimated that the cost of this program is likely to be $200,000 to $300,000.
- Carry out detailed physical and rheological testing on representative tailings samples to confirm the thickening characteristics of the tailings and the ultimate beach angle. The cost of this program is estimated to be $30,000 to $60,000.
- Carry out a preliminary borrow source characterization study to identify candidate construction materials (including for the dewatering dike and main access road route). This should include an initial air photo interpretation, followed up by preliminary ground reconnaissance. This study is estimated to cost between $50,000 and $100,000.

25.8 Dewatering Dike

A comprehensive geotechnical investigation must be carried out at the proposed dike locations to confirm foundation conditions. The investigation method must be capable of determining the in-situ geotechnical properties of the lake bed sediments and other overburden. Samples must be retrieved for laboratory testing. The cost of this program is estimated between $180,000 and $280,000.

25.9 Infrastructure

A reconnaissance survey of the road and power line routes need to be carried out to confirm the general ground conditions at these sites. The cost of this is estimated to be $10,000 to $25,000 depending on how much helicopter support would be required. In addition, a geotechnical investigation needs to be carried out at the two major river crossing sites to confirm foundation conditions. The cost of this investigation is estimated to be $75,000 to $150,000.

25.10 Environmental Studies

In support of the exploration activities and to support future EA processes and permits, Gold Canyon has initiated a variety of environmental studies to collect environmental data to characterize both the physical and biological environments. The studies include the following areas: meteorology, air quality, noise, hydrology, hydrogeology, geochemistry, terrestrial resources, fisheries resources, socio-economic, archaeology, and sediment, benthos and surface water quality.

25.11 Economic Evalutation Recommendations

For future studies, as the input data become more precise and accurate, SRK recommends that a correct practice of modelling tax depreciation, working capital, VAT and any cash flows involving the carry forward or carry back of balances is undertaken in nominal terms using a reasonable forecast of currency inflation.

25.12 Estimated Total Cost of Recommendations

The total cost for all SRK recommendations is shown in Table 25.2 where upper limit estimates have been used.

Table 25.2: Estimated Cost for Recommended Work

Recommendation	Estimated Cost (Upper Limit)
Resource Development	$11,479,000
Metallurgical Testwork	$500,000
Hydrological Characterization	$60,000
Hydrological Monitoring	$100,000
Tailings Geotechnical Investigation	$300,000
Tailings Physical Testing	$60,000
Tailings Borrow Source Characterization	$100,000
Dewatering Dike Geotechnical Investigation	$280,000
Infrastructure Reconnaissance Survey	$25,000
Infrastructure River Crossing Geotechnical	$150,000
Total Budget	**$13,054,000**

25.13 Closure

SRK believes the project should be taken to the next level of engineering study and economic assessment, typically a pre-feasibility study. It is estimated that a pre-feasibility, along with all of the accompanying engineering work would cost approximately $3.5M (exclusive of the recommendations listed above and the additional geology and drilling program required). Some of the activities involved to advance the project include:

- Initiate project permitting;
- Consummate agreements with First Nations groups; and
- Convert remaining inferred resources to indicated resources.

26 Acronyms and Abbreviations

Acronyms	
AP	Acid potential
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
COG	cut-off grade
IP	induced polarization
LOM	life of mine
Mag	magnetic
NI 43-101	National Instrument 43-101
NSR	net smelter return
NP	neutralization potential
NPV	net present value
SRK	SRK Consulting (Canada) Inc.
VLF	very low frequency
Conversion Factors	
1 oz	31.1035 g
1 tonne	2,204.62 lb
Distance	
µm	micron (micrometre)
cm	centimetre
Ha	hectare
km	kilometre
m	metre
mm	millimetre
Elements and Compounds	
Au	gold
Ag	silver
CN	cyanide
Cu	copper
Fe	iron
NaCN	sodium cyanide
S	sulphur
Mass	
g	gram
kg	kilogram
lb	pound
Mt	million tonnes
oz	troy ounce
t	tonne (metric ton)
kt	kilotonne
koz	thousand ounces
Pressure	
MPa	megapascal
Volume	
ft^3	cubic foot
L	litre
m^2	square metre
m^3	cubic metre

Other	
ºC	degree Celsius
AuEq	total revenue divided by gold price
G&A	general and administrative
hr	hour
kWh	kilowatt hour
M	million
masl	m above sea level
ppm	parts per million
s	second
Mt/yr	million tonnes per year
s.g.	specific gravity
t/d	tonnes per day

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28 Date and Signature Page

This technical report was written by the following “Qualified Persons” and contributing authors. The effective date of this technical report is March 25, 2013 and the signature date is October 7, 2016.

Qualified Person	Signature	Date
Dr. Gilles Arseneau, PGeo	“original signed”	October 7, 2016
Dr. Adrian Dance, PEng	“original signed”	October 7, 2016
John Duncan, PEng	“original signed”	October 7, 2016
Chris Elliott, FAusIMM	“original signed”	October 7, 2016
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Michael Royle, PGeo	“original signed”	October 7, 2016
Dr. Maritz Rykaart, PEng	“original signed”	October 7, 2016

Reviewed by

“Original signed”

Neil Winkelmann, FAusIMM
Project Reviewer

All data used as source material plus the text, tables, figures, and attachments of this document have been reviewed and prepared in accordance with generally accepted professional engineering and environmental practices.

APPENDIX A

Mineral Tenure Information and Legal Title Opinion

mcmillan
combining Lang Michener LLP and McMillan LLP

July 3, 2012

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Dear Sirs/Mesdames:

Re: Gold Canyon Resources Inc. - Short Form Prospectus Offering

We have acted as local counsel in the Province of Ontario for Gold Canyon Resources Inc. (the "**Corporation**") in connection with certain searches and investigations of title to:

1. certain patented lands, listed in Appendix A of this letter, in which we are advised the Corporation has an interest (collectively, the "**Patented Lands**"), which interest is registered or unregistered, as set forth below; and

2. certain unpatented mining claims registered in the name of the Corporation, listed in Appendix B of this letter (the "**Unpatented Claims**").

We have subsearched title to the Patented Lands and have reviewed uncertified copies of the Mining Recorder's electronic registers for the Unpatented Claims (the "**Ontario Mine Claim Database**") developed and maintained by the Mines and Mineral Division of the Ministry of Northern Development, Mines and Forestry (the "**Ministry**") and have relied upon the instruments of record available for public examination in the Land Registry Office of the Land Titles Division of Kenora. In addition, we have searched for executions outstanding against the registered owners of the Patented Lands with the Sheriff for the Judicial District of Kenora as of June 29, 2012.

As instructed, we have made no other searches, investigations or inquires with respect to the opinions expressed herein including, without limitation, any inquiries as to access and inquires of authorities regarding realty taxes, provincial land taxes, mining taxes, fees exigible as expressed on the Crown grant such as assurance fees, building and zoning compliance, utilities, unregistered easements, conservation or environmental matters. In addition, we have not examined any surveys of the Patented Lands or the Unpatented Claims for the purposes of this opinion and have not reviewed any of the encumbrances outstanding against the Patented Lands or the Unpatented Claims. In particular, we have not made any searches of adjoining lands to the Patented Lands to confirm compliance with the *Planning Act* (Ontario).

In conducting the searches and giving the opinions contained herein, we have assumed:

1. The authenticity of all documents that were submitted to us as originals;

2. The conformity with originals of all documents submitted or presented to us as certified or notarial copies;

3. The identity and capacity of all individuals acting or purporting to act as public officials;

4. The genuineness and authenticity of all signatures on all documents submitted or presented to us; and

5. The accuracy and completeness of the records maintained by any office of public record.

For greater clarity, our opinion is an independent opinion of our firm, however, to the extent that the uncertified copies of the Unpatented Claims posted in the Ontario Mine Claim Database developed and maintained by the Ministry are based on any assumptions or are subject to any limitations, qualifications or exceptions, our opinion given in reliance thereon is also based on each assumption and our opinion is made subject to each such limitation, qualification or exception. Further information regarding the Mining Claims Database developed and maintained by the Ministry can be found on the Ministry's website.

We are solicitors qualified in the Province of Ontario, Canada and accordingly no opinion is expressed herein as to the laws of any jurisdiction other than Ontario and the laws of Canada applicable thereto.

Based upon the foregoing, and subject to the qualifications noted below and expressed in the schedules attached, we are of the opinion that:

1. As of June 28, 2012 (being the date of our searches), the registered owners of the Patented Lands are as indicated in Appendix A under the subheading 'Springpole

Group' subject to the General Permitted Encumbrances and the specific encumbrances listed in Appendix A;

2. As of June 28, 2012 (being the date of our searches), the registered owners of the Patented Lands are as indicated in Appendix A under the subheadings 'Frahm Group' and 'Gold Canyon Group', subject to the General Permitted Encumbrances and the specific encumbrances listed in Appendix A; and

3. As of June 28, 2012 (being the date of our searches), the Corporation is listed as the registered owner of the Unpatented Claims, in the proportions listed in Appendix B, subject to the encumbrances, work required and due dates listed in Appendix B.

Qualifications

The opinions expressed above are subject to the following qualifications:

1. We make reference to an Option Agreement dated September 9, 2004 between the Corporation, Everett Williams, Patricia Williams, Douglas Hamblin, Lilian Hamblin, Neil Gaarder, Walter Howard, Dorothy Howard, Tim Howard, Suzanne Howard and The Springpole Company (the "**Springpole Option Agreement**"). There are certain discrepancies between the registered owners of the properties listed in Schedule "B" to the Springpole Option Agreement and registered owners disclosed by our searches. Further detail is provided in the exhibits attached hereto. We provide no opinion with respect to the discrepancy and have reported herein on the basis of registered ownership.

This opinion is for the use of the addressee only in connection with the qualification for sale to the public of 1,950,000 common shares of the Corporation, 7,150,000 flow-through shares of the Corporation together with an over-allotment option to issue and sell up to an additional 1,331,000 flow-through shares of the Corporation pursuant to an underwriting agreement dated effective June 15, 2012 (the "**Underwriting Agreement**") among the Corporation and a syndicate of underwriters co-led by CIBC World Markets Inc. and Fraser Mackenzie Limited, and including Dundee Securities Ltd., Haywood Securities Inc. and Dahlman Rose & Company, LLC (collectively, the "**Underwriters**") and may not be relied upon by or shown to any other party for any purpose without our prior written consent.

This opinion is provided at the request of the Underwriters pursuant to subsection 6.1(k)(v) of the Underwriting Agreement.

Yours truly,

McMillan LLP

APPENDIX A – PATENTED LANDS

General Permitted Encumbrances:

1. The reservations, limitations, exceptions, provisos and conditions, if any, expressed in the original grants from the Crown.
2. Agreements with government authorities existing as at the date of our searches;
3. Any municipal realty taxes, mining taxes, assurance fees, charges, rates or assessments, including claims for hydro, water or other utility arrears.
4. Compliance with any municipal by-laws, including building by-laws, fire department regulations and zoning by-laws.
5. Any discrepancies, defects or encroachments which might be disclosed by an up to date survey of the Patented Claims.
6. The limitations of title as set out in the *Land Titles Act*.
7. Native Land Claims, if any.

Summary of Patented Lands Ownership and Specific Encumbrances

Frahm Group

KRL No.	PIN	Parcel No.	Crown Patent No.	Registered Owner	Crown Reservations	Encumbrances
11233	42034-0832	2138	11445	Shirley V. Frahm	A	*
11234	42034-0833	2139	11446	Shirley V. Frahm	A, B	*
11235	42034-0834	2140	11447	Shirley V. Frahm	A	*
12896	42034-0844	2150	11456	Shirley V. Frahm	A, C	*
12897	42034-0845	2151	11457	Shirley V. Frahm	A, C	*
12898	42034-0846	2152	11458	Shirley V. Frahm	A	*
12899	42034-0849	2155	11461	Shirley V. Frahm	A, C	*
12900	42034-0847	2153	11459	Shirley V. Frahm	A	*

Frahm Group

KRL No.	PIN	Parcel No.	Crown Patent No.	Registered Owner	Crown Reservations	Encumbrances
12901	42034-0850	2156	11462	Shirley V. Frahm	A	*
13043	42034-0848	2154	11460	Shirley V. Frahm	A, B	*

Gold Canyon Group

KRL No.	PIN	Parcel No.	Crown Patent/ Lease No.	Registered Owner	Crown Reservations	Encumbrances
11229	42034-0829	2135	11442	Gold Canyon Resources Inc.	A, C	**
11230	42034-0830	2136	11443	Gold Canyon Resources Inc.	A, B	**
11231	42034-0831	2137	11444	Gold Canyon Resources Inc.	A, B	**
12868	42034-0837	2143	11450	Gold Canyon Resources Inc.	A, B	**
12869	42034-0838	2144	11451	Gold Canyon Resources Inc.	A	**
562895	42034-0871	2063	208469	Gold Canyon Resources Inc.	D, E, F	
562896	42034-0872	2064	208470	Gold Canyon Resources Inc.	D, E, F	
562897	42034-0873	2065	208471	Gold Canyon Resources Inc.	D, E	

Gold Canyon Group

KRL No.	PIN	Parcel No.	Crown Patent/ Lease No.	Registered Owner	Crown Reservations	Encumbrances
562898	42034-0874	2066	208472	Gold Canyon Resources Inc.	D, E, F	
562899	42034-0875	2067	208473	Gold Canyon Resources Inc.	D, E, F	
562900	42034-0876	2068	208474	Gold Canyon Resources Inc.	D, E, F	

Springpole Group

KRL No.	PIN	Parcel No.	Crown Patent No.	Registered Owner	Crown Reservations	Encumbrances
11236	42034-0835	2141	11448	Douglas A. Hamblin[1]	A, B	***
12872	42034-0841	2147	11453	Douglas A. Hamblin[2]	A	***
12903	42034-0852	2158	11464	Neil A. Gaarder	A, B	***
12907	42034-0854	2160	11466	Neil A. Gaarder	A	***
12867	42034-0836	2142	11449	Walter H. Howard	A, B	***
12873	42034-0842	2148	11454	Tim R. Howard	A, B	***

[1] The Springpole Option Agreement refers to this claim being owned by Douglas A. Hamblin, et. al

[2] The Springpole Option Agreement refers to this claim being owned by Douglas A. Hamblin, et. al

Springpole Group

KRL No.	PIN	Parcel No.	Crown Patent No.	Registered Owner	Crown Reservations	Encumbrances
12904	42034-0853	2159	11465	Everett D. Williams	A	***
12908	42034-0855	2161	11467	Everett D. Williams	A	***
12870	42034-0839	2145	11452	The Springpole Co.[3]	A, C	***
12874	42034-0843	2149	11455	Everett D. Williams	A, B	***
12905	42034-0857	2163	11473	Everett D. Williams	A	***
12909	42034-0856	2162	11468	Kenneth Gaarder[4]	A, C	***
12871	42034-0840	2146	11452A	The Springpole Co.[5]	A	***
12902	42034-0851	2157	11463	Lillian N. Hamblin[6]	A	***
12906	42034-0858	2164	11474	Lillian N. Hamblin[7]	A, C	***

[3] The Springpole Option Agreement refers to this claim being owned by Walter H. Howard

[4] The Springpole Option Agreement refers to this claim being owned by Neil Gaarder

[5] The Springpole Option Agreement refers to this claim being owned by Walter H. Howard

[6] The Springpole Option Agreement refers to this claim being owned by Douglas A. Hamblin et. al

[7] The Springpole Option Agreement refers to this claim being owned by Douglas A. Hamblin et. al

Notes to Reservations

Notation	Explanation

A. Crown Reservations

1. 5% of acreage for roads and right to lay roads where Crown or its officers deem necessary
2. All trees standing or being on said lands, together with the right to enter lands to remove timber
3. Free use, passage and enjoyment of, in, over and upon all navigable waters found on or under or flowing through land
4. Right of access to shores of all rivers, streams and lakes for all vessels, boats and persons together with right to use so much of the banks thereof, not exceeding one chain in depth from waters edge as may be necessary for fishery purposes
5. Assurance Fees of 1/4 percent on value of land and of timber and minerals removed therefrom and 1/10 percent on buildings (not less than $1) which must be paid before any dealing therewith

Exceptions and Reservations in Land Titles Parcels

1. Any unpaid Provincial or Municipal taxes, charges, rates, assessments and school and water rates or charges imposed in respect of statute labour
2. Conditions contained in section 101 of the *Mining Act* (now section 91(1) of *Mining Act*, R.S.O. 1990, c. M.14) requiring that all ores or minerals raised or removed from land shall be treated and refined within Canada
3. Exceptions and qualifications mentioned in section 9 of *Land Titles Act*, R.S.O. 1937 (now s. 45 of *Land Titles Act*, R.S.O. 1990, c. L.5)

B. Surface rights only on and over a strip of land one chain in perpendicular width along the shore of Springpole Lake

C. Surface rights only on and over a strip of land one chain in perpendicular width along the shore of Birch Lake

D. <u>Crown Lease Reservations:</u>

Lessee to pay all taxes, rates, duties, royalties or assessments imposed against the land or profits.

Subject to the provisions of the *Mining Act*, the *Mining Tax Act*, the *Forest Fires Prevention Act*, the *Ontario Water Resources Act* and any amendments or regulations.

Premises shall be used solely for the purposes of the mining industry. If in default, the premises may be declared void by the Lieutenant Governor in Council.

No surface mining operations shall be carried on within 150 feet of the limits of any highway or road maintained by the Ministry of Transportation except with written consent.

May not prevent or interfere with the free user of any public, travelled road or highway crossing the lands.

If the land is covered by navigable waters, this lease is subject to the provisions of the *Navigable Waters Protection Act* (Canada), the *Beds of Navigable Waters Act* and the *Lakes and Rivers Improvement Act*.

No restriction of fishing or fishing rights in any navigable waters covering the lands and the lessee may not do any damage to fishing nets, fishing, or the fishing industry.

No right, claim or title to the land under navigable waters which may be included within the limits of lands, but the Lessee shall have the exclusive right to extract the minerals therefrom during the term of the lease.

The lands are subject to the conditions in Section 104 of the *Mining Act* with respect to the treating and refining of ores and minerals in Canada.

E. <u>Crown Reservations</u>:

10% of the surface rights of the land for roads and the right to lay out and construct roads where the Crown may deem proper.

The surface rights on and over any public or colonization road or any highway crossing the land at the date of the lease.

All deposits of sand, gravel and peat together with the right of the Crown to enter and remove same without compensation.

Use of the land for all such works as may be necessary for the development of water power and the development, transmission and distribution of electrical power, natural gas, petroleum and petroleum products, including the construction, maintenance and operation of roads, railroads, transmission lines and stations, flumes, pipelines, dams, power houses and other works and structures.

The right to grant without compensation to any person or corporation the right of way necessary for the construction and operation of one or more railways over or across the land without hindrance from the lessee where such railway or railways shall not manifestly or materially interfere with the mining operations carried on upon the premises.

All timber and trees standing, being or hereafter found growing upon the land, and the right to enter upon such land to carry on forestry, to cut and remove any timber or trees thereon, and to make necessary roads for such purposes.

The free use, passage and enjoyment of, in, over, and upon all navigable waters on, under, or flowing through any part of the land, and the reservation of right of access to the shores of all the rivers, streams, and lakes for all vessels, boats, and persons, together with the right to use the water banks that does not exceed one chain in depth from the high watermark as may be necessary for fishery or public purposes.

F. Surface rights only on and over a strip of land along the shore of Birch Lake and which strip of land is bounded by the high water mark of said lake and by a line every point of which is distant 400 feet from the nearest point on the said high water mark.

Notes to Encumbrances

*

KRL Nos. 11233, 11234, 11235, 12896, 12897, 12898, 12899, 12900, 12901 and 13043 each have an identical notice listed on record. The notice details are as follows:

> Notice being registered October 8, 2010 receipted as Instrument No. KN36369, relating to Frahm Mineral Claims Agreement dated as of September 22, 2010 (the "**Frahm Agreement**") between Gold Canyon Resources Inc. and Shirley V. Frahm.

For the purposes of this opinion, the Frahm Agreement has not been reviewed and no opinion is given thereon.

**

KRL Nos. 11229, 11230, 11231, 12868, 12869 each have an identical notice listed on record. The notice details are as follows:

> Notice being registered July 13, 2010 receipted as Instrument No. KN34562, relating to Jubilee Claims Royalty Agreement dated as of July 12, 2010 (the "**Royalty Agreement**") between Gold Canyon Resources Inc. and Jubilee Gold Inc.

For the purposes of this opinion, the Royalty Agreement has not been reviewed and no opinion is given thereon.

***

KRL Nos. 11236, 12872, 12903, 12907, 12867, 12873, 12904, 12908, 12870, 12874, 12905, 12909, 12871, 12902 and 2906 each have an identical notice listed on record. The notice details are as follows:

> Notice being registered November 29, 2010 receipted as Instrument No. KN37360, relating to the Springpole Option Agreement.

For the purposes of this opinion, the Springpole Option Agreement has not been reviewed and no opinion is given thereon.

APPENDIX B – UNPATENTED CLAIMS

Summary of Claim Ownership and Encumbrances

Township/Area	Claim No.	Percentage Owned by Gold Canyon Resources Inc.	Reservations or Encumbrances	Work Required	Due Date
Borland Lake	KRL 4201940	40.00	A, C	$6,400	2013-Jul-04
Borland Lake	KRL 4201941	40.00	A, C	$6,400	2013-Jul-04
Borland Lake	KRL 4204641	40.00	A, C	$6,400	2013-Jul-25
Borland Lake	KRL 4204642	40.00	A, C	$6,400	2013-Jul-25
Borland Lake	KRL 4204643	40.00	A, C	$6,400	2013-Jul-25
Borland Lake	KRL 4204644	40.00	A, C	$6,400	2013-Jul-25
Borland Lake	KRL 4204682	40.00	A, C	$6,400	2013-Jun-13
Borland Lake	KRL 4204683	40.00	A, C	$6,400	2013-Jun-13
Borland Lake	KRL 4204685	40.00	A, C	$6,400	2013-Jun-13
Borland Lake	KRL 4204686	40.00	A, C	$6,400	2013-Jun-13
Casummit Lake	KRL 1184813	100.00	A, C	$1,600	2014-Jun-11
Casummit Lake	KRL 1184814	100.00	A, C	$2,400	2014-Jun-11
Casummit Lake	KRL 1185085	100.00	A, C	$2,400	2014-Apr-20
Casummit Lake	KRL 1185086	100.00	A, C	$6,400	2014-Apr-20
Casummit Lake	KRL 1185087	100.00	A, C	$6,000	2014-Apr-20
Casummit Lake	KRL 1185275	100.00	A, C	$400	2014-Sep-29
Casummit Lake	KRL 1185276	100.00	A, C	$2,400	2014-Sep-29

Township/Area	Claim No.	Percentage Owned by Gold Canyon Resources Inc.	Reservations or Encumbrances	Work Required	Due Date
Casummit Lake	KRL 1185277	100.00	A, C	$1,600	2014-Sep-29
Casummit Lake	KRL 1201989	100.00	A, C	$1,600	2014-Aug-28
Casummit Lake	KRL 1201990	100.00	A, C	$400	2014-Aug-28
Casummit Lake	KRL 1201991	100.00	A, C	$1,600	2014-Aug-28
Casummit Lake	KRL 1201992	100.00	A, C	$400	2014-Aug-28
Casummit Lake	KRL 1201993	100.00	A, C	$800	2014-Aug-28
Casummit Lake	KRL 1210701	100.00	A, C	$1,600	2013-May-08
Casummit Lake	KRL 1210702	100.00	A, C	$800	2014-May-08
Casummit Lake	KRL 1210703	100.00	A, C	$4,800	2013-May-08
Casummit Lake	KRL 1234136	100.00	A, C	$2,800	2014-Feb-28
Casummit Lake	KRL 1234137	100.00	A, C	$6,400	2013-Feb-28
Casummit Lake	KRL 1234198	100.00	A, C	$4,800	2013-Oct-06
Casummit Lake	KRL 1234316	100.00	A, C	$800	2014-Mar-24
Casummit Lake	KRL 1234317	100.00	A, C	$400	2014-Mar-24
Casummit Lake	KRL 1234318	100.00	A	$800	2014-Mar-24
Casummit Lake	KRL 1247880	100.00	A, C	$400	2014-Jul-17
Casummit Lake	KRL 1247881	100.00	A, D	$1,200	2014-Jul-17
Casummit Lake	KRL 1248691	100.00	A, C	$6,400	2014-Apr-08
Casummit Lake	KRL 3004746	100.00	A, C	$4,800	2014-Jul-09

Township/Area	Claim No.	Percentage Owned by Gold Canyon Resources Inc.	Reservations or Encumbrances	Work Required	Due Date
Casummit Lake	KRL 3018700	100.00	A, C	$400	2014-Aug-24
Casummit Lake	KRL 4205205	100.00	A, C	$1,200	2014-Feb-24
Casummit Lake	KRL 4205206	100.00	A, C	$400	2014-Feb-24
Casummit Lake	KRL 4205207	100.00	A, C	$4,000	2014-Feb-24
Casummit Lake	KRL 4205208	100.00	A, C	$4,800	2014-Feb-24
Casummit Lake	KRL 4205214	100.00	A, C	$1,600	2014-Mar-02
Casummit Lake	KRL 4205215	100.00	A, C	$6,400	2014-Mar-02
Casummit Lake	KRL 4212762	100.00	A, C	$2,400	2014-Oct-01
Casummit Lake	KRL 4212764	100.00	A, C, I	$6,400	2013-Jul-13
Casummit Lake	KRL 4224179	100.00	A, C	$4,800	2013-Aug-27
Casummit Lake	KRL 4224180	100.00	A, C	$4,800	2013-Aug-27
Casummit Lake	KRL 720373	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 720374	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 720375	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 818712	100.00	A, B, C	$400	2014-Oct-29
Casummit Lake	KRL 818713	100.00	A, B, C	$400	2014-Oct-29
Casummit Lake	KRL 818714	100.00	A, B, C	$400	2014-Oct-29
Casummit Lake	KRL 818715	100.00	A, B, C	$400	2014-Oct-29
Casummit Lake	KRL 818854	100.00	A, B, C	$400	2014-Oct-29

Township/Area	Claim No.	Percentage Owned by Gold Canyon Resources Inc.	Reservations or Encumbrances	Work Required	Due Date
Casummit Lake	KRL 818855	100.00	A, B, C	$400	2014-Oct-29
Casummit Lake	KRL 818856	100.00	A, B, C	$400	2014-Oct-29
Casummit Lake	KRL 818857	100.00	A, B, C	$400	2014-Oct-29
Casummit Lake	KRL 818858	100.00	A, B, C	$400	2014-Oct-29
Casummit Lake	KRL 818859	100.00	A, B	$400	2014-Oct-29
Casummit Lake	KRL 818866	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 818867	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 818868	100.00	A, B, C, G	$400	2014-Apr-29
Casummit Lake	KRL 818869	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 818870	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 818871	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 818872	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 818873	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 818874	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 818875	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 818876	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 818877	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 818878	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 818879	100.00	A, B, C	$400	2014-Apr-29

Township/Area	Claim No.	Percentage Owned by Gold Canyon Resources Inc.	Reservations or Encumbrances	Work Required	Due Date
Casummit Lake	KRL 818891	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 818892	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 818893	100.00	A, B, C	$400	2014-Apr-29
Casummit Lake	KRL 834734	100.00	A, B, C	$400	2014-Mar-05
Casummit Lake	KRL 834783	100.00	A, B, C	$400	2014-Mar-05
Casummit Lake	KRL 834784	100.00	A, B, C	$400	2014-Mar-05
Casummit Lake	KRL 834785	100.00	A, B, C	$400	2014-Mar-05
Casummit Lake	KRL 834788	100.00	A, B, C	$400	2014-Mar-05
Casummit Lake	KRL 845861 *	100.00	A, B, C	$400	2014-May-24
Casummit Lake	KRL 845862 *	100.00	A, B, C	$400	2014-May-24
Casummit Lake	KRL 845863 *	100.00	A, B, C	$400	2014-May-24
Casummit Lake	KRL 845864 *	100.00	A, B, C	$400	2014-May-24
Casummit Lake	KRL 845865 *	100.00	A, B, C	$400	2014-May-24
Casummit Lake	KRL 845866 *	100.00	A, B	$400	2014-May-24
Casummit Lake	KRL 870087	100.00	A, B, C	$400	2014-Aug-31
Casummit Lake	KRL 870237	100.00	A, B, C	$400	2014-Aug-06
Casummit Lake	KRL 903534	100.00	A, B, C	$400	2014-Apr-18
Casummit Lake	KRL 903535	100.00	A, B, C	$400	2014-Apr-18
Casummit Lake	KRL 903536	100.00	A, B, C	$400	2014-Apr-18