Taylor Property, Ontario, Canada

NI 43-101 Technical Report

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Report Addressed to St Andrew Goldfields Ltd

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SUMMARY

Field work carried out on the Taylor property since 2010 was used in the preparation of a pre-feasibility study report on the Taylor deposit (West Porphyry Zone only), which is located in the Taylor Township, Ontario, Canada. This National Instrument 43-101 technical report was prepared by employees of St Andrew Goldfields Ltd. (SAS), in support of the pre-feasibility study. Highlights of the report include:

- First time disclosure of approximately 1 Mt grading 5.45 g/t (173 koz) of probable mineral reserves;
- Pre-tax cumulative undiscounted cash flow of \$20 M and pre-tax $NPV_{5\%}$ of \$12 M;
- Internal rate of return of 22%
- Cash operating unit costs of US\$903/oz (\$150/t); total costs of US\$1,193/oz;
- Production at an average mining rate of 675 tpd during approximately four years and pre-production and development activities for approximately two years; and,
- The West Porphyry Zone (WPZ) hosts indicated mineral resources of 1.9 Mt grading 5.50 g/t (341 koz) and inferred mineral resources of 1.9 Mt grading 3.95 g/t (243 koz); mineral resources are inclusive of mineral reserves.

Cut and fill was selected as the mining method for the WPZ. Dilution was applied individually on each shape, ranging from 9% to 50%, and an extraction factor of 95% was set for all stopes. Based on test work performed at SGS laboratories, mill recovery averaged 94.5%.

In order to minimize the financial risk to SAS, a stepped approach is proposed, which requires the Board of Directors approval before proceeding to each subsequent step:

- Complete a bulk sample in the upper area of the mine using a contractor. The estimated cash flow is (\$1.1 M), net of credit from gold processing. It is estimated to complete the bulk sample within six months from contractor mobilization.
- Complete a bulk sample in the lower area of the mine using a contractor. The estimated cash flow is (\$15.0 M), net of credit from gold processing. It is estimated to complete the bulk sample within 12 to 14 months following the decision to proceed. Pending favourable results, seek SAS Board of Directors approval to continue the project as per the LOM plan using SAS employees. The remaining capital expenditures amount to approximately \$29.0 M.

1.0 INTRODUCTION

This National Instrument 43-101 technical report was prepared by employees of SAS, in support of SAS pre-feasibility study (PFS) pertaining to the Taylor deposit located in the Taylor Township, Ontario, Canada. While the mineral resources were estimated for the entire Taylor deposit, the PFS and associated mineral reserves estimate focused on the WPZ only.

Information was obtained through the field and technical work related to the Taylor deposit over the past few years. Most of that information was derived by SAS employees.

The two qualified persons (QP) visited the Taylor property numerous times since 2010 and participated in the direction of the field and technical work.

The units of measures used in this report conform to the metric system. Unless stated otherwise, the Canadian Dollar (CDN$) is the currency used in this technical report. A list of abbreviations is displayed in Table 1-1.

Abbreviation	Meaning
a	annum
CDN$	Canadian dollar
cm	centimetre
d	day
DDH	diamond drill hole
EM	electromagnetic
g	gram
gpt, g/t	gram per tonne
ha	hectare (2.471 acres)
HLEM	horizontal loop electromagnetic
IP	induced polarization
k	kilo
kg	kilogram
km	kilometre
L, l	litre
m	metre
M	mega
$M	million dolars
m^3	cubic metre
MASL	metres above sea level
min	minute
ODH	overburden drill hole
oz	Troy ounce (31.1035 grams)
koz	thousand ounces
ppm, ppb	part per million, part per billion
s	second
ton	short ton (0.907185 tonne)
tonne, t	metric tonne
tpa, t/a	tonne per year
tpd, t/d	tonne per day
US$	United States of America Dollar
VLEM	vertical loop electromagnetic
VLF-EM	very low frequency electromagnetic

Table 1-1: List of Abbreviations.

2.0 RELIANCE ON OTHER EXPERTS

Material information contained in this report was prepared by, or under the direct supervision of Pierre, Rocque, P. Eng. and Craig Todd, P. Geo. As a result, no other experts were relied upon.

3.0 PROPERTY DESCRIPTION AND LOCATION

3.1 Location

The Taylor property is located in the Taylor Township, approximately eight km northwest of the town of Matheson and four km, north of Highway 101, which lies within the Black River-Matheson Municipality and within Lots 5 – 8, Concessions II and III of Taylor Townships in the Larder Lake Mining Division, District of Cochrane, Ontario (Figure 3-1). The main access to the property is via Regional Road #11, north of Highway #101. Existing mine workings are located immediately north and south of Concession Road II. The Taylor property lies east of the Stock Mine property and west of the Black Fox mine, which are both owned and operated by Brigus Gold Corp.

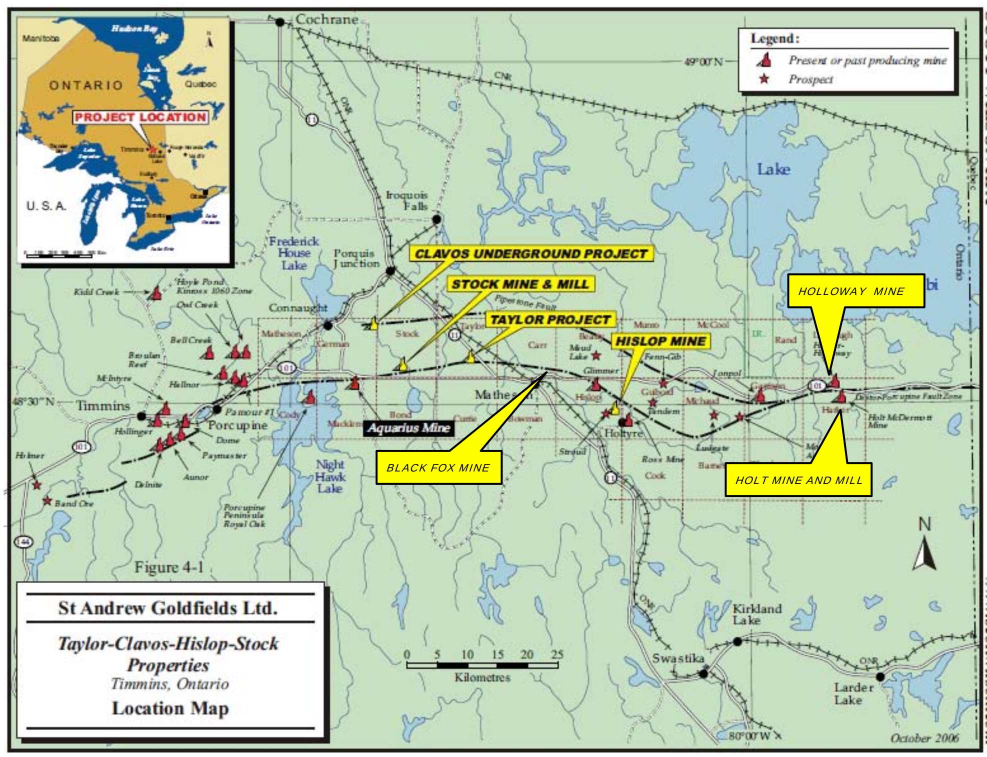

Figure 3-1: Location of the Taylor project.

3.2 Mineral Tenure and Encumbrances.

The claim group is centered at 5379000N and 529000E in NAD83, zone 17 (using UTM coordinate system).

The 3,269 ha property is comprised of 78 claims that include patented, leased, mineral claims and surface and mineral rights claims (Figure 3-2).

Various net profit payments or other royalty agreements were negotiated over the past 23 years. As shown in Table 3-1, the Shoot Zone holds a 1% and the WPZ holds an average 2% net smelter royalty respectively.

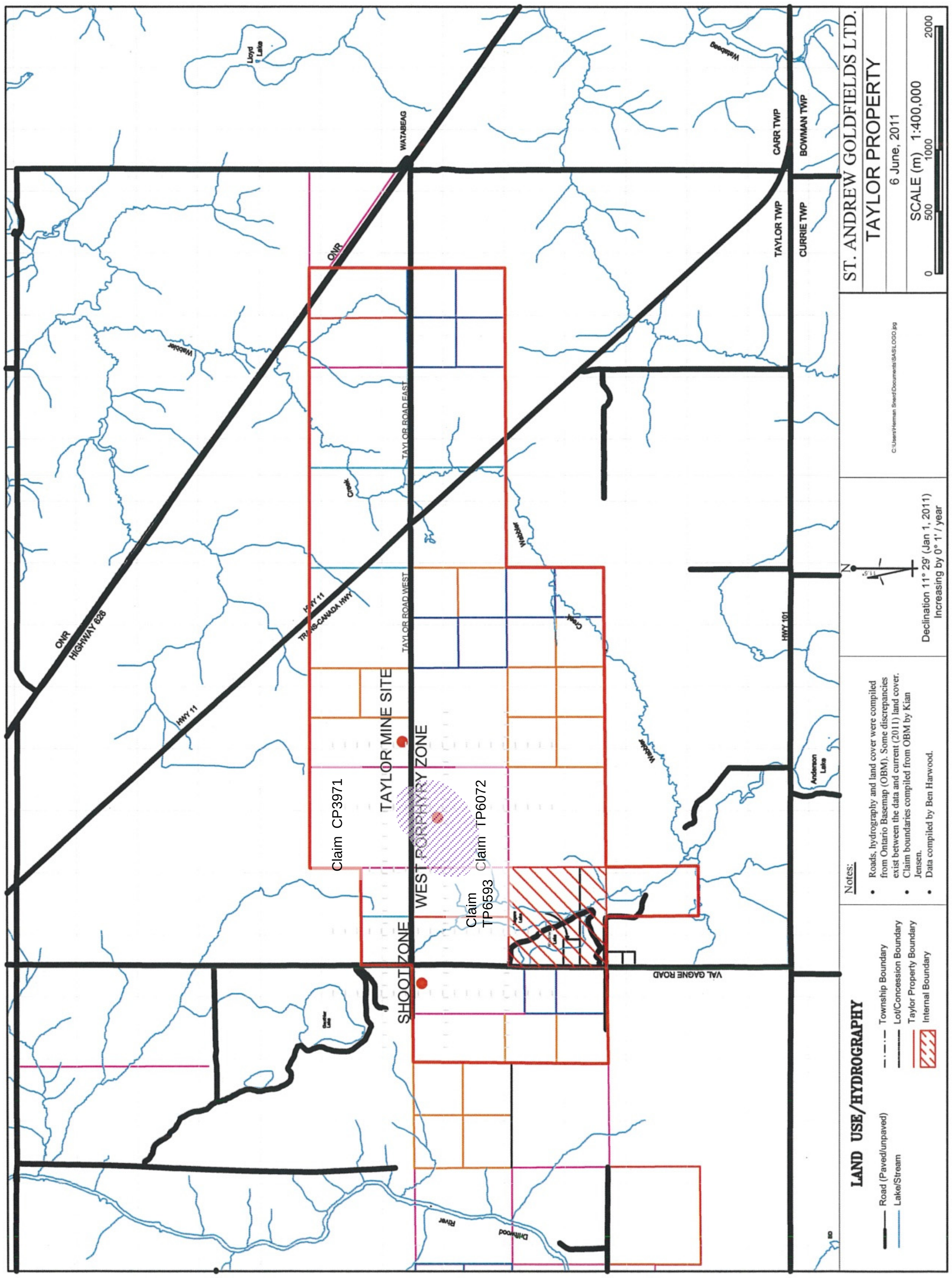

Figure 3-2: Claims boundaries.

Claim Number	PCL Number	Grant	Surf. Rights	Min. Rights	Township	Area (ha)	PIN Number	Interest	Royalty	Deposit on claim	Notes	Holders	Rate	Year
12537SEC	12537SEC		Y	Y	STOCK	72.229	65363-076	100	Nil		1	Walter Turney	2.0% NSR	2003
17109SEC	17109SEC		Y	Y	TAYLOR	32.842	65364-201	100	Nil		2	Jacques + Lise Boutin	1.5% NSR	1997
19018SEC	19018SEC		N	Y	STOCK	65.396	65363-068	100	Nil		3	Leo Alarie and Sons	2.0% NSR	1997
23574SEC	23574SEC	CP7164	Y	N	TAYLOR	33.139	65364-136	100			4	Robert Haley	2.0% NSR	1997
4842SEC	4842SEC		Y	Y	STOCK	61.991	65363-079	100	Nil		5	Robert Lalonde	2.0% NSR	2003
CP129	3314SEC	CP129	Y	Y	TAYLOR	63.415	65364-403	100	Nil		6	Ginn and Barnes	2.0% NSR	1999
CP1308	5094SEC	CP1308	Y	Y	TAYLOR	16.227	65364-135	100	Nil		7	Yvette Lapierre	2.0% NSR	2004
CP1308	24453SEC	CP1308	N	Y	TAYLOR	16.103	65364-133	100	2		8	Franco-Nevada	1.0% NSR	2006
CP181	19019SEC	CP181	N	Y	TAYLOR	59.558	65364-200	100	Nil		9	1051989 Ontario Inc.	2.0% NSR	2009
CP1823	23574SEC	CP1823	Y	Y	TAYLOR	64.459	65364-136	100	8					
CP2328	13492SEC	CP2328	N	Y	TAYLOR	30.571	65364-191	100	3					
CP351	3394SEC	CP351	Y	Y	TAYLOR	65.708	65364-139	100	1					
CP3683	18323SEC	CP3683	N	Y	STOCK	73.831	65363-073	100	Nil					
CP3971	8344SEC	CP3971	Y	Y	TAYLOR	66.558	65364-134	100	1	North end of WPZ				
CP4029	8378SEC	CP4029	Y	Y	TAYLOR	60.740	65364-155	100	Nil					
CP5323	23575SEC	CP5323	Y	Y	TAYLOR	38.021	65364-153	100	8					
CP6454	23575SEC	CP6454	Y	Y	TAYLOR	27.775	65364-153	100	8	Shoot Zone				
CP6870	23574SEC	CP6870	Y	Y	TAYLOR	34.200	65364-136	100	Nil					
CP9686	23573SEC	CP4070	Y	Y	CARR	61.526	65365-418	100	Nil					
L74172	273LC	CL286	Y	Y	TAYLOR	15.759	65364-137	100	Nil					
L74173	273LC	CL286	Y	Y	TAYLOR	15.063	65364-137	100	Nil					
L74174	1706LC	CL287	N	Y	TAYLOR	17.531	65364-151	100	Nil					
L74175	1706LC	CL288	Y	Y	TAYLOR	16.795	65364-151	100	Nil					
L74176	276LC	CL289	Y	Y	TAYLOR	16.452	65364-154	100	Nil					
L74177	276LC	CL289	Y	Y	TAYLOR	16.329	65364-154	100	Nil					
L74178	276LC	CL289	Y	Y	TAYLOR	17.167	65364-154	100	Nil					
L74179	276LC	CL289	Y	Y	TAYLOR	17.935	65364-154	100	Nil					
L74184	1707LC	CL499	Y	Y	TAYLOR	15.553	65364-160	100	Nil					
L74185	1707LC	CL500	N	Y	TAYLOR	16.555	65364-160	100	Nil					
L74186	1707LC	CL500	N	Y	TAYLOR	10.809	65364-160	100	Nil					
L74187	1707LC	CL499	Y	Y	TAYLOR	15.755	65364-160	100	Nil					
L74308	1706LC	CL288	Y	Y	TAYLOR	14.966	65364-151	100	Nil					
L74309	1706LC	CL287	N	Y	TAYLOR	15.608	65364-151	100	Nil					
L77133	1706LC	CL592	Y	Y	TAYLOR	15.991	65364-151	100	Nil					
L77134	1706LC	CL592	Y	Y	TAYLOR	16.105	65364-151	100	Nil					
L77135	1706LC	CL592	Y	Y	TAYLOR	16.322	65364-151	100	Nil					
L77136	1706LC	CL591	N	Y	TAYLOR	16.125	65364-151	100	Nil					
L77137	1706LC	CL592	N	Y	TAYLOR	15.550	65364-151	100	Nil					
L77138	1706LC	CL592	Y	Y	TAYLOR	15.754	65364-151	100	Nil					
L77139	1706LC	CL592	Y	Y	TAYLOR	16.354	65364-151	100	Nil					
L77140	1706LC	CL592	Y	Y	TAYLOR	15.500	65364-151	100	Nil					
L77638	1130LC	CL800	N	Y	TAYLOR	17.802	65364-396	100	Nil					
L77639	1130LC	CL800	N	Y	TAYLOR	15.582	65364-396	100	Nil					
L77640	1130LC	CL800	N	Y	TAYLOR	16.256	65364-396	100	Nil					
L77641	1130LC	CL800	N	Y	TAYLOR	14.313	65364-396	100	Nil					
L77642	1130LC	CL800	N	Y	TAYLOR	15.366	65364-396	100	Nil					
L77643	1130LC	CL800	N	Y	TAYLOR	15.879	65364-396	100	Nil					
L77644	1130LC	CL800	N	Y	TAYLOR	15.952	65364-396	100	Nil					
L77645	1130LC	CL800	N	Y	TAYLOR	16.583	65364-396	100	Nil					
NP3155	724SEC	NP3155	Y	Y	TAYLOR	64.036	65364-167	100	Nil					
NP6002	980SEC	NP6002	Y	Y	TAYLOR	67.131	65364-159	100	Nil					
TP2913	23573SEC	TP2913	Y	Y	TAYLOR	65.264	65364-142	100	Nil					
TP578	1371SEC	TP578	Y	Y	TAYLOR	66.098	65364-138	100	1					
TP6047	16044SEC	TP6047	N	Y	TAYLOR	31.733	65364-436	100	Nil					
TP6071	12529SEC	TP6071	N	Y	TAYLOR	65.698	65364-140	100	7					
TP6072	23574SEC	TP6072	Y	Y	TAYLOR	64.967	65364-136	100	8	Most of WPZ				
TP6593	23574SEC	TP6593	Y	Y	TAYLOR	30.522	65364-136	100	8	West end of WPZ				
TP6593	8470SEC	TP6593	Y	Y	TAYLOR	32.062	65364-152	100	9					
TP6633	23573SEC	TP6633	Y	Y	TAYLOR	37.427	65364-141	100	Nil					
TP7116	155110SEC	TP7116	N	Y	CARR	62.932	65365-393	100	6					
TP7122	2741SEC	TP7122	Y	Y	TAYLOR	64.714	65364-401	100	4					
TP7528	19017SEC	TP7258	N	Y	STOCK	56.120	65363-070	100	Nil					
TP7610	18843SEC	TP7610	Y	Y	TAYLOR	62.844	65364-157	100	Nil					
Unpatented Claims														
1218632		UPC	N	Y	CARR	131.768		20	80 to GCC					
1227976		UPC	N	Y	TAYLOR	4.452		100						
1227977		UPC	N	Y	STOCK	2.613		100						
1228935		UPC	N	Y	TAYLOR	30.346		100						
1228936		UPC	N	Y	TAYLOR	59.760		100						
1228937		UPC	N	Y	TAYLOR	64.127		20	80 to GCC					
1228938		UPC	N	Y	TAYLOR	195.734		100						
1228939		UPC	N	Y	TAYLOR	65.597		100						
1228940		UPC	N	Y	TAYLOR	64.354		100						
1228941		UPC	N	Y	TAYLOR	66.645		100						
1228942		UPC	N	Y	TAYLOR	131.718		100						
3002298		UPC	N	Y	CARR	30.918		100						
3003829		UPC	N	Y	CARR	57.400		100						
3011310		UPC	N	Y	STOCK	8.609		100						

Table 3-1: Claim ownership and associated royalties.

3.3 Permit Status

The following are valid permits pertaining to the Taylor site:

- an approved Closure Plan (the Amendment is dated December 2005);
- a Certificate of Approval – Air (COA #3041-6MGQSQ), application approved May 3, 2006);
- a Certificate of Approval – Industrial Sewage (COA #8693-7NVPHR), application approved May 11, 2009);
- a Permit to Take Water (5330-7AQLJS), which allows 2,016 m^3/day and expires on January 10, 2018.

For the proposed advanced exploration program, the following will be required:

- a modification to the existing Certificate of Approval – Industrial Sewage. This will allow for increased pumping volume and change in discharge location from the current small receiver to another location (e.g. the Driftwood River).
- a modification to the existing Closure Plan to include additional infrastructure, an updated mining plan, updated and additional underground development, and changes to surface features such as waste rock piles and overburden stockpiles. Along with this, the financial security will likely require to be updated.
- a revision of the existing Permit to Take Water to allow for a change in the receiver, year round pumping and an increased pumping rate (if deemed appropriate by SAS).
- a Certificate of Approval – Air to allow for the various site exhausts commonly associated with mining operations such as mine air ventilation and heating. This permit will be required during the second phase of the proposed work program.

SAS is currently updating the status of the four items listed above and does not anticipate delays in obtaining those in time for the start-up of the first phase of the proposed work program.

Keys to permitting for a re-activated Taylor mining program are issues concerning near surface hydrology and water discharge, both in terms of quantity and quality.

Updated underground development plans in the WPZ will also likely involve increases in mine water discharge volume from that currently permitted, but will likely not involve complex hydrology issues associated with surface mining.

The anticipated increased mine water discharge pumping will likely require a change in the discharge receiver from the present small receiver being used (i.e. Wabbler Creek), which is located just northeast of Taylor Mine site, to the Driftwood River, located approximately 2.6 km west of the WPZ.

Care will also be required to avoid issues with ammonia and suspended sediment discharge.

Finally, a Notice of Project status change is required in order to change the status from inactive to active.

3.4 Environmental Liability and Other Potential Risks

In the Qualified Person's (QP) opinion, there are no significant factors or risks that may affect access, title or the right or ability of SAS to perform work on the Taylor property.

4.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

4.1 Climate, Topography and Physiography

The climate of the area is typical of northern Ontario with cold winters, warm summers and only moderate precipitation. Climatic conditions in Timmins have been described based on meteorological information from Environment Canada[1] during the period from 1971 to 2000. The average daily temperature in the Timmins area is recorded as 1.3°C with a daily average low of -17.5°C in the month of January, and a daily average high of 17.4°C in the month of July. An extreme low of -45.6°C was recorded on February 1st, 1962 and the extreme high of 38.9°C occurred on July 31st, 1975. The yearly average precipitation for the Timmins area is 831.3 mm with approximately 67% as rain and 33% as snow. The record daily amount of rainfall, 87.6 mm, occurred on July 29th, 1990 and the record daily amount of snowfall, 48.2 cm, occurred on March 19th, 1983.

All of the Taylor property is covered by flat lying to gently rolling terrain with little topographic relief. Overburden depths range for 3 to 60 m, with average overburden depth on the property being approximately 30 m. Elevations range from approximately 250 m to 300 m above sea level. The area is reasonably well drained by creeks and small rivers, and there are numerous small swamps and marsh areas. Outcrop is limited due to an extensive blanket of overburden, mostly sand with lesser amounts of clay from the northerly trending Munro esker. The area is located within the Boreal Shield zone: tree cover is normally thick and predominantly coniferous (with black spruce and jack pine being the most common species), with lesser stands of poplar and birch. The current cover is believed to be a mix of second and third growth forest as a result of logging operations and forest fires.

4.2 Means of Access to the Property

The Taylor property can be accessed by travelling two km west on an all-weather gravel road (the “Taylor road”) off Highway #11, which is located eight km west of the town of Matheson along Highway #101. Highways #11 and #101 are part of the Trans Canada Highway system and a main transportation route between northern and southern Ontario. Alternately, the property can be accessed by travelling north on the Val Gagné Road from Highway #101, east along the Taylor Township Concession II Road.

[1] Environment Canada website: http://www.climate.weatheroffice.gc.ca

Once on the Taylor property, visitors must register with site security personnel at the security office.

Access to the property and current infrastructure are displayed in Figure 4-1.

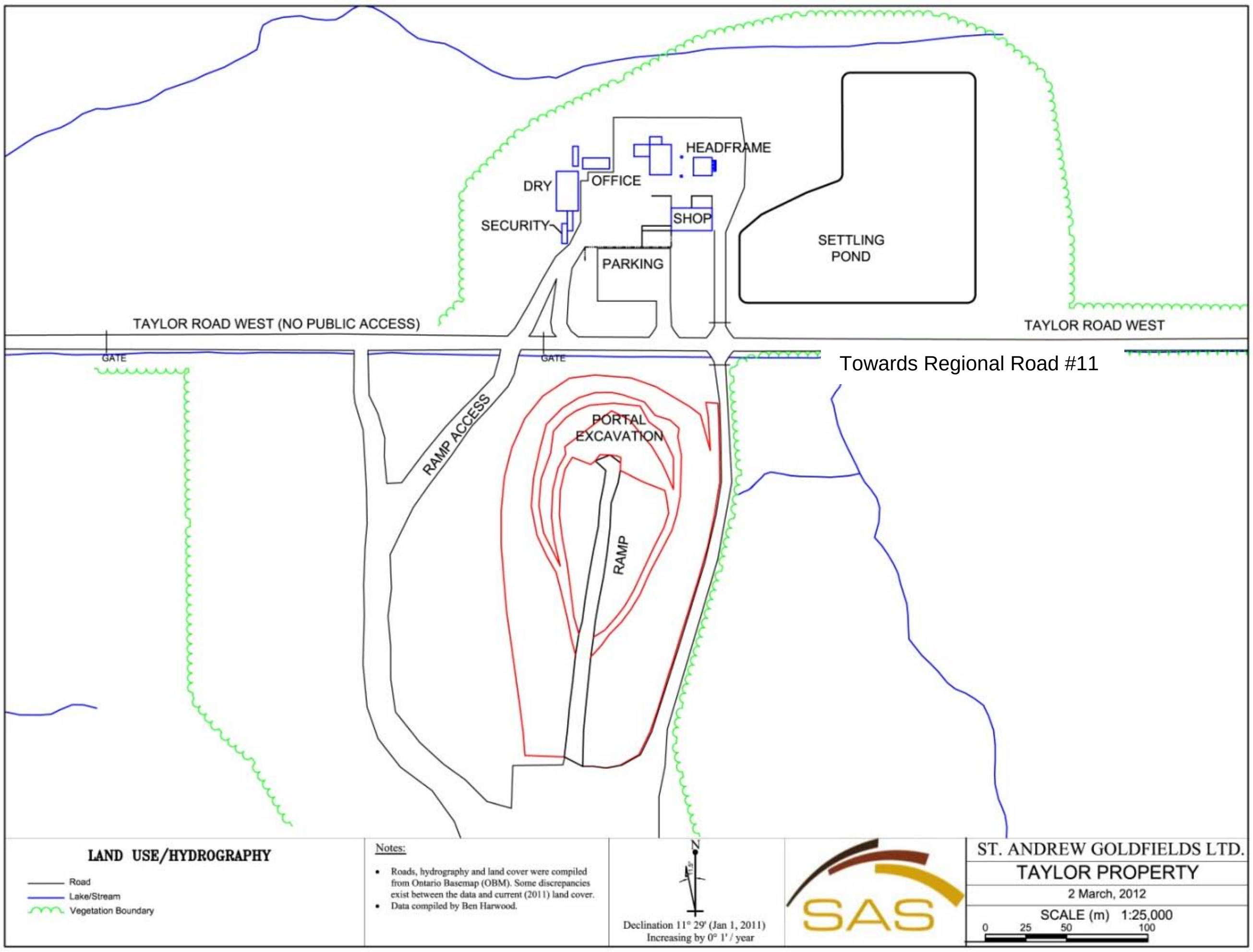

Figure 4-1: Access to the Taylor property.

4.3 Infrastructure and Local Resources

The infrastructure is well developed and can support mining activities in the area, such as the Taylor project. Power and water are already available at the Taylor site and fuel sources delivery can resume within short notice. The area is well serviced with an array of major roads and two airports (in Timmins and Rouyn-Noranda). Since the ore will be treated at the company's Holt mill, there are no requirements to store tailings at the Taylor site; waste rock storage areas were constructed during previous mining activities and can be re-activated as needs arise.

The Black River-Matheson Township (116,167 ha) has an approximate population of 2,800 residing mainly in the towns of Matheson, Shillington, Holtyre and Ramore. Further to the west are the towns and cities of Porcupine, South Porcupine, Schumacher and Timmins (approximately 45,000 residents). To the north are the towns of Iroquois Falls and Cochrane. To the south is the town of Kirkland Lake (approximately 10,000 residents).

SAS owns an office building in Matheson that is being used as its Exploration Department base. Additionally, SAS acquired two former motels in Matheson that are operated as temporary housing for relocated employees. SAS uses many local residents as support staff and local contractors to maintain the facilities.

SAS does not anticipate opposition from the local communities to the development, and the subsequent transition of the Taylor Project into production.

The primary First Nations community living close to the Taylor property is the Wahgoshig First Nation (WFN). The WFN is an Anishinaabe (Algonquin and Ojibwa) and Cree First Nation located near Matheson, in the Cochrane District of north-eastern Ontario, Canada. The reserve covers 7,770.1 ha (Abitibi 70 Indian Reserve) on the south end of Lake Abitibi. The First Nation community has approximately 270 registered people; 121 people live on the reserve, where they provide the following services: band office, health clinic, warehouse / fire hall, public works garage and a community hall. Wahgoshig is policed by the Nishnawbe-Aski Police Service, an Aboriginal staffed service.

SAS has been working for the past few years on securing a mutually agreeable impact and benefit agreement (IBA) with the Wahgoshig First Nation. Based on current permitting of the Taylor property, an IBA is not required to begin production activities, although it could be beneficial.

5.0 HISTORY

5.1 Prior Ownership

Prior to SAS' acquisition, the Taylor property area had been explored by Hollinger and by a joint venture between Labrador Mining and Exploration Company Ltd. (successor to Hollinger) and later by Esso Minerals Canada (Esso Minerals). The property included two near surface gold deposits, the Shoot Zone and the Shaft Zone and the deeper West Porphyry Zone.

5.2 Exploration and Development Work

5.2.1 Shaft Zone

The Taylor Mine Shaft Deposit was discovered by Hollinger in 1962. From 1962 to 1966, Hollinger drilled 68 surface diamond drill holes totalling 14,384 m and from 1980 to 1984 Hollinger drilled an additional 31 surface diamond drill holes totalling 3,662 m From February 1986 to July 1998, SAS drilled 42 diamond drill holes from surface totalling 13,649 m. From 1986 to 1988, an underground exploration program was undertaken on the Shaft Zone by SAS and Esso Minerals. A shaft was sunk to 172 m through 14 m of overburden. Drifting, crosscutting, some raising, and extensive underground diamond drilling were carried out on three levels. From March 1987 to October 1988, 254 holes totalling 12,108 m were drilled from underground. From April 1962 to July 1998, a total of 395 diamond drill holes aggregating 43,803 m have targeted the Shaft Deposit mineralization.

5.2.2 West Porphyry Zone

The West Porphyry Zone, located about 450 m west of the Shaft Deposit and about 400 m below surface, was discovered by Hollinger in 1962. From April 1962 to August 1966, Hollinger drilled 14 holes totalling 4,606 m. From November 1972 to June 1980, Hollinger intermittently drilled a total of 10 holes totalling 4,118 m. From February 1986 to July 1998, SAS drilled 185 holes from surface totalling 84,015 m. Up until July 1998, some 209 holes totalling 92,740 m had been drilled in the West Porphyry Deposit area and vicinity. The West Porphyry Deposit hosts the Main West Porphyry Deposit and the Upper West Porphyry Deposit.

In early 1999, another six drill holes totalling 4,831 m were completed on the Upper West Porphyry Deposit zones and some changes were made to the West Porphyry Deposit resource estimate.

In 2004, St Andrew commenced planning for the Advanced Exploration Project to test the West Porphyry deposit. In 2005, 13 surface diamond drill holes were completed in the planned portal area for condemnation purposes and rock strength testing. These holes tested the thickness of the overburden and typically cored about 3 m of rock. Detailed plans and budgets were completed. By the end of 2006, the overburden in the portal area had been excavated and development of the decline was commenced.

5.2.3 Shoot Zone

The Shoot Zone was discovered by Hollinger in 1972. From the time of the discovery until 1981, Hollinger drilled 50 holes totalling 8,263 m. From 1986 to April 1997, SAS drilled 49 holes totalling 9,960 m. The early 1997 SAS drill holes were shallow and closely spaced to investigate the Shoot Zone open pit potential. Some 99 diamond drill holes totalling 18,223 m were drilled in the Shoot Zone area from 1972 to 1997.

5.3 Historical Mineral Resources and Mineral Reserves

SAS is not treating the historical estimates as current mineral resources or mineral reserves. A qualified person has not done sufficient work to classify the historical estimates as current mineral resources or mineral reserves.

5.3.1 Shaft Zone

An internal 1988 "Feasibility Study" estimated "reserves" of 51,071 tonnes averaging 4.0 g/t Au, but indicated that profitability required a gold price in excess of C$610/oz.

5.3.2 West Porphyry Zone

In 1998, SAS re-interpreted the West Porphyry Deposit mineralization and completed a mineral resource estimate using the polygonal method. The West Porphyry Deposit indicated mineral resources were reported at 1,222,000 tonnes averaging 8.7 g/t Au at a 3.4 g/t Au cut-off grade and with high gold values cut to 34.3 g/t Au. The West Porphyry Deposit inferred mineral resources totalled 410,000 tonnes averaging 8.5 g/t Au at a 3.4 g/t Au cut-off grade and with high gold values cut to 34.3 g/t Au. These mineral resources were first updated in 2004[2].

5.3.3 Shoot Zone

In 1998, St Andrew estimated that the Shoot Zone contained indicated mineral resources totalling 670,000 tonnes averaging 5.5 Au at a 3.4 g/t Au cut-off grade. The

[2] SWRPA, "Technical Report on the Taylor, Clavos, Hislop and Stock Projects in the Timmins Area, Northeastern Ontario, Canada", October 2, 2006.

Shoot Zone inferred mineral resources totalled 106,000 tonnes averaging 5.2 g/t Au at a 3.4 g/t Au cut-off grade. Cutting of high gold values to 34.3 g/t Au had no impact on the resource estimate.

5.4 Historical Production from the Property

A 6,000-tonne bulk sample from development in the Shaft Zone at the Taylor Project was processed at the Stock mill in September 1991. Gold recovery ranged from 89% to 96% for material grading on average 2.2 g/t Au.

6.0 GEOLOGICAL SETTINGS AND MINERALIZATION

6.1 Regional Geology

The Taylor project is located along the Destor-Porcupine Fault Zone (DPFZ), a major structural feature associated with globally significant gold deposits lying within the Abitibi Greenstone Belt of northeastern Ontario and north-western Quebec. The Abitibi Greenstone Belt is typical of other Archean-aged greenstone belts in the Canadian Shield, and elsewhere in the world, in that it contains predominantly volcanic and sedimentary sequences of rocks intruded by mafic to felsic intrusions and late cross-cutting diabase dikes. Being approximately 750 km in length by 250 km in width, it is the largest greenstone belt in the world. Volcanic, sedimentary and contemporaneous intrusive rocks in the Abitibi range in age from 2,745 to 2,680 Ma.

There are three main stratigraphic units of significance to the region, from oldest to youngest:

- the <u>Deloro Group</u> consists of basal komatiitic flows overlain by calc-alkaline basalts and andesite and felsic pyroclastic volcanic rocks.

- the <u>Tisdale Group</u> consists of ultramafic to basaltic komatiitic to Mg-tholeiitic basalt, which are overlain by Fe-tholeiitic basalts, overlain by felsic calc-alkaline pyroclastic volcanic rocks.

- the <u>Porcupine Group</u> consists of metasedimentary rocks representing the infilling of a large basin via a turbiditic sequence including inter-layered greywacke, argillite and conglomerate.

All the volcanic rocks listed above were intruded by mafic and felsic intrusive bodies, including feldspar and quartz-feldspar porphyries. Late diabase dikes cross-cut all of the above stratigraphic units. There is a common association of gold deposits with porphyritic intrusions in the Porcupine camp and elsewhere.

Gold production from deposits located in proximity to the DPFZ has been prolific. Total output is estimated at over 62 million ounces of gold since the start of gold production in the Porcupine Camp.

Numerous publications describe the regional geology of the area and gold deposit models and may be referenced for a more detailed description, including Ferguson et al[3] and Pike and Jensen[4].

6.2 Local and Property Geology

The Taylor property is almost entirely covered by glacial overburden, ranging from three metres to 60 m in thickness (generally 30 m to 40 m thick). Thus, interpretations of the property geology have been made principally from diamond drill hole information as well as the underground excavations in the shaft and ramp areas developed in the Shaft Zone.

The Taylor Project is located along the DPFZ in its central portion, approximately 60 km east of the main gold producers in the vicinity of Timmins. The DPFZ in the area of the Taylor property strikes roughly east-west, and dips to the south between 40° and 60°, with the majority of the property lying to the south of the projected trace of the DPFZ. The DPFZ is a complex structural zone, and it is more accurately described as a zone of tens of metres width, along which are contained many individual zones of movement. In the Taylor property area, the footwall of the DPFZ is considered to be a thick series of relatively undeformed and unaltered metasedimentary rocks intersected to the footwall.

The rock lithology on the Taylor property can be generalized, from south to north (Figure 6-1) as follows:

- mafic volcanic rocks, which are relatively undeformed and unaltered;
- ultramafic and mafic volcanic rocks, which vary from weakly to strongly deformed and altered and contain felsic to intermediate porphyritic intrusions of varying shapes and sizes;
- metasedimentary rocks; the thick footwall sequence of metasedimentary rocks is interpreted to represent the footwall of the DPFZ on the Taylor property.

This general sequence is simplified and in detail and particularly in the mid section, the property geology is much more complex. Within the mid-section of the property, ultramafic and mafic rocks are interlayered, and exhibit varying degrees of deformation

[3] Ferguson S.A., Groen H.A., Haynes, R., 1971 Gold Deposits of Ontario, Part 1; Ontario Department of Mines MRC 13, 315p.
[4] Pyke D. R. and Jensen, L.G. 1976: Preliminary Stratigraphic Interpretation of the Timmins-Kirkland Lake Area, Ontario, Program with Abstracts, Geological Association of Canada, Vol. 1, 71p.

and alteration, sometimes intense, and are complexly intruded by dominantly felsic porphyritic intrusions. This sequence hosts the gold mineralization on the property.

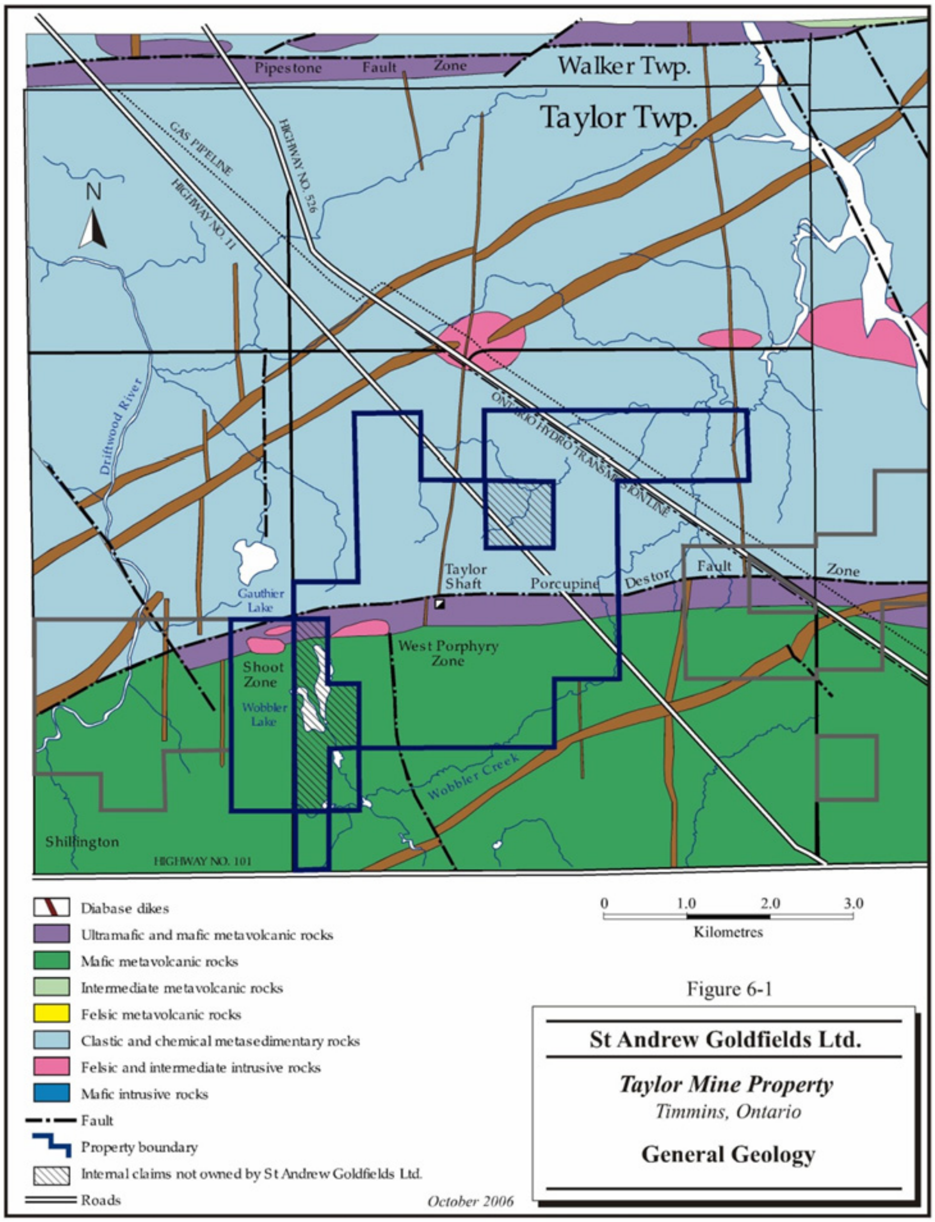

Figure 6-1: Taylor property geological map (after SWRPA, 2006).

6.3 Mineralization

The Taylor Mineralization is in close proximity, within the hanging wall, to the DPFZ. Over a strike length of 2.3 kilometres there are three mineralization zones that have been identified (Figure 6-2). From east to west these are:

- The Shaft Zone, with gold mineralization associated with felsic intrusive rocks.
- The WPZ, a system of stacked lenses, with the gold mineralization associated with felsic intrusive and altered mafic-ultramafic rocks (Green Quartz Carbonate).
- The Shoot Zone, with gold mineralization hosted by argillaceous metasedimentary rocks within a package of green quartz carbonate.

Gold commonly occurs as relatively coarse-sized free gold in quartz, but also occurs as fine particles, which may be intimately associated with sulphides (particularly pyrite and locally, arsenopyrite) both in quartz-carbonate veins or in surrounding altered host rocks. More detailed descriptions of the mineralized zones are given in the following sections.

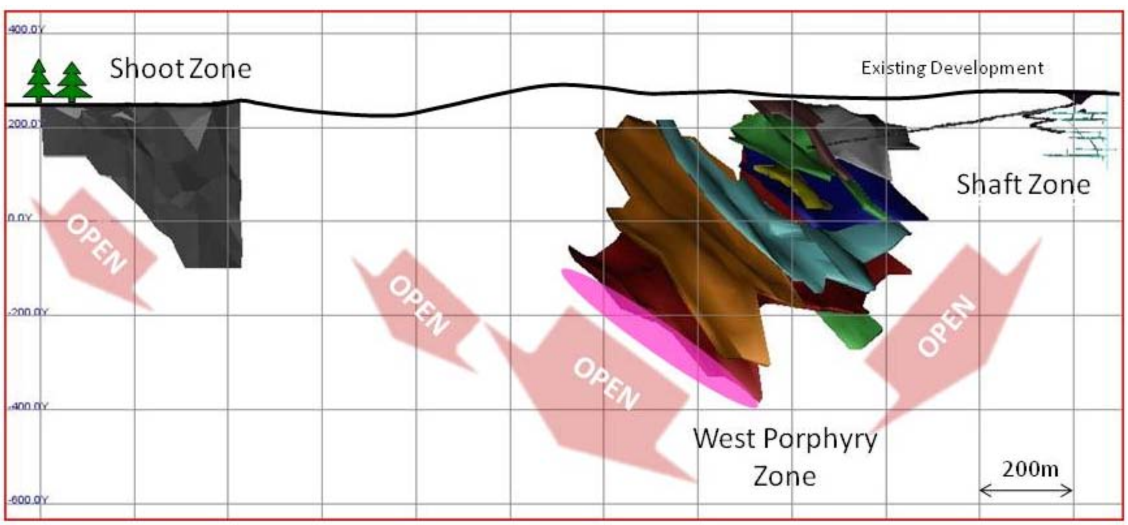

Figure 6-2: Longitudinal section showing the Shaft, Shoot and WPZ.

6.3.1 Shaft Zone

The shaft zone was explored underground in 1986 and 1987 via a 172 m shaft and three levels and again in 2006 via a surface ramp, which gave access to these levels. A large amount of information was collected by diamond drilling and drifting. At the conclusion of the earlier work, two types of gold association where recognized: Moly/Graphite and Pyrite Porphyry.

The Moly/Graphite mineralization comprises of two small lenses extending from surface to a depth of 30 to 40 m. One has a strike length of 60 m and is approximately 9 m thick; the other has a strike length of 18 m and is approximately 11 m thick. Both occur as a zone of quartz veinlets with black molybdenite and graphite fracture filling within a flat lying porphyry dike.

The Pyrite Porphyry mineralization is located 45 m to 120 m below surface and has a strike length of approximately 150 m. Horizontal width varies from 3 m to 18 m and the zone dips 50 degrees to the southeast. The zone is hosted in a brecciated, albite porphyry characterized by 10% to 30% pyrite and, to a lesser extent, red fluorapatite (5%).

In their 2006 report SWRPA concluded that "*the geometry and continuity of the Shaft Zones are complex despite the large amount of drilling and underground exploration that has been carried out. RPA considers that further compilation and interpretation is needed to determine whether a mineral resource can be estimated that meets NI43-101 definition".* This additional work has not been completed; therefore, no resources have been calculated for this zone.

6.3.2 West Porphyry Zone

The WPZ has been interpreted as a series of stacked and en échelon lenses, which contain a locus of deformation, alteration, quartz veining, and gold mineralization (Figure 6-3). For this study a total of nine lenses have been defined. The lenses are stacked with an offset to the south west and a vertical separation of about 50 m. The lenses strike approximately 060° to 070°, dip approximately 30° to 50° to the south and are locally irregular in shape (although follow similar shape patterns from lens to lens). In the lower lenses, there appears to be gold enrichment related to a minor flexure in the footwall of the alteration zone with the underlying ultramafic volcanics. This flexure is approximately parallel to another potential structure that marks the down-dip extent of the concentrated gold mineralization.

Gold mineralization is hosted by several rock types within the WPZ, but primarily within areas of increased proportions of quartz ±carbonate veins, pyrite, strongly carbonate and sericite altered volcanic rocks, and silica ±albite-rich porphyritic intrusions. The drilling completed in 2011 indicated the mineralized lenses crosscut primary lithologies.

Gold mineralization with the WPZ occurs in high grade intercepts as relatively coarse free gold in quartz, which tends to have irregular distribution, and as lower grade intercepts, which are interpreted be resultant from fine-grained gold, perhaps more evenly distributed and associated with disseminated pyrite. The shape of each gold-bearing lens was very broadly interpreted based on structure, alteration, sulphide mineralization and gold grade. Rock type was used to a lesser extent, as it was determined that the gold mineralization is more related to geologic structures that transect lithological boundaries.

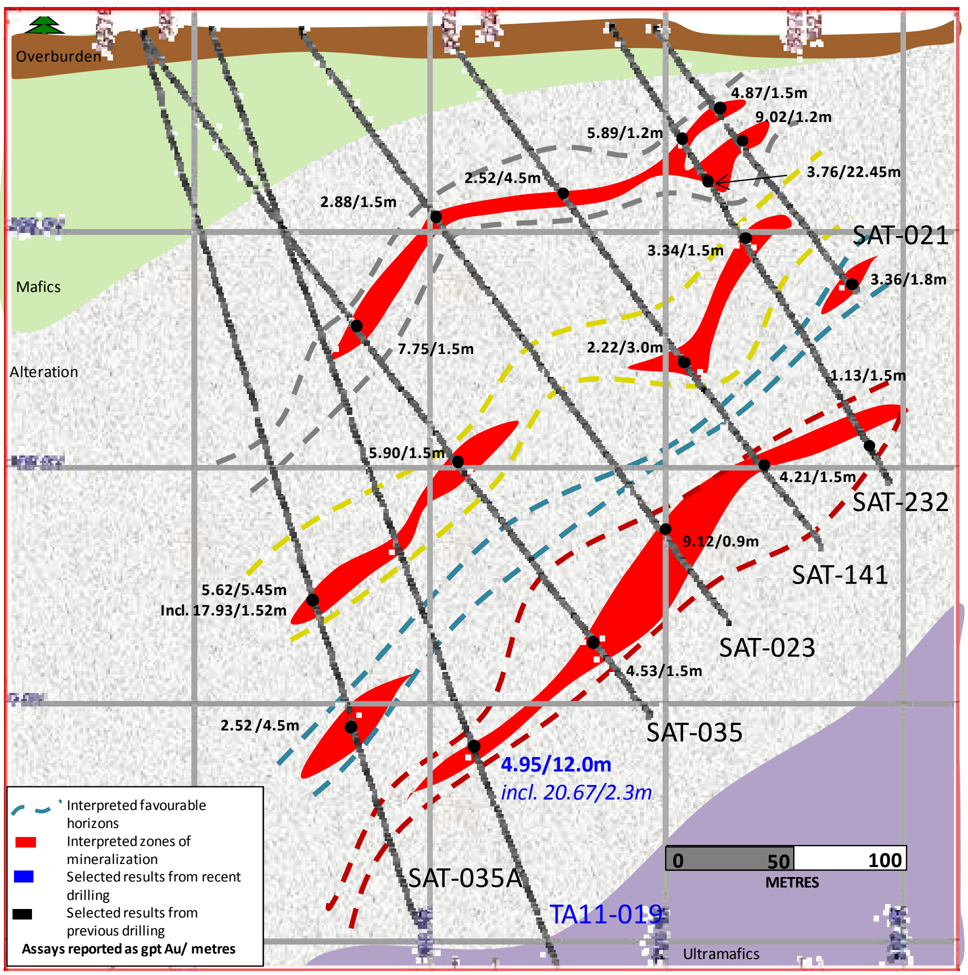

Figure 6-3: Cross section (looking west) of the WPZ.

6.3.3 Shoot Zone

The Shoot Zone consists of a single, regularly shaped, moderately dipping tabular body which strikes approximately 060° to 070° and dips 40° to 50° to the south (Figure 6-4). The zone has a strike length of about 350 m and has been traced from surface to a vertical depth of 350 m. The zone is controlled by a metasedimentary unit which includes greywacke and argillite. This unit hosts gold bearing quartz carbonate veins. The unit is thickest near surface averaging about 13 m and thins down dip and to the west.

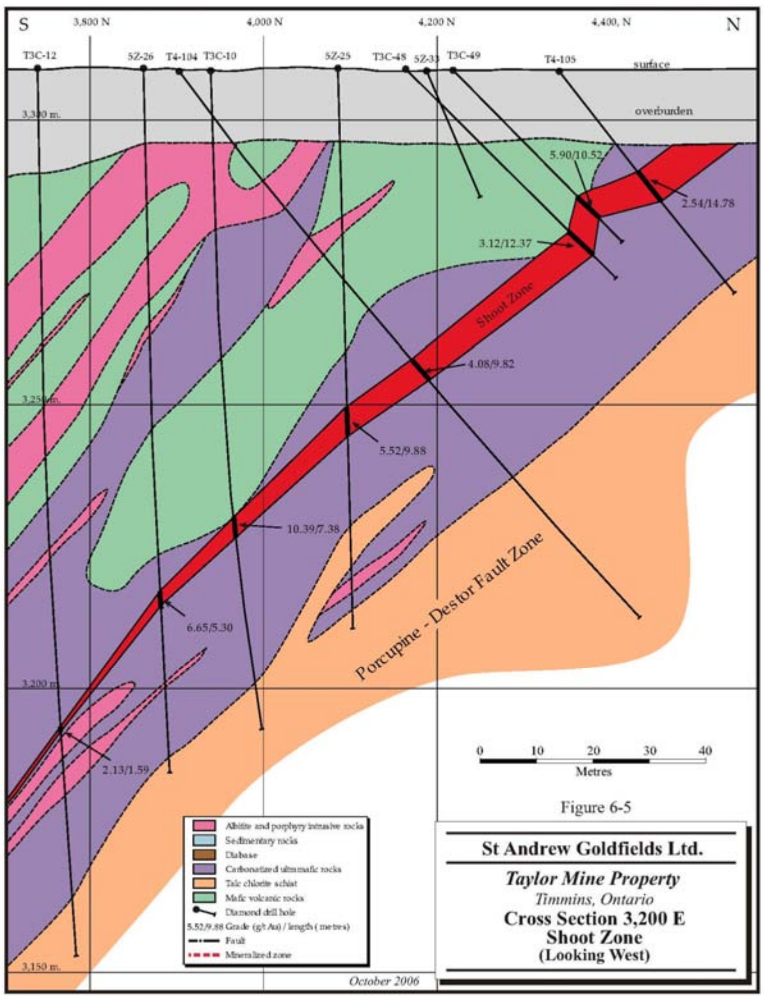

Figure 6-4: Cross section (looking west) of the Shoot Zone.

7.0 DEPOSIT TYPE

Numerous gold deposits occur in the vicinity of the DPFZ and related structures such as the Pipestone Fault. These include the major mines of the Timmins camp (Dome, Hollinger, McIntyre, and Pamour). A number of gold deposits have been discovered in more recent years, including the Holt-McDermott Mine, Holloway Mine, Owl Creek Mine, Bell Creek Mine, Hoyle Pond Mine, Aquarius Mine, Maude Lake Deposit, Glimmer (Black Fox) Mine, Stroud Deposit, Fenn-Gib Deposit, Ludgate Deposit, Jonpol Mine and a number of other prospects.

Some of the DPFZ gold deposits extend from surface to over 1,000 m below surface, and some are blind deposits, in that they do not reach bedrock surface. The top of the Holloway deposit (i.e. the Lightning Zone), for example, is over 240 m below surface.

The following description of potential gold deposit types on the SAS Timmins area claims is from Reid (2003). Deposit types and exploration models can generally be characterized as one of three main types, although they tend to merge with each other at times. The deposit types may have more to do with the different host rocks than a genetic difference. Proximity to the main break(s), associated splays, presence of hydrothermal alteration, Timiskaming sediments or high level porphyries are common to all. The three main types are as follows:

- **Green Carbonate Hosted**: Nighthawk Lake, Aquarius, Stock, West Porphyry, and Black Fox all fall into this classification. Gold is generally present as free gold in quartz veins or with disseminated sulphides associated with small intrusive rocks or albitic alteration in completely carbonate altered ultramafic flows. Carbonate alteration is up to 200 m wide and can be traced for thousands of metres discontinuously on strike. The gold is often in cross cutting or conformable features. Timiskaming conglomerates are often proximal or part of the package.

- **Felsic Intrusive Related**: Ronnoco, Pominex, parts of the Taylor Shaft and Hislop are examples of this type. The intrusive rocks vary from feldspar (plus or minus quartz) porphyry in the west to more syenitic in the east. Mineralization is characterized by both cross cutting to stockwork quartz veins, disseminated sulphides and/or contact skarns or hornfels, depending on host rock. Carbonate alteration is still quite common in the host rocks with silica, sericite, and hematite more within the intrusive.

- **Mafic Volcanic Hosted:** Holloway, Holt and Hoyle Pond are examples. Ubiquitous carbonate alteration with iron carbonate, albite, silicification and sericite more proximal to ore. Quartz veins and/or albitized variolitic mafic flows are often central to the zone and often found near the mafic/ultramafic contact.

8.0 EXPLORATION

A number of geophysical and geochemical surveys have been carried out over the Taylor property over the past few decades. Geological mapping is hampered by the lack of exposures in the general area, and bedrock geology relies on interpretation of drill hole information and geophysical surveys. In 1998, Realsection IP geophysical surveys and enzyme leach and sodium pyrophosphate geochemical surveys were conducted on the property.

In 2011, spectral analysis of approximately 3,000 m of drill core was conducted by Photonic Knowledge in an attempt to better define alteration and mineralization patterns in the West Porphyry Zone. The drill core was chosen to be representative of the typical alteration and mineralization assemblages in the WPZ; however, the complexity of the alteration hampered calibration of the spectral data with the alteration types. Consequently, this data was not used when compiling the current resource estimate.

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9.0 DRILLING

Diamond drilling has taken place from surface and underground in various campaigns since Hollinger Gold Mines Ltd's first exploration on the property in 1962.

There are 833 surface and underground diamond drill holes present in the drill hole database for the Taylor project. Drilling done to date is summarized in Table 9-1. These have been compiled from a variety of sources into one secure database. The data has been standardized to:

- UTM NAD 83 metric co-ordinates;
- All Assays reported as g/t Au;
- Common Lithology Legend

Company	Year	Collar	Series	# of Holes	Metres cored	Target
		Surface	HT	86	20,098	SZ-Expl
		Surface	NAT	15	2,768	Expl
		Surface	QS	35	8,739	Expl
		Surface	SAT	247	108,331	WPZ-SZ- Shaft Z- Expl
		Surface	SZ	47	9,858	SZ
		Surface	T1A	1	269	Expl
		Surface	T2B	37	6,662	Shaft Z-WPZ-Expl
		Surface	T3C	23	3,741	SZ - Expl
		Surface	T4	2	713	WPZ
		Surface	T4	32	5,319	SZ
		Surface	T5	1	369	Expl
		Surface	T84	6	1,309	Expl
		Surface	T86	8	1,545	Expl
		Surface	T98	5	1,724	Expl
		Surface	T99	3	797	Expl
		Underground	T1	69	4,253	Shaft Z
		Underground	T1S	52	2,228	Shaft Z
		Underground	T2	26	944	Shaft Z
		Underground	T2S	29	723	Shaft Z
		Underground	T3	48	2,998	Shaft Z
		Underground	T3S	29	989	Shaft Z
		Surface	TC	14	433	Expl
SAS	2006	Surface	TA06-	11	1,789	WPZ
SAS	2010	Surface	TA10-	17	7,528	WPZ-SZ
SAS	2011	Surface	TA11-	35	17,780	WPZ
SAS	2011	Surface	TA11-	5	1463	SZ
Total				883	213,370	

Table 9-1: Summary of drilling on the Taylor Project.

Until 2006, the surface and underground drill collar locations where surveyed by theodolite and distomat to the local imperial mine grid systems. Some of the older holes are located by cut grid co-ordinates, which have since been converted to imperial mine grid coordinates. Starting in 2006, the drill co-ordinates where converted to the UTM NAD 83 systems. Many of the older holes were mathematically converted and some of the available collars were resurveyed using Topcon GR3 GPS Survey Instrument. Up to and including drill hole number TA11-019, collar locations were surveyed by an independent contractor using a Topcon GR3 GPS Survey Instrument. The remaining holes have been located using a hand held GPS survey instrument.

Drilling since 2006 has had the down hole surveys done by FLEXIT, a down hole survey instrument that measures deviation and records it digitally. Down hole deviation in historic holes where determined by Tropari instruments, Sperry Sun Single Shot and/or by acid tests which give a dip measurement only.

The drill logs provide sufficient description and recognition of the lithology, alteration, geological structures, and mineralization to correlate mineralization between holes and sections. The mineral resource estimate is based entirely on diamond drilling data. The diamond drill spacing varies on a property wide scale. Drill spacing along the plane of the mineralization by zone is listed in Table 9-2.

ZONE	Diamond Drill Spacing (approx)	
	Indicated	Inferred
Shoot	30 m x 30 m	30 m x 60 m
WPZ		
1003		60 m x 100 m
1004-1	30 m x 30 m	70 m x 70 m
1004-2	30 m x 30 m	35 m x 35 m
1006	38 m x 30 m	60 m x 30 m
1008-1	30 m x 30 m	35 m x 35 m
1008-2	39 m x 30 m	39 m x 45 m
1009	30 m x 35 m	30 m x 60 m
1010	30 m x 30 m	100 m x50 m
1011		40 m x 40 m

Table 9-2: Summary of drill spacing on the Taylor Project.

The following older series of drill holes are stored at the Taylor site:

- SAT-***,GK-***, H99-***,H98-***, H97-***
- T98-***, T86-***, T84-***
- T1-***, T2-***, T3-***, T15-***, T25-***

Except as noted the core from drill programs since 2006 is stored at the Taylor site. Core from TA10-001 to 012 are stored in racks at the Matheson Exploration Office.

In the QP's opinion, there are no drilling, sampling or recovery factors that could materially impact the accuracy and reliability of the results.

10.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

10.1 Sampling Method and Analytical Techniques

Drill core sampling conducted by SAS during its drill campaigns followed a sampling protocol. For the 2010 drill campaign, this protocol was documented[5,6] and geologists and technicians were trained on using the protocol.

Briefly described, samples are to be laid out based on geologic contacts and are to be a minimum 0.3 m and maximum of 1.5 m in length. Samples are to be taken on the up hole and down hole side (i.e. “shoulders”) of intervals were gold mineralization is prospective. Gaps of seven metres, or less, between prospective intervals are to be sampled. Each sample is assigned a unique sample number, preferable six digits long, as recorded on pre-printed sample tag books. Sample data are entered in the DHLogger program as the samples are being laid out and this information is confirmed by the logging geologist prior to placing the core in the queue for cutting. During sampling, one portion of each tag is placed in each numbered sample bag, while another portion remains in the core box at the end of each sample, and another portion remains in the tag book which contains all records for that sample.

10.2 Sample Preparation, Analysis and Security

Industry standard analytical techniques are utilized by third party labs in the performance of gold analyses. SAS routinely analyzes its samples via gold Fire Assay with atomic absorption (AA) or induction coupled plasma (ICP) finish. Gravimetric finish may be used where an initial result is greater than 3 gpt Au. From the start of drilling on the Taylor project in 2010 to drill hole TA10-015, 30 gram assay aliquots were used by labs. Commencing with TA10-016, 50 gram assay aliquots have been used and are recommended to be continued to be used. “Screened Metallic” analysis is requested where visible gold has been identified in core, and if fire assay results are returning values greater than 5 gpt Au. Check analyses are performed over selected intervals using fire assay and screened metallic procedures.

All drill core samples are analyzed by independent laboratories, which are either certified or are commonly used in the mining industry for such analyses. For the majority of the spring 2010 drill program, drill core samples were analysed by SGS Mineral Services Ltd., which is an accredited lab. For TA10-001, 002, and 003, Swastika lab was used, which is not an accredited lab but has a long history of use by

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[5] SAS, “Technical Procedure for Core Sampling”, internal document, March 22, 2010.
[6] SAS, “Technical Procedure and Guidelines for Core Cutting and Handling”, internal document, July 13, 2010.

the mining industry. For the fall 2010 drill program, drill core analysis has been completed by ALS and SGS labs, which are certified labs, and also by Cattarello Assayers Inc., which is not an accredited lab. During the 2011 drilling program both ALS Minerals and Accurassay Laboratories where used for gold analysis. Accurassay Laboratories is an accredited lab.

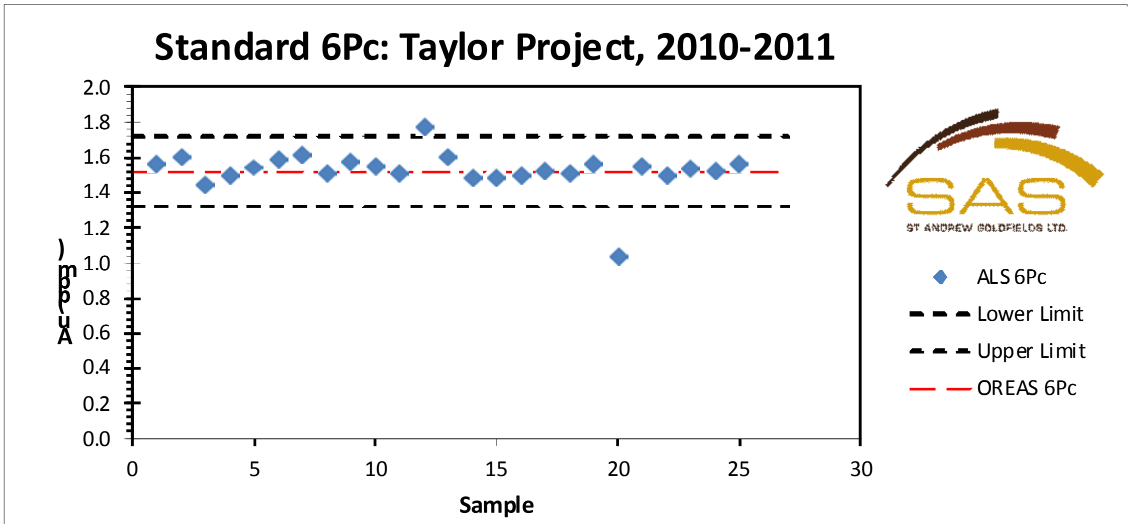
Quality assurance and quality control (QA-QC) procedures are in place for all drill core sampling conducted by SAS and also by the analytical labs during the analytical process. A significant QA-QC measure undertaken by SAS includes the insertion of standard reference samples and blank sample materials within sampled drill core intervals, as can be referenced in the above cited SAS Technical Procedures. Standard reference materials are purchased in 60 gram foil packets from a qualified third party vendor (ASL), which has been subject to auditing by other labs. The distribution of assays for all the sample standards used during the 2010-2011 drilling program is shown in Figure 10-1 to Figure 10-8. The results are split by analytical laboratory and sorted by date analyzed. The red dashed line shows the expected grade, the two black dashed lines indicate performance gates (3 standard deviations, as reported in the OREAS documentation). Blank materials are obtained from diabase dike intersected in Taylor drill holes, which are confirmed to contain no or insignificant gold. The distribution of assays for all the blanks used during the 2010-2011 drilling program is shown in Figure 10-9 to Figure 10-11. The results are split by analytical laboratory and sorted by date analyzed. The black dashed line shows the cut-off grade, above which the blank is considered to have “failed”. The standards and blanks are inserted at a rate of 1 in 20 drill core samples at irregular intervals in the drill core sample sequence and are ‘blind’ to the analytical lab. Results reported by the lab for submitted standards are compared against expected values. If the returned value lies within allowed limits (typically 3 standard deviations of the mean value for that standard), the standard, along with the “lot” of samples analyzed along with the standard is consider to “pass”. Should a result lie outside allowed limits, the lot of samples analyzed along with that standard is considered to have “failed” and that lot may be re-analyzed if considered to be in the sequence of rocks where significant gold content may be expected (i.e. not all failed lots may be re-analyzed). Similarly for blanks, lots where blank samples analysed at values greater than background levels may be re-analyzed if deemed warranted.

Trends for standard and blank results are also assessed on a periodic basis with respect to any high or low bias which may be apparent at any particular lab.

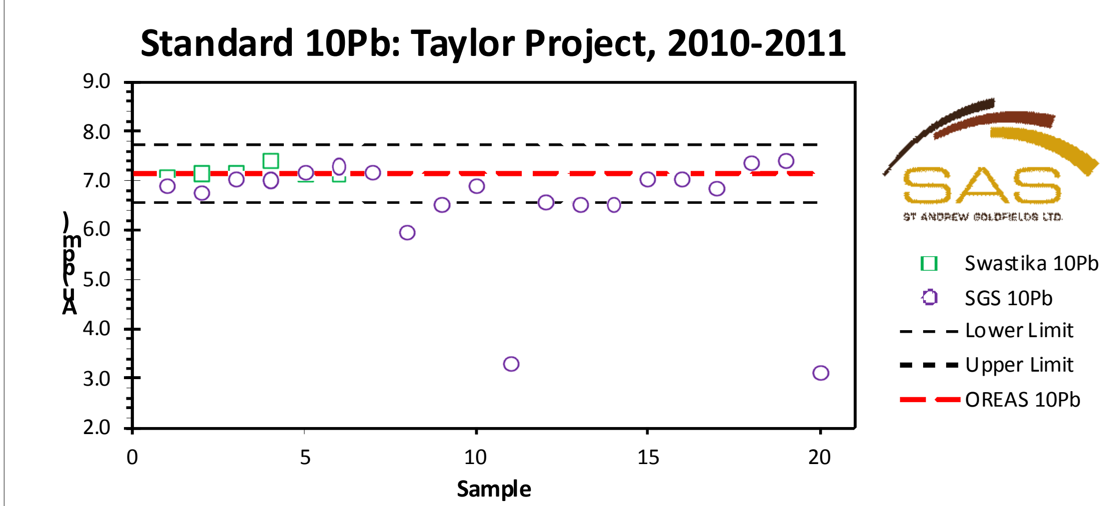
SAS records and tracks its shipments of drill core samples to analytical labs using Chain of Custody documents. A Chain of Custody document is prepared for each sample shipment, which records the sample numbers shipped, number of samples, who packed the samples, and security seal information. SAS employs plastic tamperproof security seals to ensure no tampering of any sample has taken place between preparation and packing of each sample by the sampling technician, and receipt at the analytical lab. Security seals are embossed with a unique number and the shipping bag number and corresponding security seal number is recorded on the Chain of Custody document. The Chain of Custody document is checked by the lab upon receipt of each shipment, and if any discrepancy is noted between the information recorded on the Chain of Custody and the shipment, SAS is informed and the samples in that shipment are not proceeded with until the issue is resolved.

Each third party analytical lab employs its own QA-QC procedures involving the use of standards and blank materials to clean out equipment between samples. These procedures can be referenced in each lab's published QA-QC procedures.

Upper and lower limits are "3 standard deviations" above and below reported average Au concentration. Major discrepancies from expected values were typically human errors during sampling; however, significant samples associated with these standards were re-analyzed and Au values confirmed.

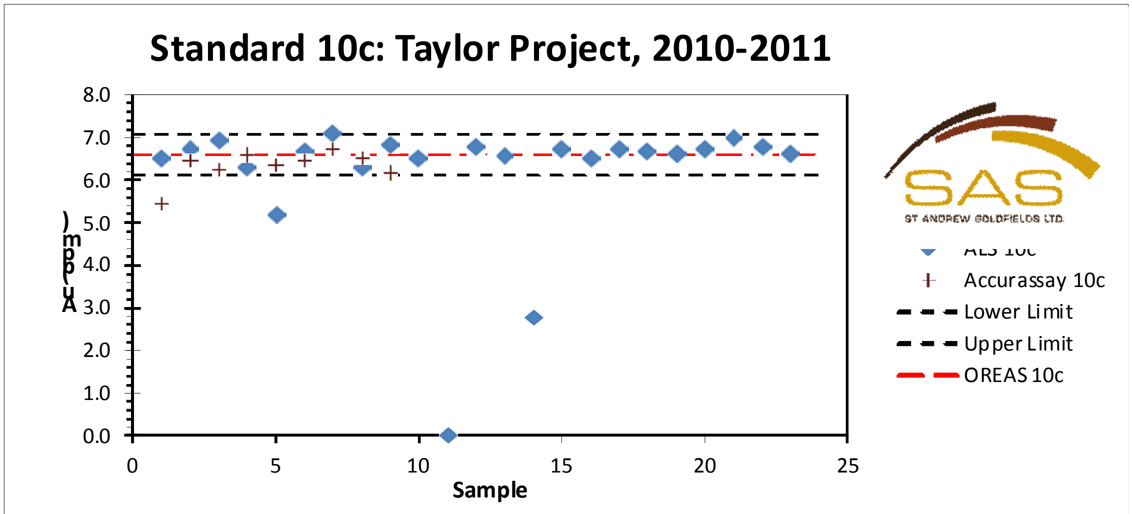


Description of Material	CRM Code	Unit Size	Au Concentration (ppm)	95% Confidence Interval (ppm)	95% Tolerance Interval (ppm)	LOW (3σ)	HIGH (3σ)
Metasediment	OREAS 6Pc	60g	1.52	± 0.04	± 0.08	1.32	1.72

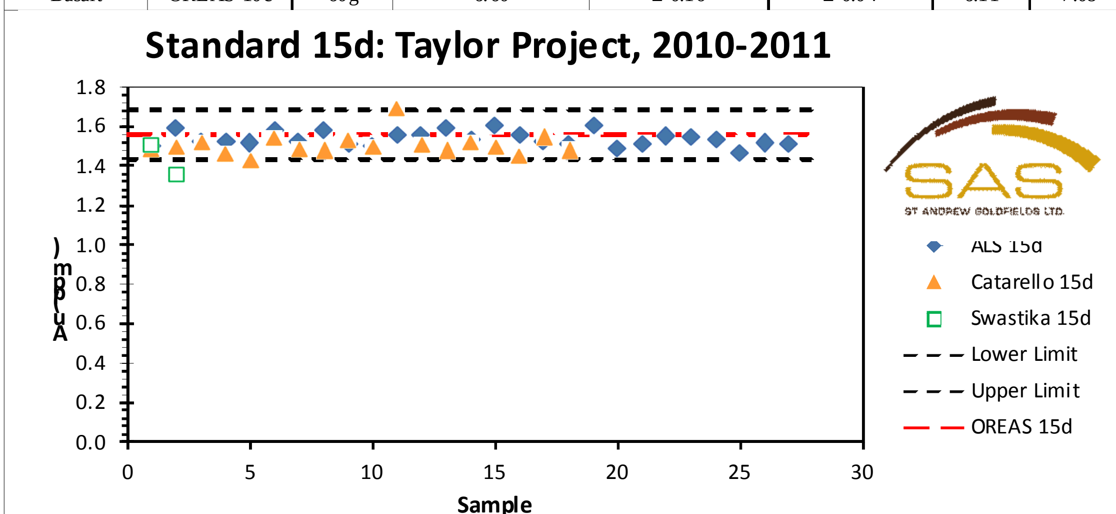


Description of Material	CRM Code	Unit Size	Au Concentration (ppm)	95% Confidence Interval (ppm)	95% Tolerance Interval (ppm)	LOW (3σ)	HIGH (3σ)
Basalt	OREAS 10Pb	60g	7.15	± 0.11	± 0.07	6.57	7.72

Figure 10-1: Analysis of OREAS Standards 6Pc and 10Pb.

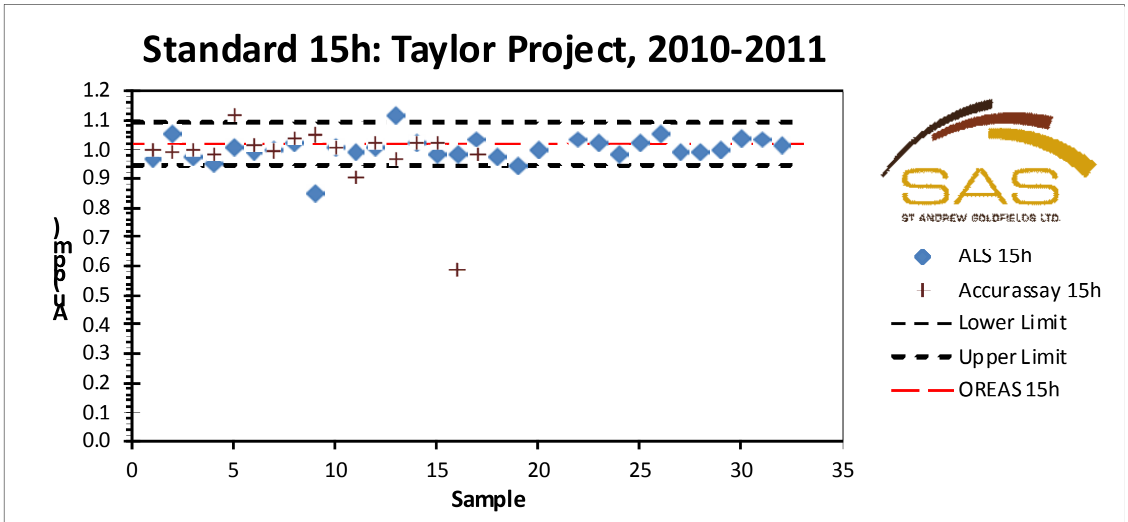


Description of Material	CRM Code	Unit Size	Au Concentration (ppm)	95% Confidence Interval (ppm)	95% Tolerance Interval (ppm)	LOW (3σ)	HIGH (3σ)
Basalt	OREAS 10c	60g	6.60	± 0.16	± 0.04	6.11	7.08

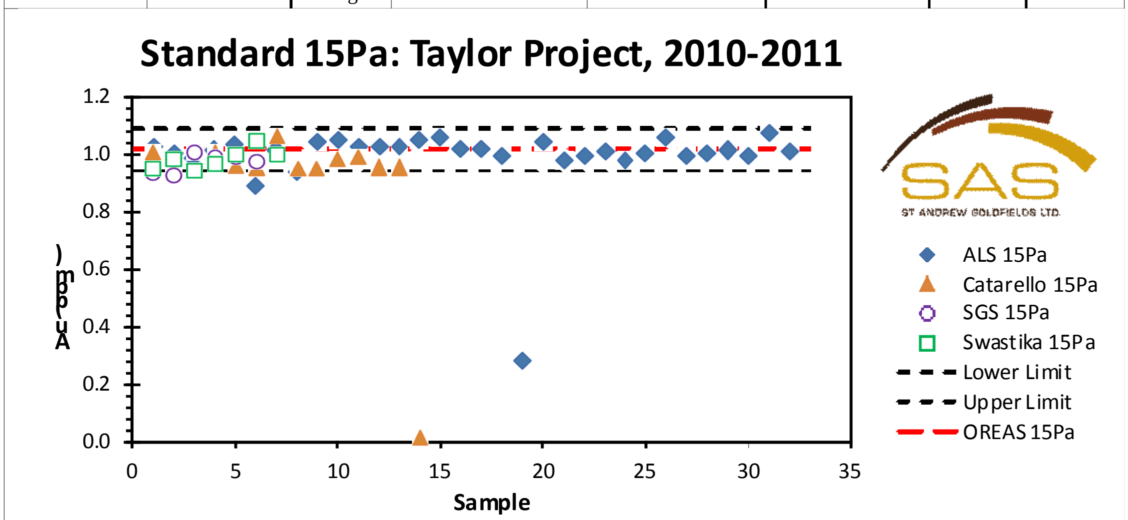


Description of Material	CRM Code	Unit Size	Au Concentration (ppm)	95% Confidence Interval (ppm)	95% Tolerance Interval (ppm)	LOW (3σ)	HIGH (3σ)
Basalt	OREAS 15d	60g	6.60	± 0.02	± 0.01	1.433	1.685

Figure 10-2: Analysis of OREAS Standards 10c and 15d.

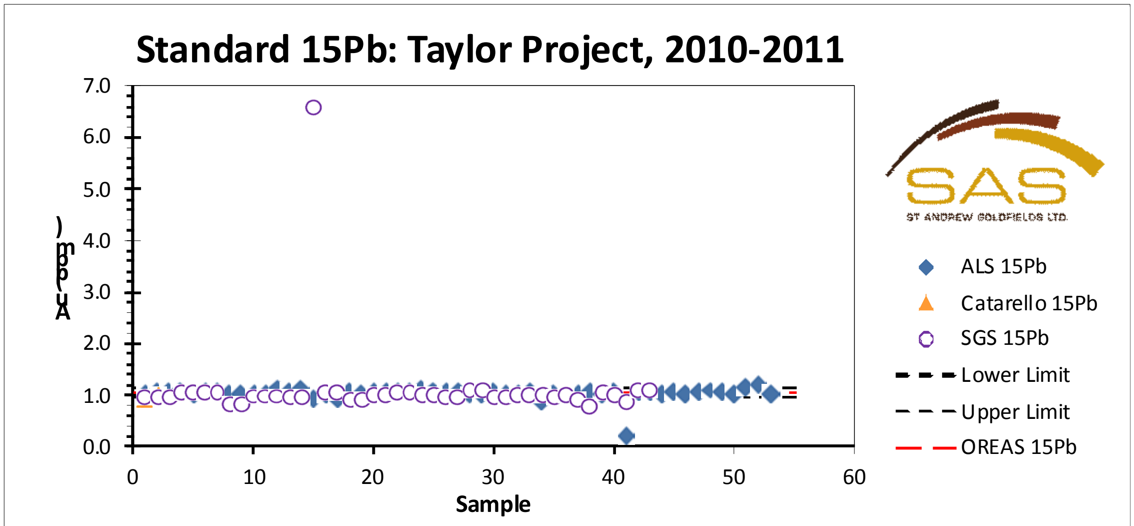


Description of Material	CRM Code	Unit Size	Au Concentration (ppm)	95% Confidence Interval (ppm)	95% Tolerance Interval (ppm)	LOW (3σ)	HIGH (3σ)
Basalt	OREAS 15h	60g	1.019	± 0.007	± 0.005	0.945	1.093

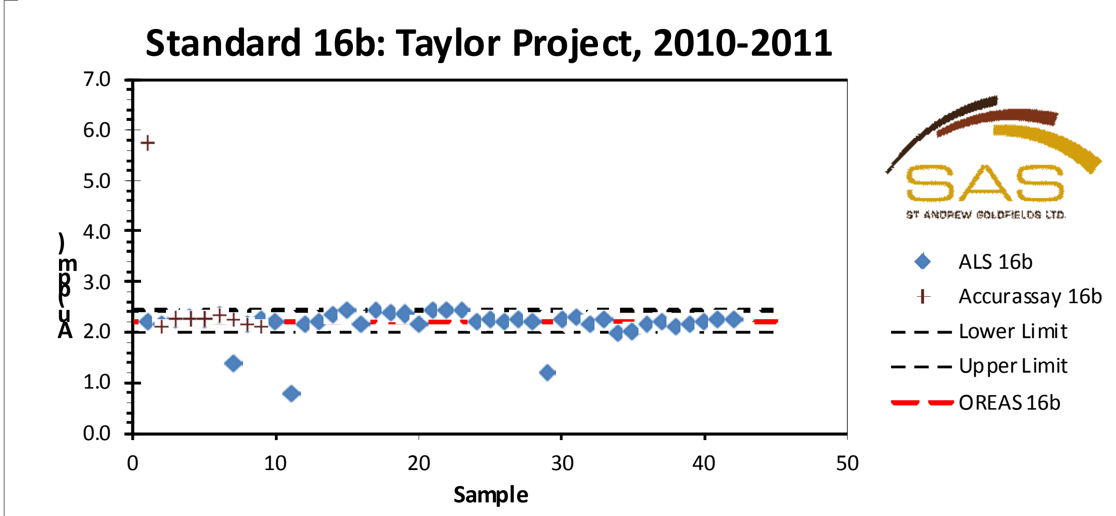


Description of Material	CRM Code	Unit Size	Au Concentration (ppm)	95% Confidence Interval (ppm)	95% Tolerance Interval (ppm)	LOW (3σ)	HIGH (3σ)
Basalt	OREAS 15Pa	60g	1.02	± 0.02	± 0.03	0.94	1.09

Figure 10-3: Analysis of OREAS Standards 15h and 15Pa.

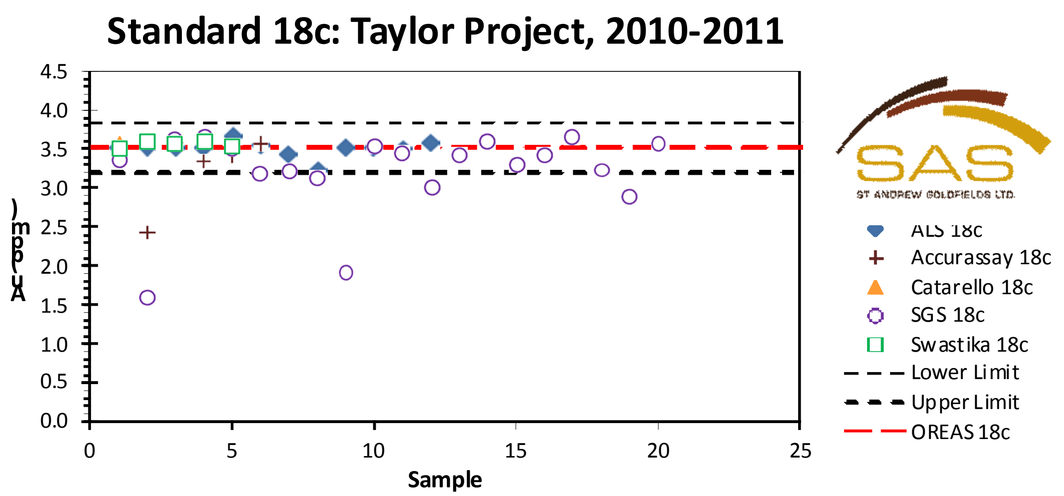


Description of Material	CRM Code	Unit Size	Au Concentration (ppm)	95% Confidence Interval (ppm)	95% Tolerance Interval (ppm)	LOW (3σ)	HIGH (3σ)
Basalt	OREAS 15Pb	60g	1.06	± 0.01	± 0.02	0.97	1.14

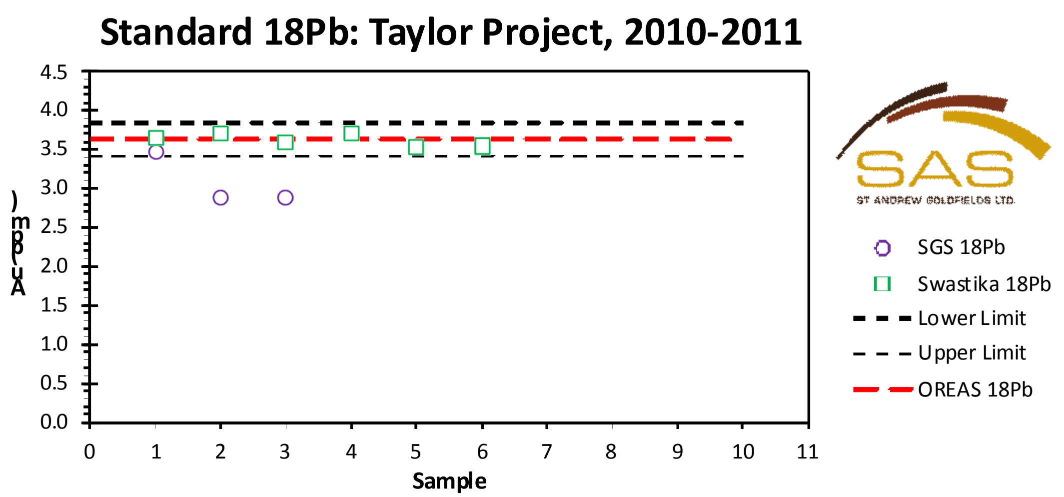


Description of Material	CRM Code	Unit Size	Au Concentration (ppm)	95% Confidence Interval (ppm)	95% Tolerance Interval (ppm)	LOW (3σ)	HIGH (3σ)
Basalt	OREAS 16b	60g	2.21	± 0.03	± 0.01	1.99	2.43

Figure 10-4: Analysis of OREAS Standards 15Pb and 16b.

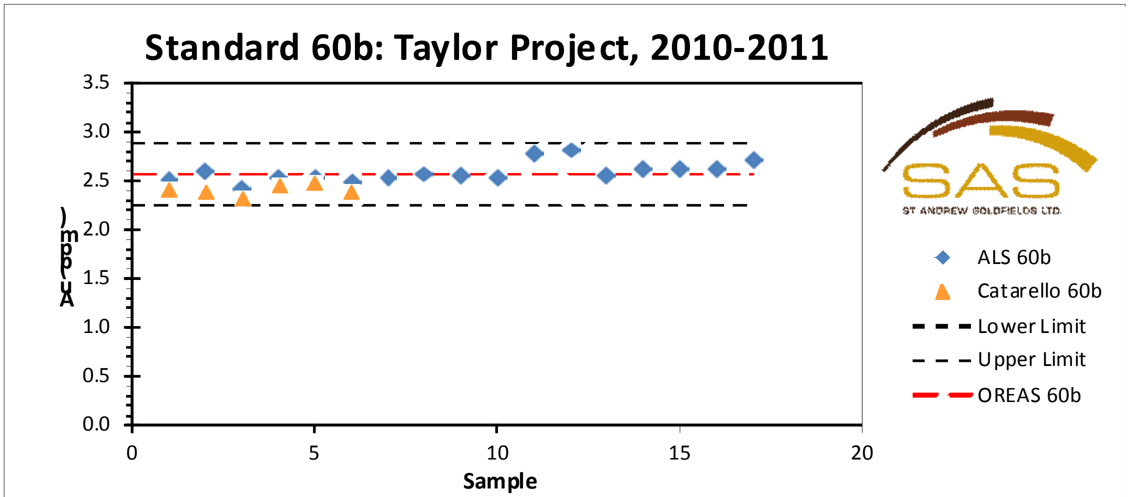


Description of Material	CRM Code	Unit Size	Au Concentration (ppm)	95% Confidence Interval (ppm)	95% Tolerance Interval (ppm)	LOW (3σ)	HIGH (3σ)
Basalt	OREAS 18c	60g	3.52	± 0.05	± 0.01	3.20	3.84

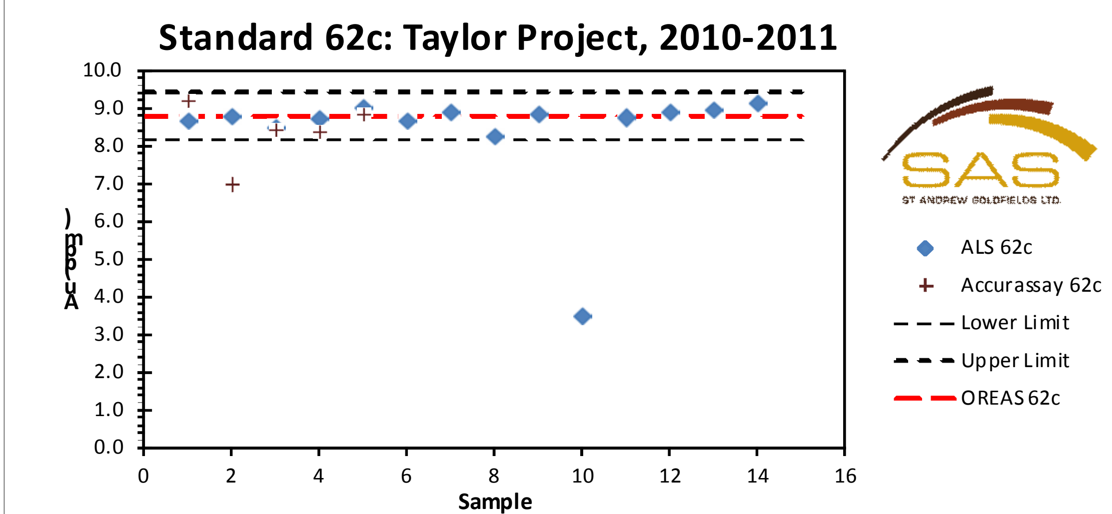


Description of Material	CRM Code	Unit Size	Au Concentration (ppm)	95% Confidence Interval (ppm)	95% Tolerance Interval (ppm)	LOW (3σ)	HIGH (3σ)
Basalt	OREAS 18Pb	60g	3.63	± 0.03	± 0.01	3.42	3.84

Figure 10-5: Analysis of OREAS Standards 18c and 18Pb.

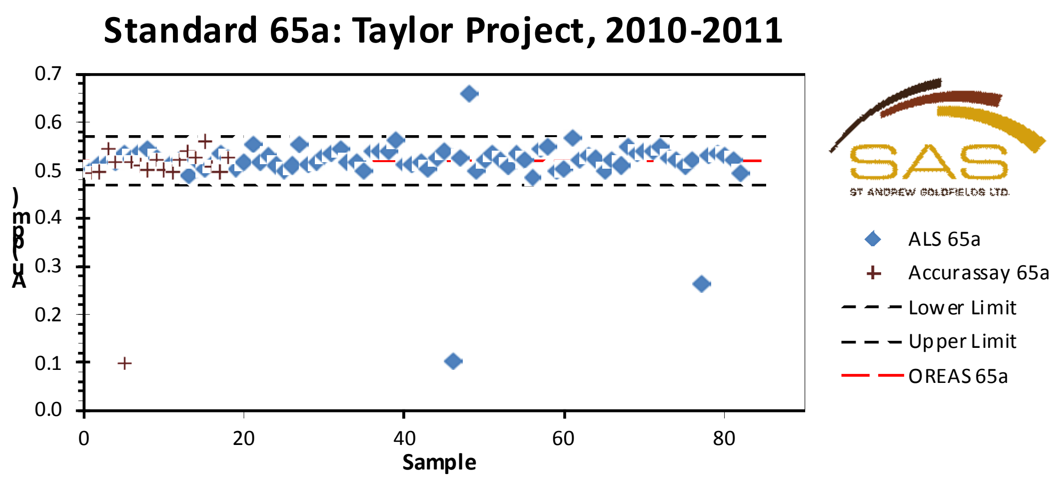


Description of Material	CRM Code	Unit Size	Au Concentration (ppm)	95% Confidence Interval (ppm)	95% Tolerance Interval (ppm)	LOW (3σ)	HIGH (3σ)
Meta-andesite	OREAS 60b	60g	2.57	± 0.04	± 0.01	2.25	2.89

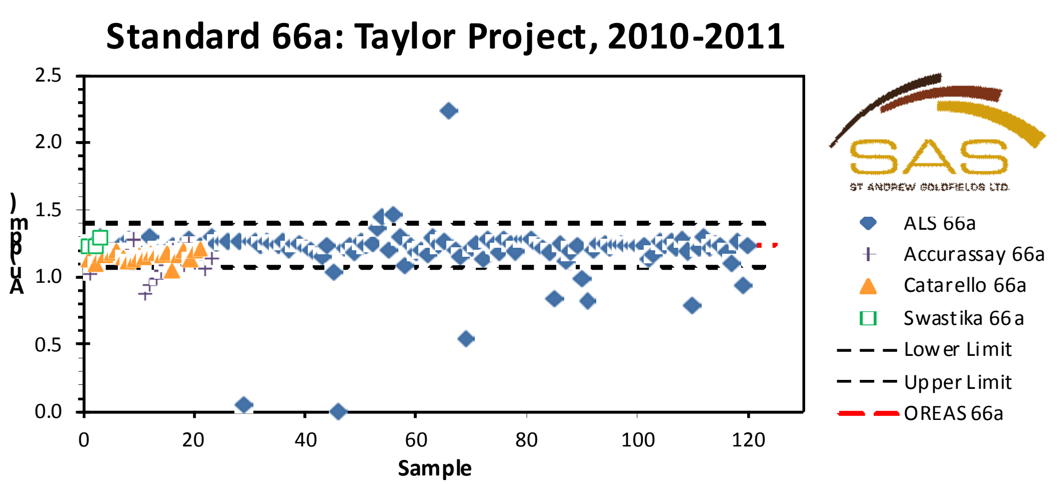


Description of Material	CRM Code	Unit Size	Au Concentration (ppm)	95% Confidence Interval (ppm)	95% Tolerance Interval (ppm)	LOW (3σ)	HIGH (3σ)
Andesite	OREAS 62c	60g	8.79	± 0.09	± 0.013	8.15	9.42

Figure 10-6: Analysis of OREAS Standards 60b and 62c.

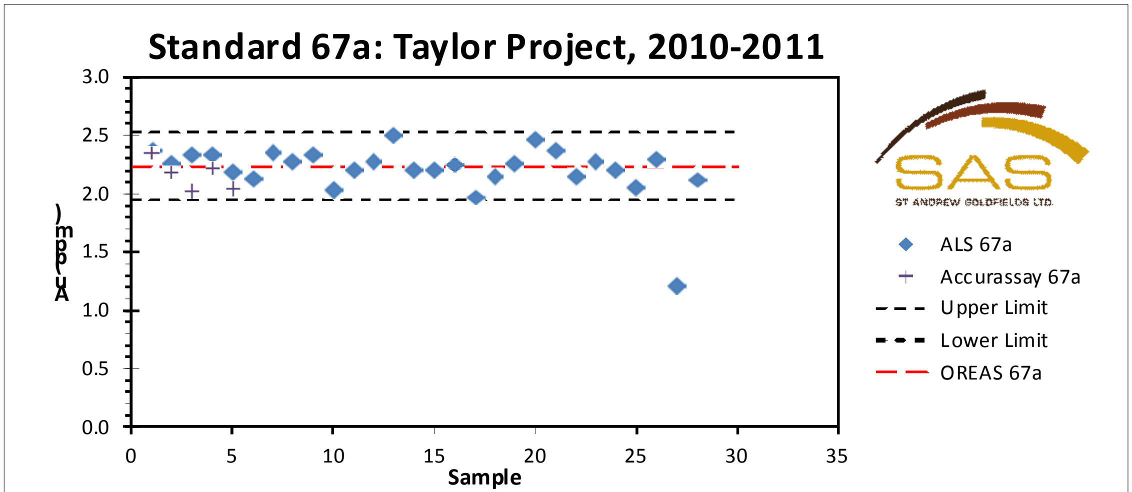


Description of Material	CRM Code	Unit Size	Au Concentration (ppm)	95% Confidence Interval (ppm)	95% Tolerance Interval (ppm)	LOW (3σ)	HIGH (3σ)
Basalt	OREAS 65a	60g	0.520	± 0.08	± 0.003	0.469	0.571

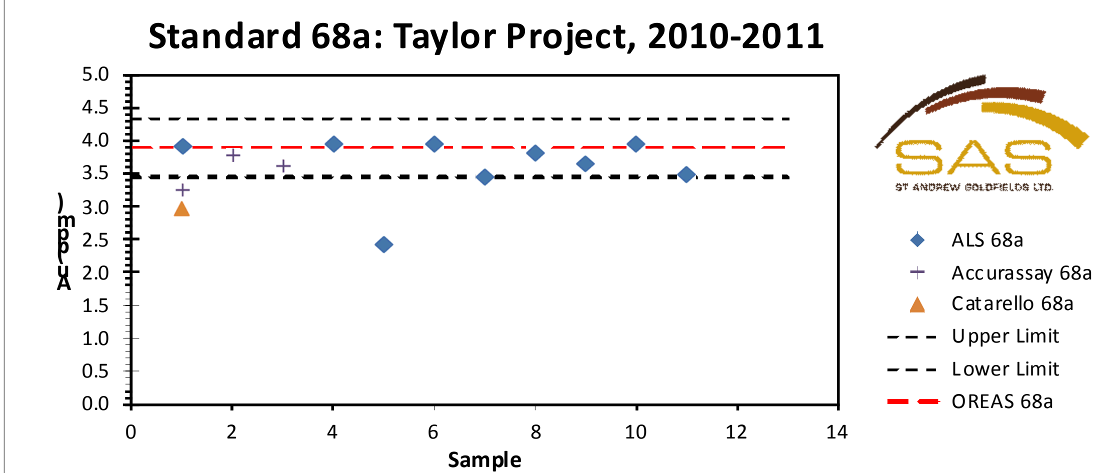


Description of Material	CRM Code	Unit Size	Au Concentration (ppm)	95% Confidence Interval (ppm)	95% Tolerance Interval (ppm)	LOW (3σ)	HIGH (3σ)
Basalt	OREAS 66a	60g	1.237	± 0.026	± 0.007	1.075	1.399

Figure 10-7: Analysis of OREAS Standards 65a and 66a.

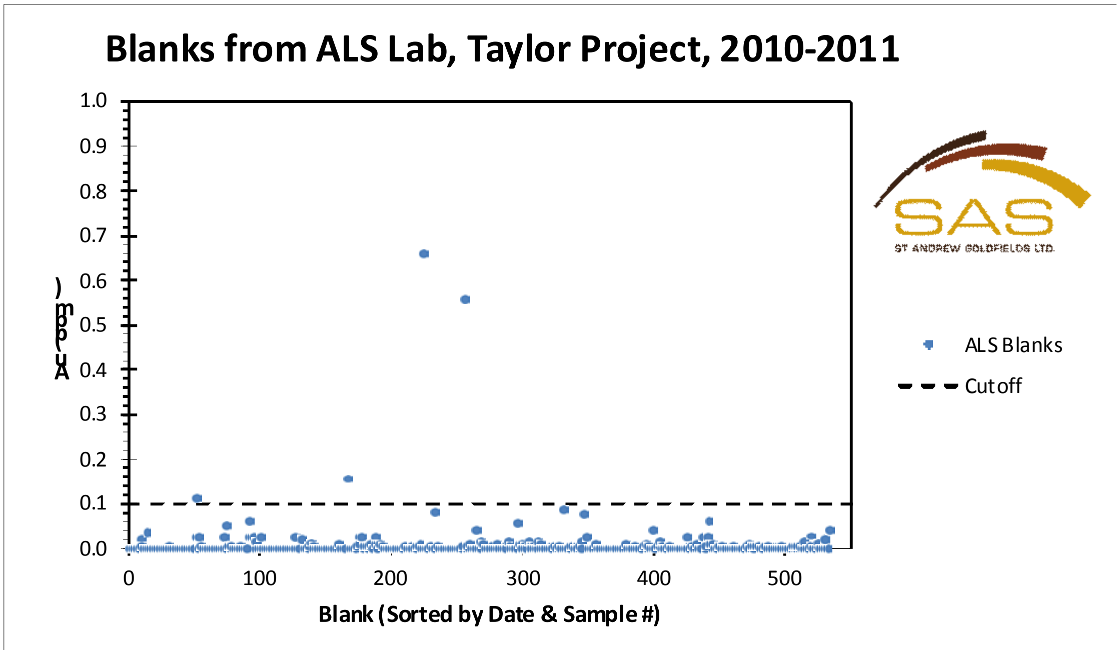


Description of Material	CRM Code	Unit Size	Au Concentration (ppm)	95% Confidence Interval (ppm)	95% Tolerance Interval (ppm)	LOW (3σ)	HIGH (3σ)
Basalt	OREAS 67a	60g	2.238	± 0.044	± 0.013	1.950	2.526



Description of Material	CRM Code	Unit Size	Au Concentration (ppm)	95% Confidence Interval (ppm)	95% Tolerance Interval (ppm)	LOW (3σ)	HIGH (3σ)
Basalt	OREAS 68a	60g	3.89	± 0.06	± 0.021	3.45	4.33

Figure 10-8: Analysis of OREAS Standards 67a and 68a.



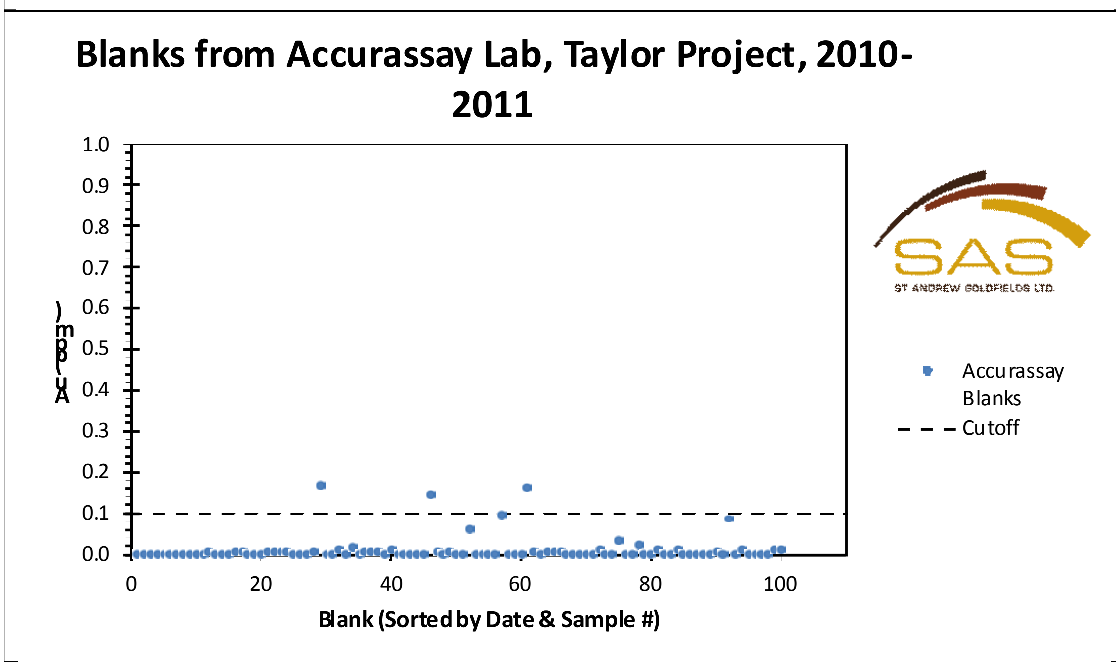
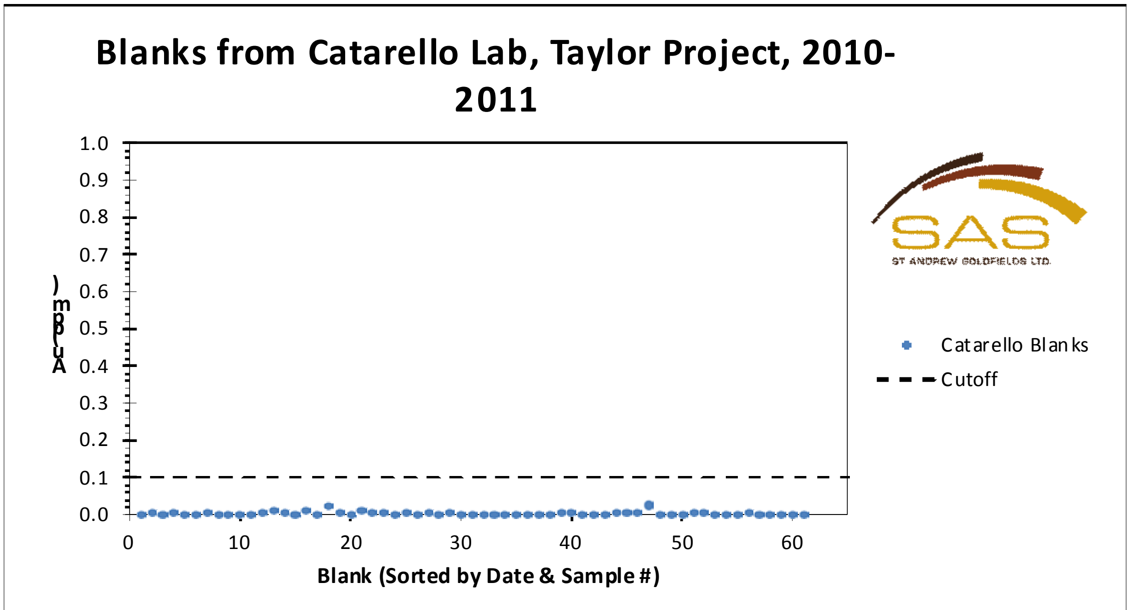


Figure 10-9: Analysis of blank material by ALS and Accurassay Labs.



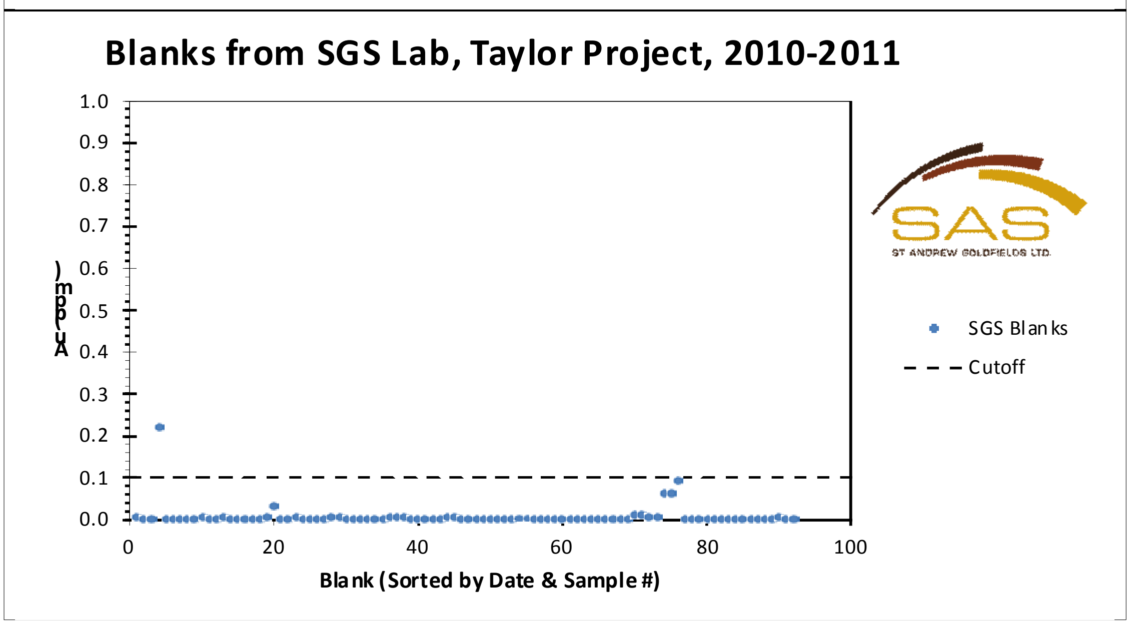


Figure 10-10: Analysis of blank material by Catarello and SGS Labs.

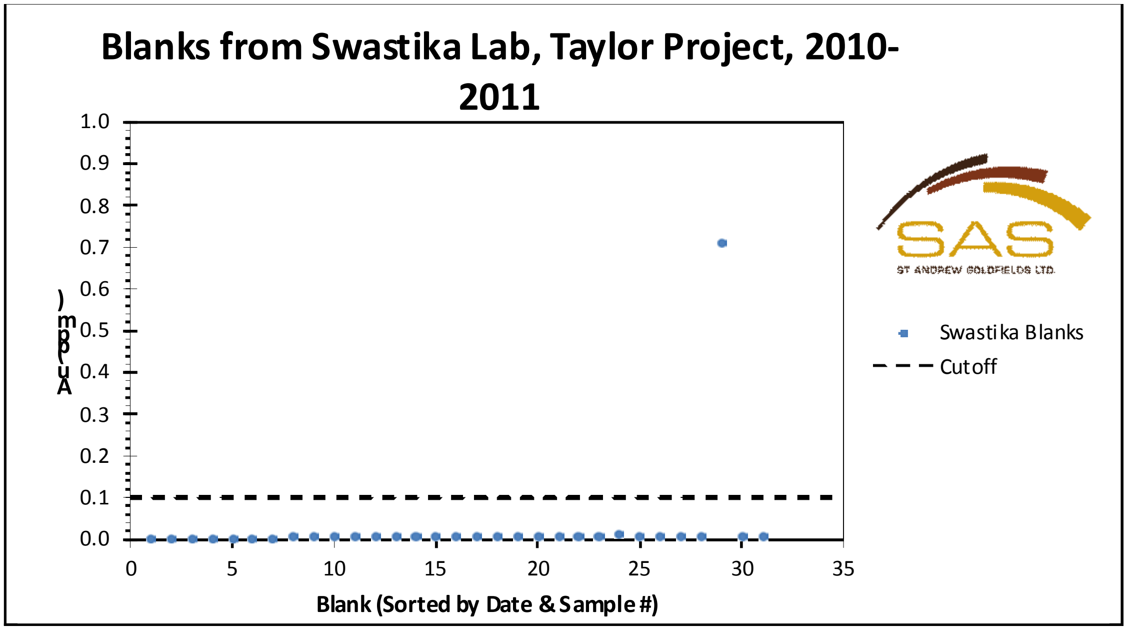


Figure 10-11: Analysis of blank material by Swastika Lab.

Samples containing potentially significant Au values (>0.1 g/t) in batches that contained failed blanks were re-analyzed.

In the QP's opinion, the sample preparation, security and analytical procedures are adequate.

11.0 DATA VERIFICATION

Historical and more recent drill records have been compiled in a digital database for the project and verified to the maximum extent possible. Data verification was undertaken by Scott Wilson RPA during their resource estimation conducted in 2003[7]. During June and July 2010, (i.e. prior to the September 2010 resource estimation conducted by SAS personnel), issues with the digital database, such as inclusion of the azimuth component of downhole survey readings were addressed, to the maximum extent possible dependant on if original hardcopy records could be located. Other data verification was also completed at this time, including correction of minor differences during conversion of imperial data to metric data, as well as checking gold assays, assay duplicates, checks for duplicate or misidentified holes, and missing drill holes.

In the QP's opinion, the data are adequate for the purposes used in this technical report.

[7] Roscoe, W. E., Gow, N. N. (SWRPA) 2006;"Technical report on the Taylor, Clavos, Hislop and Stock projects in the Timmins area, northeastern Ontario, Canada.", prepared by SWRPA, October 2006

12.0 MINERAL PROCESSING AND METALLURGICAL TESTING

12.1 Metallurgical Test Work

SAS requested that SGS Minerals Services (SGS) perform a grindability and metallurgical characterization of two composites from the Taylor Mine. The samples were subjected to grindability testing and metallurgical test work (i.e. gravity separation and cyanidation). Five composites were also provided for gravity recoverable gold (GRG) and cyanide leaching test work. The composites were submitted for grindability and metallurgical testing. This report presents the grindability, gravity separation and the cyanidation test results.

12.1.1 Grinding Summary

To predict milling rates, conclusions are drawn by comparing data from previous milling campaigns of Taylor ore, along with current milling and production data.

A 10,000 tons bulk sample from Taylor Shaft Zone was processed at the Holt Mill in 2007. The mill configuration was such that only the SAG mill and one ball mill was being used, thus reducing the overall throughput. That was due to the production demands at the time, not justifying running the mill in its optimum configuration. The throughput of Taylor ore was slightly better than that of Holloway ore (another mine operated by SAS), with a difference of 4 tonnes over a 24 hour period; Holloway ore has a Bond Work Index (BWI) value of 17.1 while Taylor ore has an average BWI of 15.9. It is anticipated that Taylor's average milling rate will be equal to, or slightly higher than the average milling rate for Holloway ore of 125 tph. The projected milling rate for Taylor ore will be 125 tph with a final product size of 80% passing 325 mesh.

12.1.2 Overall Recovery Summary

Using data collected at SGS, FLD Smith-Knelson was able to model a gravity circuit design that predicts an average primary circuit gravity recovery of 19.8% and a secondary gravity circuit recovery of 13.8%, yielding an overall gravity recovery of 33.5%.

CIL leaching of the gravity tailings yielded a recovery of 88% to 97%. The combined recovery will be within the range of 93% to 99%, with the feed head grade being the largest contributing factor for the variation in the overall recovery.

An average recovery of 94.5% was achieved with previous whole ore leaching test work done by SGS in 2006 (without gravity concentration taken into account) and is proposed for the PFS.

12.1.3 Grindability Testing

Bond Ball Mill Grindability Test

Five samples were submitted for Bond ball mill grindability testing at a closing screen size of 100 mesh (150 microns). The BWIs test results were consistent, ranging from 15.6 kWh/t to 16.4 kWh/t. The samples were categorized as moderately hard.

SMC Testing

The SMC test is an abbreviated version of the standard JK drop-weight test performed on rocks from a single size fraction (-22.4/+19 mm in this case). The SMC test was performed on all five samples. The “A x b” parameter ranged from 46.6 to 30.0, which corresponds, respectively, to the “moderately hard” to “hard” category. The density ranged from 2.77 g/cc^3 to 2.89 g/cc^3.

12.1.4 Metallurgical Testing

The metallurgical test program examined the response of four Taylor Mine composite samples to gravity separation, gravity tailing cyanide leaching and extended gravity recoverable gold separation (e-GRG).

Gravity Separation

The response of the Taylor Mine composites to gravity separation for the recovery of free gold was examined on four kilogram charges of each sample. The tests were performed at a target grind size of a P_{80} of 45 microns, which produced tailings for the gravity tailing cyanidation testing. The gravity separation tests were performed using a Knelson MD-3 concentrator. The Knelson concentrate was recovered and further upgraded by treatment on a Mozley mineral separator to a low weight and high grade concentrate. The Mozley concentrate sample was assayed in its entirety. The Mozley and Knelson tailings were combined and forwarded to gravity tailing cyanidation testing. A summary of the test conditions and results are given in Table 12-1.

		Gravity Concentrate			Gravity Tails		Gravity Recovery		Head Grade			
Test Sample	K_{80}	Wt	Au	Ag	Au	Ag	Au	Ag	Calc. Au	Direct Au	Calc. Ag	Direct Ag
	μm	%	g/t	g/t	g/t	g/t	%	%	g/t	g/t	g/t	g/t
G1-RT-1	29	0.028	7,568	1,252	1.71	< 0.5	55.3	n/a	3.82	3.02	0.85	2.6
G1-RT-2	32	0.055	3,451	643	2.29	< 0.5	45.2	n/a	4.18	4.87	0.85	0.7
G1-RT-3	32	0.065	993	122	1.16	< 0.5	35.7	n/a	1.80	1.66	0.58	< 0.5
G1-RT-4	25	0.031	5,026	517	1.54	1.4	50.5	n/a	3.11	2.36	1.56	< 0.5

Table 12-1: Summary of gravity test results.

The direct head analyses of the four composites reported gold grades of 3.02 g/t, 4.87 g/t, 1.80 g/t and 3.11 g/t for samples RT-1 through RT-4 respectively. The tests on the composites resulted in relatively high gold recoveries ranging from approximately 36% to 55%. Gold gravity tailing analyses were in the range of 1.16 g/t to 2.29 g/t.

<u>Gravity Tailing Cyanidation</u>

Each gravity test tailings had two, one-kilogram charges split out for a cyanidation testing. Standard leach conditions were applied and performed, which included:

- 40% solids;
- a pH of 10.5 to 11.0;
- a solution concentration of 0.5 g/L NaCN;
- a carbon concentration of 10 g/L; and,
- the tests were carried out for 24 and 48 hours for each composite.

Upon completion of the tests, the final pulp was poured through a screen to remove the carbon and onto a filter to separate the solids from the solution. All three test products were submitted for chemical analyses. For the tests completed, the gold recoveries ranged from 88% to 97%. The NaCN consumption ranged from 1.77 kg/t to 2.63 kg/t and the lime consumption ranged from 0.22 kg/t to 0.51 kg/t for all of the tests performed. The tailings assays for the tests were quite low, ranging from 0.050 g/t Au to 0.150 g/t Au. These results, combined with the gravity results, showed overall gold recoveries ranging from 93% to 99% for the four composite samples. A summary of the test results from gravity tailing cyanidation is shown in Table 12-2.

Test Sample	K80 µm	Retention Time h	Residue Weight g	Reagent Addition (kg/t) NaCN	CaO	Reagent Consumption (kg/t) NaCN	CaO	Recovery Au %	Ag %	Residue Au g/t	Ag g/t	Carbon Au g/t	Ag g/t	Barren Au mg/L	Ag mg/L	CN Head Calc. Au g/t	Ag g/t	CN Head Direct Au g/t	Ag g/t	O'all % Recovery Gravity/Cyanidation Au	Ag
CIL-1-RT-1	29	24	969	2.64	0.32	2.43	0.23	97.0	n/a	0.050	< 0.5	97	23.5	< 0.01	< 0.03	1.69	0.94	1.71	< 0.5	98.7	n/a
CIL-2-RT-1	29	48	1,001	3.09	0.49	2.48	0.40	97.3	n/a	0.045	< 0.5	107	25.5	< 0.01	< 0.03	1.64	0.90	0.78	< 0.5	98.8	n/a
CIL-3-RT-2	32	24	1,028	2.49	0.22	2.30	0.14	93.8	n/a	0.150	< 0.5	137	27.0	0.01	< 0.03	2.34	0.97	2.29	< 0.5	96.6	n/a
CIL-4-RT-2	32	48	1,021	2.97	0.36	2.34	0.27	94.7	n/a	0.115	< 0.5	142	26.8	0.01	< 0.03	2.17	0.90	2.29	< 0.5	97.1	n/a
CIL-5-RT-3	32	24	1,022	2.36	0.23	1.77	0.15	92.0	n/a	0.100	< 0.5	72	17.7	< 0.01	< 0.03	1.25	0.82	1.16	< 0.5	94.9	n/a
CIL-6-RT-3	32	48	951	2.88	0.33	2.63	0.30	88.4	n/a	0.140	< 0.5	69	16.6	< 0.01	< 0.03	1.20	0.80	1.16	< 0.5	92.5	n/a
CIL-7-RT-4	25	24	987	2.50	0.27	2.31	0.19	96.6	n/a	0.060	< 0.5	94	22.1	< 0.01	< 0.03	1.63	0.91	1.51	1.4	98.3	n/a
CIL-8-RT-4	25	48	1,052	2.75	0.51	2.18	0.42	96.4	n/a	0.055	< 0.5	105	22.0	< 0.01	< 0.03	1.51	0.84	1.51	< 0.5	98.2	n/a
Average				2.71	0.34	2.31	0.26	94.5	n/a	0.089	< 0.5	103	22.7	0.01	< 0.03	1.68	0.89	1.55	1.4	96.4	n/a

Table 12-2: Summary of CIL test results.

Extended Gravity Recoverable Gold (EGRG) Test work

EGRG testing was conducted on composite samples RT-1 through RT-4. The procedure used was developed by Knelson Gravity Solutions of British Columbia and involves the recovery of gold from a sample ground to progressively finer sizes, with size analyses of the gravity concentrates and tailings at each stage. This test allows for the determination of the GRG value (theoretical maximum amount of gold recoverable) as a function of the size distribution.

For stage 1, a 20 kg sample of each composite was processed through the Knelson concentrator, producing a gravity concentrate and tailings. The first pass was performed on minus 20 mesh crushed material. The K_{80} range for the four samples was 497 to 559 microns. The concentrate was filtered and submitted for Au size fraction analysis. The tailings sample was filtered and sub-sampled (~200 g to 300 g) for Au size fraction analysis. The remainder of the tailings was split into two 10 kg charges, pulped to approximately 65% solids and ground in a 10 kg rod mill, targeting a K_{80} grind of 150 to 200 microns. The two charges were then combined for stage 2.

During stage 2, the gravity separation, sampling, and size fraction assaying procedure was repeated. The remaining stage 2 Knelson tailings was split into two 10 kg charges, pulped to approximately 65% solids and ground in a 10 kg rod mill, targeting a K_{80} grind of 50 to 70 microns. The two charges were then combined for stage 3.

Stage 3 was performed as per the prior stages, repeating the gravity separation, sampling and size fraction assaying procedure. All the concentrates were assayed to extinction.

For the RT-1 composite, a GRG number of 57.2 was obtained, indicating that gravity processes could recover approximately 55% to 60% of the gold. The distribution of gold in the gravity concentrate was similar to that seen in the conventional gravity test G1 conducted and reported at 55%. The calculated head grade from the EGRG test for the RT-1 composite was 5.91 g/t Au.

For the RT-2 composite, a GRG number of 45.4 was obtained, indicating that gravity processes could recover approximately 45% to 50% of the gold. The distribution of gold to the gravity concentrate was similar to that seen in the conventional gravity test G2 conducted and reported at 45%. The calculated head grade from the EGRG test for the RT-2 composite was 4.90 g/t Au.

For the RT-3 composite, a GRG number of 52.9 was obtained, indicating that gravity processes could recover approximately 50% to 55% of the gold. The distribution of gold to the gravity concentrate was similar to that seen in the conventional gravity test

G3 conducted and reported at 36%. The calculated head grade from the EGRG test for the RT-3 composite was 2.40 g/t Au.

For RT-4 composite, a GRG number of 62.1 was obtained, indicating that gravity processes could recover approximately 60% to 65% of the gold. The distribution of gold to the gravity concentrate was similar to that seen in the conventional gravity test G4 conducted and reported at 51%. The calculated head grade from the EGRG test for the RT-4 composite was 3.48 g/t Au.

12.1.5 Gravity Circuit Simulations

Modeling results are presented in Table 12-3 and in Figure 12-1 to Figure 12-2.

Ore Sample	Knelson Model	Tonnage to primary gravity (tph)	Tonnage to secondary gravity (tph)	Primary Gravity Recovery (%)	Secondary Gravity Recovery (%)	Overall Gravity Recovery (%)
GRG-1	QS48	200	200	24	14	38
GRG-2	QS48	200	200	15	13	28
GRG-3	QS48	200	200	15	12	27
GRG-4	QS48	200	200	25	16	41

Table 12-3: Gravity recovery modelling results.

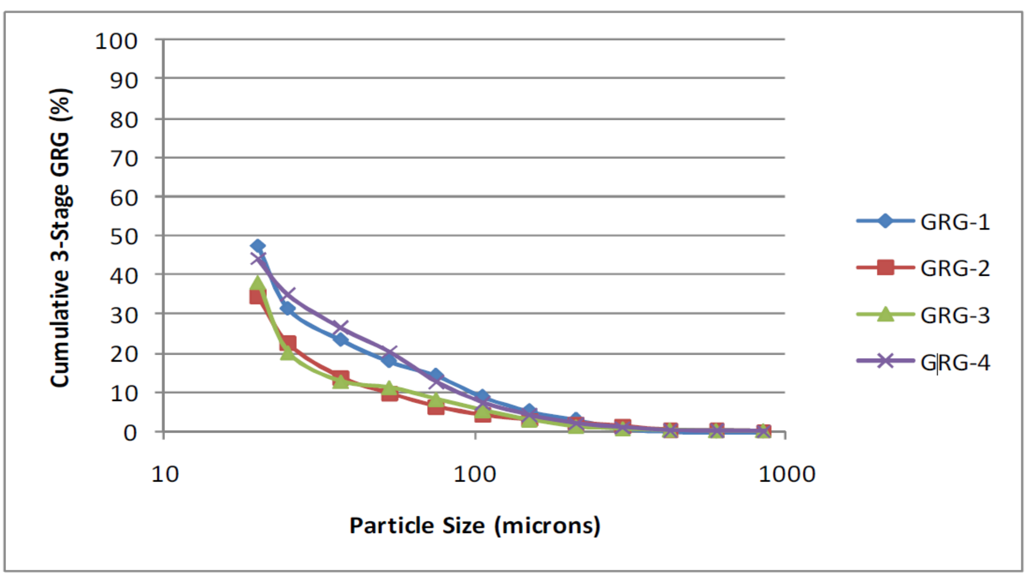


Figure 12-1: Primary circuit cumulative GRG recovery versus particle size.

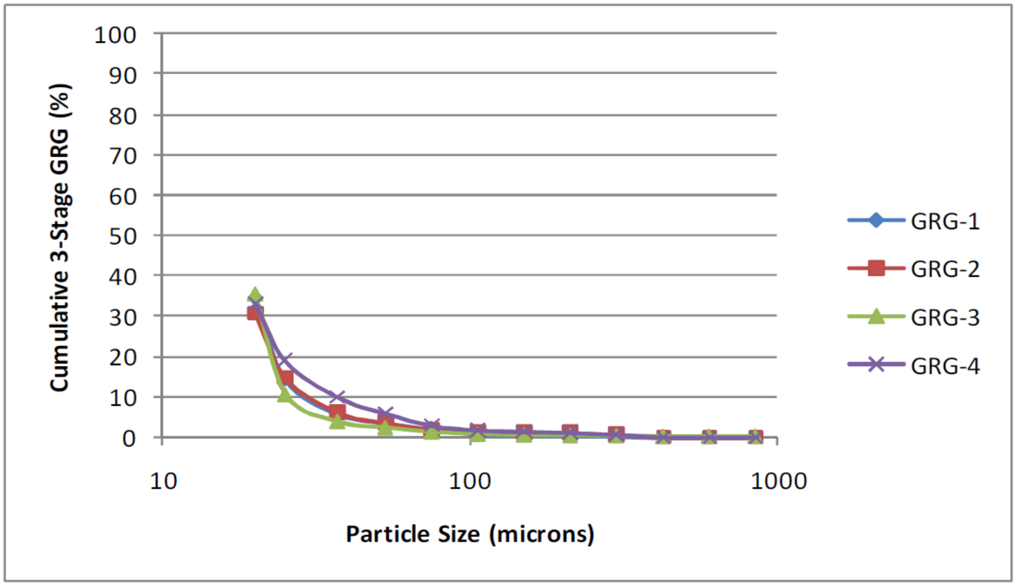


Figure 12-2: Secondary circuit cumulative GRG recovery versus particle size.

13.0 MINERAL RESSOURCE ESTIMATES

The Mineral Resources effective December 31, 2011 are summarized in Table 13-1.

Area	Category	Tonnes	Grade gpt Au	Contained Oz Au
West Porphyry Zone	Indicated	1,929,000	5.50	341,400
Shoot Zone	Indicated	696,000	5.19	116,100
Total Indicated Resources		**2,625,000**	**5.42**	**457,500**
West Porphyry Zone	Inferred	1,910,000	3.95	242,500
Shoot Zone	Inferred	19,000	5.19	3,100
Total Inferred Resources		**1,929,000**	**3.96**	**245,600**

Notes

1. CIM definitions (2010) were followed for Mineral Resources
2. Mineral Resources were estimated at a block cut-off grade of
   2.5 gpt for the West Porphyry Zone
   3.0 gpt for the Shoot Zone
3. Mineral Resources are estimated using a long term gold price of US$1,200/oz
   and an exchange rate of C$1.00 = US$0.98
4. A minimum mining width of 2.0 m was used
5. A bulk density of 2.84 t/m^3 for the WPZ and 2.67 t/m^3 for the SZ was used
6. Totals may not add exactly due to rounding

Table 13-1: Mineral Resources for the Taylor Project, as of 31 December 2011.

13.1 Database

The database comprises of 883 drill holes with 213,370 m of drilling for an average hole length of 241 metres. The number of holes and composites within each mineralized zone/lens that were subsequently used for grade interpolation are summarized in Table 13-2. Holes were composited to a maximum length of 1.5 m in the West Porphyry Zone and a Maximum of 1.0 m in the Shoot Zone.

Zone	Lens	# Holes	# of Composites	Average Core Length (m)
West Porphyry[1]	1003	37	564	23
	1004-1	188	2,417	19
	1004-2	54	335	9
	1006	131	1,229	14
	1008-1	122	1,222	15
	1008-2	40	452	17
	1009	71	520	11
	1010	56	470	13
	1011	33	317	14
Shoot[2]		79	823	10

Notes
1. Composited to 1.5 m
2. Composited to 1.0 m
General note: A single hole may pierce more than 1 solid

Table 13-2: Summary of drill hole intercepts by zone and lens.

13.2 Geological Interpretation and 3D Solid Modelling

In order to update the Mineral Resources for the Taylor Project, SAS personnel interpreted the geology and gold mineralization and constructed three-dimensional (“3D”) solid body models to better constrain the gold mineralization during grade interpolation.

For the WPZ, the shape of each gold-bearing lens was very broadly interpreted based on structure, alteration, sulfide mineralization and gold grade. Rock type was used to a lesser extent, as it was determined that the gold mineralization is more related to geologic structures that transect lithological boundaries. This recent interpretation and 3D solid modelling of the gold mineralization is considered to be a more realistic representation of the mineralized zones, as the interpretation is based on alteration and mineralization based on a sound geologic model. Previous resource estimates (i.e. SWRPA, 2006) extrapolated the better gold grades from hole to hole, with no consideration for the alteration and/or overall mineralized envelope. This assumed that the higher grade gold mineralization is continuous from hole to hole, which has yet to be confirmed with detailed underground exploration.

In the current model, the gold mineralization is modelled as nine broad mineralized lenses (Figure 13-1). These lenses are vertically stacked with an offset to the west as depth increases, with an average strike direction of 067° and dipping to the south at 33°. The lenses have been identified using a numbering scheme of 1003 (deepest lens) to 1011 (shallowest lens).

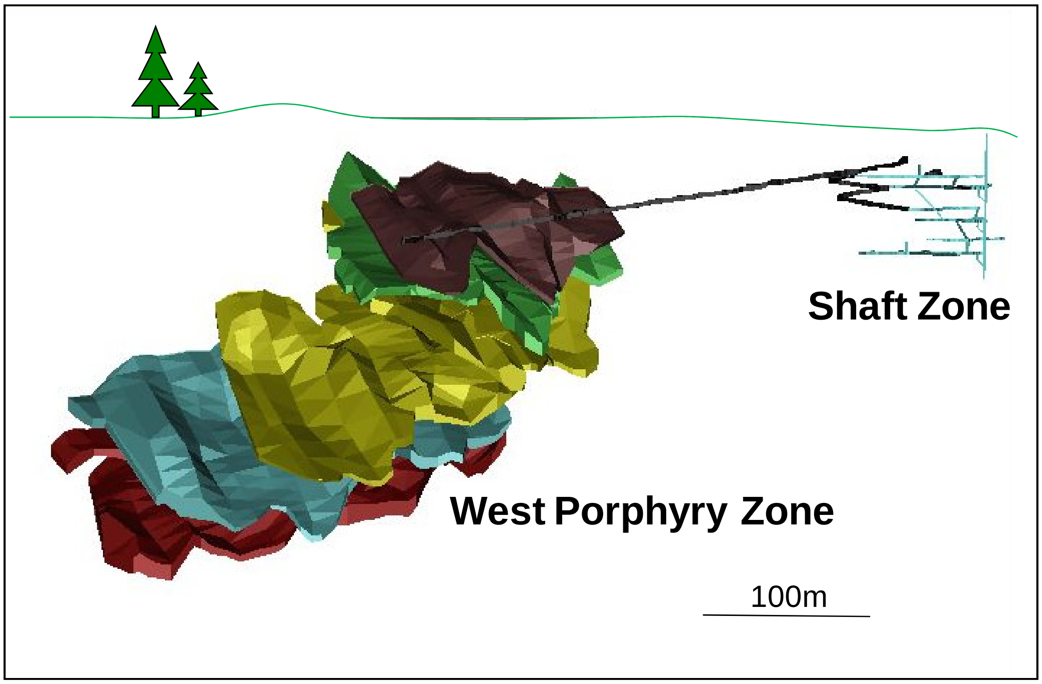


Figure 13-1: Isometric view of the WPZ (looking northeast).

13.3 Density Data

During the 2010-2011 drill programs, additional density measurements were taken and the results of metallurgical testing were used to confirm the density of Taylor ore at 2.84 t/m^3.

13.4 Assay Composites

In order to normalize the assay data, the 8,707 assays were composited into 1.5 m length; 87% of the assays were less than 1.5 m long and 44% were less than 1.0 m long).

Since no correlation between grade and density was identified, only sample length was used to weight the grades during compositing. Only assays occurring within the mineralized zones were composited. Each composite was identified with the mineralized solid shape. During the compositing process, missing or unsampled areas are assumed to have zero grade. These unsampled intervals only account for approximately 10 to 20% of the intervals within the mineralized zones.

Composites were calculated within the mineralized zones from the contact closest to the collar to the toe of the hole. A small number of composites are less than the full length due to a reduced interval between the previous sample and the edge of the solid. For the WPZ, all composites greater than 0.75 m were used.

13.4.1 Capping of High Gold Values

Each zone was analyzed separately to establish a capped grade for the composite values. Because of the large, low grade envelope used in the modelling there was a strong low grade bias. To limit the influence of lower values, all composites grading less than 0.05 g/t Au were ignored. Commonly used capping criteria (e.g. 99th percentile, mean, etc.) were assessed for the various lenses of the WPZ. The selected capping grades varied by lenses from 12 g/t to 30 g/t. In addition, log probability plots were used to better analyse the frequency distribution.

In the WPZ, the mineralization is characterized as a low grade envelope with a population between 0.03 g/t to 4 g/t Au, with a second, higher grade population ranging from 1 g/t to 116 g/t Au. There are three (of the nine) zones that have a greater percentage of the higher grade population, namely 1004-1, 1008-1 and 1010. Of significance is zone 1004-1 which contains a large portion of the resources. A capping factor of 30 g/t Au has been used. This factor is very conservative and reflects the concern of projecting the influence of the high grade zones with little or no supporting lithology/alteration constraints.

The sensitivity of capping values on the WPZ 1004-1 lens is shown in Table 13-3. Lowering the capped grade from 80 to 30 g/t Au resulted in a loss of 56,000 ounces of gold in the Indicated and Inferred Resources, while further reducing the capped grade from 30 to 20 g/t Au resulted in the loss of an additional 76,000 ounces of gold. Given the sensitivity of the average resource grade with the capped grade, it is recommended that any future underground exploration and possible bulk sampling be completed in order to better define the correct capping grades. In addition, further studies should be conducted to define any geologic controls of the higher grade zones in order to better delineate any higher grade shoots and allow a more accurate model to be constructed.

	Indicated			Inferred		
Capping Grade	Tonnes	Grade	Ounces	Tonnes	Grade	Ounces
Uncapped	1,190,284	8.43	322,757	712,156	4.76	108,987
80	1,188,115	7.27	277,705	711,958	4.70	107,521
30	1,167,761	6.07	227,895	707,587	4.45	101,129
20	1,231,769	4.69	185,537	575,971	3.62	66,996

Table 13-3: Impact of Different Capping Grades on WPZ 1004-1 lens.

13.5 Semi-Variograms

In order to define the amount of grade variability and the orientation of maximum grade continuity, a suite of semi-variograms were constructed from the composite values.

Omni (non-directional) semi-variograms were constructed to establish a nugget and sill value. Subsequently, preliminary semi-variograms where calculated in order to define the various ranges (i.e. distances) for grade interpolation, which considered the following:

- Orientation of the mineralized envelope, along strike and dip;
- Orientation based on a visual inspection of a preliminary block model high grade trends; and
- Incremental models.

The WPZ 1004-1 lens has a relative nugget value of 31% of the total sill value. This indicates that the local variability of the gold is higher at the WPZ.

Although a number of directions were investigated, semi-variogram models could not be created for the different directions. Essentially, the semi-variograms indicate that the grade continuity is similar both down dip and along strike. As such, the omni variogram ranges were used to interpolate gold grades into the block model.

13.6 Block Model

A three-dimensional block model was constructed to include the interpolated gold grade. The block model is constructed by filling the solid with the predefined blocks oriented in an east west direction in line with the UTM grid.

The WPZ solid was filled with 8m x 2.5m x 1.5m blocks and subdivided into 4 sub blocks to fill the edges of the solid.

13.7 Grade Interpolation

Gold grades were interpolated into the block model utilizing the inverse distance squared (ID^2) method. Only composites within the solid being modelled were used in the calculation. For each lens, parameters used in the calculations are summarized in Table 13-4.

Zone	Lens	Search Ellipsoid						Composites			Cap
		Azimuth	Dip	Plunge	Xm	Ym	Zm	Min	Max	per Hole	(gpt)
Shoot		45	-20	-10	32	32	10	2	8	2	20
WPZ	1003	58	-25	-15	40	30	10	2	8	2	20
	1004-1	58	-25	-15	40	30	10	2	8	2	30
	1004-2	58	-25	-15	40	30	10	2	8	2	16
	1006	58	-25	-15	40	30	10	2	8	2	20
	1008-1	58	-25	-15	40	30	10	2	8	2	16
	1008-2	58	-25	-15	40	30	10	2	8	2	25
	1009	58	-25	-15	40	30	10	2	8	2	12
	1010	58	-25	-15	40	30	10	2	8	2	12
	1011	58	-25	-15	40	30	10	2	8	2	12

Table 13-4: Summary of grade interpolation parameters.

The search ellipsoid is oriented sub-parallel to the orientation of the modelled mineralized zones and lenses. The primary search ellipsoid is approximately double the range as defined by the omni semi-variograms. Once the first pass is complete, the search distance is doubled and grades are calculated for any blocks not previously calculated. This is meant to populate the remaining cells with gold values projected a large distance from drill hole information and to populate cells that may have been blank due to folds and bends in the solid model.

The composite limitations results in block values to be calculated based on at least one drill hole and a maximum of four drill holes within the search radius, which was used in order to reduce the possibility of overly averaging the gold values.

13.8 Resource Estimate and Classification

The resource classification is essentially based on the density of drill hole information and the continuity of gold grades.

The shallowly dipping mineralized zones combined with the steeply dipping drill holes results in the composites being at approximately right angles to the mineralization. Therefore, the limits to the drill hole density are primarily along strike and down dip.

To define the indicated and inferred resource categories a multi-step process was used. The resource solid for each zone was displayed in plan view with the block model turned on with the distance to the nearest data point labelled. A closed string bounding the blocks having less than the omni range (13.0 m in the WPZ) and having grade continuity was used to create a new solid from the resource solid. For the WPZ 1004-1 lens, the average distance to data was 13.4 m, indicating a continuous Indicated Resource 3D solid shape (Figure 13-2).

The actual calculations used the total resources from the blocks (above a block cut-off grade) contained within the original solid minus the indicated resources from the contained blocks from the shape created from the original shape with the balance being the inferred resource.

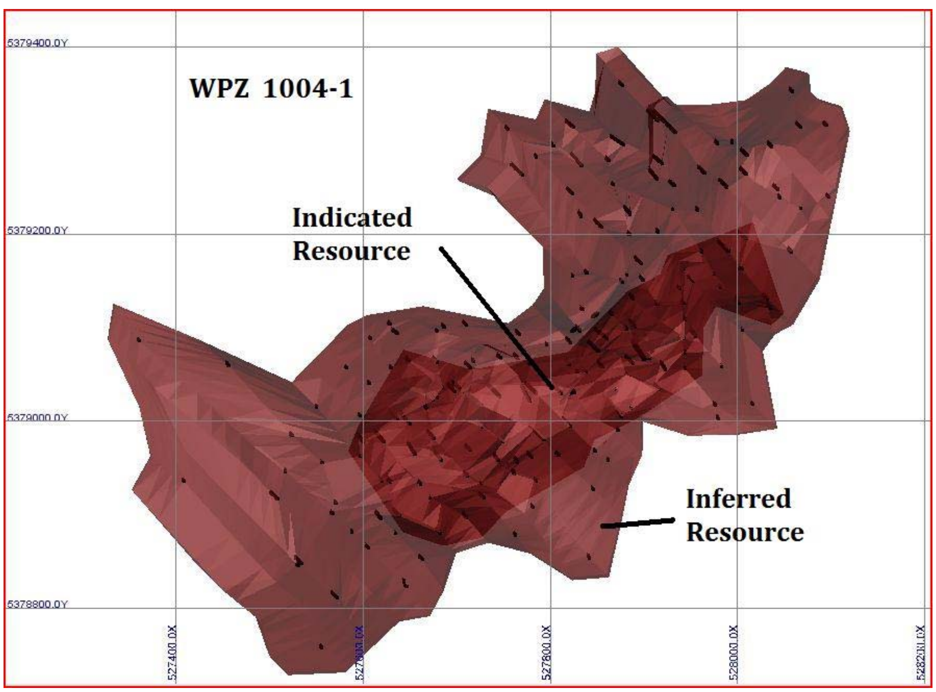


Figure 13-2: WPZ 1004-1 lens indicated and inferred mineral resources (plan view).

Mineral resources disclosed for the WPZ are shown in Table 13-5.

	Indicated			Inferred			
Lens	Tonnes	Grade	Ounces	Tonnes	Grade	Ounces	Cap
1003				617,949	3.53	70,212	20
1004-1	1,167,761	6.07	227,895	707,587	4.45	101,310	30
1004-2	21,496	4.45	3,073	43,600	3.16	4,428	16
1006	203,386	4.76	31,100	380,965	3.65	44,745	20
1008-1	161,303	4.40	22,808	50,432	4.25	6,886	16
1008-2	58,004	7.24	13,496	10,105	9.46	3,074	25
1009	199,133	4.58	29,297	18,211	4.17	2,442	12
1010	118,117	3.62	13,728	42,809	3.63	4,996	12
1011				38,832	3.49	4,358	12
Total	1,929,200	5.50	341,397	1,910,490	3.95	242,450	

Notes

1. CIM definitions (2010) were followed for Mineral Resources
2. Mineral Resources were estimated at a block cut-off grade of 2.5 g/t Au
3. Mineral Resources are estimated using a long term gold price of US$1200/oz and an exchange rate of C$1.00 = US$0.98
4. A minimum mining width of 2.0 m was used
5. A bulk density of 2.84 t/m^3 was used
6. Totals may not add exactly due to rounding

Table 13-5: Mineral resources by lens for the WPZ, as of 31 December 2011.

In the QP's opinion, there are no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that could materially affect the mineral resources estimate.

13.9 Exploration Potential

In addition to the known zones of gold mineralization and the existing mineral resource base, there remains additional exploration potential along the plunge of the known deposits comprising the WPZ and Shoot zone.

At the WPZ, recent drilling and re-interpretation indicates that wider and higher grade gold mineralization is spatially associated with "kinks" or bends along the footwall contact of the alteration zone with the underlying ultramafic volcanics. These "kinks" define a footwall contact that flattens to a shallower dip then the overall alteration zone, which is interpreted to be caused by thrusting. It is these areas where the higher grade and wider zones of gold mineralization occur. It is possible that these flatter zones have the potential to repeat along the contact down dip (Figure 13-3).

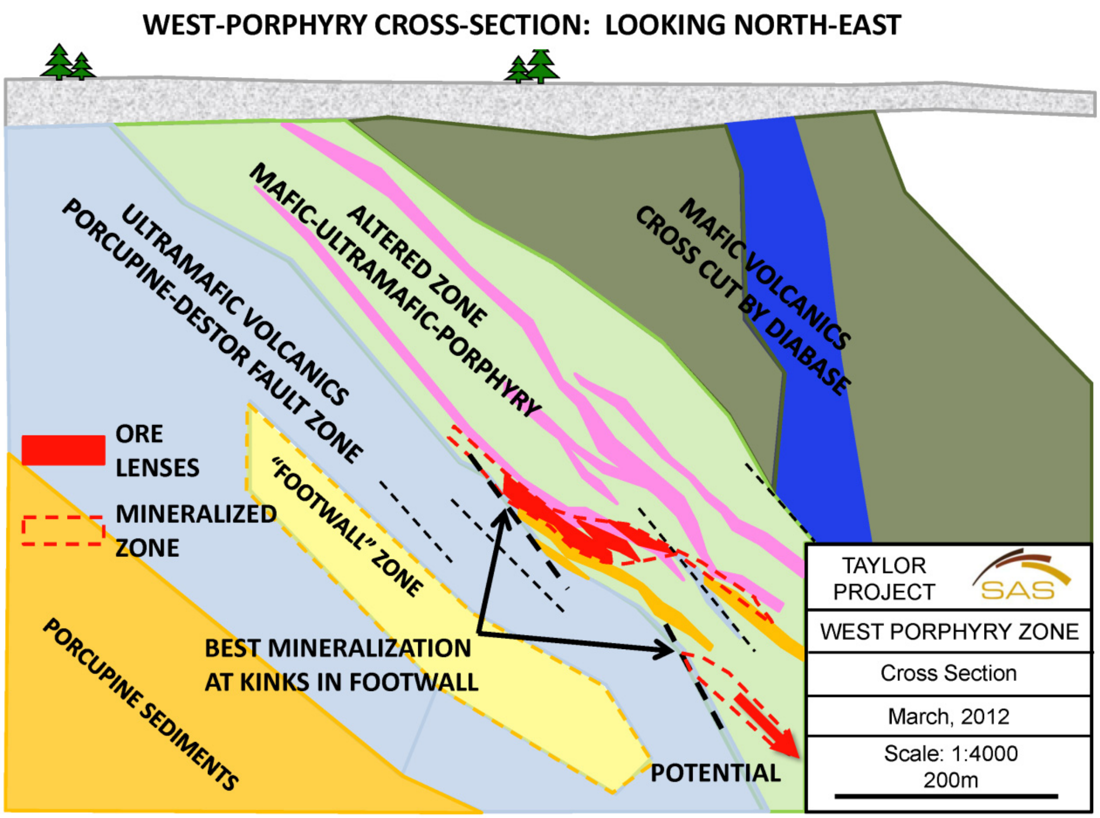


Figure 13-3: Cross section of the WPZ (looking east).

Additionally, one hole drilled approximately 200 metres down plunge from the WPZ intersected several zones of alteration and gold mineralization similar to the WPZ, and is interpreted to be the down plunge extension. This hole returned 16.3 g/t Au over 2.4 metres and 8.8 g/t Au over 1.2 metres (Figure 13-4).

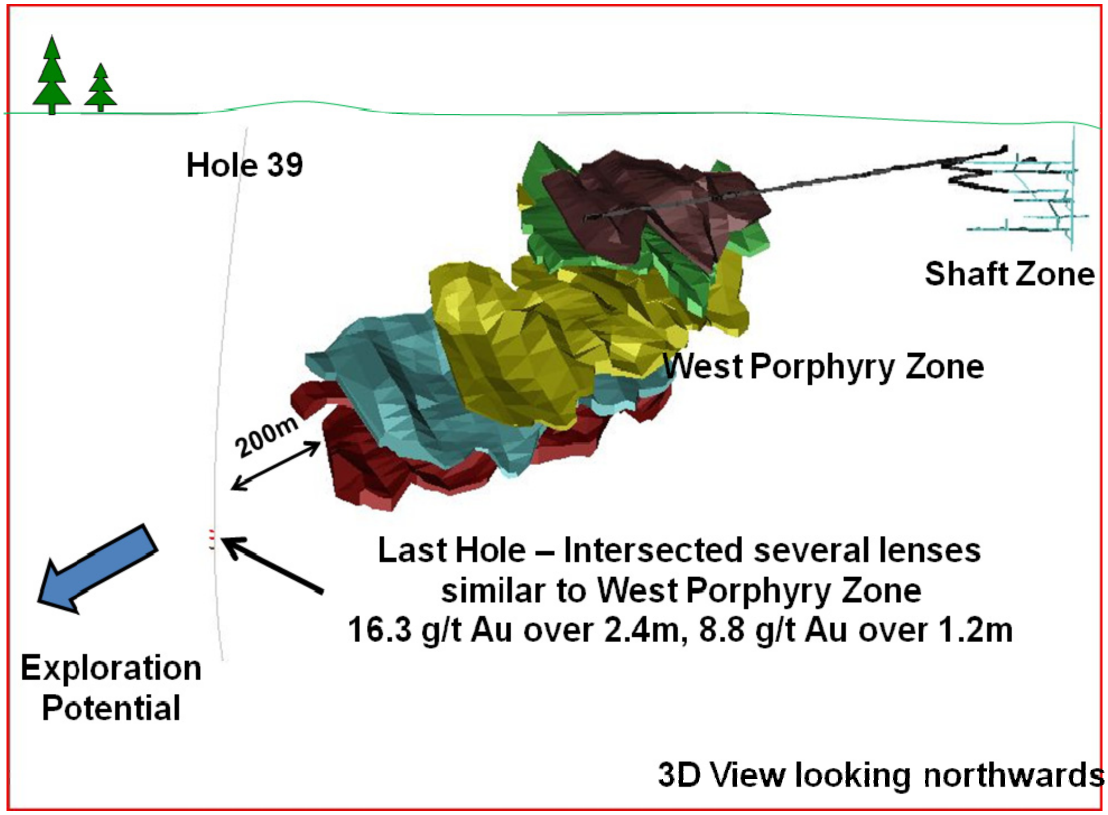


Figure 13-4: Three-dimensional view of the WPZ and the position of hole 39.

At the Shoot zone, limited drilling down dip and along strike to the west indicates the potential for extension in these directions (Figure 13-5).

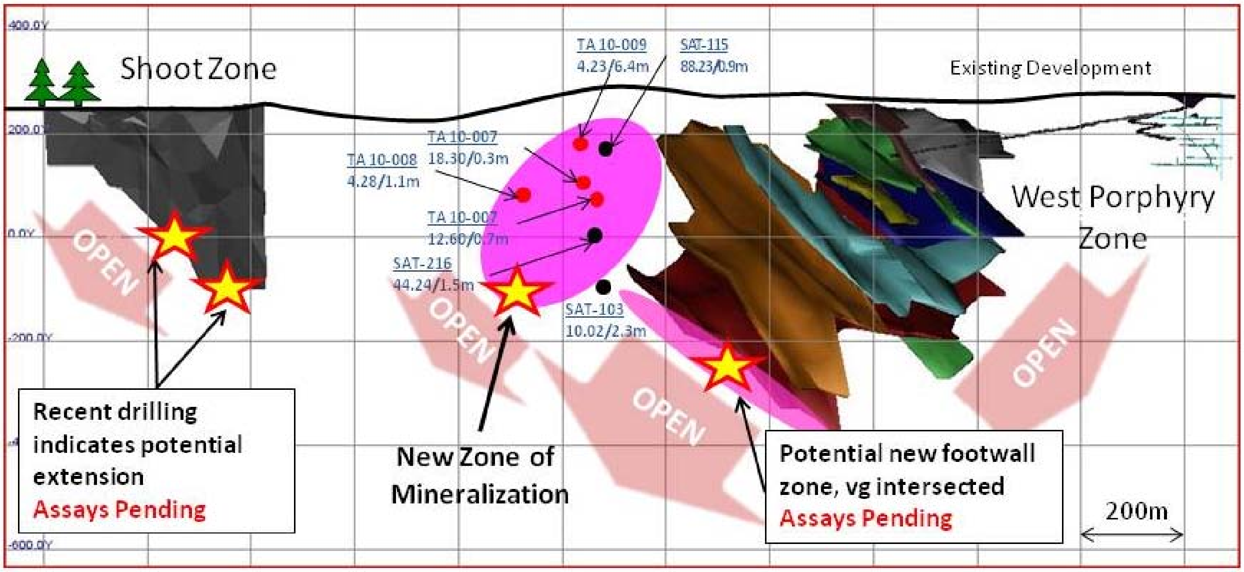


Figure 13-5: Exploration potential (longitudinal vertical view looking northeast).

14.0 MINERAL RESERVES ESTIMATE

The mineral reserves at the WPZ stand at approximately 985,000 t grading 5.45 g/t (173,000 ounces in-situ) and is effective December 31, 2011. Details by lens are shown in Table 14-1. Mineral reserves are included in the indicated mineral resources.

WPZ		Probable	
Lens	Tonnes	Grade	Ounces
1003	0	0.00	0
1004-1	881,534	5.34	151,441
1004-2	26,590	6.46	5,526
1006	38,320	5.13	6,320
1008-1	27,707	5.63	5,014
1008-2	11,107	12.11	4,324
1009	0	0.00	0
1010	0	0.00	0
1011	0	0.00	0
Total	985,257	5.45	172,625

Table 14-1: Mineral reserves at the WPZ.

The following assumptions were used in converting mineral resources to mineral reserves:

- Price of gold: US$1,300/oz;
- Currency exchange rate: $CDN1.00 = US$0.98;
- Stope cut-off grade was calculated on an individual basis;
- Dilution ranged from 9% to 50% (applied by geologist on mining shapes);
- Dilution grade varied from 0.5 gpt to 1.0 gpt (applied by geologists);
- Mining Extraction: 95%;
- Milling Recovery: 94.5% (based on test results from SGS);
- Stopes needed to show positive operating cash flow to be included.

In the QP's opinion, there are no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that could affect materially the mineral reserves estimate.

15.0 MINING METHODS

15.1 Design Criteria

The Taylor Project consists of two distinct zones: the Shoot Zone, which is located on the west side of the property, and the WPZ, which is located on the east side of the property (Figure 15-1); the Shaft Zone is also illustrated for the purpose of scale and completeness even though it is not considered in the Project. Based on the conclusions and recommendations of an internal scoping study[8], only the WPZ is considered for the purpose of the PFS.

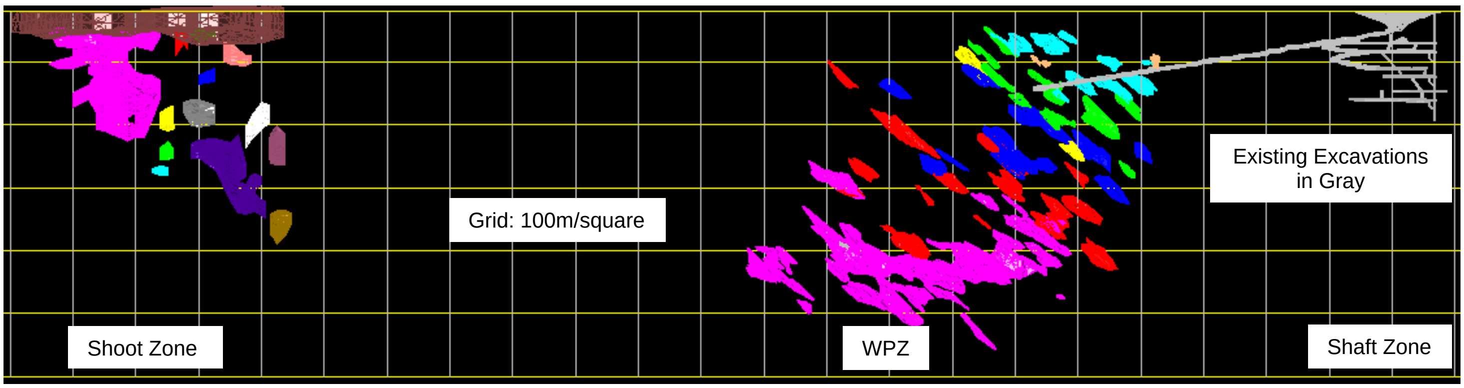


Figure 15-1: Longitudinal view of the Taylor Project (looking Northwest).

The WPZ extend vertically about 600 m respectively and is mostly open at depth. The WPZ will be accessed via a ramp and mined by overhand cut and fill method. Ore and waste will be trucked to surface where the ore will be loaded in surface trucks for haulage to the Holt mill and the waste will be stockpiled on designated surface areas.

Ventilation will be forced underground initially under positive pressure. During commercial production stage, additional design work will be completed and the airflow may later be pulled under negative pressure. Auxiliary fans will be required for adequate airflow distribution.

Results from geomechanical test work averaged the rock uniaxial compressive strength at 124 MPa. Backfill will be introduced in cut and fill stopes; cement may be added where warranted (e.g. local stability, stope cycle).

[8] Rocque et al., “Taylor Property Scoping Study”. St Andrew Goldfields internal document. May 5th, 2011.

Underground water will be pumped to a collector pond on surface prior to be discharged in the environment (for example, in the Driftwood river); alternatively, the option of pumping the underground water to existing excavations in the Shaft Zone will be assessed during development work. The current pumping rate is capped at 2,016 m^3/day, which exceeds the anticipated water inflow at the mine; surface water inflow towards the portal will be collected and mixed with the underground water prior to be discharged.

Underground infrastructure will be re-installed underground as the work progresses (e.g. power, leaky feeder system, piping, etc.).

Existing ground support systems will be tested and, if determined below current SAS ground support installation standards, be replaced or enhanced.

Due to current adverse labour market conditions and an early start-up date, the business case considered for the PFS includes initially, a contractor for all "below collar work". (i.e. miners, supervision, maintenance, construction, etc.). SAS workforce will consist of technical services and administration personnel. Security on site will remain contracted out. As work progresses towards commercial production, there will be an opportunity to substitute the contractor with a SAS workforce.

15.2 Mining Shapes and Associated Tonnage, Grade and Metal Contents

Mineral resources were modelled in 3D using Datamine, with mining shapes directly drawn on the sections. The longitudinal and side views of all of the mining shapes by lenses are displayed in Figure 15-2 and Figure 15-3. Mining shapes considered in the PFS are illustrated in Figure 15-4 using the same colour legend.

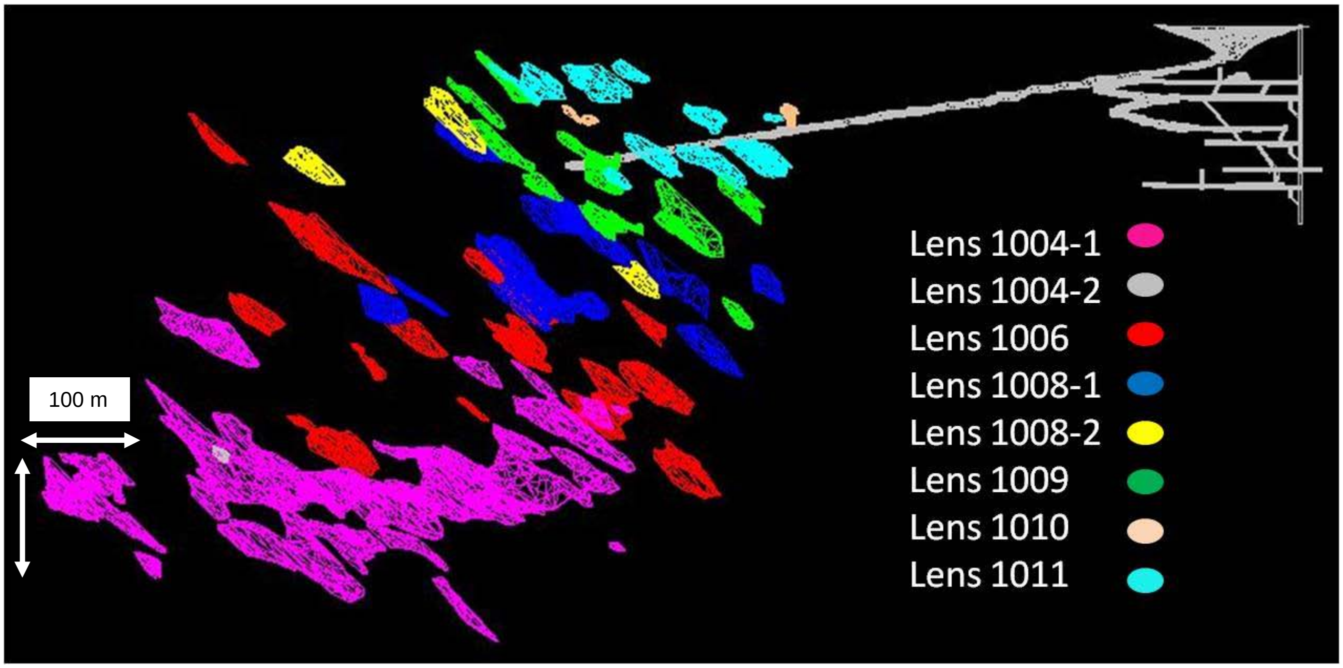


Figure 15-2: WPZ all mining shapes numbering (longitudinal view looking Northwest).

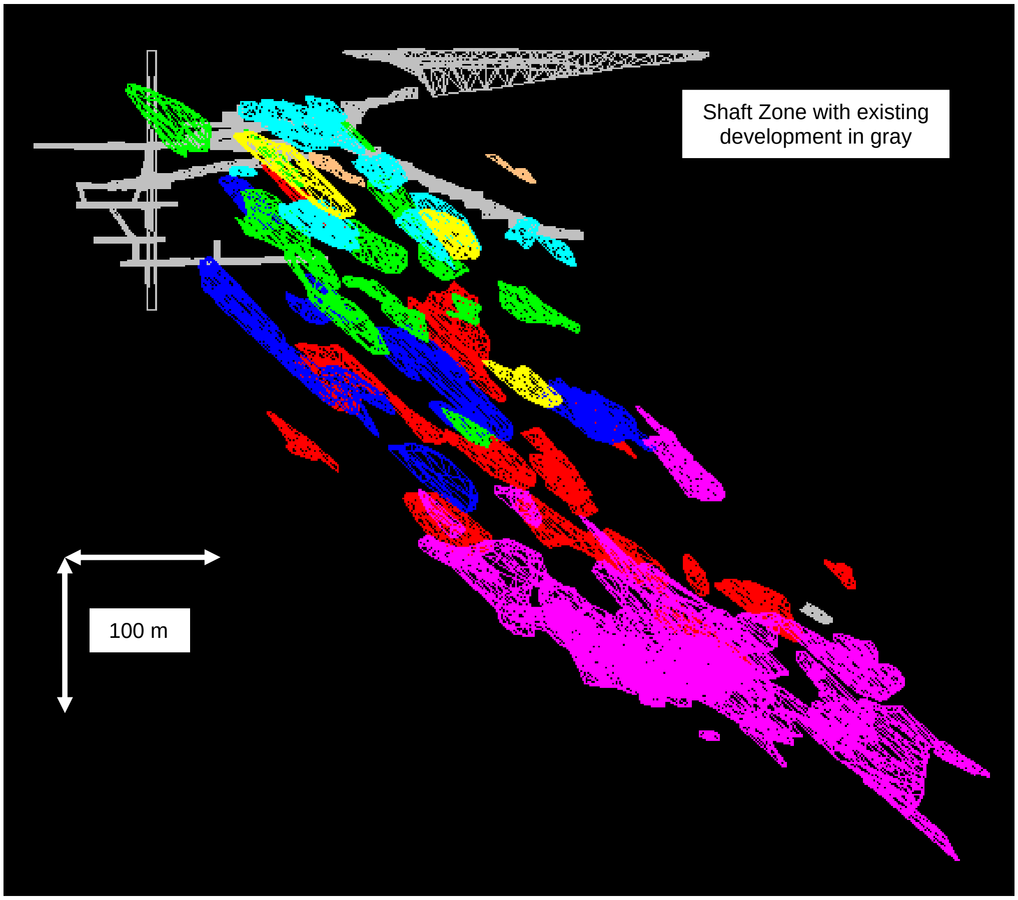


Figure 15-3: Side view of the WPZ (looking Northeast).

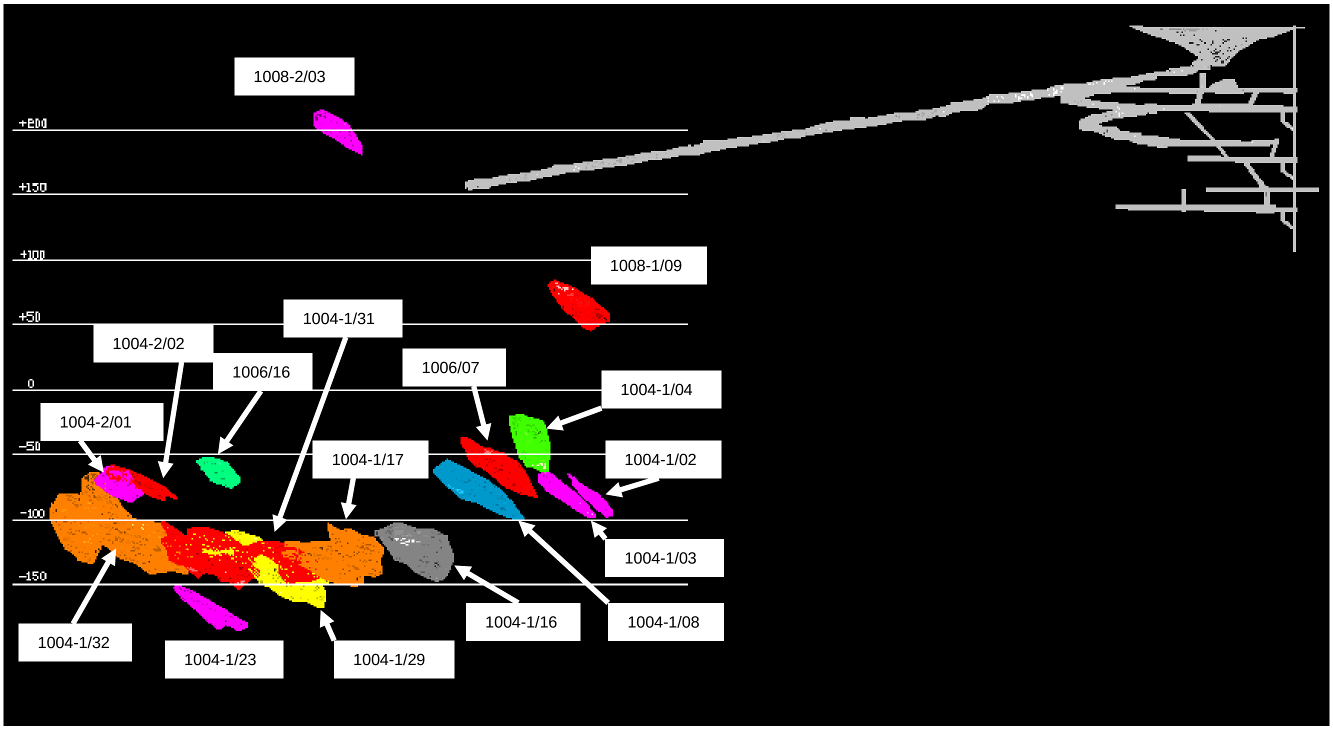


Figure 15-4: WPZ stopes in the mining plan (longitudinal view looking Northwest).

The block model was run against the mineral resources shapes and yielded 1.9 Mt grading 5.50 g/t for 301 koz in-situ (at a cut-off grade of 2.5 g/t). After applying dilution and extraction factors each shapes grading above 3.5 g/t was kept for further economic analysis. Finally, each resulting shape was assessed independently and only the shapes that returned a positive operating cash flow were included in the mineral reserves. Details are displayed in Table 15-1.

Lens	Mineral Resources Indicated (COG=2.5 g/t)			Mining Shapes > 3.5 g/t Diluted and Extracted			Mineral Reserves Probable		
	Tonnes	Grade	Ounces	Tonnes	Grade	Ounces	Tonnes	Grade	Ounces
1003	0	0.00	0	0	0.00	0	0	0.00	0
1004-1	1,167,761	6.07	227,895	1,146,550	5.02	185,232	881,534	5.34	151,441
1004-2	21,496	4.45	3,073	26,590	6.46	5,526	26,590	6.46	5,526
1006	203,386	4.76	31,100	157,065	4.47	22,586	38,320	5.13	6,320
1008-1	161,303	4.40	22,808	56,062	5.17	9,324	27,707	5.63	5,014
1008-2	58,004	7.24	13,496	39,177	6.60	8,311	11,107	12.11	4,324
1009	199,133	4.58	29,297	137,276	4.07	17,941	0	0.00	0
1010	118,117	3.61	13,728	42,813	4.21	5,796	0	0.00	0
1011	0	0.00	0	0	0.00	0	0	0.00	0
Total	1,929,200	5.50	341,397	1,605,533	4.93	254,716	985,258	5.45	172,625

Table 15-1: Mining shapes tonnage, grade and metal contained by lens.

15.3 Mining Method

Criteria used for selecting the appropriate mining method(s) were discussed in the previous Scoping Study. Since the updated mining shapes are not significantly different than the previous ones, albeit dipping shallower, Overhand Cut and Fill (OC&F) and Drift and Fill (D&F) were selected as the most suitable mining methods:

- Where the ore is dipping at less than 45° and the ore horizontal width is narrower than 10 m, OC&F is the method of choice;

- Where the ore is dipping at less than 45° and the ore horizontal width exceeds 10 m, D&F will be the method of choice.

It is anticipated to use waste rock generated by development activities as the main source of backfill. Mine design optimization will determine the backfill characteristics requirements (e.g. rock fill, cemented fill, etc.).

Upon examination of the mining shapes, the geologist applied a dilution factor varying from 9% to 50% and a dilution grade ranging from 0.5 g/t to 1.0 g/t. Mining extraction was set at 95%, based on experience at SAS and other operations.

15.4 Geomechanical

One hole was drilled for the purpose of recovering core for geomechanical logging and test work[9]. The hole was drilled from the hanging wall side and went through mineralized zones located in the upper area of the WPZ.

An external consultant logged the core and classified the rock mass using the RMR and Q systems[10]. As a result from the analysis, the mineralized areas were assigned an RMR value of 65 (Q rating of 20) and the waste areas were assigned an RMR value of 62 (Q rating of 20).

The RMR rating was input in the “Critical Span Graph”[11] in order to estimate the largest stable stope excavation. The resulting range of stable stope spans varies from 9 m to 12 m. As more information is gathered through development and stoping, the assumptions leading to stope design will be reviewed accordingly.

The Q ratings were used for estimating the length of ground support for general applications in development headings and stopes.

15.5 Mine Access and Development

The WPZ will be accessed via the existing decline located near the shaft. Both excavations are currently flooded. Consequently, a dewatering program will be required prior to the resumption of development activities. The extension of the decline will be located on the hanging wall side of the WPZ, addressing exploration and production requirements. Drill bays located along the decline will be used for definition drilling of the WPZ, as required.

Access from the decline will vary in length, depending on the location of the stope. Generally, one access will be required to mine six to seven cuts per stope. In some cases, the geometry of the stope and access will permit more, or less, cuts to be mined from that location.

Three mining fronts will be created in order to mine the stopes in sequence and in the shortest time. Total development requirements are detailed in Table 15-2 (note: Year 1 is 2012).

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[9] Queen’s University at Kingston, “Rock Core Strength Testing – Taylor Property”, August 22, 2011.
[10] SRK, “Geotechnical Field Program”, July 22, 2011.
[11] Pakalnis et al., Update of Span Design Curve for Weak Rock Masses, Presented at 2004 AGM-CIM Edmonton.

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
Total Capital Development	1,323	2,291	495	180	20	0	4,309
Total Operating Development	0	475	765	455	995	310	3,000
Total Development	1,323	2,766	1,260	635	1,015	310	7,309

Table 15-2: Total development requirements (metres).

15.5.1 Capital Development

Details of capital development are listed in Table 15-3.

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
DECLINE							
Ramp	976	1,405	0	0	0	0	2,381
Remucks and Safety Bays	122	196	0	0	0	0	318
Sub-total	*1,098*	*1,601*	*0*	*0*	*0*	*0*	*2,699*
VENTILATION SYSTEM							
East Vent Raise-top access	25	0	0	0	0	0	25
East Vent Raise-bottom access	50	0	0	0	0	0	50
East Vent Raise-raise	100	0	0	0	0	0	100
West Vent Raise-top access	50	0	0	0	0	0	50
West Vent Raise-access #2	0	50	0	0	0	0	50
West Vent Raise-access #3	0	25	0	0	0	0	25
West Vent Raise-access #4	0	80	0	0	0	0	80
West Vent Raise-raise	0	460	0	0	0	0	460
Sub-total	*225*	*615*	*0*	*0*	*0*	*0*	*840*
OTHER LATERAL DEVELOPMENT							
1004-1/08 from -95 elev	0	0	75	0	0	0	75
1004-1/02 from -40 elev	0	0	300	0	0	0	300
1006/07 from -40 elev	0	0	0	180	0	0	180
1004-1/29 from -155 elev	0	55	0	0	0	0	55
1004-1/29 from -140 elev	0	0	80	0	0	0	80
1004-1/29 from -120 elev	0	0	0	0	20	0	20
1004-2/01 from -80 elev	0	20	0	0	0	0	20
1004-2/01 from -60 elev	0	0	40	0	0	0	40
Sub-total	*0*	*75*	*495*	*180*	*20*	*0*	*770*
Total Capital Development	1,323	2,291	495	180	20	0	4,309

Table 15-3: Capital development breakdown (metres).

Decline

A 5.0 m by 5.0 m ramp, driven at -15% grade, is proposed for accessing the WPZ. The designed length is 2,381 m, or approximately 2,700 m including safety bays and remuck bays.

A vertical longitudinal view is shown in Figure 15-5. The view is cut on strike with the deposit (approximately in a south-west to north-east direction).

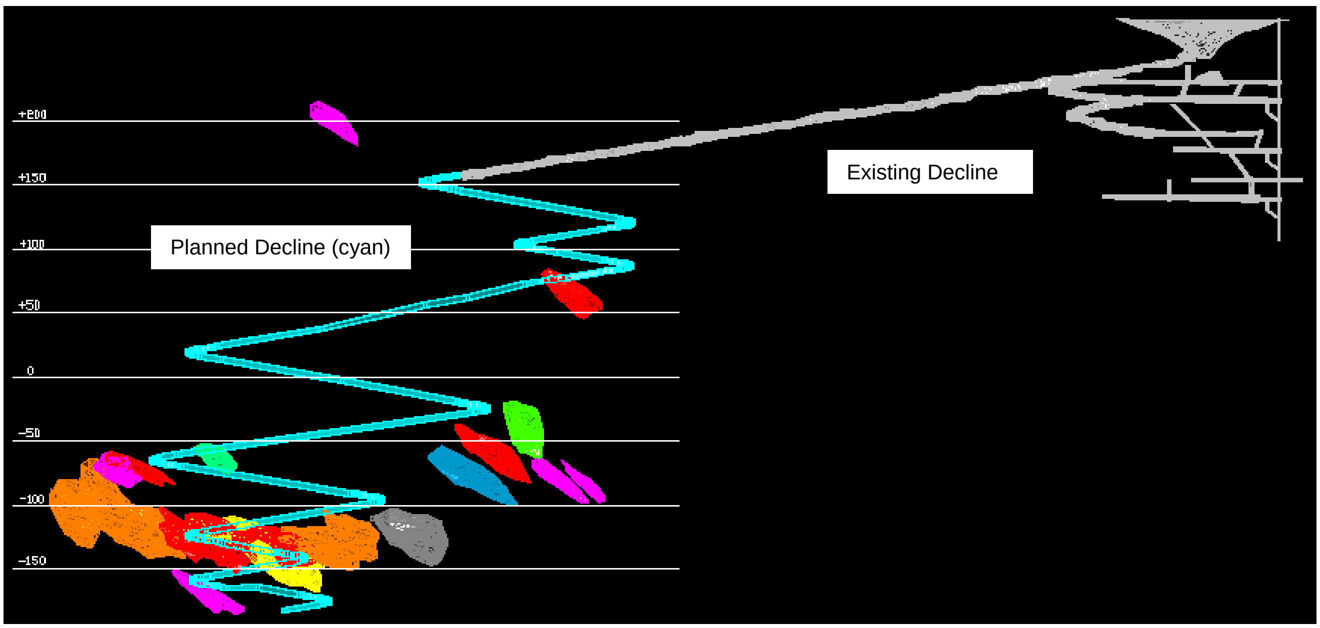

Figure 15-5: Longitudinal view showing the planned decline access to the WPZ.

Lateral

Capital lateral development for stope accesses and infrastructure amounts to approximately 770 m.

Ventilation Raise and Escapeway

A 4.3 m circular raise, or equivalent area, approximately 460 m long, is proposed (Figure 15-6). Additionally, a shorter raise (approximately 100 m) is required at the start of the ramp development to better manage the fresh airflow into the mine. Lateral accesses to the ventilation raises amount to approximately 280 m.

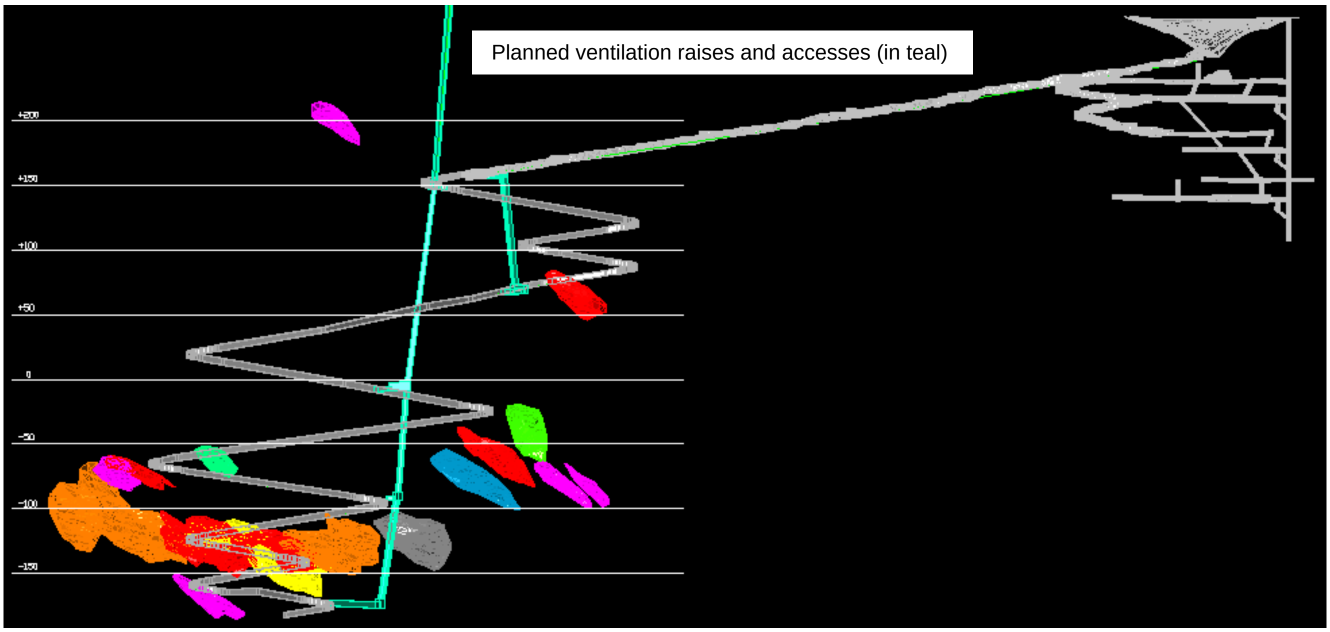

Figure 15-6: Longitudinal view of the planned ventilation raises and accesses.

15.5.2 Operating Development

Operating development consists of various accesses from capital headings or linking stopes. Lateral drifts are designed as 5 m by 5 m excavations; however, the final size will be set once the designed stopes are finalized. Details are provided in Table 15-4.

Stope	Lens	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
		0	0	0	0	0	0	0
1004-1	2	0	0	65	0	0	0	65
	3	0	0	65	0	0	0	65
	4	0	0	0	120	125	0	245
	8	0	0	130	0	0	0	130
	16	0	325	0	0	0	0	325
	17	0	0	45	0	25	0	70
	23	0	0	125	80	0	0	205
	29	0	80	20	0	70	0	170
	31	0	0	65	0	10	0	75
	32	0	0	40	195	145	190	570
	Sub-total	*0*	*405*	*555*	*395*	*375*	*190*	*1,920*
1004-2	1	0	35	20	0	0	0	55
	2	0	35	20	0	0	0	55
	Sub-total	*0*	*70*	*40*	*0*	*0*	*0*	*110*
1006	7	0	0	60	60	40	0	160
	16	0	0	110	0	0	0	110
	Sub-total	*0*	*0*	*170*	*60*	*40*	*0*	*270*
1008-1	9	0	0	0	0	200	120	320
	Sub-total	*0*	*0*	*0*	*0*	*200*	*120*	*320*
1008-2	3	0	0	0	0	380	0	380
	Sub-total	*0*	*0*	*0*	*0*	*380*	*0*	*380*
Total Operating Development		0	475	765	455	995	310	3,000

Table 15-4: Operating lateral development breakdown (metres).

15.6 Equipment

The list of proposed major mobile equipment is shown in Table 15-5.

Item	Quantity	Comment
LHD 6yd^3	3	To be used in ramp development: 5 m wide by 5 m high
LHD 3yd^3	2	To be used in stopes (cut and fill, drift and fill, all approx. 3m wide)
Truck 30t	6	Haulage on max 15% grade over approx. 3 km
2 Boom jumbo	2	To be used in ramp development (5 m wide by 5 m high)
1 Boom jumbo	3	To be used in stopes (cut and fill, drift and fill, all approx. 3m wide)
Personnel carrier	2	Vehicle for carrying workers in and out of the mine
"Toyota" trucks	5	Vehicle for supervisors, etc.
Scissor lift	2	For special tasks (ground support, service, etc.)
Light vehicle	2	Vehicle for surveyor, etc.
Crane truck	1	For maintenance tasks

Table 15-5: WPZ list of major mobile equipment.

15.7 Production Rate and Life of Mine Plan

A mining rate varying from 600 tpd to 700 tpd was selected, mainly based on Taylor's Law[12] and the authors' experience related to this type of deposit. A pre-production period of approximately 18 months for dewatering and ramp development will be followed by approximately four years of production at an average mining rate of 675 tpd.

The LOM plan is shown in Table 15-6.

	2013	2014	2015	2016	2017	Total
Mining						
Average Daily Mining Rate (tpd)	230	624	673	655	745	
Total Tonnes Mined	41,389	218,326	235,536	229,255	260,750	985,256
Head Grade (g/t)	5.22	5.61	5.44	5.23	5.56	5.45
Ounces Mined	6,946	39,378	41,195	38,549	46,611	172,680
Milling						
Average Daily Milling Rate (tpd)	230	624	673	655	745	
Operating days per year	180	350	350	350	350	
Tonnes milled	41,389	218,326	235,536	229,255	260,750	985,256
Head Grade (g/t)	5.22	5.61	5.44	5.23	5.56	5.45
Ounces Mined	6,946	39,378	41,195	38,549	46,611	172,680
Mill Recovery	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%
Ounces Poured	6,564	37,213	38,929	36,429	44,048	163,182

Table 15-6: LOM plan.

[12] Taylor's Law (tpd) = $0.014*(\text{reserves tonnage})^{0.75}$

16.0 RECOVERY METHODS

16.1 Summary of Laboratory Test Work

The reader is referred to section 12.

16.2 Process Plant Flow Sheet

Description of the current milling process is summarized from a previous NI 43-101 technical report[13]; the process for treating Taylor ore is not expected to vary from the process described below.

The Holt Mill was constructed in 1988 and was originally designed for a throughput of 1,360 tpd. Expansions in 1988 and 2001 increased the throughput to 2,500 tpd and 3,000 tpd, respectively.

Surface ore storage is a total of 4,900 t in three silos, the Holt headframe bin (900 t) and two other separate storage bins (1,000 t and 3,000 t). Ore can be delivered to the mill from the Holt Mine by conveyor or from a separate surface dump that enters a 100 tonne hopper, and then can be fed to either of the two storage bins.

The grinding circuit consists of a 5 m diameter by 6.1 m long Allis Chalmers ball mill, converted to a SAG mill, a 4 m diameter by 5.5 m long Allis Chalmers ball mill and a 3.6 m diameter by 4.9 m long tertiary ball mill, all operating in series and in closed circuit. The details of the grinding circuit are shown below in Table 16-1. The grinding circuit is controlled by an expert system and fuzzy logic.

The primary cyclone cluster consists of six 381 mm (15") Krebs D15B cyclones. A secondary cyclone cluster consists of twelve 254 mm (10") Krebs gMAX cyclones with an Outokumpu PSI-200 online analyzer. The secondary cyclone cluster feeds a 27 m (90 ft) Eimco thickener. The thickener underflow feeds six carbon-in-leach (CIL) tanks. The tank system is conventional gravity flow for slurry with counter-current carbon advancement

Precious metal stripping is performed in batch operations, advancing 2.7 t of loaded carbon through a 1.2 m by 2.4 m (4 ft x 8ft) Simplicity screen. Carbon is transferred to an adsorption column where a Zadra process is utilized as the gold elution method. Barren solution is circulated through two shell and tube heat exchangers and a 360 kW electric inline heater.

[13] Technical Report on the Holloway-Holt Project, Ontario, Canada. NI 43-101 technical report from SWRPA. July 9th, 2008.

The resulting pregnant solution is pumped from the solution tank to an electro-winning cell. The gold precipitate is further refined using a 125 kW Inductotherm furnace and the doré bars are poured in a seven mould cascade arrangement. After stripping, the carbon is regenerated in a rotary kiln, quenched, screened and returned to the process. Carbon fines are collected in a tank, filtered in a Perrin press, and packaged for sale.

The process flow sheet is shown in Figure 16-1.

Reagents and operating supplies for the mill, such as process chemicals and grinding steel, are stored in the reagent storage building attached to the concentrator at the south end of the building.

Tailings Disposal

The tailings storage facility is located 2.5 km south of the process facilities. Tailings are pumped (at about 50% solids) via primary and booster pumps in a 254 mm HDPE pipe. The tailings pipe is equipped with two magnetic flow meters – one at the mill and the other at the tailings pond to monitor pressure differential. Excessive differences in flow will trigger an audible alarm. The tailings pipe is laid in a secondary containment capable of handling rock slurry in the event of a line burst.

The tailings storage facility is subdivided into four separate areas to receive the mill tailings and treat the solution that results from the milling of up to 3,000 tpd.

The quality of water discharge relies on natural degradation of cyanide and precipitants of other metals through the use of ferric sulphate. The annual discharge to the environment is closely monitored. The annual allowable discharge is a function of the water quality in the polishing pond and the flow rate in the receiving waters.

All of the tailings dams were designed by Golder, and Golder has carried out the dam safety inspections. The tailings management facility (TMF) is divided into a series of four basins: the North basin, the Southwest Basin, the Southeast Basin and the Polishing Pond (Figure 16-2). The North Basin was originally formed in a valley by the construction of perimeter dams. The capacity of the North Basin was increased in 1995 by raising the dams and constructing additional dams. The Southwest Basin, Southeast Basin and the Polishing Pond are contained by dams.

To date all of the tailings deposition has been into the North and Southwest basins. The current practice is to transfer tailings contact water from the North and Southwest basins into the Southeast Basin by pumping. Water is then transferred to the Polishing Pond from which it is released to the Magusi River. The effluent discharge is not *permitted to exceed one-tenth of the flow in the Magusi River in the summer months and one-fifth during spring and in the fall.*

There are five stages of TMF development, with associated permitted water levels and tailings levels. Up to 3.6 Mt of the tailings will be deposited in the Southwest and Southeast basins. Tailings deposition design is based upon an in situ void ratio of 1.0, a specific gravity of 2.65 and an inferred dry density of 1.33 t/m^3. Slopes of 0.6% were assumed for sub-aerial deposition and slopes underwater were assumed to be 2.5% slopes.

The ultimate storage capacity of the Southwest Basin is calculated to be 1.43 Mt of tailings. There is space for the deposition of 0.7 Mt before the dam heights must be increased. The Southeast Basin has the capacity for 2.17 Mt of tailings. Golder has also developed the water management plan to be used with the deposition plan.

Laboratory

The assay laboratory is located at the Holt site in an area near but separate from the mill and previously used as an assay lab. The building was renovated and a sample preparation area, fire assay facilities and an AA facility were established to provide analytical services for the site."

Data	Primary SAG mill	Secondary Ball mill #1	Tertiary Ball mill #2
Diameter (m)	5.0	4.0	3.6
Length (m)	6.1	5.5	4.9
Motor (hp)	3,400	1,650	1,250
Ball charge (%)	8-12	45	40
Grinding media	5" balls	2" balls	1" slugs
Media consumption (kg/t)	0.75	0.30	0.45
Speed (rpm)	13.9	16.2	17.3
Critical speed (%)	72.5	76.5	71.0
Circulating load (%)	10-15	350	225
Power draw (kWh)	2,250	1250-1450	750-900
Lifters	Polymet	Rubber	Rubber
Liners	Polymet	Rubber	Rubber
Discharge grates (mm)	18-30 mm by 40 mm	Overflow mill	

Table 16-1: Details of the grinding circuit.

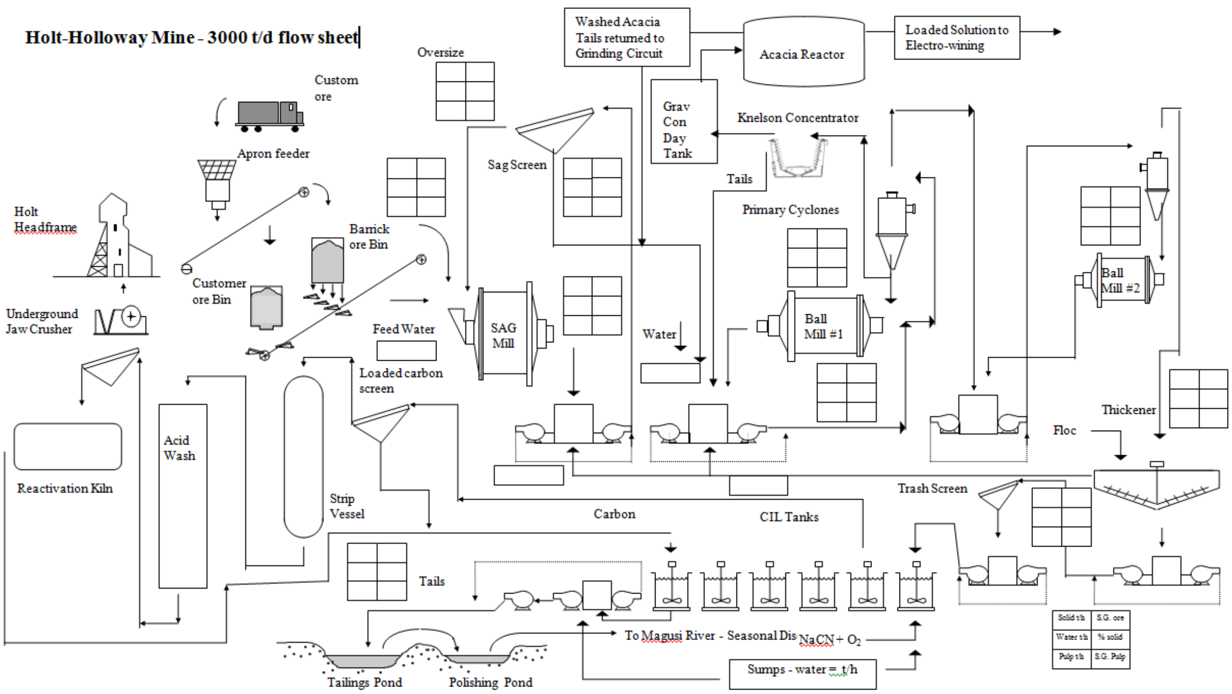

Figure 16-1: Process flow sheet.

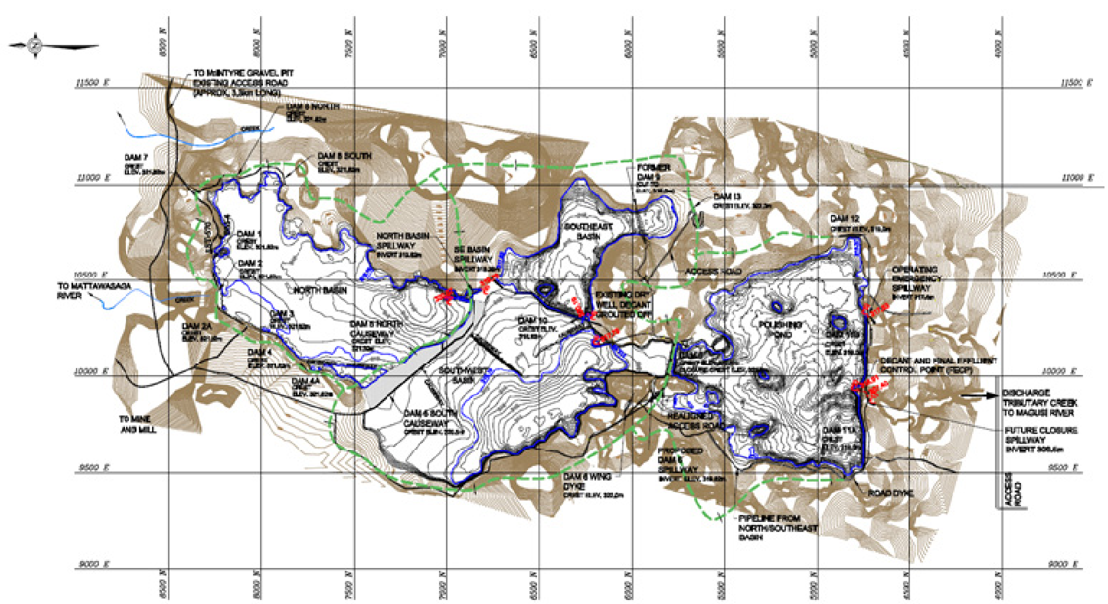

Figure 16-2: Tailings management facilities.

16.3 Potential Gravity Recovery Circuit Design

A potential gravity recovery circuit would consist of adding two Knelson concentrators to the current process flow: one receiving the primary cyclone underflow and the second being fed from the secondary cyclone underflow. The primary Knelson tails would be directed to the ball mill #1 and the primary Knelson concentrate would be collected for 24 hours in the gravity concentrate tank/ACACIA reactor feed tank. The secondary Knelson tails would be directed to the ball mill #2 and the secondary Knelson concentrate would be collected for 24 hours in the gravity concentrate tank/ACACIA reactor feed tank. The reactor would be filled once daily with concentrate. The previous day's concentrate, now reactor tails, would be returned to the grinding circuit after being washed free of residual cyanide.

In the QP's opinion, there are no processing factors or deleterious elements that could have a significant effect on potential economic extraction at Taylor.

17.0 PROJECT INFRASTRUCTURE

17.1 Surface Buildings

A few buildings were erected on site during a previous exploration and development phase of the Taylor Shaft Zone, namely:

- A security office;
- A hoist house (223 m^2) housing a 1.524m by 1.829 m double drum hoist;
- A collar house (130 m^2) and headframe, 30 m in height and constructed of steel;
- an electrical substation (electrical power is currently distributed to site by this substation);
- Engineering, Geology and Administration offices;
- Change room, wash room, shower and dry buildings;
- Mine office, meeting room and wicket area;
- Surface maintenance shop;
- Core logging and sampling facilities; and a,
- Cold storage building.

Some maintenance and structural repairs will be required for the offices and changing facilities. The associated costs are included in the Project capital expenditures.

17.2 Road Upgrade and Ore Transportation

Ore will be transported along the existing 1.7 km long access road to the Taylor site, then along Regional Road #11, and finally along Highway #101 until it reaches the Holt mill. The Taylor site access road will require to be widened and its base material be upgraded in order to accommodate the anticipated traffic and truck haulage flow. The associated costs are included in the Project capital expenditures.

17.3 Surface Stockpiles

Temporary ore stockpile are located with 200 m of the portal. Permanent waste stockpiles were permitted during the earlier stage of exploration and development work at the Taylor Shaft Zone; those stockpile areas are permitted and can be re-activated with minimal surface work. The associated costs are included in the Project capital expenditures

17.4 Tailings Deposition and Storage

The process plant for the Taylor ore is located at SAS' Holt mill, an operating facility that is fully permitted and can accommodate the tailings generated by processing the Taylor ore.

17.5 Power

The Taylor Project is currently serviced by a 1.5 MW sub-station, which has enough capacity for the execution of the current scope of work at Taylor.

Maintenance and upgrade of some elements of the distribution circuit will be required to bring the equipment at par with current regulation. The associated costs are included in the Project capital expenditures.

17.6 Underground Mine Dewatering and Fresh Water Requirements

Water in the existing underground excavations was estimated to be 142,000 m^3. Based on information received from personnel who previously worked at the Project site, there was no unusual amount of water being generated through the underground excavations. Consequently, the "steady-state" de-watering requirements were estimated at approximately 2,000 m^3/day, which is the daily limit of SAS current permit. Underground water will be pumped to settling ponds located on surface and then discharged in an appropriate receiver upon meeting water quality regulations.

Fresh water requirements are minimal since there is no process plant on site; this activity is covered by the appropriate and valid permit to take water.

17.7 Underground Mine Ventilation

Underground ventilation requirements and design were completed by an external consultant[14].

[14] Hatch, "St Andrew Goldfields Ltd. - Taylor Project Ventilation Design", January 3, 2012.

The "steady-state" ventilation system design for the Taylor Project will be a push system, in which fresh air is delivered through a series of intake ventilation raises and ultimately exhaust up the decline through the portal at surface. Based on the expected fleet of equipment that will be operating at the Taylor Project, the fresh air requirements are estimated at 203 m^3/s. Two 400 kW surface fans will be operating in series, each delivering approximately 102 m^3/s at a total pressure of 3.3 kPa.

Initially it is envisioned that the current decommissioned surface fans (115 kW each) can be re-used to deliver approximately 30 m^3/s air to the first fresh air raise (FAR) connection through steel spiral ducting. Once commissioned, the new fresh air system will deliver air through the FAR. This should cover the development activities of the Taylor Project, after which the ventilation system will be upgraded to the 400 kW fans as mentioned above.

Down ramp ventilation will rely on booster fans and steel spiral ducting leapfrogging from raise breakthrough to raise breakthrough, while ventilating the decline development.

Level development and stope ventilation will be provided by using auxiliary fans and PVC ducting.

Climate data for this report was based on Timmins, Ontario. Mine air will be heated to 2°C through the use of propane direct-fired heaters.

Ventilation associated costs are included in the Project capital and operating costs schedule.

17.8 Underground Material Handling

Simulations were undertaken with an external consultant to assess the truck and LHD fleet size requirements[15].

Results from the simulation work confirmed the adequacy of SAS' proposed fleet for loading and hauling (as per Table 15-5).

17.9 Communications

A leaky feeder system will be installed underground, facilitating radio communications.

[15] SANDVIK, "Loading and Hauling Simulations", personal communication, September 2011.

18.0 MARKET STUDIES AND CONTRACTS

18.1 Market for the Product

The QP has reviewed SAS contract with the refiner and he is satisfied that the contract reflects industry norms and reasonable market terms for selling Taylor gold production.

18.2 Material Contracts

SAS will negotiate with one or more contractors for the development and initial mining of the proposed bulk samples. It is anticipated to hire a contractor during the second quarter of 2012.

SAS anticipate extending the contract in force at another operation to haul the Taylor ore to the Holt mill facility.

Security services at the Taylor site are provided by Garda, an independent contractor.

19.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The Taylor site utilizes an Environmental Management System (EMS). This system embodies a recurrent review process of site environmental policies, procedures, permits and approvals. The EMS system continually audits and supports the waste and hazardous waste management plan, the water and wastewater treatment plan and environmental monitoring programs throughout the site.

This process is kept current though EMS revisions, which are included as part of the continuous improvement review cycle. Thus, the EMS forms the basis for the monitoring, sampling, and reporting program requirements under each of the relevant governmental agencies. More importantly, it verifies that all the activities at the Taylor site are in compliance with governments and company standards.

Underground and surface water are used as part of Taylor site sewage works. Water is collected, monitored, treated and released through an approved regulated industrial sewage works permit. This results in primarily controlled effluent discharge to the environment.

19.1 Summary of Environmental Studies

19.1.1 Terrestrial Environment

Surveys were undertaken in the past to provide further details on terrestrial vegetation and wildlife in areas that may be affected by mining activity, such as in the vicinity of the overburden and waste rock storage piles. Depending on the final stockpiles designs, additional studies may be warranted.

19.1.2 Hydrogeological Characterization

A number of investigations were completed to support the characterization of the groundwater regime in the vicinity of the Taylor project. Monitoring of groundwater levels in exploration holes was conducted, in order to determine the characteristics of the overburden aquifer. Data obtained during these tests were used to estimate the amount of groundwater that would potentially report to the mine from the overburden aquifer. Additional testing is underway on selected wells to help approximate in-situ hydraulic conductivity values for each screened interval. A three dimensional conceptual groundwater model was developed from the field data to predict the potential effects of mine development activities on the local groundwater and surface waters (e.g. drawdown effects).

19.1.3 Hydrological and Aquatic Habitat Assessments

Previous hydrological assessments were in large part developed by pro-rating regional flow data to the local watershed areas. Current studies are focusing on developing more accurate estimates of stream flows, runoff volumes and site drainage patterns associated with the existing mine site and future developments. Efforts were spent on detailed watershed mapping initiatives, as well as the development of a stream flow monitoring station on Wabbler Creek. This information will be crucial in assessing potential adverse environmental effects to the downstream aquatic receiving environment and assisting in storm water management planning activities.

Past aquatic habitat assessments were based on data collection initiatives recommended in prior studies, within the context of the proposed project; additional sampling of stream sediments, water chemistry and benthic macro invertebrates were also undertaken. Future aquatic assessment programs will be expanded to include areas that could potentially be affected by future mining activity. Of particular importance is the comprehensive assessment of potential fisheries habitat areas in the areas of proposed mine development.

19.1.4 Waste Characterization Studies

A detailed geochemical characterization of all mine waste materials will be completed in support of the development of an integrated water and waste management plan for the site.

In developing the mine model, waste and host rock materials have undergone a comprehensive geological classification to ascertain the total volumes of materials that will be generated. Representative samples from each type of waste material were selected and tested for their acid generating and metal leaching potential as per the relevant guidance documents. An eight sample study of waste rock deposited on surface was completed and results indicated very good neutralizing potential.

19.2 Tailing Management Plan

No process plant or tailings storage facilities are currently planned during the development and production activities at Taylor. Ore will be processed at the Holt site process plant where there are four individual ponds: two tailing ponds, one sludge precipitate pond and one polishing pond. Within these tailings facilities are 18 individual dam structures, a total of 465.4 ha of watershed area and 212 ha of tailings area. The remaining storage capacity is approximately 1.9 Mm^3. The tailings facilities are in compliance with all governmental regulations.

19.3 Permits Status and Posted Bonds

The reader is referred to section 3.3.

19.4 Social and Community

The permitting process requires Aboriginal and public consultations. As the project moves forward, permit changes will be required, which will include opportunities for First Nations and public feedback. No delays are anticipated for the development and operation of the Taylor deposit.

19.5 Closure Plan

As part of the Taylor site development phase, a closure plan was submitted to the appropriate government agencies. The mine received government approval of this closure plan in 2005. Amendments to the current closure plan include updated infrastructure details, mining plan, additional underground development and changes to surface features such as waste rock piles and overburden stockpiles.

20.0 CAPITAL AND OPERATING COSTS

20.1 Capital Costs

Capital costs were estimated by SAS at $48M (including 10% contingencies), or US$293/oz (over the LOM). The capital expenditures schedule is shown in Table 20-1.

	Total	2011	2012	2013	2014	2015	2016	2017
Total	**$47,506,755**	$238,040	$9,501,250	$18,752,614	$17,711,351	$984,500	$319,000	$0
DESCRIPTION								
Pre-feasibility report								
Enviro work	**$0**	$0	$0	$0	$0	$0	$0	$0
Metallurgical work	**$100,000**	$100,000	$0	$0	$0	$0	$0	$0
Geomechanical work	**$16,400**	$16,400	$0	$0	$0	$0	$0	$0
External consultants (vent, bfill, etc.)	**$100,000**	$100,000	$0	$0	$0	$0	$0	$0
Dewatering	**$1,000,000**	$0	$1,000,000	$0	$0	$0	$0	$0
Equipment								
Truck AD 30 (30 t)	**$5,305,446**	$0	$0	$1,768,482	$3,536,964	$0	$0	$0
LHD (2x10 t or 6 yd3 and 2x 3yd3)	**$4,113,513**	$0	$0	$881,549	$3,231,964	$0	$0	$0
Jumbo DD420 (1 Boom and 2 Boom)	**$4,536,000**	$0	$0	$1,079,700	$3,456,300	$0	$0	$0
Other mobile equipment	**$2,215,000**	$0	$0	$1,000,000	$1,215,000	$0	$0	$0
Ramp (@$5,100/m)	**$13,764,900**	$0	$5,599,800	$8,165,100	$0	$0	$0	$0
Stope/level access (@$5,100/m)	**$3,162,500**	$0	$0	$382,500	$1,980,000	$720,000	$80,000	$0
Escapeway raise								
Access (@$5,100/m)	**$0**	$0	$0	$0	$0	$0	$0	$0
Raise (@$3,500/m)	**$0**	$0	$0	$0	$0	$0	$0	$0
Ventilation/escapeway raise								
Access (@$5,100/m)	**$1,428,000**	$0	$637,500	$790,500	$0	$0	$0	$0
Raise (@$3,500/m)	**$2,835,000**	$0	$450,000	$2,070,000	$315,000	$0	$0	$0
Infrastructure								
Ventilation (fans, heaters)	**$4,043,200**	$0	$382,200	$910,000	$2,366,000	$175,000	$210,000	$0
Surface electrical	**$295,000**	$0	$295,000	$0	$0	$0	$0	$0
Surface building/road	**$273,000**	$0	$273,000	$0	$0	$0	$0	$0
Contingency (@10%)	**$4,318,796**	$21,640	$863,750	$1,704,783	$1,610,123	$89,500	$29,000	$0

Table 20-1: Capital expenditures schedule.

20.1.1 Basis of Estimate

Capital costs estimate for major items is based on budgetary quotations from suppliers in the industry. In order to account for unknown factors that should have been included in the estimate, a contingency of 10% was applied to the total.

20.1.2 Cost Estimate

Mining

SAS intends on completing the capital work using contractors. Based on quotes received from four contractors, a fair and reasonable unit cost was calculated (i.e. it is not an average of the quotes). Results are displayed in Table 20-2.

Capital costs related to development activities amount to $21.2 M ($23.3 M including contingencies).

Item	Range of unit costs	Selected costs
Ramp development (4m wide by 4.5m high; -15% grade)	$3,840/m - $9,036/m	$5,100/m
Sub-level and various accesses (+/- flat grade)	$3,840/m - $9,036/m	$5,100/m
Ventilation raise (4.3m dia.; include the surface prep work)	$4,257/m - $9,286/m	$4,500/m
Egress raise (1.5m by 1.5 m; includes ladderway)	$2,170/m - $5,605/m	$3,500/m
Dewater portal (flooded for 3 years)	$600,000-$3,800,000	$1,000,000
Cut and fill/drift and fill stoping (3m wide by 3m high)	$107/t - $132/t	$120/t
Open stoping (20m sub spacing; 3m wide on average; 50 deg. dip)	$25/t - $77/t	$50/t

Table 20-2: Budgetary quotations from various mining contractors.

Equipment

Budgetary quotations were received for the equipment and amounted to $16.2 M ($17.8 M including contingencies):

- Trucks (30 t payload): $885,000 per unit (total of $5.31 M);
- Load-Haul-Dump (6 yd3 or 10 t): $882,000 per unit (total of $2.65 M);
- LHD (3 yd3 or 5 t): $735,000 per unit (total of $1.47 M);
- Jumbo (2-boom): $1,079,000 per unit (total of $2.16 M);
- Jumbo (1-boom): $793,000 per unit (total of $2.38 M);
- Scissor lift: $360,000 per unit (total of $0.72 M);
- Personnel carrier: $290,000 per unit (total of $0.58 M);
- “Toyota” type vehicle: $100,000 per unit (total of $0.50 M);

- Light vehicle: $50,000 per unit (total of $0.10 M)
- Crane truck: $316,000 per unit (total of $0.32 M).

Based on a “contractor-operated” scenario for the first 24 months, no capital expenditures for mobile equipment are included in the early years of the Life-of-Mine (LOM) economic model.

Infrastructure

Infrastructure costs were estimated at $4.6 M ($5.1 M including contingencies).

Quotations were received for upgrading the Taylor site surface infrastructure ($0.6 M) and ventilation equipment ($4.0 M).

Other capital costs

Dewatering and rehabilitation of the existing decline was estimated at $1.0 M, based on quotes from contractors.

Test work for the PFS was estimated at $0.2 M.

Contingency

A contingency of 10% was applied to the total capital estimate to cover the estimate accuracy (± 25%) and items that were omitted unknowingly.

20.2 Operating Costs

Total operating costs are estimated at $148.5 M, broken down as follows:

- Mining: $109.5 M (or $111.2/t)
- Milling: $17.7 M (or $18.0/t)
- G&A: $7.0 M (or $7.1/t)
- Trucking: $10.0 M (or $9.9/t)
- Royalties (2%): $4.3 M (or $4.4/t)

Operating unit costs amounts to $150.7/t or US$903/oz.

The operating costs schedule is shown in Table 20-3.

	2013	2014	2015	2016	2017	Total
Mining	$4,610,000	$24,436,000	$25,911,000	$25,252,000	$29,328,000	$109,537,000
Milling	$745,000	$3,930,000	$4,240,000	$4,127,000	$4,694,000	$17,735,000
G&A	$295,000	$1,554,000	$1,677,000	$1,632,000	$1,857,000	$7,015,000
Trucking	$414,000	$2,183,000	$2,355,000	$2,293,000	$2,608,000	$9,853,000
Royalties	$175,000	$990,000	$1,036,000	$968,000	$1,171,000	$4,340,000
Total	$6,238,000	$33,094,000	$35,219,000	$34,271,000	$39,657,000	$148,480,000

Table 20-3: Operating costs schedule.

20.2.1 Basis for Estimate

Quotations were obtained for units of work that will be contracted out.

Operating costs for units of work that will be carried out by SAS personnel were based on SAS budget figures for 2012.

20.2.2 Cost Estimate

Breakdown of operating unit costs by stope is shown in Table 20-4.

Stope		Stope preparation	Infrastructure	Stoping	G&A	Milling	Trucking	Royalties	Unit Costs	Total Costs
1004-1	2	$26.74/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$6.50/t	$174.36/t	$720,562
1004-1	3	$10.66/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$6.38/t	$158.16/t	$1,639,591
1004-1	4	$13.78/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$4.11/t	$159.02/t	$4,804,630
1004-1	8	$6.61/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$4.09/t	$151.82/t	$5,073,224
1004-1	16	$6.77/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$3.86/t	$151.75/t	$12,389,280
1004-1	17	$1.03/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$4.15/t	$146.30/t	$16,958,705
1004-1	23	$17.76/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$7.34/t	$166.22/t	$3,261,213
1004-1	29	$2.23/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$4.30/t	$147.65/t	$19,150,349
1004-1	31	$0.93/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$4.80/t	$146.85/t	$20,112,065
1004-1	32	$3.03/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$4.06/t	$148.22/t	$47,364,249
1004-2	1	$8.71/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$6.61/t	$156.44/t	$1,678,864
1004-2	2	$5.90/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$4.29/t	$151.30/t	$2,399,435
1006	7	$10.66/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$4.06/t	$155.83/t	$3,977,582
1006	16	$14.61/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$4.32/t	$160.06/t	$2,047,977
1008-1	9	$19.63/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$4.55/t	$165.30/t	$4,579,903
1008-2	3	$58.16/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$9.79/t	$209.07/t	$2,322,060
Average or total		$5.18/t	$6.00/t	$100.00/t	$7.12/t	$18.00/t	$10.00/t	$4.41/t	$150.70/t	$148,479,690

Table 20-4: Operating unit cost break down by stope.

Mine

The average unit operating costs of $111/t were largely based on SAS budget figures for 2012 with the exception of stoping costs, which were derived from contractor stoping quotes. In summary:

- Stoping unit costs: $100/t;
- SAS surface and infrastructure cost: $6/t;
- Stope preparation and development unit cost: $5/t;
- Development unit costs: $1,700/m.

Mill

The average unit operating costs of $18/t were based on SAS budget figures for 2012.

G&A

The average unit operating costs of $7/t were based on SAS budget figures for 2012.

Haulage to Mill

The average unit trucking costs of $10/t were estimated from the current contract in place at the Hislop mine.

Royalties

Royalties were calculated at 2% of the revenues from gold produced.

21.0 ECONOMIC ANALYSIS

21.1 Summary results

Results from the economic analysis demonstrate:

- A pre-tax cumulative undiscounted cash flow of $20 M;
- A pre-tax $NPV_{5\%}$ of $12 M;
- A pre-tax internal rate of return of 22%; and,
- Pre-tax operating cash unit costs of US$903/oz ($150/t); pre-tax total unit costs of US$1,193/oz.

21.2 Methodology

The economic model was valued using the traditional discounted cash flow method (DCF). The price of gold varied yearly according to a schedule provided by SAS financial advisor, which reflects "street consensus" (average of US$1,319/oz over the life of the project). The currency exchange rate also followed a schedule provided by SAS financial advisor (on average, the Canadian dollar is at par with the US dollar over the life of the project).

No escalation was applied to operating and capital costs.

A net present value (NPV) was calculated using a discount rate of 5%.

The pre-tax undiscounted cash flow model (and associated NPV) represents the "base case scenario" of the economic model.

Sensitivity analyses on the undiscounted cash flow and NPV were completed by varying the price of gold, head grade, operating costs and capital costs by a range of ±25%.

21.3 Annual Cash Flow Forecast

The annual cash flow forecast is presented in Table 21-1.

		2011	2012	2013	2014	2015	2016	2017	Total
Market Criteria									
Gold Price (US$/Oz)		$1,575	$1,750	$1,650	$1,491	$1,300	$1,250	$1,200	
CDN$/US$		0.99	1.02	1.00	0.99	1.01	1.01	1.01	
Gold Price (CDN$/oz)		$1,559	$1,785	$1,650	$1,476	$1,313	$1,263	$1,212	
Depletion									
Tonnes at beginning of year		985,256	985,256	985,256	943,867	725,541	490,005	260,750	
Grade (g/t)		5.45	5.45	5.45	5.46	5.41	5.40	5.56	
Metal at beginning of year (oz)		172,638	172,638	172,638	165,692	126,313	85,118	46,569	
Mining									
Average Daily Mining Rate (tpd)		0	0	230	624	673	655	745	
Total Tonnes Mined		0	0	41,389	218,326	235,536	229,255	260,750	985,256
Head Grade (g/t)		0.00	0.00	5.22	5.61	5.44	5.23	5.56	5.45
Ounces Mined		0	0	6,946	39,378	41,195	38,549	46,611	172,680
Milling									
Average Daily Milling Rate (tpd)		0	0	230	624	673	655	745	
Operating days per year		0	0	180	350	350	350	350	
Tonnes milled		0	0	41,389	218,326	235,536	229,255	260,750	985,256
Head Grade (g/t)		0.00	0.00	5.22	5.61	5.44	5.23	5.56	5.45
Ounces Mined		0	0	6,946	39,378	41,195	38,549	46,611	172,680
Mill Recovery		0.0%	0.0%	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%
Ounces Poured		0	0	6,564	37,213	38,929	36,429	44,048	163,182
Ounces lost		0	0	382	2,166	2,266	2,120	2,564	9,497
Revenues		**$0**	**$0**	**$10,830,835**	**$54,929,254**	**$51,114,424**	**$45,991,168**	**$53,385,618**	**$216,251,299**
Operating Costs									
Mining		$0	$0	$4,610,213	$24,435,853	$25,911,464	$25,251,642	$29,328,082	$109,537,254
Processing		$0	$0	$745,001	$3,929,868	$4,239,653	$4,126,598	$4,693,505	$17,734,625
G&A		$0	$0	$294,689	$1,554,481	$1,677,018	$1,632,299	$1,856,542	$7,015,030
Trucking		$0	$0	$413,890	$2,183,260	$2,355,363	$2,292,555	$2,607,503	$9,852,570
Total Mine Cash Costs		$0	$0	$6,063,793	$32,103,461	$34,183,498	$33,303,094	$38,485,631	$144,139,478
Royalty, NSR	2%	$0	$0	$174,610	$990,164	$1,035,556	$968,383	$1,171,470	$4,340,182
Cash Operating Costs		**$0**	**$0**	**$6,238,403**	**$33,093,625**	**$35,219,054**	**$34,271,477**	**$39,657,100**	**$148,479,660**
Operating Income		**$0**	**$0**	**$4,592,431**	**$21,835,629**	**$15,895,370**	**$11,719,691**	**$13,728,518**	**$67,771,639**
Capital Costs		**$238,040**	**$9,501,250**	**$18,752,614**	**$17,711,351**	**$984,500**	**$319,000**	**$0**	**$47,506,755**
Pre-Tax Cash Flow		**-$238,040**	**-$9,501,250**	**-$14,160,183**	**$4,124,278**	**$14,910,870**	**$11,400,691**	**$13,728,518**	**$20,264,884**
Cumulative Pre-Tax Cash Flow		-$238,040	-$9,739,290	-$23,899,473	-$19,775,194	-$4,864,324	$6,536,366	$20,264,884	
Mine Unit Cash Cost Per Ounce	US$/Oz	$0	$0	$950	$898	$896	$931	$891	**$903**
Non Cash Unit Costs	US$/Oz	$0	$0	$291	$288	$294	$294	$294	**$293**
Pre-Tax Unit Total Costs	US$/Oz	$0	$0	$1,242	$1,187	$1,190	$1,226	$1,185	**$1,197**
Tax	US$/Oz	$0	$0	$0	$0	$0	$0	$0	$0
Total Unit Cost Per Ounce	**US$/Oz**	**$0**	**$0**	**$1,242**	**$1,187**	**$1,190**	**$1,226**	**$1,185**	**$1,197**

		Before Tax			Before Tax
NPV	**0%**	$20,264,884	Value $/Share @ NPV	**0%**	$0.05
	3%	$15,091,536		**3%**	$0.04
	5%	$12,263,366		**5%**	$0.03
	8%	$8,767,375		**8%**	$0.02
	IRR	22%			

Table 21-1: Annual cash flow forecast.

21.4 Discussion on NPV, IRR and Payback

In the QP's opinion, the economic analysis was conservative by using a discounted price of gold from the current price (i.e. approximately US$1,300/oz versus US$1,700/oz observed in March 2012), capital costs and mining costs based on budgetary quotations that can improve during the construction, development and production phases of the Taylor Project; nonetheless, the NPV and IRR values calculated from the DCF analysis meet or exceed SAS requirement for determining the project as positive. All parameters were calculated as of 2011, including the payback of approximately four years.

21.5 Summary of Taxes, Royalties and Other Government Levies

SAS' current fiscal situation includes a "tax pool" of approximately $212 M, which is intended to be used to reduce future income tax payments until approximately 2017 and future mining tax payments until approximately 2013. Consequently, the impact of taxes on the project cash flow was ignored.

Royalties were calculated at 2% NSR for the WPZ.

No other government levies are foreseen or included in the economic analysis.

21.6 Sensitivity Analysis

Results from a sensitivity analysis are plotted in Figure 21-1 for the pre-tax undiscounted cash flow and Figure 21-2 for the pre-tax NPV; it is apparent that gold price and implicitly the head grade have the strongest influence on the economic performance of the Taylor Project, followed by operating costs and, to a lesser degree, capital costs.

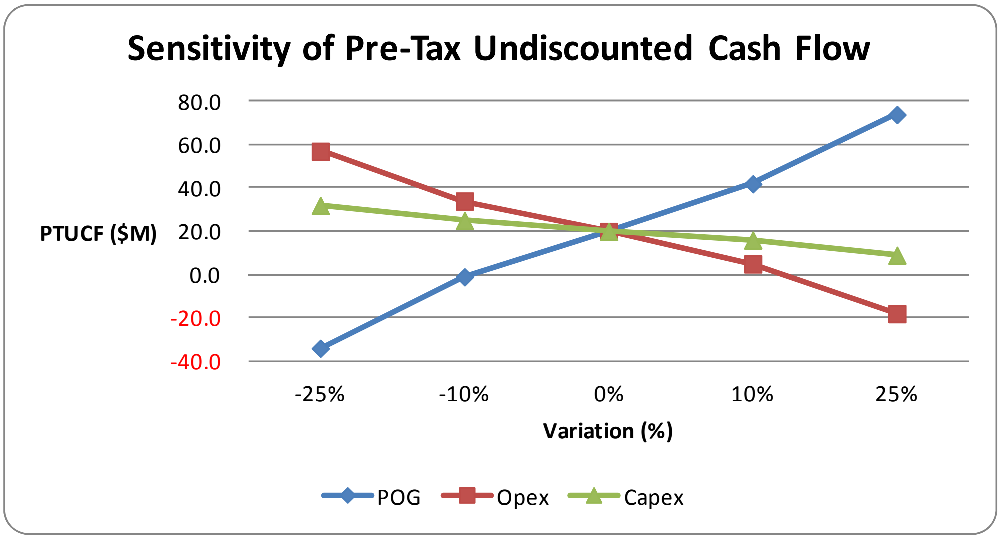

Figure 21-1: Sensitivity analysis of the pre-tax undiscounted cash flow.

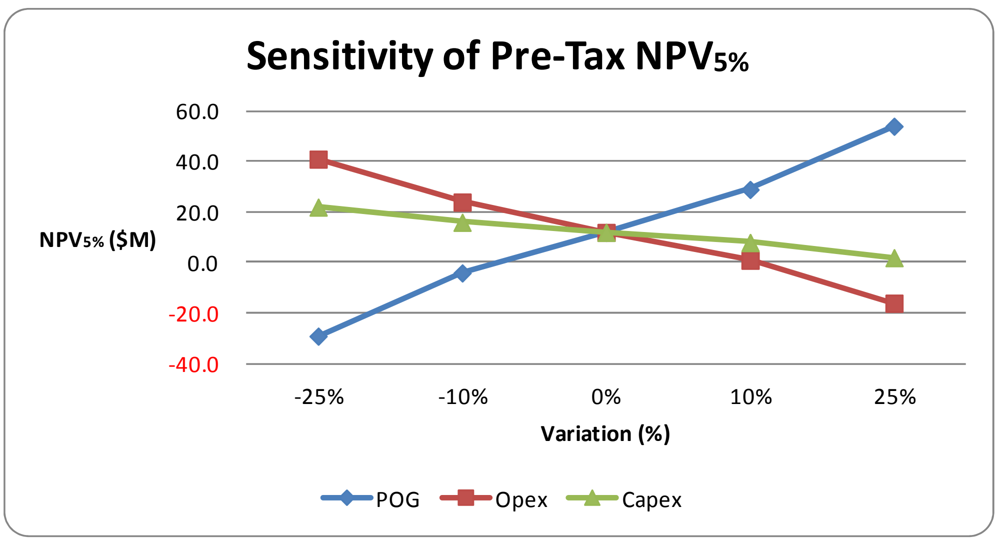

Figure 21-2: Sensitivity analysis of the $NPV_{5\%}$.

22.0 ADJACENT PROPERTIES

There are no adjacent properties to the Taylor Project that are material to the scope of this technical report.

23.0 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information on the Taylor Project known to the QPs that if undisclosed would make this NI 43-101 technical report misleading or more understandable.

24.0 INTERPRETATION AND CONCLUSIONS

24.1 General

Field work carried out on the Taylor property since 2010 was compiled by SAS employees who updated the 2009 mineral resources. Further technical work was completed in 2010, 2011 and 2012, leading to the preparation of a pre-feasibility study report (PFS) on the West Porphyry Zone (WPZ) only. Highlights of the PFS include:

- First time disclosure of approximately 1 Mt grading 5.45 g/t (173 koz) of probable mineral reserves;
- Pre-tax cumulative undiscounted cash flow of $20 M;
- Pre-tax NPV5% of $12 M;
- Internal rate of return of 22%
- Cash operating unit costs of US$903/oz ($150/t); total costs of US$1,193/oz;
- Production at an average mining rate of 675 tpd during approximately four years;
- Pre-production and development activities for approximately two years.

The WPZ hosts indicated mineral resources of 1.9 Mt grading 5.50 g/t (341 koz) and inferred mineral resources of 1.9 Mt grading 3.95 g/t (243 koz). The indicated mineral resources were converted into approximately 1 Mt grading 5.45 g/t (173 koz).

Cut and fill was selected as the mining method for the WPZ. Dilution was applied individually on each shape, ranging from 9% to 50%, and an extraction factor of 95% was set for all stopes. Based on test work performed at SGS laboratories, mill recovery averaged 94.5%.

Based on the above and in order to minimize the financial risk to SAS, it is concluded that a stepped approach is warranted and appropriate to pursue development and exploration activities at the Taylor Project.

A review of opportunities and risks associated with the development of the West Porphyry Zone at the Taylor Project is presented in section 24.2 and section 24.3, respectively.

24.2 Opportunities

- Contractor rates used in operating and capital costs estimates are budgetary quotes; the actual rates may improve;

- The installation of a gravity recovery circuit may improve the overall recovery by 1% to 2% based on recent test work;

- Displacing Hislop ore feed with Taylor's does not make a significant difference on SAS cash flow (i.e. company-wide); however, produced ounces are improved by 25% over the LOM by introducing Taylor ore in the ore feed stream;

- It is postulated that the mill capacity is increased when processing Taylor ore instead of Hislop ore due to favourable properties of the Taylor ore (i.e. higher processing rate of Taylor ore should translate in an overall increase in processed tonnage over the year for all feed sources). As a result, processing Taylor ore instead of Hislop ore should lead to a reduction in operating costs and unit costs;

- The current and short-term price of gold is higher than the price input in the economic model (e.g. approximately US$1,600/oz versus US$1,325/oz over the LOM);

- Mineral reserves were converted conservatively using individual stopes operating cash flow in the cut-off grade determination. For example, eight marginal stopes (i.e. with negative operating cash flow of less than approximately -$350,000) may turn a positive cash flow by increasing the price of gold in the range of US$1,350/oz to US$1,640/oz, or by reducing the operating costs in the range of 1% to 19%.

- Geology re-interpretation based on information gained through additional drilling and underground sampling may lead to additional mineral resources (and possibly to additional mineral reserves).

24.3 Risks

- Continuity of ore zones not well defined or understood;

- Cost estimates from the PFS lower than "actual" costs expensed during the project life;

- Price of gold decreasing below the modelled range.
- Delayed project start-up date;
- Contractor or workforce availability;
- Conditions of existing infrastructure; and,
- Although an IBA is not required under current legislation to develop and bring the Taylor Project into production, SAS is continuing negotiations with First Nations.

25.0 RECOMMENDATIONS

In order to minimize the financial risk to SAS, a stepped approach is proposed, which will require SAS' Board of Directors approval before proceeding to the next step:

1. Complete a bulk sample in upper area of the deposit ($1 M):

    - Dewater and rehab the existing ramp using contractors ($1.1M);
    - Upgrade surface infrastructure (power, offices, dry, etc.) ($1.5M);
    - Validate geological model with drilling from underground ($0.2M);
    - Drive approximately 300 m to the 1008-2/03 stope ($1.5M);
    - Mine approximately 11,000 t grading 12 g/t and confirm mining method is appropriate ($2.0M);
    - Assess milling results;
    - Estimated cost: ($6.3M);
    - Estimated revenues: 4,086 oz at $1,300/oz = $5.3M;
    - Estimated cash flow: ($1.0M);
    - Estimated time: approximately 6 months following contractor mobilization.
    - Pending favourable results, seek SAS Board of Directors approval for the next phase.

2. Complete a second bulk sample near the -90 elevation ($15 M):

    - Upgrade ventilation system and surface infrastructure ($1.5M);
    - Develop the ramp down approximately 2,000 m ($11.2M);
    - Ventilation raiser #1 access: 125 m ($0.7M);
    - Ventilation raise #1: 100 m ($0.4M);
    - Lateral development from the ramp = 205 m ($1.2M);

- Validate geological model by drilling underground ($0.5M);
- Mine partially from a few stopes at -100 elevation, approximately 7,000 t grading 6 g/t and confirm mining method is appropriate ($1.2M);
- Assess milling results;
- Estimated cost: ($16.7M);
- Estimated revenues: 1,293 ounces at $1,300/oz = $1.7M;
- Estimated cash flow: ($15.0M);
- Estimated time: approximately 12 to 14 months;
- Pending favourable results, seek SAS Board of Directors approval to continue the project as per the LOM plan using SAS employees.

26.0 REFERENCES

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27.0 SIGNATURE PAGE AND DATE

The undersigned prepared this technical report titled "Taylor Property, Ontario, Canada, NI 43-101 Technical Report". The effective date of this Technical report is December 31, 2011 and the disclosure date is March 30, 2012.

Signed,

"signed and sealed"

Pierre Rocque, P. Eng.	March 29, 2012	St Andrew Goldfields Ltd. 20 Adelaide Street East, Suite 801 Toronto, Ontario, M5C 2T6 Canada

"signed and sealed"

Craig Todd, P. Geo	March 29, 2012	St Andrew Goldfields Ltd. 489 MacDougall Avenue, P. O. Box 249, Matheson, Ontario, P0K 1N0 Canada

CERTIFICATE OF QUALIFIED PERSON

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I, Pierre Rocque, P. Eng., am employed as a Director of Engineering with St Andrew Goldfields Ltd.

This certificate applies to the technical report entitled "Taylor Property, Ontario, Canada, NI 43-101 Technical Report" with an effective date of December 31, 2011.

I am a member of Professional Engineers of Ontario and Ordre des ingénieurs du Québec. I graduated in 1986 from École polytechnique de Montréal with a Bachelor's degree in Mining Engineering (BIng.) and in 1992 from Queen's University at Kingston with a Master's degree in Mining Engineering (M.Sc.Eng.).

I have practiced my profession for twenty five years. I have been directly involved in mine design of underground gold mines and, since 1997 I have overseen the mining engineering department at three narrow veins underground gold mines, providing relief to the Mine Manager and General Manager on site. Since 2008, I have provided corporate direction for the engineering function at junior gold exploration and producing companies.

I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101 for this report.

I visited the Taylor property on numerous occasions since 2010.

I am responsible for the preparation of the Summary and Sections 1 to 4, 12, 14 to 21 and 23 to 26 of the technical report entitled "Taylor Property, Ontario, Canada, NI 43-101 Technical Report" dated March 29, 2012 and with an effective date of December 31, 2011.

I am not independent of St Andrew Goldfields Ltd. Independence is not required under Section 5.3 (3) of NI 43–101.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"Signed and Sealed"

Pierre Rocque, P. Eng.
Director of Engineering

CERTIFICATE OF QUALIFIED PERSON

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I, Craig Todd, P.Geo., am employed as the Exploration Manager with St Andrew Goldfields Ltd.

This certificate applies to the technical report entitled "Taylor Property, Ontario, Canada, NI 43-101 Technical Report" with an effective date of December 31, 2011.

I am a member of Association of Professional Geoscientist of Ontario. I graduated in 1979 from Laurentian University in Sudbury with an Honours B.Sc. in Geology.

I have practiced my profession for over thirty years. I have been Chief Geologist at several gold and base metal mines since 1995 and have been responsible all geological functions including calculating and reporting Resources and Reserves. Since 2010, I have been Exploration Manager overseeing surface exploration activities on the company's extensive land package.

I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101 for this report.

I visited the Taylor property on numerous occasions since 2010.

I am responsible for the preparation of the Summary and Sections 5 to 11, 13 and 22 to 26 of the technical report entitled "Taylor Property, Ontario, Canada, NI 43-101 Technical Report" dated March 29, 2012 and with an effective date of December 31, 2011.

I am not independent of St Andrew Goldfields Ltd. Independence is not required under Section 5.3 (3) of NI 43–101.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"Signed and Sealed"

Craig Todd, P. Geo.
Exploration Manager