Updated NI 43-101 Technical Report on the Timmins Mine Property, Ontario, Canada

PREPARED FOR: **LAKE SHORE GOLD CORP.**

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Authors:
George Darling, P. Eng.
Todd Fayram, Q. P. MMSA
Robert Kusins, P. Geo.
Jacques Samson, P. Geo.
Heather Miree, P. Geo.

EXECUTIVE SUMMARY (ITEM 3)

History

The Timmins Mine Project (Project) of Lake Shore Gold Corp. (LSG) is located in Northern Ontario, within the city limits of Timmins (population 45,000). Timmins was established in 1912 as a by-product of the Porcupine Gold Rush and it is still one of the richest mineral producing areas in the western hemisphere. Strategically located in the heart of the Porcupine gold camp, it is easily accessible to the more densely populated areas of Southern Ontario, 300 km north of Sudbury and 700 km north of Toronto, by highway, rail and air transport. The Project property is located at the intersection of Highways 101 and 144 approximately 20 km west of Timmins.

Gold was discovered on the Project property in 1911 but did not receive serious exploration attention until the 1990's when Holmer Gold Mines Limited ("Holmer") completed 44 diamond drill holes totalling about 9,000 m and issued a mineral resources estimate. LSG entered into an option agreement with Holmer in 2003, continued drilling and updated the mineral resources estimate in 2004, in accordance with National Instrument 43-101 ("NI 43-101"), to 1.3 Mt at a grade of 10.96 g/t of gold in the Indicated category.

In December 2004, LSG acquired a 100% interest in the Project and immediately conducted an aggressive deep drilling campaign that was completed in October 2006.

In January 2007, LSG signed a Letter Agreement with Goldcorp Canada Ltd., manager of the Porcupine Joint Venture (a joint venture between Goldcorp Canada Ltd. and Kinross Gold Corporation), to acquire the Bell Creek mine and mill facilities, adjacent to LSG Vogel-Schumacher properties.

In April 2007, LSG received notice of acceptance of its certified closure plan for the Project from the Ontario Ministry of Northern Development and Mines, allowing the initiation of an advanced exploration program (AEP).

In August 2007 LSG released the results of a prefeasibility study for the Project, which supported the economic development of the property. In October 2007 LSG filed a NI 43-101 Technical Report on the Project property, estimating Indicated mineral resources of 3.3 Mt grading 8.62 g/t Au (905,000 contained ounces of gold) and Inferred mineral resources of 0.97 Mt grading 5.62 g/t Au. The mineral reserves were estimated at 3.4 Mt grading 7.59 g/t (826,000 contained ounces of gold) in the Probable category.

At the end of July 2008, LSG commenced shaft sinking at the Project location. The 710 m shaft is expected to be completed during the third quarter of 2009. A program of level development, bulk sampling and underground diamond drilling is scheduled for the remainder of 2009. In early September 2008, LSG commenced driving a ramp from surface to access mineral reserves above the 400 meter level.

In March 2009, LSG began processing development material from the Project at the Bell Creek Mill, which was refurbished to a capacity of 1,500 tpd. LSG forecasts pouring 30,000 ounces of gold in 2009 from the processing of development material.

In August 2009 LSG and WTM agreed to a business combination which triggered an update to the current NI43-101 Technical Report on the Project.

Mineral Resources Estimate

Mineral resources of 3.2 Mt grading 8.56 g/t Au (893,000 contained ounces) or 12.24 g/t Au uncut (1,278,000 oz) were estimated in the Indicated category. The estimation process followed CIM guidelines, in accordance with NI 43-101 definitions.

In addition, Inferred mineral resources of 0.89 Mt grading 5.74 g/t Au (165,000 contained ounces) were estimated. The deposit remains open down-plunge.

The gold mineralization occurs in ten geological zones. In the Main Zone and the three Vein zones, mineralization is associated with quartz/tourmaline veining and stringers along with small, varying amounts of pyrite and arsenopyrite. Mineralized zones are typically 1 to 7 m wide in the Veins and Main zone. In the three Ultramafic and three Footwall Zones, the gold values occur mainly within the alteration halo adjacent to the veins in zones up to 20 m wide and are closely related to the pyrite content. These types of mineralization are typical of deposits in the Porcupine mining camp.

Mineral Reserves Estimate

Mineral reserves of 3.4 Mt grading 7.52 g/t (812,000 ounces contained) were estimated in the Probable category, using a 3.00 g/t cut-off grade. The estimation process followed CIM guidelines, in accordance with NI 43-101 definitions.

The Consultant's estimate of underground mineral reserves is based on LSG's polygonal and block model (the latter covering the veins and main zones within 120 metres from surface) used for mineral resources estimation. The Consultant converted the mineral resources models into several wire frames and modeled them using Mine2-4D software.

The Consultant then validated this conversion by cross-referencing back to the original LSG mineral resources estimate.

Mine Plan

The mineral reserves estimate was based on the following parameters developed by the Consultant as part of the mine plan:

- Mining recovery: 86 to 92%;
- Dilution: 12% for cut and fill mining and 27% for long-hole stoping, using dilution grades of 0 and 1 g/t Au respectively;
- Minimum mining width: 2.0 to 3.5 m;
- Mill recovery: 95%; and
- Total site operating cost: $95.08/t ore processed.

Project parameters and highlights are shown in Table S-2.

The mine plan is based on utilizing the shaft and ramp to surface facilities that remain from the Advanced Exploration Project (AEP). The Consultant has estimated that after completion of the AEP ($140M), a pre-production capital cost (Capex) of $33M will be required to start production. Once completed, the production phase Capex is estimated at $29M. The planned production rate of 1,500 t/d was based on the selected mining methods and the steeply dipping orebody geometry. Ore will be loaded onto highway trucks and hauled to the Bell Creek Mill, approximately 42 km from the mine site. Mine waste rock will be utilized as backfill material in the mine.

The total Capex is estimated at $202M.

Processing

The Project gold deposit appears to be very amenable to cyanide leaching, yielding a high recovery rate, typically in excess of 95%. Based on extensive test work, a preliminary process design was developed using conventional ball milling followed by agitated cyanide leaching and carbon in pulp (“CIP”) gold recovery.

The initial ore treatment design rate is 1,500 tpd with approximately 29,000 ounces of gold produced per quarter over the seven year production life (i.e. approximately 772,000 ounces recovered).

The original mill had been expanded a number of times until the 1,500 tpd capacity was reached in 2001. The main equipment includes two grinding mills, rated at 1,400 and 400 hp, five leach tanks, eight CIP tanks, and a gold recovery circuit based on carbon - pressure. The mill was under care and maintenance from 2002 until start-up in May 2009.

The BC Mill also includes an adjacent conventional tailings disposal facility that has been determined to be suitable and expandable to accommodate the present mineral reserves at the Project. Permits are still valid as the entire complex has been kept on 'operational' status with the regulators. Modifications to account for the properties of the Project's ore were incorporated in the implementation program and are consistent with the planned initial 1,500 tpd mining rate.

The re-commissioning of the BC Mill, including upgrading of the tailings and water management installations was completed for $1.8M. Expansion of the tailings facility to ultimately accommodate about an additional 5 Mt has been determined to be feasible. Staged expansions have been estimated at a cost of $2.4M for the 3 million tonnes of tailings expected from the planned Project ore over the seven year mine life. Total milling costs have been estimated at $17.62/t ore processed.

Work Force

Total payroll is calculated at 129 at the Project site and 34 at the BC Mill. Staff of 12 management and administration personnel will be required, for a total workforce of 175.

Economics

Cost estimates and discounted cash flow analysis indicate that the Project will be potentially economic, as shown in Table S-3 (Canadian dollars unless stated otherwise); the execution of an AEP is therefore justified.

Only the pre-tax economics are presented as the Project benefits from large tax write-off pools that are expected to reduce the after-tax internal rate of return (IRR) by only minimal amounts. The pre-tax economic indicators are positive, as indicated by an IRR of 28% (US$950/oz gold price) when taking the AEP cost into account or 240% when ignoring AEP costs expensed to date.

Table S-2: Project Updated Parameters and Highlights.

Mineral reserves		3.4 Mt	7.52 g/t Au (cut); 812,000 oz. Au (base case)
Mining rate		1,500	Tonnes per day
Tonnage mined per year at Timmins Mine		532,500	Tonnes
Mine life		7.5	Years
Minimum mining widths		2.0	Metres
Mining method: primary			Open stoping (long hole sublevel and Uppers mining)
Mining method: secondary			Mechanized cut and fill
Dilution		12 to 27 %	
Milling:			Cyanide leaching and CIP recovery
Metallurgical recovery		95%	
Trucking distance (app.)		40.0	Kilometres
Capital expenditures			
Pre-production advanced exploration capital	$	140	Million
Capital	$	33	Million
Sustaining capital (including mine closure)	$	29	Million
	$	202	Million total Capex
Operating Costs			
Mining	$	69.66	Per tonne of ore
Processing	$	17.62	Per tonne of ore (assumes no other source of mill feed)
Trucking	$	5.44	Per tonne of ore
G&A	$	2.36	Per tonne of ore
Project operating costs	$	95.08	Per tonne of ore total
Cash cost per ounce	US$	369	Per ounce base case (production phase)
Project payback		3	Years
Production start		Q4	2010
IRR (pre tax, production phase)		28% 240%	(Base case including AEP costs) (excluding AEP costs expensed to date)
NPV for the Base Case At 0% At 5% At 8% At 15%	$M	 300 184 135 62	 (No AEP cost: $425M) (No AEP cost: $295M) (No AEP cost: $240M) (No AEP cost: $153M)

Table S-3: AEP Stage Economics

Present Day (2009) Economics (Base Case)					
	Total Cost		Costs/tonne	Economics Pre-tax 100% Equity	
AEP: (net of gold credit)	202M	Mining Ore	69.66/t	Gold Price	
Timmins Mine	252M	Processing	17.62/t	Price	US$950/oz
BC Mill	60M	Trucking	5.44/t	Pre-Tax	
	8M	G & A	2.36/t	100% Equity	
Total Project Expenditures	522M	Total Ore	95.08/t		
Overall Economic Return on Investment with AEP costs:				28% IRR @ US$950/oz	
Overall Economic Return on Investment without AEP costs:				240% IRR @ US$950/oz	

The expenditures of $140M for the AEP should be viewed with the same caution and risks associated with other mining exploration programs. The resources for the Project have been carefully and prudently established from significant surface drilling, which limits the potential downside risk that the actual ore grades encountered during the mine life could differ from those estimated to date. The bulk sampling program will determine the orebody grade more accurately than estimates from surface drilling.

In addition, there are a number of potential factors that could change the economics of the Project:

- Increased grades from re-calculation of the cutting factor;
- Optimized mining methods to minimize dilution;
- Higher sustained mining rates during early years of production;
- Increased external dilution from plan;
- Continuity of the vein structures to be tested during bulk sampling and pre-production.

LAKE SHORE GOLD CORP.

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1.0 INTRODUCTION (ITEM 4)

1.1 Terms of Reference and Purpose of the Report

In August 2009, West Timmins Mine (WTM) and LSG entered into a business transaction that will result in the creation of one company. This agreement is awaiting approval of WTM shareholders, who will vote on this matter at the shareholders meeting on November 5, 2009. This report is triggered by section 4.2(1) (c) of NI43-101 and addressed to LSG.

Terms of Reference

The purpose of this report is to update the current NI43-101 report (October 2007).

On August 27, 2009 George Darling, P. Eng., Todd Fayram, Q.P. (MMSA), were retained by LSG to provide an update on the current underground mine plan, mine production schedule, mineral reserve estimate, process plant, mine capital and operational costs for the Timmins Mine Project. Geology related updates are provided by Robert Kusins, P. Geo., Jacques Samson, P. Geo., and Heather Miree, P. Geo., all LSG employees. Subsequently Mr. Darling's commissioning was expanded to include the compilation of the overall NI43-101 Technical Report based on the studies and findings provided by other consultants that were separately commissioned by LSG.

The results of this report by the Authors are not dependent on any prior agreements concerning the conclusions to be reached; however, documents specified in Section 1.3 of this report are taken into consideration.

Unless stated otherwise, metric units and second quarter 2009 Canadian dollars are used throughout this report.

1.2 Consultants Contributing to Report

The technical report update was completed by the following Qualified Persons:

- George B. Darling P. Eng. Qualified person responsible for sections 1, 2, 3, 4, 16.7, 16.8, 17 (co-author), 18 (except 18.4), 19 (co-author) and 20 (co-author).
- Todd Fayram, Q.P. (MMSA). Qualified person responsible for sections 15, 18.4., and 18.10.
- Robert Kusins P. Geo. Qualified person responsible for sections 6, 7, 8, 9, 10, 11, 13, 14, 15, 16.1 to 16.5, 19 (co-author) and 20 (co-author).

- Jacques Samson, P. Geo. Qualified person responsible for section 12 (co-author)
- Heather Miree, P. Geo. Qualified person responsible for section 12 (co-author)

1.3 Sources of Information

- WGM Technical Review Report Timmins West Gold Project, dated January 3rd, 2007 (WGM Technical Review Report);
- L.D.S. Winter NI43-101 Technical Report, Timmins Gold Project, January 25th, 2006;
- SGS Lakefield Research Ltd.: investigation of the recovery of gold from samples from TM Property;
- RPC - The Technical Solutions Centre report: LSG Project Beneficiation Test (metallurgy test), reference # PET-J1478&1533;
- Consultant's Mine Plan and Reserve Estimate Timmins Mine Project
- LSG Timmins West Pre-feasibility Study Report by SRK ("PFS Report");
- Cook/Dumas, (Thunder Bay, ON, Canada), 2007, LSG Gold Project AE Study Hoisting Plan and Shaft, Report;
- AMEC, LSG, BC Tailings Facility Report, 2007;
- EHA, LSG, TW Project review of metallurgical test work and process design, 2007;
- BHM; BC Mill for LSG electrical and structural report, 2007;
- CSA National Instrument NI43-101 – Standards of Disclosure for Mineral Projects;
- CSA Form 43-101F1 to above instrument and Companion Policy 43-101CP.

Please refer to complete reference list specified in Section 21 of this report.

1.4 Site Visit

On September 11, 2009, George Darling, P. Eng., visited the Timmins Mine site. Mr. Darling collected updated information since his last visit (January 15, 2007).

Todd Fayram, Q.P. (MMSA), visited the mill site numerous times in 2008 and 2009 for the purpose of collecting information and providing technical opinions to LSG regarding the process plant. Mr. Fayram's last visits occurred on October 4, 2008 and January 18 to 22, 2009.

Robert Kusins, P. Geo., Jacques Samson, P. Geo., and Heather Miree, P. Geo., are employed by LSG and based at LSG's Exploration Office or at the Timmins Mine site. As a result they have frequently visited the property and are familiar with the Project.

2.0 RELIANCE ON OTHER EXPERTS (ITEM 5)

Mr. Darling's opinion contained herein and effective at the date of this report, is based on information provided during the course of Mr. Darling's investigations. This in turn reflects the various technical and economic conditions at the time of writing this report. Given the nature of the mining business, these conditions can change significantly over relatively short periods.

The results of this technical report are not dependent on any prior agreements concerning the conclusion to be reached, nor are there any undisclosed understandings concerning any future business dealings.

Technical economic projections include forward-looking statements that are not historical facts. These forward-looking statements are estimates and involve a number of risks and uncertainties that could cause actual results to differ materially from the estimates.

This report includes technical information that requires subsequent calculations to derive sub-totals, totals, and weighted averages. Such calculations inherently involve a degree of small errors due to rounding. Where these occur, the authors do not consider these to be material.

The authors have relied on baseline environmental studies made by Golder Associates Ltd.

The authors have not verified title to the Property, but have relied on information supplied by LSG in this regard. The authors have no reason to doubt that the title situation is other than which was reported by LSG.

3.0 PROPERTY DESCRIPTION AND LOCATION (ITEM 6)

3.1 Property Location

The Property is located in the south-west corner of Bristol Township, District of Cochrane, in the Porcupine Mining Division at an approximate Longitude of 81°33'W and Latitude of 48°23'N (at UTM coordinates 5,359,200N and 459,000E using NAD 83 Zone 17).

The Timmins Mine (TM) site is comprised of a mix of natural and disturbed areas. Throughout the site, anthropogenic disturbances of logging, tree planting and exploration activities were observed.

The disturbed area at TM will comprise approximately 30 hectares (“ha”), including the mine site with waste rock and mine water storage.

The perimeter of the Property was re-surveyed and marked out by an Ontario Land Surveyor in 2006.

3.2 Mineral Tenure

The Property, which consists of a contiguous block of 23 claims (11 individual patented claims and 12 leased claims, which are grouped into two 21-year leases) that covers approximately 395 ha. All of the claims cover both mining and surface rights. The individual patented claims have no expiry date provided a small annual fee of $4.00/ha is paid. The 12 leased claims are subject to an annual fee of $3.00/ha. The claim block has maximum dimension of approximately 2.6 km in an east-west direction and 2.0 km in a north-south direction.

One claim is subject to a royalty as described in Section 3.4 of this report.

The claims comprising the Property are owned by LSG or its wholly owned subsidiary LSG Holdings Corp. (LSG Holdings), as set out in Table 3-1 below, shown in Figure 2.

Table 3-1: Lake Shore Gold TW Project Claims Holding

Mining Claim #	Description	Recorded Holder	Area (ha)	Expiry
INDIVIDUAL PATENTED CLAIMS				
P-4039	Mining & Surface Rights	LSG Holdings	19.63	NA
P-4040	Mining & Surface Rights	LSG Holdings	17.28	NA
P-9389	Mining & Surface Rights	LSG Holdings	15.50	NA
P-9391	Mining & Surface Rights	LSG Holdings	15.74	NA
P-9392	Mining & Surface Rights	LSG Holdings	14.33	NA
P-9393	Mining & Surface Rights	LSG Holdings	16.23	NA
P-9580	Mining & Surface Rights	LSG Holdings	15.34	NA
P-9581	Mining & Surface Rights	LSG Holdings	19.34	NA
P-9586	Mining & Surface Rights	LSG Holdings	17.97	NA
P-9587	Mining & Surface Rights	LSG Holdings	18.74	NA
P-4227	Mining & Surface Rights	LSG Holdings & LSG	16.67	NA
LEASED CLAIMS - LEASE 107874				
P-21981	Mining & Surface Rights	LSG Holdings & LSG	15.52	July 31, 2027
P-18749	Mining & Surface Rights	LSG Holdings & LSG	17.43	July 31, 2027
P-18750	Mining & Surface Rights	LSG Holdings & LSG	14.84	July 31, 2027
P-21980	Mining & Surface Rights	LSG Holdings & LSG	14.54	July 31, 2027
P-19564	Mining & Surface Rights	LSG Holdings & LSG	19.34	July 31, 2027
P-18751	Mining & Surface Rights	LSG Holdings & LSG	27.13	July 31, 2027
LEASED CLAIMS - LEASE 106634				
P-55902	Mining & Surface Rights	LSG Holdings	15.22	July 31, 2013
P-55903	Mining & Surface Rights	LSG Holdings	18.72	July 31, 2013
P-55904	Mining & Surface Rights	LSG Holdings	18.11	July 31, 2013
P-55905	Mining & Surface Rights	LSG Holdings	21.65	July 31, 2013
P-55906	Mining & Surface Rights	LSG Holdings	21.08	July 31, 2013
P-55907	Mining & Surface Rights	LSG Holdings	4.99	July 31, 2013

3.3 Location of Mineralization

The TM deposit consists of veins and stock work zones in close proximity to each other within a lithologic package consisting of altered sediments, mafic to ultramafic volcanics and an intrusive pyroxenite complex. The plan area of the overall deposit is roughly 250 m long on strike and 150 m wide. This is the horizontal expression of a pipe-like trend of mineralization that straddles a volcanic/sedimentary/ultramafic contact in a folded sequence that plunges 60° to the north. Mineralization has now been defined within a block approximately 760 m long by 300 m wide and to a depth of 1,300 m (Figures 3 and 4).

3.4 Royalties, Agreements and Encumbrances

Claim P-4227 is subject to a 1.5% net smelter return (NSR) royalty payable to Mr. Lorne Lebrash of Sudbury. The current extents of the mineral resources model are not located on this claim.

3.5 Environmental Liabilities and Permitting

The development of the mine will create a disturbance footprint on the terrestrial environment. Baseline work did not identify any provincially or federally listed fauna species on the site that will trigger concern. At closure, the site will be rehabilitated in accordance with closure plans filed with the Ministry of Northern Development and Mines. The intent is to return the site to a productive use (i.e. forestry) resulting in limited long-term impact to the area.

3.5.1 Required Permits and Status

LSG has in hand all permits that are required for mining plant operations in the province of Ontario:

- Ministry of Northern Development and Mines (MNDM)
- Ministry of the Environment (MOE)
- Ministry of Natural Resources (MNR)
- Ministry of Transportation (MTO)
- Technical Standards and Safety Authority (TSSA)
- Ministry of Labour (MOL)
- Occupational Health and Safety
- Explosives
- Notification of Commencement of Construction and Operation

3.5.2 Federal Permits

LSG has in hand all permits that are required for mining plant operations in Canada:

- Department of Fisheries and Oceans Canada (DFO)
- Natural Resources Canada (NR CAN) – Explosives Regulatory Division (ERD)

- Environment Canada (EC)

Closure plan for the bulk sampling is to be filed in October 2009 and the commercial production closure plan to be filed in 2010.

3.5.3 Consultation

Consultation is being undertaken with regulatory agencies, the general public, the Métis Nation of Ontario, Wabun Tribal Council and the First Nation communities of Flying Post First Nation and Mattagami First Nation, who are represented by Wabun Tribal Council. Consultation provides an opportunity to identify/address the concerns of external stakeholders and expedite the authorization process.

The consultations have been held in order to comply with LSG corporate policy and the provincial requirements of Ontario Regulation 240/00 and the Environmental Bill of Rights.

4.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY (ITEM 7)

4.1 Topography, Elevation and Vegetation

The Project site topography is undulating from 300 to 350 m above sea level. Several wetland areas that are believed to be typical of the region exist within the TM Project boundary. Two distinct wetland areas were observed. The largest wetland area, the flood plain bordering Thunder Creek, is a shrub-dominated fen. The second wetland observed within the Project boundary is located in the upland area south of Thunder Creek. This small marsh type wetland is contained in the bedrock depressions. Wetlands were not evaluated under the wetland evaluation system; however no significant features were identified during the baseline studies (Golder 2007). If wetlands are impacted as part of the mine development and it has been identified through the consultation process to be of significance then further biological studies will be carried out to evaluate the biological feature. The current proposed mine plan would not impact wetland features identified on the site.

The site is located on the edge of the clay belt and within the broader confines of the boreal forest region (Legasy 1995). Forest cover typical of this region is mostly coniferous, but includes a mix of deciduous trees such as white birch (betula papyrifera) and trembling aspen (populus tremuloides). Provincial Forest Resource Inventory stand numbers provided by TEMBEC indicated that most of the forestry related disturbance to the area occurred approximately 20 years ago, and that the forest communities are composed of poplar, jack pine, white birch, black spruce and white spruce. Terrestrial surveys completed as part of the baseline studies did not identify any features of significance (Golder 2005 and Golder 2007) that will be impacted by the proposed mine plan.

No significant wildlife species were identified from the file review, or through discussions with local government agencies, so no specific wildlife surveys were conducted for the Project site during the baseline biological program. Instead, wildlife species noted during the field assessments were recorded as incidental observations within a specific habitat type. All incidental wildlife noted to use the site is commonly found in Northern Ontario (Golder 2005 and Golder 2007).

Baseline fish community and fish habitat surveys were completed for both the Thunder Creek system that flows east across the site and for the Tatachikapika River located approximately 1.5 km downstream of the site boundary. Once a preliminary project description was presented for the site a number of tributaries of Thunder Creek and Tatachikapika River were identified within the proposed project footprint. Baseline fish and fish habitat mapping was also completed for these systems.

A fish habitat survey of Thunder Creek in 2004 noted that it provides suitable fish habitat for all life stages, foraging, feeding and over wintering for all species of fish captured in the system. Sediment sampling completed in 2002 and 2003 as part of the benthic program indicates that current sediment quality for Thunder Creek exceeds the MOE Provincial Sediment Quality Guidelines (PSQG) (Lo Effects Level) for total nickel and total chromium. The 2004 sediment samples reported no metals concentration exceeding the PSQG.

4.2 Climate and Length of Operating Season

The climate is typical North American with extreme seasonal variations, leading to cold winters and warm summers. Average daily winter temperatures range from -24°C to -11°C and average daily summer temperatures range from 10°C to 24°C. Annual precipitation averages 930 mm, about half in the form of snow.

The prevailing winds for the area are from the south and have average wind speed of 11.9 km/h.

Work can be carried out on the Property twelve months a year.

4.3 Physiography

The TM Project is situated in an area that encompasses a large portion of the Canadian Shield. It is characterized by a distinctive geological legacy dating back some three billion years.

4.4 Access to Property

The access road to the Property is located at the intersection of Highway 101 and Highway 144 (see Figures 1 and 5).

The access road from Highway 101 to the site is approximately 800 meters. This road crosses the Thunder Creek by a single-lane logging bridge.

Ore is transported by trucks from the site via Highway 101 to the BC Mill.

4.5 Local Resources and Infrastructure

Timmins has evolved from a mining town to a full service, modern city. Small businesses have played an important role in job creation. Timmins is now a focal point of north-eastern Ontario, and home to approximately 45,000 people (2006 Census). Given its strategic location, local companies attract significant business from outlying communities. Timmins draws consumers from throughout the Cochrane district, the James Bay coastal area, and nearby communities such as Chapleau and Kirkland Lake. The regional market territory is approximately 118,000 people.

Timmins is served by air, rail, and road transportation. The local airport provides flights to and from Toronto as well as other locations several times a day.

The local economy is dominated by mining and smelting activities along with logging, numerous government facilities, and a community college. The downtown business core provides a multitude of offices and convention facilities.

An electrical power line crosses the Property (Figure 2). There is ample supply of power for the mining operation.

The Property has sufficient surface area to support the planned infrastructure.

The Timmins labour force is educated and well trained. According to the 2001 Census for the Timmins West area, 41% of the Timmins population has completed post-secondary education or certification. The workforce also receives constant training and upgrades, making Timmins residents among some of the most qualified mining workers in Canada.

5.0 HISTORY (ITEM 8)

5.1 Exploration History and Historic Mineral Resource Estimates

There has been prospecting activity in the Porcupine camp since the late 1800's and the first significant gold discovery was made in 1909. The early major producers were the Dome and Hollinger Mines in Tisdale Township. In 1911, gold was discovered on the Timmins Mine Property and claims staked by T. McAuley. McAuley, his brother and their partner, J. Brydge, sank two exploration shafts in the area of the main showing. These shafts are reported to be approximately 15 m in depth.

Orpit Mines Ltd. (Orpit) acquired the claims in 1938 and did 7,620 m of diamond drilling. Piccadilly Porcupine Mines Ltd. acquired the Property in 1945 and by 1953 control of the Property was held by Stanwell Oil and Gas Ltd. In 1957, the Ontario Geological Survey (OGS) reported one shaft and a series of pits on the main showing on claim P-4040 and the other shaft on claim P-4039. OSG further reported that 87 drillholes had been completed, and the 60 holes for which logs were available totalled 12,600 meters. Most of the drilling was done in the Main Zone.

In 1959, Paul Meredith purchased the Property from Stanwell Oil and Gas Ltd. In 1963, the Property was transferred to Holmer. United Buffadisson Mines Ltd. (Buffadisson) optioned the Property in 1964 and constructed a road between the showing and Highway 101. Buffadisson drilled 10 holes totalling 2,116 m to confirm Orpit's results and did some test trenching in which high-grade gold values were reported. Buffadisson concluded that gold mineralization occurred in stacked en-echelon quartz veins that dip 50° north.

Holmer became the operator in 1968 and over the following 13 years drilled 45 steep dipping to vertical holes. A total of 10,512 m of BQ drilling was conducted focusing on the McAuley-Brydge showing (another name for the Main Zone, also referred to as the Folkler Zone). Additional stripping, channel sampling, and 'bulk sampling' were also performed. Based on this work, it was proposed that there were two mineralized zones, the Main Zone and Shaft Zone. The Western or Main Zone was reported to have a northern strike and steep dip south. The Eastern or Shaft Zone was reported to strike north-easterly with an unknown dip. It was concluded that "probable reserves" of 653,000 t grading 4.25 g/t Au were in the Main Zone

Additional exploration work on the Property at that time consisted of ground Electromagnetic surveying and limited exploration drilling.

Holmer optioned the Property to Noranda Exploration Company Limited (Noranda) in 1984. Noranda conducted an airborne magnetic and electromagnetic survey and follow-up ground geophysics. Four NQ holes totalling 1,465 m were drilled west of the major diabase dyke. A review of the previous resource estimate by Noranda concluded that several of the assumptions made were incorrect. Noranda estimated that the Property hosted a "resource" of 785,000 t grading 2.4 g/t Au. This included a higher-grade core of 159,000 t grading 4.46 g/t Au.

In 1987, the Property was optioned from Holmer by Chevron Minerals Ltd. (Chevron). A comprehensive exploration program was conducted in the following two years. Line cutting, ground geophysics (magnetic, VLF and IP) and geological mapping were conducted over the entire Property. A large area over the Main Zone was stripped, detailed mapped and channel sampled. Twenty-nine BQ diamond drillholes totalling 6,115 m tested the Main Zone as well as several geophysical targets. Known mineralization was extended to a vertical depth of 360 meters.

In 1996 and 1997, a program of ground geophysics, humus sampling, mapping, and rock sampling was carried out by Holmer. Sixty-six diamond drillholes totalling 25,380 m were also completed. Fifty-four of these were directed at expanding "resources" in the main deposit area and 12 were drilled to test geophysical anomalies elsewhere on the Property.

The historic "Mineral Resource" estimates referred to above were prepared prior to the implementation of NI 43-101. These are presented because LSG consider the information to be relevant and of historic significance. These estimates should not be relied on.

In 1998, Holmer completed twenty-two diamond drillholes totalling 3,923 m to test the continuity of the mineralized zones at shallow depths. In 1999, St. Andrew Goldfields Ltd. (St. Andrew) optioned the Property and drilled 10 shallow holes totalling 1,341 m to explore the open pit potential of the deposit.

In 2002, an additional 22 holes totalling 5,220 m were drilled by Holmer to complete definition drilling at 25 m centres and a Mineral Resource estimate was completed by Holmer and was audited and revised by WGM. Holmer began environmental baseline studies by initiating a surface water-sampling program during the year.

An updated Mineral Resource estimate was prepared by LSG in September 2004 and once again in late 2006. These NI 43-101 estimates were also audited and revised by WGM, and are described in the WGM Technical Review Report. Utilizing WGM's Report, LSG engaged SRK's Sudbury office to provide a Mineral Reserve estimate and a preliminary mine production plan which was reported in SRK's "NI 43-101 Technical Report Lake Shore Gold Corp. Timmins West Project Timmins, Ontario" dated Oct 12, 2007. This report forms the basis of the current report which updates the work that Lakeshore has completed since 2006. The updated resources are summarized in Section 16 of this report.

In February 2009, an update of the Timmins Mine resources was undertaken by AMEC. Due to differences in opinion on the interpretation of zones, this work was not completed.

5.2 Historic Production

Prior to March 2009 there has been no production activity at the TM Property.

Mine workings on the property are currently under development. Access to lower areas underground was commenced in July 2008 with the excavation of a 5.5 m (completed inside) diameter shaft (6.1m diameter excavation). Access to upper areas underground was commenced in September 2008 with the development of a portal and ramp. To this time, the ramp has been developed to 200 Level, accessing advanced exploration targets on 50 m, 60 m, 80 m, 90 m, 110 m, 120 m, 140 m, 150 m, 170 m, 180 m and 200 Level. At present, the shaft has been excavated to 682 m depth below surface. Advanced exploration of several potential ore zones is presently being conducted as accessed by the ramp. The Ultramafic Zone and Footwall Zones are the main targets being assessed from the shaft. Shaft stations have been developed on the 200, 400, 525 and 650 Levels at this time.

Ancillary to the advanced exploration underground activities, a waste water collection pond has been constructed, waste dumps are actively being accumulated on surface, and several buildings have been constructed in support of the underground operations.

6.0 GEOLOGIC SETTING (ITEM 9)

6.1 Regional Geology and Structure

The Property lies in the south-western part of the Abitibi Greenstone Belt, within the Superior Province. Most of the rocks in the southern Abitibi Greenstone Belt are Archean in age with ages ranging from 2,730 to 2,670 Ma. The overall geometry is of east-west trending lithographic sequences that vary in composition from ultramafic through to felsic and are primarily of volcanic origin. These units are capped by local occurrences of narrow, east-west trending sedimentary sequences consisting of fine, turbiditic rocks (e.g., Porcupine Group, Kewagama Group). These turbiditic sequences are spatially related to less common, younger, coarse clastic rocks of the Timiskaming Group and its equivalents that are generally thought to be 2,677 ±2 Ma. Volumetrically minor felsic intrusions and their extrusive equivalents occur in many areas, including the Porcupine area (~2,690 Ma), the Kirkland Lake area (2,677 ±2 Ma) and in the Duparquet area (2,689 to 2,682 Ma), just inside Québec, east of Matheson and north of Rouyn-Noranda. The rocks described above have all undergone regional green schist grade metamorphism. The majority of rocks in the immediate Property area are either volcanic or sedimentary in origin with only minor volumes of intrusives known. On a regional scale, the volcanic sequences have been intruded by volumetrically significant mafic to felsic batholiths that are mostly dated between 2,707 to 2,696 Ma.

The volcanic rocks in the Timmins area have been subdivided into two principal groups; the Deloro Group (~2,725 Ma and older) and the overlying Tisdale Group (~2,705 Ma). In the main gold-producing areas near Timmins (Deloro and Tisdale Townships), these groups are separated by the Destor-Porcupine Fault Zone ("DPFZ"), which is a crustal-scale fault trending 070° to 090° and extends from west of Timmins, east and then southeast to the Destor area and beyond in Québec. The third major group of rocks in the area is composed of a series of clastic sediments that lie unconformably on top of both the Deloro and Tisdale Groups (Figure 3).

The DPFZ is the most important regional structure. In addition to defining the contact between the Tisdale and Deloro Groups in the immediate Timmins Mine area, it, and the similar Larder Lake-Cadillac Break in the Kirkland Lake area, these rocks are associated with large-scale elongate lozenges of sedimentary rocks. These sedimentary sequences are spatially associated with pre-existing, first-order, east-west trending structures and are also spatially related to all the known major gold deposits in the region. Despite the spatial relationship, the sediments are younger than the major structures and the major gold deposits post-date the deposition of the sediments. The first-order structures (DPFZ, Larder Lake-Cadillac Fault) most frequently occur at the boundaries between different volcanic or sedimentary sequences but as they also cut certain sequences, they are not always terrane boundaries.

In general, the Deloro Group is composed of ultramafic, mafic, intermediate and felsic volcanics with narrow bands of iron formation near the stratigraphic top of the group. The Tisdale Group is composed mostly of ultramafic and mafic volcanics with only a minor felsic component. There are also compositional differences in that the Deloro Group is largely calc-alkaline in nature and the Tisdale Group has a larger ultramafic and tholeiitic basalt component. Pillowed, amygdaloidal basalts are common in the Tisdale Group and iron formations are rare.

The clastic sediments that overlie the Deloro and Tisdale Groups are grouped into the Porcupine Group (~2,695 to ~2,680 Ma), which is a widespread, primarily turbidite sequence that is in turn overlain, generally unconformably, by a slightly younger, polymictic conglomerate, the Timiskaming Group. In the Porcupine Camp, the Timiskaming Group is often spatially associated with the DPFZ and it is also often proximal to the larger gold deposits (e.g., Dome, Pamour). At the base of the Porcupine Group in the Timmins area, there are places where a characteristic felsic tuff unit, known as the Krist Fragmental, occurs. This unit is thought to be an extrusive unit related to the feldspar porphyry intrusives that occur in the vicinity of the Hollinger and McIntyre Mines. Minor occurrences of a unit similar to the Krist Fragmental occur on the Property. Other small felsic intrusions, commonly feldspar or quartz-feldspar porphyries, occur in the Tisdale Group, and although these are often proximal to producing mines, age dating suggests that gold mineralization is more than 10 Ma younger than these intrusions. Diabase dykes constitute a much younger set of intrusions that cut all other units.

6.2 Property Geology

The eastern part of the TM Property is covered by considerable amounts of overburden with only local minor outcrops. The geology in this area is largely inferred from drilling and geophysics. The western part of the Property has abundant outcrop exposure. The Property is underlain by volcanic rocks on the west and north and by sedimentary rocks on the east and south. The volcanic rocks are interpreted to be part of the Tisdale Group and the sedimentary rocks those of the Porcupine Group. The contact between the two groups is likely unconformable and has been folded so that it is rotated from trending east-northeast at the north-eastern part of the Property to north-south trending and then folded into a tight fold in the immediate area of the TM Deposit back to principally east-northeast trending further to the southwest of the Deposit (Figure 4).

In the vicinity of the Deposit there are also ultramafic flows (komatiites?) and a pyroxenite intrusion that occur along the volcanic-sedimentary contact. The exact nature of these rocks has been somewhat enigmatic but they are important hosts for gold mineralization, which is referred to as the Ultramafic Zone. On a detailed scale, there is considerable strain and likely faulting of the sedimentary-volcanic contact in the vicinity of the gold mineralization at the Deposit. The following description of the intrusive rocks on the Property is taken from Rhys (2003):

"…at and southwest of the Holmer deposit, a band of south-westerly widening ultramafic rocks is initially present at the volcanic-sedimentary contact at Holmer, but progressively enters the mafic volcanic sequence to the southwest. Possible spinifex textures in stripped portions of the south-eastern Holmer outcrops suggest that part of this unit comprises komatiitic flows. Dykes and bodies of a mafic to ultramafic intrusion(s) of potentially alkalic composition intrude the massive ultramafic unit."

"Several large quartz-feldspar porphyry bodies intrude the basal portions of the turbiditic sedimentary sequence to the northeast and south of the Holmer Property, within 1.4 km of the contact with the mafic volcanic sequence. Lateral porphyry breccia and possible tuffs are spatially associated with these intrusions (J.Samson, pers. comm., 2003), suggesting the presence of volcanic-intrusive complexes in the sequence. While felsic volcanic units are not mapped in Bristol Township, a Krist-age felsic volcanic component to the sequence is suggested in the area by the relationships described above, and by the presence of probable felsic tuff and tuffaceous sedimentary rocks in drill core and outcrop on the Holmer Property. Further porphyry bodies may also be present to the east of the contact near the Holmer deposit, forming potential favourable hosts for Au vein systems where intersected by shear zones."

Massive, fine to medium-grained diabase dykes on the Property strike approximately north-northwest and persist for a number of kilometres. The most significant of these occurs to the west of the Main Zone in surface exposure, around section 4800E. This dyke dips more or less vertically and cuts the down plunge extension of the Deposit; however, there does not seem to be any significant displacement of mineralization along the fracture hosting the dyke, only dilation.

Table 6-1is a Table of Formations of the rock types found on and around the Property and is taken largely from the Timmins area classification of Pyke (1982).

Table 6-1: Table of Formations

TABLE OF FORMATIONS

Diabase Dykes

------------------------------------Igneous Contact------------------------------------

Timiskaming Group
Conglomerates, Wackes & Slates
2679±2 Ma

------------------------------------ Disconformity ------------------------------------

Porcupine Group
Turbidic Sediments
*conformable on top of the Krist at 2688?
2696 to <2688? Ma

Quartz-Feldspar Porphyries
(e.g., Pearl Lake Porphyry)
2688 Ma

-------------------- Disconformity and Igneous Contact --------------------------

Tisdale Group
Ultramafic Flows, Tholeiitic Flows, Upper Mafic to Felsic Flows
2710 – 2700 Ma

------------------------------------Structural Contact ?------------------------------------

Deloro Group
Ultramafic Flows, Tholeiitic Basalts, Calc-alkaline Andesites, Basalt, Pyroclastic Tuffs, Iron Formation
2730 – 2725 Ma

* It should be noted that if the Krist Formation is related to the porphyries that date from 2688 Ma then at least part of the Porcupine Group must be younger than 2688Ma as the Porcupine Group sediments sit conformably on top of the Krist.

6.3 Host Rocks

The principal lithological units at the TM Deposit are mafic volcanics, sediments, ultramafic rocks thought to be likely komatiitic flows, and ultramafic rocks that may be an alkalic ultramafic intrusion. The distribution of these rocks on a Property scale is considerably less complex than in the immediate area of the Deposit.

6.3.1 Mafic Volcanics

These are described as being dark green, occasionally pillowed and amygdaloidal and are interpreted to be tholeiitic in composition (Holmer, 1997).

6.3.2 Ultramafic Rocks

A variably altered talc-chlorite ± carbonate unit occurs in the immediate vicinity of, and hosts some of, the TM Deposit, although it appears to be almost conformable between mafic volcanic units to the south end, and sedimentary units to the north and east, its geometry when seen on sections is considerably more complex. It could be a komatiitic flow, which with the mafic volcanics, would constitute rocks equivalent to the Tisdale Group seen elsewhere, or it could be a similar age intrusion.

A second ultramafic unit has been tentatively identified as an ultramafic alkaline intrusion (carbonatite in Holmer's 1997 report). This unit occurs in association with and is interpreted to intrude, the ultramafic unit described above. It consists of diopside, apatite, magnetite and minor phlogopite and amphibole (H.Marsden, per. comm. 2004), based on petrographic studies completed for LSG. This rock is also described as having magnetite-garnet-apatite phases with chlorite, biotite and fine grained quartz (Rhys, 2003). Its significance is that at depth it appears to be a preferential host for gold mineralization.

6.3.3 Sediments

The majority of the sediments in the deposit area are described as thinly bedded turbidic greywackes, siltstones, and mudstones. There are minor occurrences of sericitic rocks that may be sediments or felsic tuffs (Holmer, 1997; WGM, 1998; Rhys, 2003). These could be the stratigraphic equivalent to the Krist Fragmental unit seen at the base of the Porcupine Group in the Tisdale Township area.

6.4 Alteration

There is a systematic but irregular increase in carbonate alteration (calcite→Fe-carbonate) in proximity to mineralization. The degree to which this alteration is manifested is to a large extent controlled by the composition of the host rocks. Quartz, carbonate, sericite, albite, sulphides, and tourmaline all occur in the vicinity of the veins (see below). In addition, there is a significant amount of disseminated magnetite developed as an alteration halo associated with the ultramafic intrusion.

6.5 Veining

There are pervasive carbonate stringers evident in the more altered ultramafic and mafic rocks. These do not appear to be directly related to the gold mineralization. Gold is hosted by quartz-tourmaline, tourmaline and quartz stringers, and veins that cut all of the rock types described above. Sericite, albite, quartz-carbonate, and sulphides are all developed in association with these veins. Rhys (2003) concludes that there are two dominant styles of veining:

- quartz + tourmaline + carbonate ± pyrite ± arsenopyrite veins mostly in clastic sediments or at sedimentary-volcanic contacts with associated tourmaline and sericite-carbonate envelopes; and
- quartz + carbonate + albite ± tourmaline ± pyrite ± arsenopyrite ± pyrrhotite veins mostly in mafic and ultramafic rocks with associated albite-sericite-carbonate inner alteration envelopes and carbonate-sericite-chlorite outer alteration envelopes.

Almost all of the gold mineralization occurs in association with quartz-tourmaline veins.

Drilling at depth has revealed a style of veining in the ultramafic units characterized by quartz-tourmaline veins with considerable amounts of gold related to sulphides on the margins of the veins in addition to, and in some cases instead of, the primarily vein-hosted gold mineralization seen nearer surface.

There has been a high degree of deformation in terms of shearing, folding, and likely faulting in the TM Deposit and the relationship of the primary lithologies is more complex in three dimensions than surface plans suggest.

6.6 Structural Geology

The DPFZ has been variably interpreted to pass more or less through the TM Deposit (Ferguson, 1957) or in more recent interpretations, 5-6 km to the south parallel to the regional sedimentary-volcanic contact in Thornloe Township. The Bristol Fault, which is likely related to the faulting associated with the development of the DPFZ, is located approximately 150 m north of the Deposit. It generally trends east-northeast, parallel to the DPFZ, and is characterized by intense deformation over widths of 100 to 200 m (WGM, 1998). Rhys (2003) describes the faulting in the immediate deposit area as follows:

"The east-northeast trending Bristol Fault is developed immediately to the north of the Holmer deposit. It comprises one or more seams of chloritic clay gouge. The fault accommodates several hundred meters of apparent left lateral displacement, and accentuates the left lateral, shear zone related deflection that the Holmer deposit is developed at. Magnetic patterns suggest diabase dykes may be displaced along it, which is consistent with relationships on similar faults elsewhere in the Timmins area. Minor brittle faults are present within the Holmer surface showing, where they locally may modify ore shoot geometries."

"The Holmer deposit occurs within an approximately 600-800 m left handed deflection of the northeast to north trending mafic volcanic – sedimentary contact. Between 100 and 250 m of this is accommodated by the late, brittle Bristol Fault, as indicated by displacements of Proterozoic diabase dykes, but the remainder is related to a network of west and northwest trending D2 and D3 shear zones that collectively are here termed the Holmer Shear Zone (previously termed the Holmer fault). The shear zone is host to, and genetically related to the formation of the Holmer Au deposit."

7.0 DEPOSIT TYPE (ITEM 10)

7.1 Deposit Type

Gold mineralization on the Property is typical of Archean mesothermal gold deposits. On a deposit scale, gold mineralization, alteration, and veining are better developed in areas that are sheared and/or occur in areas of structural heterogeneity such as near major lithological contacts and near intrusions.

Examples in the Timmins area of these types of deposits include the Dome and Pamour mines at unconformable mafic-sedimentary contacts, the Dome and Hoyle Pond mines at mafic, ultramafic, and sedimentary contacts, and the Hollinger and McIntyre mines that were mafic volcanic flow hosted but were also proximal to felsic intrusions.

The style of veining at the Timmins Mine Property is somewhat similar to that of the Delnite, Aunor and Buffalo-Ankerite mines in Deloro Twp. These mines had ore principally hosted by pillowed and massive mafic volcanics that were associated with ultramafic rocks (historically described as serpentinite) and also in association with porphyry bodies. Veins in these deposits were composed of 50% or more of tourmaline in association with quartz and carbonate.

8.0 MINERALIZATION (ITEM 11)

8.1 Mineralized Zones

The plan area of the overall deposit is roughly 250 m long on strike and 150 m wide. This is the horizontal expression of a pipe-like trend of mineralization that straddles a volcanic/sedimentary/ultramafic contact in a folded sequence that plunges 60° to the west-northwest. Please refer to Figures 4 and 6.

Gold mineralization on the Property occurs as three distinct types as follows:

<u>Quartz-tourmaline veins and stock works in Veins 1, 2 and 3 and the Main Zone.</u>
Within the quartz-tourmaline veins, the gold appears to be related to the presence of tourmaline and coarse visible gold is commonly present. The quartz-tourmaline veins attain widths of 1 to 7 metres and contain pyrite and arsenopyrite mineralization. Alteration surrounding the veins is in the form of sericite and ankerite. The veins generally form at or near the sediment/volcanic contacts.

<u>Footwall type mineralization</u>
The Footwall mineralization occurs within a unit of mafic tholeiitic volcanic rocks near or at the contact of the ultramafic complex. For one of the Footwall subzones there is a spatial relationship between the Footwall mineralization and an embayment or secondary fold in the ultramafic contact. Gold mineralization is concentrated in highly sheared and mylonitized rocks that are associated with silica, albite and pyrite alteration. Pyrite is the most common sulphide mineral with local concentrations of up to 10%. Accessory pyrrhotite and rare arsenopyrite may also be present. Locally, quartz and iron carbonate veining are common. Minor tourmaline is present in some of the veins; however, gold values are directly related to sulphide content as indicated by assay results and thin sections. Visible gold has only occasionally been observed in the Footwall Zone but is commonly observed in thin sections. The Footwall zones have a plunge similar to the other mineralized zones and gold values appear to vary with depth.

<u>Ultramafic Zone mineralization</u>
Drilling indicates that the zones are associated with a coarse-grained pyroxenite intrusive rocks and fine-grained carbonatite units within ultramafic komatiitic flows. Feldspars have been locally sausseritized and iron carbonate alteration is common. Deformation in this zone is ductile and brittle with fractures filled with chlorite or quartz. The pyroxenite ranges from massive and under formed to locally intensely foliate. Mineralization is associated with white to grey translucent quartz and quartz-tourmaline veins occur within altered zones and sulphides commonly occur as halos around the

veins. Up to 10% sulphide mineralization, in the form of pyrite and rare chalcopyrite, occurs in the strongly altered zones. Gold values have a strong correlation with the sulphide content. In drill holes, coarse visible gold is locally observed in the quartz veins. The altered host rock and quartz veins can both contain significant gold mineralization. The ultramafic zones exhibit the same plunge as the encompassing geology and other zones in the deposit.

Typically, only parts of the quartz-tourmaline veins are mineralized, usually along the vein-wallrock contacts. WGM believes that the location of the gold is governed by minor folding and/or fault structures and as such could likely be selectively mined. The overall continuity can only be determined with underground exposure but generally, a fair three-dimensional alignment in section and plan is evident. Such alignment can be shown not only where the veins are stratigraphically conformable but also where they pass from sediments into the volcanics. Bulk sampling will be necessary to determine the grade of these structures reliably.

The various mineralized zones appear to be located close to the apexes of anticlinal folds and are clustered at two locations along the overall plunge. The best parts of the No. 2 and 3 Veins occur between elevations 600 and 700 m (surface at approximately 1,000 m elevation) while the main Footwall and ultramafic zones occur immediately below elevation 450 m (Figures 7 and 8). Veins 1, 2, and 3 occur at shallow depth where the sedimentary-volcanic contacts form a tight anticline with sub parallel limbs, in proximity to the ultramafic rocks (Figures 9 and 10). Going with depth, the structure plunges away from the ultramafics. To date, below elevation 450 m the best mineralization occurs at the base of the mafic volcanic sequence and within the underlying ultramafics.

Photo 8-1 is a photograph the Main Zone outcrop showing the sharp limits of the stockwork style of mineralization. Photo 8-2 illustrates the complex age relationships of the various types of quartz veining in the Main Zone outcrop.

Typical underground exposures of mineralized zones are depicted in Photo 8-3 through Photo 8-5.

Photo 8-1: Photograph of Main Zone outcrop showing sharp contact of zone

Photo 8-2: Photograph of complex ore relationships of quartz veining in Main Zone

Photo 8-3: Photograph of Vein 1, Round 16 60m Level

Photo 8-4: Photograph of Vein 2, Round 16 50m Level

Photo 8-5: Photograph of Main Zone 1, Round 6 80m Level

9.0 EXPLORATION (ITEM 12)

9.1 General

After it took over operatorship of the Project in 2003, LSG focussed on exploring the down plunge extension of the Footwall and Ultramafic zones. This led to the preparation of a new NI 43-101 Mineral Resource estimate in 2004, followed by an updated resource in 2006 and 2009. Since 2007, most of the surface drilling was focused on exploring for extension to the Vein zones and testing new exploration targets.

9.1.1 2003

Drilling completed in 2003 by LSG consisted of 53 NQ holes and/or wedges totalling 17,146 m. Twenty-three were deep Mineral Resource expansion holes. Nine were exploration holes completed away from the known resource and 21 short holes were drilled in an area above 100 m vertical in a program designed to define an open pit resource. The nine exploration holes tested geological/structural/alteration targets and encountered no significant gold values.

Other work completed in 2003 included a detailed (75 m line spacing) airborne magnetic survey completed by Fugro Airborne Surveys of Ottawa, a detailed structural study of the exposed portion of the TM Deposit, and Mobile Metal Ions ("MMI") soil sampling on the overburden-covered eastern and north-eastern portions of the Property.

Preliminary metallurgical test work was carried out on four composite samples of historic core by SGS Lakefield Mineral Services ("SGS"). This work continued into early 2004.

LSG exploration expenditures on the TM Property for 2003 amounted to $2,006,572.

9.1.2 2004

An additional 38 holes totalling 17,655 m were drilled, for the most part to extend the resources from 4715E to 4595E and to a depth of 850 m. Thirty-five holes were directed towards Mineral Resource expansion and three were exploration holes directed towards various exploration targets on the Property. None of the exploration holes encountered significant results.

Environmental baseline surface water sampling continued. No other surface work was carried out except drill site reclamation.

LSG exploration expenditures on the Property for 2004 amounted to $2,339,220.

9.1.3 2005

An additional 58 holes totalling 28,875 m were drilled. Forty-seven holes were directed towards Mineral Resource expansion and 11 were exploration holes directed towards exploration along the sediment-volcanic contact to the north and east of the Deposit. No significant values were obtained in this exploration, with the exception of minor anomalous gold values in the northeast section of the Property.

Metallurgical testwork and process design for a new on-site 1,500 t/d mill were begun by EHA. Drill core samples were forwarded to the RPC testing laboratory in Fredericton, N.B. in May and a report was issued in mid-2006. This work is discussed further in Section 17.3 of the WGM Technical Review Report.

Environmental baseline surface water sampling continued.

LSG exploration expenditures on the Property for 2005 amounted to $4,895,090.

9.1.4 2006

LSG completed an additional 55 drill holes totalling 28,313 m. Thirty-seven holes were directed towards down plunge Mineral Resource expansion and 25 were exploration holes. The exploration holes focused on attempting to expand the deposit along strike and dip, as well as various other exploration models. The resource expansion holes were largely successful as discussed in Section 16 of the WGM Technical Review Report. The exploration holes have not encountered any significant gold values, except for TG06-115, which possibly extends the V3 further down dip on section 5000E.

LSG also carried out a small amount of stripping and diamond-saw channel sampling in the original showing stripped area and in addition, the perimeter of the Property was re-surveyed and marked out by an Ontario Land Surveyor.

LSG initiated a Prefeasibility Study, to be completed by SRK, for a 1,000 t/d underground mining operation. These studies will continue into 2007. LSG has engaged several consulting firms and service providers to carry out various activities as summarized below:

- AMEC Americas Ltd. of Oakville, Ontario completed planning for an Advanced Exploration Program/permit ("AEP") based on shaft access for underground drilling stations and bulk sampling;
- RPC delivered a metallurgical testwork report to EHA in July 2006. The results are discussed in Section 17.3 of the WGM Technical Review Report;

- Mill-Ore Industries Inc. of Timmins was contracted to complete a survey of available mills/milling capacity in the Timmins area;
- LSG has hired an environmental coordinator to shepherd and manage all environmental work, including the application process for the AEP;
- Tailings handling and storage issues are being studied by Golder Paste Technology Ltd., Sudbury;
- Golder Associates Ltd., Sudbury are continuing with aquatic and terrestrial baseline studies for mine and mill sites as proposed in the base case for the Prefeasibility Study;
- Golder Associates Ltd. is also directing an Environmental Impact Assessment;
- A tentative exploration shaft location has been chosen;
- Discussions have been held with the Ministry of Transportation and Hydro One regarding upgrading an access road off of Highway 101 and running a power line into the site; and
- A background study of First Nations' issues was prepared by Blue Heron Solutions for Environmental Management Inc., Timmins.

LSG exploration expenditures on the Property for 2006 amounted to $7.4 million.

9.1.5 2007

LSG drilled 20 diamond drill holes (10,959 m) in 2007. Most were drilling exploration targets with some testing down-plunge extensions of the vein mineralization. No new mineralization has been discovered in the exploration drilling with limited success in the attempt to follow the veins down-plunge, mainly due to deviation problems in the drill holes. Two sub-vertical shaft pilot holes were completed, and several old holes were re-collared and extended past the proposed shaft area, in order to explore and record additional structural data. Geotechnical core logging and packer testing was also carried out on selected holes by Golder Associates, of Sudbury.

Numerous holes drilled into the deposit and around the proposed mining infrastructure were cemented and plugged, and several casings were removed.

LSG has also continued with the planning and initial implementation of the AEP throughout 2007 as covered in the 2007 SRK Technical Report.

Total LSG exploration expenditures on the Property in 2007 amounted to $11.7 million which includes AEP expenditures.

9.1.6 2008

LSG completed 47 surface diamond drill holes in 2008, for a total of 9,357 m completed. Twenty-one holes shallow holes (877 m) were designed to assist in the definition of a ramp area in the vicinity of the Main Showing. Twenty-six holes were focused on exploring to the east and south of the mine site, mainly targeting the volcanic-to-sedimentary contact zone. No significant mineralization was intersected.

Historical drill holes into the deposit and near mine developments were also cemented.

Underground diamond drilling using electric drill units commenced on the property in October 2008, initially via ramp access. In 2008, a total of 41 NQ-sized (47.6mm core diameter) underground drill holes were completed, totalling 3,185.5m, including 3 service holes totalling 137.5m. Underground drilling during late 2008 targeted upper areas accessible from the ramp being developed at the time. This included drilling of the Main Zone, Vein 1 and Vein 2.

Total LSG exploration expenditures on the Property in 2008 amounted to $41.0 million which includes AEP expenditures.

9.1.7 2009-present

In 2009, two surface holes (1,836 m) targeted the volcanic-to-sediment contact zone south of the mine site. As of August 28 2009, 3 infill holes were also in progress, and one drill hole was exploring for the down-plunge extension to the deposit, on section 4300E (1750m in progress).

In the period from January 1, 2009 to August 28, 2009, a total of 259 drill holes were completed, totalling 21,387m, including 10 service holes totalling 323m. Of these, 214 holes (12,906m) were completed in the ramp area, and 45 holes (8,481m) were completed in the shaft area. The majority of underground holes are BQTK in diameter (40.7mm core diameter), but HQ-sized holes (77.8mm core diameter) are typically drilled for service holes, and AQTK-sized (30.5mm core diameter) holes are sometimes drilled using air-powered drilling units. All underground drill holes are cemented and/or have a grout plug installed.

As the ramp progressed to ever deeper levels, drilling of several targets followed shortly thereafter. Drill targets included the Main Zone, Vein 1, Vein 2, and Footwall Zones.

As soon as access and support were available in the shaft, drilling commenced in the shaft area with drilling on the 525m Level, followed shortly after by drilling on the 650m Level. Drill targets in the shaft area include the Ultramafic Zone and Footwall Zones.

Underground drilling was conducted by Forage Azimut Inc. of Rouyn-Noranda from October 2008 to March 2009. Underground diamond drilling has been conducted by Boart-Longyear since April 2009 to present. Management of all drill programs is by Lake Shore personnel and geologists under contract to Lake Shore.

Total LSG exploration expenditures on the Property in the first half of 2009 amounted to $23.8 million which includes AEP expenditures.

Throughout the remainder of 2009, Lakeshore will continue to complete underground definition diamond drilling from the surface ramp as it advances below the 120m level as well as on the 525 and 650m levels. Total diamond drilling is forecasted to cost approximately $1.3M direct drill cost for the remainder of 2009. Deep surface drilling is also in progress testing gaps in the Resource/Reserve model below the 800mL as well as testing the down plunge extension of the orebody below the 1300m level. In 2010, estimated costs for this drilling will be $1.0M.

In 2010, diamond drilling will continue to define the orebody as development advances in the surface ramp and from the 525 and 650m levels. Underground diamond drilling for 2010 will involve approximately 30,000 metres for $2.1M direct drill cost. At the same time deep surface exploration drilling is planned to test the down plunge extension of the orebody with approximately 4000 metres of drilling for $0.8M.

10.0 DRILLING (ITEM 13)

10.1 General

Between 1938 and 1980, 144 diamond drill holes totalling 26,285 m were drilled on the Property. The information from this work is either missing or unreliable/incomplete and has not been considered in the LSG Mineral Resource estimates.

Since 1984, exploration on the Property has been covered by the same north-south cut-grid, which has been refurbished as required. The portion of the grid which deep holes have been and are being drilled has been transit surveyed.

From 1984 to August 28, 2009, 432 surface holes (including wedged and abandoned holes) totalling 171,241 m have been drilled on the Property. Four additional drill holes (1,750 m) are still in progress at the time of writing.

The vast majority of the drilling, including that of LSG, has been NQ-sized (47.6 mm diameter core). Chevron and Holmer drilled a total of 37 BQ-sized holes (36.5 mm diameter core). LSG has occasionally reduced to BQ-size in cases of difficult ground conditions. Bradley Bros. Limited of Timmins carried out all the diamond drilling on the Property for St. Andrew, Holmer and LSG using a variety of drill rigs. Core recovery has consistently been close to 100%.

Casings were generally left in the hole and capped. The hole number was stamped on the cap or indicated by a labelled steel bar emplaced at the collar. Most of the holes were initially not cemented; in 2007 and 2008, cement was pumped down all of the casings which were relatively easy of access. Drilling details from 1984 to August 28, 2009 are provided in Table 10-1.

Please refer to Figure 11 for the surface drill collar locations.

Table 10-1: Timmins Mine Property – Surface Drilling Statistics

Company	Year	No. of Holes *	Metres
Noranda	1984	4	1,465
Chevron	1987	31	7,870
Holmer	1996 to 2002	114	46,425
St. Andrew	1999	10	1,341
Subtotal Others		159	57,101
LSG	2003	53	17,146
	2004	38	17,655
	2005	58	28,875
	2006	55	28,313
	2007	20	10,959
	2008	47	9,357
	2009	2	1,836
Subtotal LSG		**273**	**114,140**
Total to August 28/09*		**432**	**171,241**

refers to holes actually **completed during the period indicated; value includes wedged and abandoned holes; an additional 4 holes or 1750 m still in progress at present time.*

Underground drilling programs are summarized on Table 10-2 and described in section 9.1.7. Underground drilling has focussed on further assessment and delineation of known and projected mineralization as part of an Advanced Exploration program. As drill platforms are established successively deeper in the underground workings, drilling of nearby potential mining areas has commenced. Underground drilling targets have included the Main Zone, Vein 1, Vein 2, Footwall, and Ultramafic Zones.

Table 10-2: Timmins Mine Property – Underground Drilling Statistics

Company	Year	No. of Holes *	Metres
LSG	2008	41	3,186
	2009*	259	21,387
Total to August 28/09		**300**	**24,573**

10.2 LSG Core Handling and Logging Protocols

The drill core is delivered to the core shack by drill contractor personnel. LSG geologists take rock quality designation ("RQD") measurements, the core is photographed, and geological logging is done directly on a laptop computer using DHLogger software, and more recently using GemsLogger. Pick-lists are used to record some data and free-form comments/descriptions are also entered. The logs and logging detail are of very good quality.

Core logging, sampling, QA/QC and handling procedures are consistent between those performed on surface drill holes and underground drill holes. Similar to practices for surface drill holes, all drill holes generated from underground are logged and photographed prior to sampling. Approximately 20% of all underground holes are also geotechnically (RQD) logged. Care is taken to ensure geological and technical consistency between surface and underground drill core logging.

10.3 Hole Collar and Down-hole Attitude Surveys

Surface drill collars are spotted by chaining along the cut grid lines from the baseline, or by chaining from pre-surveyed stations. The head angle is set using an inclinometer. The majority of the drilling has been on the long-established north-south grid at horizontal intervals of 50, 25, and 12.5 m, with vertical hole spacing on section usually ranging between 15 and 40 m. Collar locations of all holes are transit surveyed after completion.

Downhole surveys for the Holmer drilling were done at 50 m intervals using a Sperry Sun instrument. LSG drill holes have been surveyed with an EZ-Shot instrument in combination with a magnetic susceptibility meter to measure rock magnetism. Surveys are done at 30-50 m intervals (depending on the intended depth of the hole) while the hole is in progress, for use in directional control when necessary. In the magnetite bearing ultramafic rocks, or elsewhere where magnetic influences were recorded, the unreliable data are discarded. More recently, in particular for the deep holes and wedges, certain holes have been gyro surveyed. This work is carried out by Haliburton Group Canada Inc. of North Bay, Ontario.

Underground drill hole collar locations and azimuths are established using survey control. The same grid coordinate system is utilized underground in the ramp and shaft areas as is used on surface. Drill hole collars are surveyed after completion. Down hole directional surveying is completed on all BQTK and HQ-sized drill holes using ‘Reflex’ or ‘EZ-Shot’ instruments. Azimuth and dip information is collected at 9 to 15m intervals downhole, along with magnetic susceptibility readings. Magnetic susceptibility is assessed for each directional reading data point to ensure azimuth readings have not been made inaccurate by highly magnetic materials in proximity to the azimuth data point.

10.4 Results

LSG has created a Gemcom GEMS database of all underground and surface drilling done to date on the property. This database was built from a combination of the mine’s Amine Corelog database and ASCII csv files of collar location, collar and down hole survey, lithology, and assay data from the surface exploration group.

LSG is currently in the process of establishing a master database based on Gemcom GEMS SQL which would integrate the surface and underground programs into a single database. This would eliminate the need to combine databases in the future.

The database utilized in the Mineral Resource estimation process consisted of an Access based Gemcom GEMS database of collar location, collar and down hole survey, lithology, and assay data.

The parameters of the drill hole database are summarized in Table 10-3.

Table 10-3: Summary of Database Parameters

Table Name	Table Description	Fields
Header	Drill hole collar location data in local grid co-ordinates	Hole-ID Location X Location Y Location Z Length Collar_Azimuth Collar Dip
Survey	Down hole survey data of direction measurements at down hole distances	Hole-ID Distance Azimuth Dip
Assays	Sample interval assay data with Au units grams per tonne	Hole-ID From To Au_GPT_FIN Au_GTT_AA Au_GPT_GRA Au_GPT PM
Lithomaj	Major logged rock type intervals down hole	Hole-ID From To Rocktype
Lithomin	Minor logged rock type intervals down hole	Hole-ID From To Rocktype

11.0 SAMPLING METHOD AND APPROACH (ITEM 14)

11.1 General

Sampling methods have varied somewhat between operators but have been consistently of a high standard. Drill core recovery has always been close to 100% and all of the surface core drilled by Holmer and LSG is stored in core racks for future reference. A portion of the drill core drilled from underground platforms is 'whole core sampled', thus, is no longer available for reference. Photographs and logs of these holes are available for reference. Also, care is taken is ensure equally representative geology is available for reference in nearby drill holes, which were not whole core sampled.

11.2 LSG 2003-present

Sample lengths within the well-mineralized sections of core are 0.5 m with minor variations determined on the basis of lithologies and vein contacts. Sample intervals are increased up to as much as 1.5 m where sparse mineralization is encountered. The sample intervals are determined by the logging geologist, then marked on the core, and recorded in the drill log. The core is split by LSG technicians using a diamond saw and half of the core is placed in a plastic sample bag. The remaining half is returned to the core box and retained for future use and to serve as a permanent record. The archived core is stored in racks adjacent to the field office, which is within a gated compound adjacent to the office and warehouse of Bradley Bros. Limited. This facility is located on the north side of Highway 101, about halfway between downtown Timmins and the Property. LSG uses sequentially numbered triplicate sample tags. One portion goes in the sample bag; one goes into the core box at the end of the sample interval and the third stays in the sample book.

Prior to early 2007, the samples were transferred in security-sealed bags and transported by Manitoulin Transport to the ALS Chemex Prep Lab in Mississauga (2003 to 2005), and then to Sudbury (2006 to 2007).

Since early 2007, samples are being delivered by LSG personnel or commercial couriers directly to the ALS Chemex Prep Lab in Timmins. The pulps created in Timmins are then shipped to the ALS Chemex Assay Laboratory in Vancouver, B.C or Rouyn-Noranda, PQ.

11.3 Holmer 1998-2002

All core was delivered by the contractor to a secured location at the Holmer core shack. The core was logged and samples marked on the basis of geological divisions. All core to be sampled except for the quartz-tourmaline veins of the Main Zone and Hangingwall

Zone mineralization in which visible gold was observed was split mechanically. The suspected higher-grade intercepts with visible gold were sent for assay as whole core.

Sample length averaged 1.20 m but was shorter in the well-mineralized sections. The samples remained at the secure site until delivery to the shipping company. The samples were then transported to Accurassay Laboratories (Div. of Assay Laboratory Services Inc.) ("Accurassay") in Thunder Bay by BPX.

11.4 Holmer 1996-1997

Sample lengths were based on geological features, with lengths ranging from 0.14 m to 4.0 m. Sampling and assaying procedures were identical to those in 2002 except for holes 96-01 to 96-10 and 97-01 to 97-06 in which all core was split and no whole core was submitted to the lab.

Holes 97-07 to 97-57 were sampled using the same procedures as in 2002. Sampling lengths were based on geological features, with lengths ranging from 0.14 to 1.75 m.

11.5 Relevant Samples

The LSG and Holmer drill core is stored adjacent to the field office in core racks and WGM collected two independent samples of the archived half of the split core from LSG drill holes during its site visit. A description of these samples is contained in the WGM Technical Review Report.

12.0 SAMPLE PREPARATION, ANALYSIS AND SECURITY (ITEM 15)

12.1 Sample Preparation and Assay Procedures

12.1.1 General

Sample preparation for all samples in the database used for Mineral Resource estimates on the Property has been carried out at the primary commercial laboratory used by the program operator. Except as described above in Section 11 Sampling Methods and Approach, no employee, officer, director or associate of LSG was involved in any aspect of the sample preparation.

The following assaying and check assaying procedures are largely as described in a report entitled "QC/QA Procedures for 2004 Resource Calculation" by LSG. All of the supporting details of the standards and check assay results obtained by LSG up to the time of the resource estimate of 2004 and 2006 are also summarized in the WGM Technical Review Report.

12.1.2 LSG 2008-present

LSG's samples are prepared at the ALS Chemex facility in Timmins, and the pulps are then analysed by ALS Chemex Laboratory in Vancouver, B.C or Rouyn-Noranda, PQ. ALS Chemex is an ISO 9001-2000 registered laboratory.

Most strongly mineralized zones are assayed for gold by pulp and metallic screen analysis while minor veining or alteration are submitted for standard fire assay with an atomic absorption ("AA") finish.

The samples sent for "regular" assaying are entirely crushed to ≥70 % passing 2 mm mesh. The crushed samples are split and a 250 g sub-sample is pulverised to ≥85% passing <75 micron using a ring & puck pulveriser (PREP-31). A 30 g aliquot is taken from the pulp and analysed by fire assay and atomic absorption methods (Au-AA23). For samples that return a value greater than 3 gpt Au, another pulp is taken and analysed using a gravimetric finish (Au-GRAV21).

Samples from prospective veins and significantly mineralized zones are analysed using "pulp and metallic/screen fire assay" methods (Au-SCR21). The samples are crushed to ≥70 % passing 6 mm mesh, and the entire sample is then pulverized to ≥85 % passing 75 micron (PREP-21). The pulp is passed through a 100 micron stainless steel screen and the entire (+) fraction is analysed by fire assay and gravimetric finish. The (-) fraction is homogenized and two 30 g aliquots are analysed by fire assay and atomic absorption finish (Au-AA25 and Au-AA25D). The total gold content is then calculated by combining the weighted averages of the two fine fractions with the grade of the coarse fraction.

A check-analysis is also done by the lab by producing and analysing a second pulp from the coarse reject on every 25th sample received.

The analytical methods as described in this section represent a small divergence from what was previously done during the 2004 and 2006 resource drilling campaigns (see Section 12.1.3 below): Since August 2008, samples from surface exploration holes are not analysed for arsenic, as it appears unnecessary. The fusion weights were also lowered from 50g to 30g aliquots in December 2008, in order to harmonize the protocols with the ones recommended for the adjacent Thunder Creek project.

A special "AU DUP Protocol" was requested on selected samples from the 21 ramp definition holes which were drilled in January and February 2008. This method implied pulverising the entire sample and taking a duplicate split of the pulp (SPL-24, 150g), followed by regular fire assay and ore-grade atomic absorption finish on both splits (Au-AA26, upper limit of 100g). This exercise was meant as a substitute to pulp and metallics method, and was meant to test for repeatability of the results on mineralized samples.

The analytical method used for the underground drill program has been Fire Assay with Atomic Absorption finish, and should a result exceed 10 ppm, then a Fire Assay with Gravimetric finish is also completed, both using 50g aliquots.

For a short time between the commencement of underground drilling at the project in October 2008 and February 2009, a portion of underground drill core samples were prepared and analyzed by Swastika Laboratories of Swastika Ontario. Swastika Laboratories are not an accredited analytical laboratory.

With the commencement of underground development and grade control activities combined with the commencement of gold analytical capability at Lake Shore's Bell Creek Complex, a minor portion of the gold analyses completed on underground drill core samples has been completed at Lake Shore's Bell Creek lab. These analyses have been completed on AQTK-sized drill core, in areas where Mineral Resources have previously been established and where the information is being sought very quickly and a very detailed manner in order to assist in daily geology control activities (commonly, these are termed 'production drill holes').

Also for a short time between June and July 2009, ACTLabs of Hamilton Ontario was used to analyze a portion of the underground drill samples generated from the project. These samples were prepared using ACTLabs affiliated facility in Timmins, True North Lab. Again, these analyses were completed on AQTK-sized drill core, in areas where Mineral Resources have previously been established and where the information is being sought very quickly and in a very detailed manner in order to assist in daily geology control activities (again, these are commonly termed 'production drill holes').

Again, in certain instances, underground drill core is 'whole core sampled', that is the entire core is sent for analysis rather than cutting and retaining half the core.

At present, all un-sampled underground drill core is stored on site either stacked on pallets or in core racks at the Timmins Mine property.

12.1.3 LSG 2003-2007

LSG's samples are prepared at the ALS Chemex facility in Mississauga (from 2003 to 2005), Sudbury (from 2006 to early 2007), and Timmins (Feb 2007 to present). Sample pulp sub-samples are shipped to and analyzed by ALS Chemex at its ISO 9001:2000 and ISO/IEC Standard 17025 accredited laboratory in North Vancouver.

All strongly mineralized zones are assayed for gold by pulp and metallic screen analysis while minor veining or alteration are submitted for standard fire assay with an atomic absorption ("AA") finish. All samples are also analysed for arsenic (As), by aqua regia digestion and atomic absorption scanning.

The samples that have not been selected for pulp and metallics assaying are crushed to 70% passing 2 mm mesh. The crushed sample is split and a 250 g sub-sample pulverised using a ring and puck pulveriser. A 50 g aliquot is taken from the pulp and analysed using fire assay methods with an AA finish. For samples that return a value greater than 3 g Au/t another pulp is taken and fire assayed with a gravimetric finish.

For the samples subjected to metallic screen assaying, the grade of the pulp is determined by averaging the results from two 50 g aliquots that are fire assayed with an AA finish. The entire sample (approximately 1 kg) is used for the metallic screen assay with the coarse fraction fire assayed with a gravimetric finish.

ALS Chemex inserts a duplicate sample as a check for every 25 samples. Standards are also inserted on a regular basis.

12.1.4 Holmer 1996-2002

Accurassay in Thunder Bay served as Holmer's primary laboratory. While Holmer procedures varied somewhat over the period, in general higher grade whole core samples were analyzed by pulp and metallic screen analysis in a manner similar to that described above, while 20 g aliquots of split core samples were assayed by fire assay with an AA finish.

Accurassay inserted a duplicate pulp every 10th sample and also inserted standards into the sample stream.

12.2 Quality Assurance/Quality Control

12.2.1 LSG – 2003

LSG's quality assurance/quality control ("QA/QC") procedures were based on advice from Mr. John Reddick, an independent consultant, near the start of the program in 2003. No standards and blanks were submitted for the first two holes and the pulp metallic assay procedure described in Section 13.1.2 of the WGM Technical Review Report varied slightly using only 250 g of the coarse material for the coarse fraction assay. The controls described below were used for all remaining drilling.

One field standard and one field blank were inserted into the sample stream with every 25 samples prior to shipment to the lab. Blanks were prepared using barren diabase drill core. Standards of varying gold content prepared by Ore Research and Exploration Pty. Ltd. ("OREAS") of Australia and provided by Analytical Solutions Ltd. of Toronto were used and are summarized in the following Table 12-1.

Table 12-1: LSG QA/QC Procedure Standard Samples

Standard	Source	Value	Comment
A-Au42b	Accurassay	0.59 g /t Au	Poor standard
O-6Pb	OREAS	1.42 g /t Au	
O-15Pz	OREAS	1.27 g/t Au	
O-61Pa	OREAS	4.45 g/t Au	
O-61Pb	OREAS	4.75 g/t Au	

In a few cases, there was a significant variation from the expected value, as summarized in the Table 12-2 below.

Table 12-2: LSG QA/QC Procedure Variations from Supplied Standards

Standard	Value	Number Submitted	# with > 10% variation from expected value	# with >20% variation from expected value	# with Au > expected value
A-Au42B	0.59	22	17	3	0
O-6Pb	1.42	56	10	3	0
O-15Pz	1.27	84	6	1	1
O-61Pa	4.45	27	3	0	0
O-61Pb	4.75	116	1	5	0

Standard A-Au42b was clearly of very poor quality and was not used after submitting the initial 22 samples. Only one standard in the entire database of samples deviated from the expected value by more than 10%, sample 241727 that assayed 1.47 g/t Au, 15.75%

higher than the expected value of 1.27 g/t Au. Although the other standards all have a small percentage of samples with >10% variation from the expected value, in all cases the reported value was less than the expected value. Considering the consistency in the re-assay program (see below), this was interpreted to reflect variation of distribution of gold within the standard rather than reflecting on the reliability of the lab analyses.

12.2.2 LSG 2003 Check Analysis

Check assaying was carried out at Accurassay in Thunder Bay. The first check assay program was carried out after completion of the 2003 drilling. Every 20th sample from the entire LSG-generated portion of the database as well as every 20th sample from the portion of the LSG- generated portion of the database that reported in excess of 1.0 g/t Au was re-assayed. Samples initially assayed by pulp metallics had no coarse reject, therefore only the pulp was re-assayed. For samples initially assayed by 50 g fire assay, a re-assay was done on both the existing pulp and on a new pulp prepared by Accurassay from the coarse reject.

Overall the check assays were very consistent. Two hundred and five fire assay pulps were re-assayed and the data show a 0.996 correlation coefficient. The maximum variation was from a single high-grade sample that assayed 15.25 g /t Au at ALS Chemex and 17.85 g /t Au at Accurassay. One hundred and ninety-eight coarse rejects were prepared and re-assayed at Accurassay. The sample populations show a 0.97 correlation coefficient. As with the pulps, the only samples to show any significant variation are in the >10 g /t Au range, with sample 239103 assaying 8.59 g /t Au at ALS Chemex and 12.84 g /t Au at Accurassay. These samples were then re-assayed using both the pulp and coarse reject. The ALS Chemex assay and the Accurassay check show a 0.98 correlation.

The pulp and metallics samples were checked by re-assaying the pulps from 72 samples at Accurassay. The samples show a 0.97 correlation coefficient. Only two high-grade samples showed any notable variation. Sample 250913 assayed 15.85 g/t Au at ALS Chemex and 23.4 g Au/t at Accurassay and sample 241781A assayed 29.7 g/t Au at ALS Chemex and 21.7 g/t Au at Accurassay. Considering the high grade of the samples and the presence of even higher gold in the coarse fraction, these differences are not considered significant.

12.2.3 LSG 2004-2006

LSG has continued the QA/QC protocol for analytical data established in 2003 and as described above. WGM reviewed the data collected and results reported by LSG for the 2004 to 2006 period. The QA/QC program for 2004-2006, the period since the previous resource estimate and technical review by WGM is summarized in Table 12-3.

Table 12-3: QA/QC Sample Statistics – 2004-2006

Control Sample	Number
Standard Reference Material Samples	789
Blanks	841
Check Assays 2004-2005	577
Check Assays 2005-2006	143
Total	**2,350**

During this period, LSG analyzed approximately 17,500 samples. The LSG-inserted 'in-stream' control analyses (standards and blanks) therefore represent approximately 8.5% of analyses during the programs. A total of 720 check assays has been completed during the period, which represents approximately 1 in 25 samples analyzed.

As per the previously reported period, LSG has used certified gold standards prepared by OREAS and provided by Analytical Solutions Ltd. The standards used during the 2004-2006 period are summarized in Table 12-4.

Table 12-4: List of Gold Standards and Performance Gates Used By LSG (2004-2006)

Standard Ref.	Expected Mean Value	Performance Gate, g/t	Max, g/t	Min, g/t
O-6Pb	1.43	0.16	1.58	1.27
O-7Pb	2.77	0.16	2.93	2.61
O-15Pz	1.27	0.12	1.39	1.15
O-60P	2.6	0.2	2.8	2.4
O-61Pa	4.46	0.4	4.86	4.06

The pass-failure summary of the 789 standard analyses is contained in Table 12-5. A total of 73 samples was outside of the 3X standard deviation range and are thus considered failures. However, only 12 (2%) returned values greater than expected value and if the five from standard O-7Pb are not considered, as it is postulated by LSG that it may be an unreliable standard due to the high number of failures, then only seven (1%) of the standards failed high. LSG has further investigated the control samples associated with the sample batches from the standards that analyzed high, including the lab internal controls as well as the repeat standards. All of these tests were positive and it is suggested that the high values may be statistically representative of the sample population.

Blanks consist of barren diabase core inserted into the sample stream randomly at a rate of approximately one per 20-21 samples. A total of 841 blanks were inserted for the reporting period. Using a threshold of 100 ppb, 18 samples (2%) are considered anomalous. All of these samples were part of pulp and metallic batches and were likely contaminated following high-grade samples.

Table 12-5: Summary of Standards Submitted for Analysis during the 2004-2006 Program

Standard Reference	Expected Mean Value (g/t)	Number Submitted	# outside of the 3 x standard deviation range	# with >20% variation from expected value	# with Au > expected value
O-6Pb	1.43	236	18 (8%)	7	2
O-7Pb	2.77	114	21 (18%)	2	5
O-15Pz	1.27	35	2 (6%)	0	0
O-60P	2.60	105	8 (8%)	3	1
O-61Pa	4.46	12	0 (0%)	0	0
O-61Pb	4.75	248	19 (8%)	4	4
O-62Pb	11.33	39	5 (13%)	4	0
Total		**789**	**73 (9%)**	**20 (3%)**	**12 (2%)**

All check assays were completed at Accurassay Labs in Thunder Bay, Ontario. The samples re-assayed include cuts of original pulps and pulps prepared from coarse rejects (Table 12-6). Pulps analyzed include both those prepared for standard FA-AA and those prepared for screen fire assay.

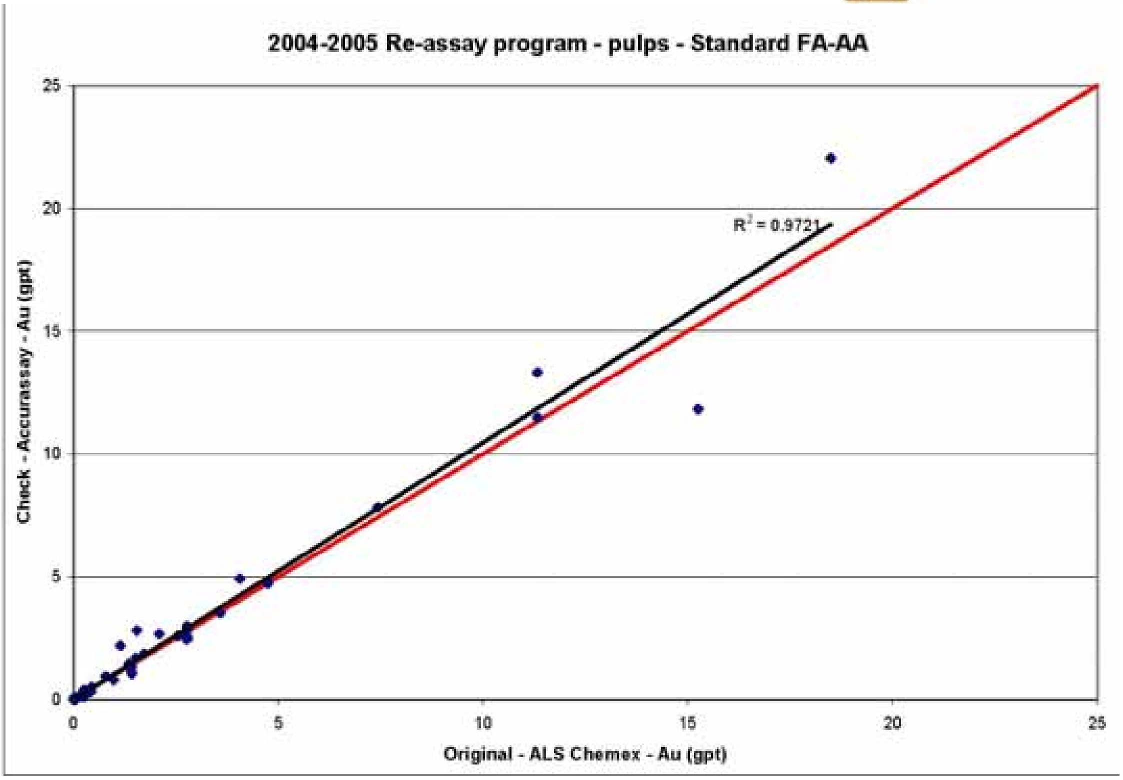
Table 12-6: Check Assay Samples 2004-2006

Sample Description	Number of re-assays
2004-2005 Re-assay program	
Pulps assayed by standard FA-AA	274
Pulps prepared from coarse reject	167
Pulps assayed FA-AA from SFA* samples	136
2005-2006 Re-assay program	
Pulps assayed FA-AA from SFA samples	143

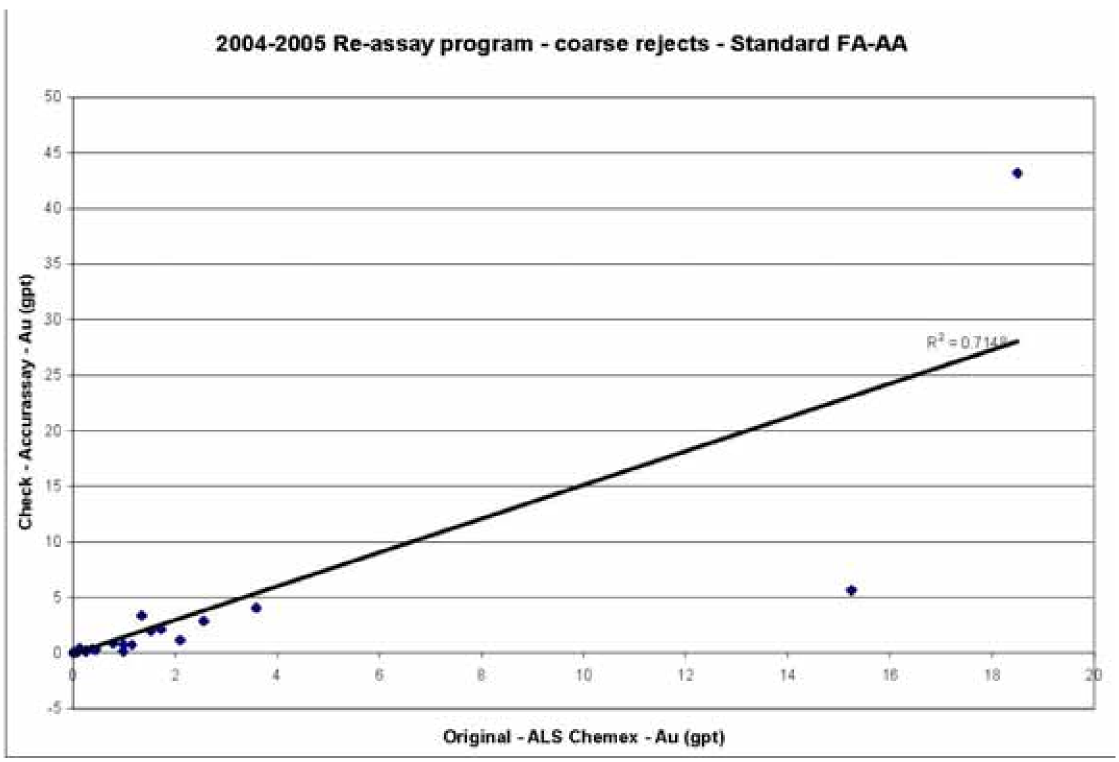
* SFA = Screen Fire Assays

Scatter plots of original and check assay data generally form linear arrays (Graph 12-1 to Graph 12-5). The sub-equal distribution of data points about the one-to-one lines indicates no apparent bias between the original and check assay data sets and this, for the most part, is supported by the calculated correlation coefficients. In the case of the check data for the coarse rejects from the 2004-2005 data, two higher grade samples have variable results although the general population (<5 g Au/t) does not demonstrate a clear bias but does show some spread in original and check values. As these samples are from the coarse rejects, the spread is attributed to natural variability or the "nugget effect”. In the 2005-2006 check data of pulps from Screen Fire Assays, three samples have erratic results. Two of the samples have check values greater than the originals and one sample has an original value three times the check value. Although natural variability can be expected in the analyses of pulps, it should be less than that from the coarse rejects. WGM suggests that the discrepancy of the three samples from the 2005-2006 check assay data probably has a different origin such as simple "mix-ups" or possibly sample segregation due to transport from original laboratory to the check laboratory and improper re-homogenization at the check laboratory.

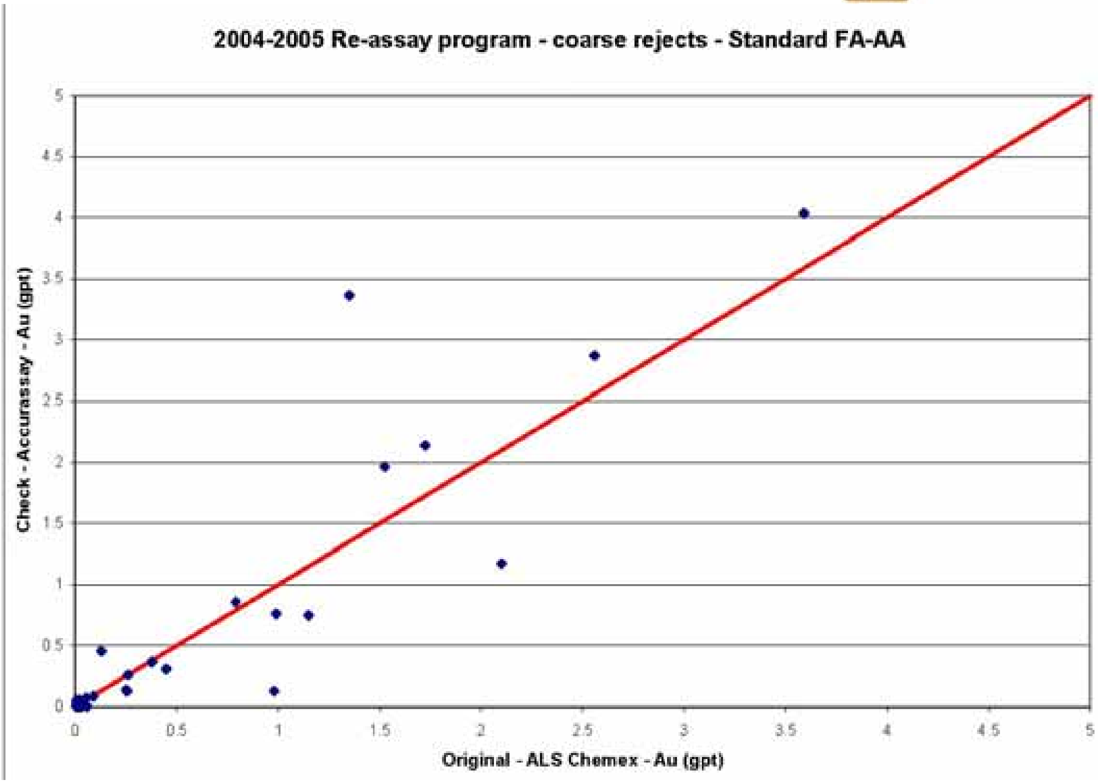
LSG has been very diligent and rigorous with the administration of QA/QC on the Timmins Mine assay data. WGM is of the opinion that the LSG QA/QC program meets or exceeds industry standards, that it demonstrates that there is no significant systematic error or bias present in the ALS Chemex assays and that the assay database is suitable for use in the preparation of Mineral Resource estimates. WGM recommended, however, that check assaying be carried out on a regular and more frequent basis, so corrective measures can be undertaken in a timely manner in the unlikely event that a problem is detected.



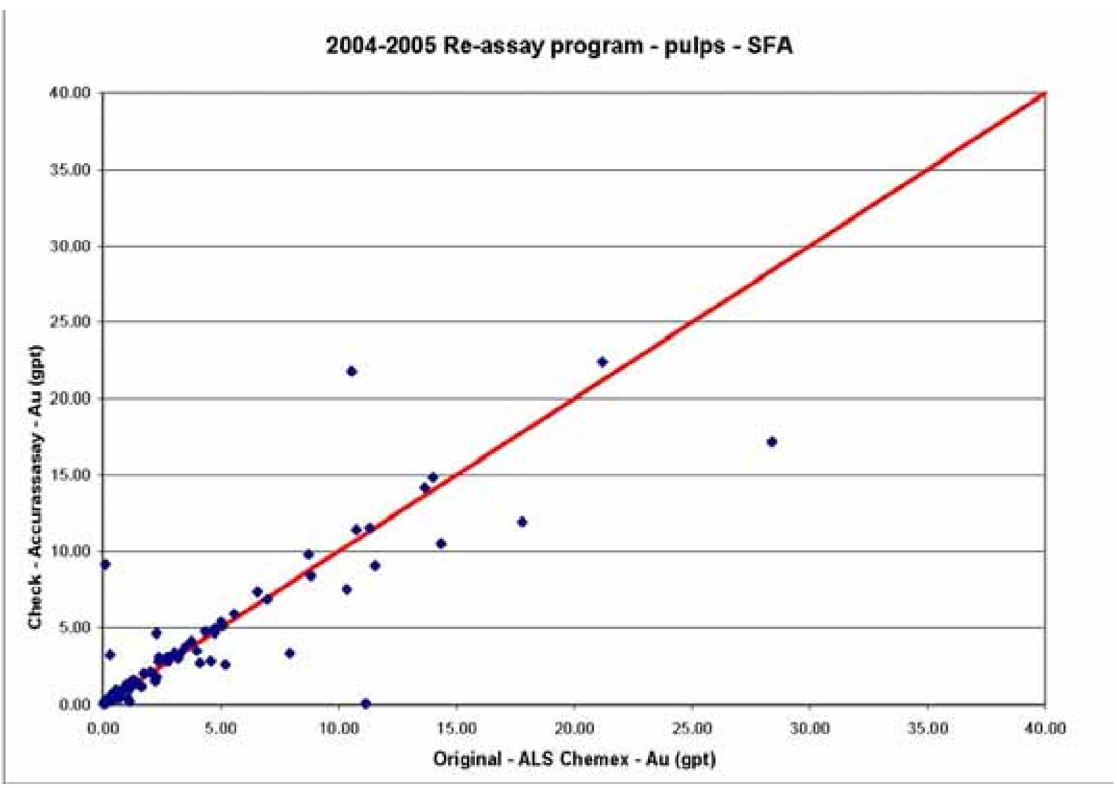
Graph 12-1: Scatter plot of check assay data, 2004-2005 program, pulps, standard FA-AA



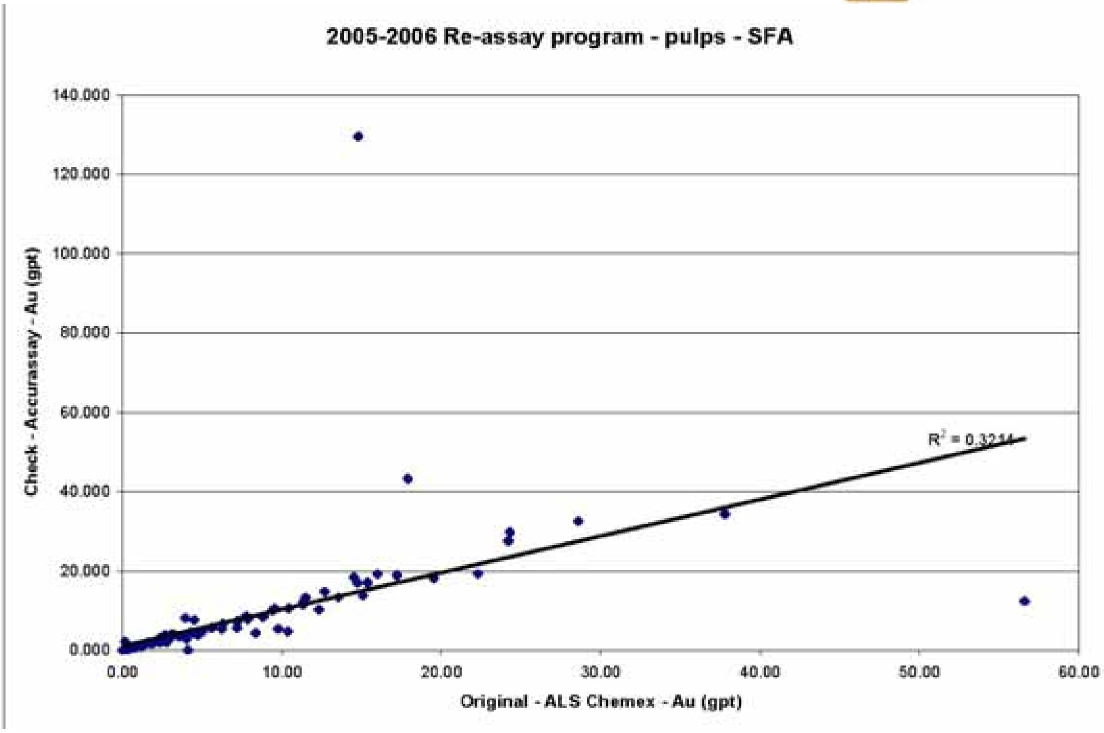
Graph 12-2: Scatter plot of check assay data, 2004-2005 program, coarse rejects, standard FA-AA

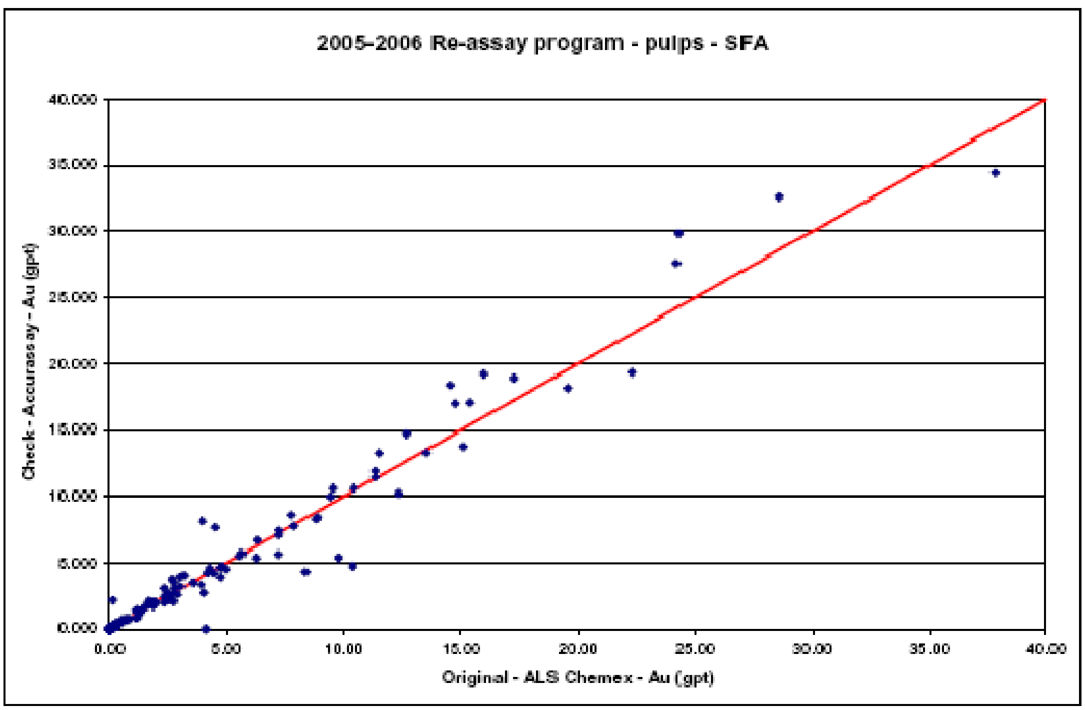


Graph 12-3: Scatter plot of check assay data, 2004-2005 program, coarse rejects, standard FA-AA



Graph 12-4: Scatter plot of check assay data, 2004-2005 program, pulps, SFA





Graph 12-5: Scatter plot of check assay data, 2005-2006 program, pulps, SFA

12.2.4 LSG 2007-present

LSG's quality assurance/quality control ("QA/QC") procedures have remained the same since 2007, as the protocols described above.

One field standard and one field blank were inserted into the sample stream with every 25 samples prior to shipment to the lab. Blanks were prepared using barren diabase drill core. Standards of varying gold content prepared by Ore Research and Exploration Pty. Ltd. ("OREAS") of Australia and provided by Analytical Solutions Ltd. of Toronto were used.

Since 2007, LSG has maintained the same quality assurance/quality control ("QA/QC") procedures as the protocols described above. A statistical analysis of the results is still in progress. Most of the drilling during that period was focused on exploration targets outside of the resource. Since no significant assays were obtained and results bear no impact on the resource, a check assay program was not carried out.

A check analysis program from underground drill samples has been recently initiated. QA/QC results for underground drill sampling are currently will be compiled and assessed for more detailed analysis of the precision and accuracy of completed gold assays.

12.2.5 Holmer 1996-2002

Field standards and blanks were not submitted during the 1998 and 2002 programs. Quality control was maintained only at the lab where every 10th sample was re-analysed.

A check assay program was completed in 1997. A total of 247 sample rejects from mineralized and non-mineralized zones was sent for re-assay to X-RAL Laboratories ("X-RAL") (now a division of SGS Canada Inc., Mineral Services ("SGS")).

Samples from both the Vein and Footwall Zones were selected for check assays. The average grade of all samples remained almost identical, decreasing by a minor factor of -0.24% when re-assayed by X-RAL. The median increased slightly by 1.1%. The correlation coefficient between Accurassay and X-RAL assays was 0.928. There were several outliers from the high end of the assay range from samples taken from the Vein Zone associated with coarse gold mineralization in quartz-tourmaline vein systems. These were probably due to the nugget effect associated with coarse gold and not analytical error.

12.3 LSG Security

The drill core is transported to a secure logging and sampling site where logging and sampling takes place. Assay samples are marked and identified by a numbered sample tag, a duplicate of which is placed in the sample bag. The tag placed in the bag contains no indication of the hole number or meterage. The shipping bags are sealed with a numbered security seal as noted on Chain of Custody documents which accompany shipments and are verified by the receiving lab facility as being intact. There is little opportunity for anyone to tamper with the samples in any organized manner during the sampling or sample shipping process.

13.0 DATA VERIFICATION (ITEM 16)

Information relating to past work on the TM Property was primarily obtained from LSG and previous 43-101 reports completed by WGM which validated work up until 2006. Subsequent to 2006, the author has visited both the LSG field office and core storage facilities adjacent to the Bradley Bros. Ltd. operations base on Highway 101 just west of Timmins and the mine core shack located adjacent the portal at the Timmins Mine on a number occasions and observed operating drills on both surface and underground.

Logging and sampling procedures were meticulous and "general housekeeping" at the site, core shack, field and mine office was very good.

The LSG and Holmer drill core is stored adjacent to the field office in sturdy racks or at the Timmins Mine property, most of which are well preserved. Mine underground core has been labelled and cross-piled adjacent to the portal area at the Timmins Mine site. WGM collected two independent samples of the archived half of the split core from LSG drill holes the results of which are summarized in the WGM Technical Review Report.

13.1 Limitations

Assay certificates are maintained at the LSG field and mine offices. LSG is currently in the process of centralizing the certificates at the Timmins Mine office complex.

14.0 ADJACENT PROPERTIES (ITEM 17)

14.1 Statement

The WGM Technical Review Report concluded the following:

- There are no significant past or presently producing mines in the immediate vicinity of the TM Deposit and the Property, although there is virtually no open ground.
- West Timmins Mining Inc., from whom LSG has optioned claims on its southern and western borders, holds a large block of claims surrounding the Property. LSG has conducted drilling programs on this Property and has encountered encouraging gold values.

The style of mineralization, type of host rocks, proximity to major lithological contacts, and proximity to major through-going structures found on the Property are features that are similar to those found in association with many gold mines in the Porcupine Camp. Some of these have been world class in size (e.g., Hollinger-McIntyre Mines with about 30 million ounces of combined production, and Dome Mine with approximately 15 million ounces production).

15.0 MINERAL PROCESSING AND METALLURGICAL TESTING (ITEM 18)

15.1 Initial Bell Creek Mill Start-up

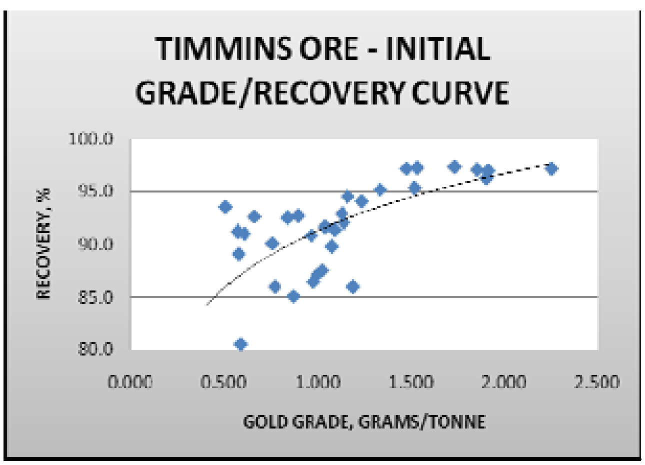
Starting in May 2009, the Bell Creek Mill has been in start-up mode, processing all available material from the Timmins ore body. The ore has been processed in campaigns when sufficient material was available to operate for approximately 5 days or longer. The mill has currently processed 16,888 dry tonnes in 3 campaigns. Two campaigns were in May and one in July 2009.

For these campaigns, the mill has operated with only the large ball mill (3,658 x 4,877 mm (12' Ø x 16') operating. The mill has typically operated at a maximum tonnage of approximately 500 tons per day but when ore was available, the single ball mill has comfortably obtained grinding rates of approximately 1,000 tonnes per day.

Initially in May, most of the feed material to the mill was low grade ore of <1.5 grams Au/tonne. The recoveries of this material were consistently above 90%. Milling in July was with ore typically having higher grade material consistently >1.5 grams Au/tonne. The recovery of this material was consistently above 95%. One day in July operated with nearer normal head grades of 5.96 grams Au/tonne and had a recovery of 99.3%. The gravity circuit has been collecting approximately 40% of the gold in the circuit. These recoveries are in-line with projections.

Figure 15-1 shows the current Au grade/recovery curve. This curve is estimating recoveries as projected at or above 97% Au recovery at grades of 9.5 grams per ton.

Figure 15-1: Initial Gold Grade/Recovery Curve



During the initial start-up, the mill operated at projected cyanide levels of 300 ppm with no abnormal cyanide usage noted.

As start-up progresses, more data will become available. At the time of writing, this author sees no material issues that are significantly different then initial projected for the processing facilities.

Prior to the Bell Creek Mill start-up, there had been no mining of ore from the Lakeshore Properties on the Timmins Property; therefore no mineral processing had taken place. To the best of the authors knowledge, no other historic operators/owners other then LSG carried out any metallurgical testwork which is identified below.

15.2 Metallurgical/Rheology Testing History

An extensive series of metallurgical/rheological testing has been completed for Timmins LSG ores. Test work has followed industry accepted standard practices and is believed to be technically sound and representative for the deposit, although there can be no guarantee that all mineralogical assemblages have been tested.

The following companies have been involved with various aspects of the Lakeshore Gold Project metallurgical evaluations:

- SGS Lakefield Research Limited, Lakefield, Ontario (“SGS”);
- EHA Engineering Ltd., Richmond Hill, Ontario (“EHA”);
- RPC Engineering, Fredericton, New Brunswick (“RPC”);
- Pocock Industrial, Inc, Salt Lake City, Utah, USA (“Pocock”); and
- Golder Associates, Sudbury, Ontario (“Golder”).

In a synopsis of the test work, RPC and SGS tested samples of the ore types as composites as well as individual samples. The test programs consisted of bottle rolls to determine the metallurgical response of the ore types to cyanide recovery along with tests to determine gravity concentration, pulp agglomeration, flotation, and cyanide leaching of the flotation tailings and concentrates. RPC performed crushing, grinding and abrasion indices determinations. Pocock and Golder performed flocculent screening, gravity sedimentation, and pulp rheology on leached tailings samples. SGS also performed preliminary Sag mill testing. EHA evaluated work completed by RPC.

Below is a brief review of the test work completed by the various metallurgical companies. Please review the actual documents for the full text of each document.

15.2.1 SGS Preliminary Metallurgical Testing: The Recovery of Gold from Samples from the Timmins Property (February 16, 2005)

In November 2003, preliminary metallurgical test work on LSG's Timmins ore samples was conducted by SGS Lakefield Research. This program used assay reject samples to chemically characterize and determine the amenability of the LSG samples to conventional cyanidation and gravity processing.

Conclusions to this study are identified as follows:

- Samples responded well to both direct cyanidation and gravity separation followed by cyanidation of the gravity tails
- Gold extractions were good with recoveries between 92 and 99%
- There was little performance difference between footwall and ultramafic rock type recovery
- High wall and main samples contained higher percentage of gravity recoverable gold with all samples containing significant gravity recoverable gold
- Cyanide consumption was estimated at 0.2 to 0.4 kg/tonne
- Lime consumption was estimated at 1.0 to 1.7 kg/tonne

15.2.2 EHA Report: Review of Metallurgical Testwork and Process Design (May 2006)

In May 2005 a metallurgical test work program under the direction of EHA Engineering Ltd. (EHA) was initiated at the Research and Productivity Council (RPC) in Fredericton, New Brunswick. Gravity, flotation and cyanidation test work was conducted to support a planned pre-feasibility study. Drill core samples from the designated Vein Zones, Main Zone, Ultramafic Zone, and Footwall Zone were used for this program. Based on preliminary test work, two main composites were prepared for detailed test work with Main and Ultramafic Zones accounting for most of the mineral resource with much of the test work conducted on a 50:50 blend of the two.

The metallurgical test work completed by RPC was reviewed by EHA and used as a basis for the development of process design criteria, supporting material balance, preliminary capital and operating costs in support of the pre-feasibility study.

Conclusions reached by EHA are identified as follows:

The Timmins deposit:

- Identified that relatively fine grinding at a grind of 80% minus 50 microns followed by 48 hours of conventional cyanidation and carbon in pulp (CIP) processing for recovery of gold yielded a design gold extraction of 95.5 %
- Flotation with concentrate cyanide leaching is a viable process that should be considered further
- Material from all zones is substantially free of deleterious elements likely to interfere with processing or present an environmental concern

Certain metallurgical testwork was incomplete and recommended prior to detailed design:

- Semi-autogenous amenability tests
- Confirmatory cyanide destruction tests
- Thickening and rheological testwork to confirm paste thickener sizing and to provide data for detailed design of pumps and lines
- Additional flotation locked-cycle testwork

15.2.3 RPC Testwork: Lake Shore Gold Project Preliminary Beneficiation Results (July 20, 2006)

RPC's Process and Environmental Technology group was contracted by LSG, to develop and test extraction strategies for gold bearing samples provided from the LSG Timmins project. The contract was awarded in May 2005 and finished in July 2006. These tests were independently conducted at RPC's facilities in Fredericton, New Brunswick, under support from subcontractors as required. EHA Engineering Ltd. provided test procedures and direction throughout the entire program. The test program consisted of sample characterization, mineralogy and a series of metallurgical and environmental tests (grinding index, flotation, gravity, CN leaching, SO_2 destruction, CIL, CIP, ABA, column, humidity cell, and tailings physical/chemical characterization).

Results reached by RPC are as follows:

- Direct cyanide leaching achieved high gold extractions (>95%) in all four samples evaluated (FW, UM, VM and FW/UM)

- Gravity table tests achieved from 45% to 74% Au recovery in the concentrate in the VM and UM/FW samples
- Combined recovery with flotation of the table middling's and tails was 91% for the FW/UM sample and up to 96% for the VM sample
- Rougher flotation recovery on the head composite samples (FW, UM and VM) ranged from 90% to 95% and the FW/UM blend achieved 91% to 93% gold recovery with grades up to 100 g/t
- Cleaning tests carried out on the FW/UW sample achieved an increase in grade up to 171 g/t Au with recoveries of up to 95%
- CIL tests were carried out on FW/UM and VM and Au extracted into the carbon was 97%
- CIP modeling tests were then carried out on the FW/UM sample using a grind size of P80 of 53 microns, pulp density of 45%, 0.5 g/L NaCN and 48 hour leach retention time
- Limited cyanide destruction and testing work was completed

The following metallurgical testwork is incomplete:

- Lock cycle flotation testing is required to confirm the flotation test results

15.2.4 SGS Sag Mill Testing: Proposed Grinding Systems for the Timmins West Gold Project Based on Small Scale Date (August 15, 2008)

Two samples from the Timmins LSG deposit were submitted for preliminary Sag Mill testing and sizing. The samples consisted of a large ultramafic sample and a Main/Footwall zone sample. The ultramafic sample was tested using normal testing parameters with the Main/Footwall zone sample tested using the JKTech drop-weight test.

The following are the conclusions as identified by SGS:

- The test results categorized the sample as medium in terms of resistance to impact breakage, with the Main/FW zone sample categorized as hard with respect to resistance to abrasion breakage
- The grindability parameters for a theoretical "Overall Composite" were calculated from the individual test results assuming a 40% Main/FW composite and 60% Ultramafic ratio. This ratio was used to develop a suitable grinding circuit capable of milling 113.2 tonnes/hour to a final grind size, P_{80}, of 75 microns

- The simulations investigated several second-hand SAG mill sizes, as well as the effect of the classification screen size on the ball mill and overall power requirements

15.2.5 Pocock Industrial: Flocculent Screening Gravity Sedimentation and Pulp Rheology Studies (October 2008)

Solids liquid separation (SLS) tests were conducted on two samples of tailing material varying in targeted grind size for the LSG Bell Creek Mill. The samples were prepared by The Center for Advanced Metallurgical and Mineral Processing (CAMP), and shipped to Pocock Industrial, Inc. for SLS testing in slurry form in late September of 2008. Flocculent screening and conventional (static) thickening tests were conducted on the sample to select a flocculent product for testing, and to develop a general set of data for conventional thickener design. Dynamic (continuous) thickening tests were conducted on the sample to determine the hydraulic parameters/limitations suitable for high-rate type thickener design. Thickener underflow produced from static and/or dynamic thickening tests for the sample were used for viscosity testing to determine the rheological properties of the thickened pulp. Viscosity tests were conducted with a FANN viscometer to develop data for standard thickener and pump/pipeline design. No unusual results were obtained.

15.2.6 Golder Associates Ltd: Results of Initial Laboratory Testing on Lake Shore Gold Tailings (July 6, 2006)

LSG requested Golder to assess tailings disposal options, based on existing data, to support LSG's prefeasibility study. For the prefeasibility study, two types of tailings were evaluated, specifically slurry and paste tailings. In reviewing the options, Golder completed a laboratory testing program to determine the suitability of paste tailings. Particle size analysis, specific gravity determination, water retention, thickening, settling and filtration analysis, and unconfined compressive strength (UCS) at varying cement contents and tailings/sand blends were completed. No unusual results were obtained.

15.3 Metallurgical Test Results

Metallurgical testwork results of interest are as follows:

Combined recovery with flotation of the table middling's and tails was 91% for the FW/UM sample and up to 96% for the VM sample. Rougher flotation recovery on the initial three composite samples (FW, UM and VM) ranged from 90% to 95% and the FW/UM blend achieved 91% to 93% Au recovery with grades up to 100 g /t Au.

Cleaning tests carried out on the FW/UW sample achieved an increase in grade up to 171 g Au/t with recoveries of up to 95%. Locked cycle tests would be required to confirm these results.

Direct CN leaching achieved high Au extractions (>95%) in all four samples evaluated (FW, UM, VM and FW/UM) at grinding sizes of 74 microns or less. Gravity table tests achieved from 45% to 74% Au recovery in the concentrate in the VM and UM/FW samples.

CIL tests were carried out on FW/UM and VM and Au extracted into the carbon was 97%.

Based on previous testing, EHA selected conditions for the CIP modelling testwork. CIP modelling tests were then carried out on the FW/UM sample using the following selected cyanidation criteria: grind size of P80 of 53 microns, pulp density of 45%, 0.5 g/l NaCN and 48 hour leach retention time. Gold recoveries after 5, 24 and 48 hours were 86.1, 94.8 and 95.7% respectively.

Based on the simulation studies, available Sag Mills ranging in size of 24' x 8', 24' x 7', 20' x 11' SAG mills were found to be suitable for the Timmins gold circuit with a high recirculating load and small classifier slot. The SAG ball charge ranged from 8 to 11% and the gross SAG power requirement varied from 12.8 to 14.0 kWh/t.

The rheology testing identified a decreasing apparent viscosity or "shear thinning" of the underflow pulp. Shear thinning is characteristic of the pseudoplastic class of non-Newtonian fluids. It demonstrates the need to achieve and maintain a specific velocity gradient or shear rate in order to initiate and maintain flow. Testing also identified that the potential to yield underflow pulps with yield values in excess of 30 N/m^2. This identified that the underflow pulps from the thickeners need to remain under 60% solids to minimize pumping problems.

SO_2 CN destruction procedures were provided by EHA. One acid base accounting ("ABA") test was carried out on a UM/FW tailings sample after SO_2 CN destruction and showed that the tailings sample was not an acid producer.

Results of the RPC work indicate that conventional processing of "ore" from the TM Deposit would achieve recoveries greater than 95% with no deleterious processing effects related to the ore.

16.0 MINERAL RESOURCES AND MINERAL RESERVE ESTIMATES (ITEM 19)

16.1 General

LSG has prepared an updated Mineral Resource estimate for the Timmins Mine, building on the 2006 estimate prepared by LSG and audited by WGM in January 2007, by incorporating the results of diamond drilling carried out since that time. Drilling since the previous estimate focussed on delineating and validating Mineral Resources above the 120m Level. All drilling in this area was completed from underground from a ramp and level system designed to access this mineralization for development and validation. A total of 300 holes were completed from underground, of which, 86 have been used in updating the Mineral Resource above the 120m Level.

The current Mineral Resource estimate is based on a block model prepared by LSG for those zones where a 3-D model could be generated above the 120m Level, and a traditional polygonal Mineral Resource model prepared by LSG and audited by WGM, for those zones below the 120m Level. The latter is described in SRK's October 2007 Technical Report ('2007 Estimate'), and is illustrated in Figure 6. Those resource blocks above the 120m Level which were previously included in the 2007 polygonal resource estimate have been replaced by the updated block model tonnes and grade from the current estimation. The block model Mineral Resources of those zones above the 120m Level represents only a small portion of the total Mineral Resource representing approximately 2% of the total tonnes and ounces. Since the estimation of the bulk of the Mineral Resource is covered by the 2007 Estimate, relevant sections from this report have been reproduced here to provide a more comprehensive overview of the previous resource estimate. Readers are referred to the technical report for additional comments and suggestions that were provided by WGM in auditing of the LSG Mineral Resources. The current block model Mineral Resource estimate has not been audited by an independent consultant.

Three main mineralized areas have been defined from this work and are consistent with the previous interpretation. These three zones are collectively grouped into the Vein Zones comprised of Vein 1, Vein 2 and Main Zone. The Main Zone, for the purpose of Mineral Resource estimations, has been further subdivided into five zones which have been designated MZ1, MZ2, MZ2a, MZ3 and MZ4. Development and drilling have shown that the upper zones can be traced as geological zones for about 100 metres on individual levels with potentially economic mineralization occurring over a more restricted distance.

Mineral Resources below the 120m Level (9880 elevation) have not been adjusted from the 2007 Estimate, as insufficient new drilling has been completed to warrant a re-estimation. This lower portion of the Mineral Resource is comprised of the continuation of the Vein zones to depth, as well as the Footwall and Ultramafic Zones, which will be tested from a validation and bulk sampling program off the 650m Level (9365 elevation).

Mineralization has now been defined within a block approximately 760 m long by 300 m wide and to a depth of 1,300 m. The mineralization envelope forms an elongate or pipe-like shape with a relatively well-defined moderate to steep plunge to the west. LSG has defined three zones of mineralization, within which occur several sub-zones, based on style of mineralization, mineralogy, and host rock (Table 16-1 and Figure 6).

Table 16-1: Summary of Mineralization Types at the TM Deposit

Zone	Sub-zones	Host Rock	Description
Vein	Main (MZ1, MZ2, MZ2a, MZ3, MZ4), Vein 1 (V1), Vein 2 (V2), Vein 3 (V3), Deep Zone (DZ)	Sedimentary rocks, sedimentary-volcanic contact	Tourmaline-quartz veins, py+aspy up to 2%, common coarse gold
FW (Footwall)	Footwall (FW) and Parallel Footwall (2FW), Upper Footwall (UFW), Contact Zone (CT)	Mafic volcanic rocks	10-20% quartz+/-tourmaline veins in strongly sheared and carbonate and albite altered host rocks, up to 10% py, generally finer gold not commonly visible
UM (Ultramafic)	Upper (UMU), north (NUM), south (SUM)	Ultramafic	Quartz+tourmaline veins, up to 10% py in halo to veins, gold is variable fine to relatively coarse

The Indicated and Inferred Mineral Resources for the TM Deposit as at August 28, 2009, as estimated by LSG, are documented in Table 16-2.

Table 16-2: Timmins Mine Deposit Mineral Resource Estimates

(Prepared by LSG – August 2009)

Resource Classification and Zone Name	Grade cut to 3.00 (g/t Au)		Contained	Uncut Grade	Top Cut Grade
	Tonnes	g/t Au	Gold (ounces)	(g/t Au)	(g/t Au)
Indicated Mineral Resources					
Vein Zone	323,000	9.42	97,900	17.48	50
Footwall Zone	1,185,000	7.27	277,100	7.57	30
Ultramafic Zone	1,737,000	9.27	517,600	14.46	50
Total Indicated	**3,245,000**	**8.56**	**892,700**	**12.24**	
*Inferred Mineral Resources					
Vein Zone	470,000	5.92	89,300	7.76	50
Footwall Zone	340,000	5.95	65,000	6.27	30
Ultramafic Zone	85,000	3.89	10,600	3.89	50
Total Inferred	**894,000**	**5.74**	**164,900**	**6.83**	

* Note: Inferred Mineral Resources are reported in addition to Indicated Mineral Resources. All tonnage and total contained ounces have been rounded and may not add up due to rounding.

LSG has classified the Mineral Resources according to the definitions of National Instrument 43-101 and the CIM Standards. For the purposes of this report, the relevant definitions for the CIM Standards are contained in the Glossary at the end of this report.

16.2 Estimation Method

16.2.1 Estimation Method and Parameters

LSG has completed both a cross sectional polygonal estimate for the lower portion of the mine (below the 120m Elevation) and a block model method for the upper part of the mine (above the 120m Elevation) to estimate the Mineral Resources at the Timmins Mine. The bulk of the Mineral Resource remains estimated by the polygonal method, which represents approximately 98% of the total tonnes and 98% of the total ounces. The method and parameters for the polygon estimate are summarized below, followed by those for the block model estimate.

The general procedures for the polygonal Mineral Resource estimate include:

1) Determination of intersections based on established cut-off grade and minimum width criteria;

2) Creation of polygons on section limited by the calculated intersections, drawn consistent with geological interpretation of continuity, and limited to one-half the distance to the adjacent drill holes on the section;

3) Calculation of polygon volume where volume is calculated as the product of the polygon area and the polygon thickness and where the polygon thickness is determined as the sum of one-half of the distance to each of the adjacent sections,

4) Estimation of tonnage; which is the product of the polygon volume and density; and

5) Tabulation of tonnage and weighted average grade. Examples of sections through the deposit area are illustrated on Figures 7 to 10.

LSG has used a cut-off grade of 3.00 g/t Au and a minimum intersection width of 1.5 m as criteria for intersection definition. The cut-off grade chosen is based on a preliminary compilation and review of parameters impacting the economic viability of a potential mining operation at Timmins Mine and includes mine dilution, metallurgical recoveries, operating costs, and gold price. A gold price of US$500/oz has been assumed by LSG. Identical parameters were used for the block model estimate which follows.

The general procedures for the block model Mineral Resource estimate include:

- Database compilation and verification in Gemcom GEMS (“GEMS”).
- Interpretation of the mineralization on 12.5m wide sections with a minimum intersection width of 1.5m. This width is consistent with the polygon estimate.
- Construction of closed polylines “snapped to” drill hole assays based on limits of similar zones and tagged by rock code. Each polyline was assigned an appropriate rock type and stored with its section definition in the GEMS polyline workspace.
- Zones are defined by more than one intersection that forms a continuous band of mineralization.
- The sectional interpretations are then strung together by tie lines and 3-D solids or wireframes are generated that represent the mineralized zones that are used for estimation of tonnes and grade. Outside edges of the 3-D model are extruded half the distance to the next section, or 6.25m. In total, seven 3-D solids were constructed to enable individual volumes, tonnages and grades to be reported for

each individual zone. A three dimensional view of the interpreted zones are shown in Figure 12.

- Solid intersection composites are generated from all drill holes intersecting the 3-D Mineral Resource solids. Corresponding entry and exit points are saved to the drill hole workspace and back coded with a zone identifier.

- Individual 1m composites are generated from the assay table based on down-the-hole averaging within the limits of the solid intersection composites.

- The 1m composites are then used to generate a block model grade based on an Inverse Distance Squared ("ID²") interpolation that encompasses the 3-D wireframes that were assigned a unique rock code. Indicated Mineral Resources are based on a 20m search radius, with individual blocks requiring values from at least two drill holes and not more than two composites from the same hole. Inferred Mineral Resources are based on a 40m search radius, with similar individual block requirements.

- 3-D Excavation solids are constructed to reflect the development completed on the individual zones.

- The Mineral Resources are estimated for the individual zones and tabulated on the basis of Indicated or Inferred and whether inside or outside the excavation solid. The tonnes and grade of the excavation solid was removed from the Mineral Resource estimate.

16.2.2 Database

As mentioned previously in Section 10.4 of this report, the database is comprised of a Gemcom GEMS (Microsoft Access) database which was compiled from the mine's Amine Corelog SQL database and ASCII files provided by the exploration group. The GEMS database was used for the Mineral Resource estimation process and consisted of header, survey lithology and assay data and is summarized in Table 16-3.

Table 16-3: Summary of GEMS Access Database Used by LSG

Table Name	Table Description	Fields
Header	Drill hole collar location data in local grid co-ordinates	Hole-ID Location X Location Y Location Z Length Collar_Azimuth Collar Dip
Survey	Down hole survey data of direction measurements at down hole distances	Hole-ID Distance Azimuth Dip
Assays	Sample interval assay data with Au units grams per tonne	Hole-ID From To Au_GPT_FIN Au_GPT_AA Au_GPT_GRA Au_GPT PM
Lithomaj	Major logged rock type intervals down hole	Hole-ID From To Rocktype
Lithomin	Minor logged rock type intervals down hole	Hole-ID From To Rocktype

The following validation steps were taken to insure the integrity of the database:

1) Plotting of plans and sections to check for location, elevation and downhole survey errors.

2) Checking for any gaps, overlaps and out of sequence intervals for assay and lithology data.

3) Checking for any data beyond the end of hole.

4) Verifying both random and significant assay values against drill logs and assay certificates.

Only minor discrepancies were noted, which would have no significant impact on the Mineral Resource estimate. These discrepancies were corrected in the database prior to the estimation of the resources.

LSG is currently in the process of centralizing it's mine and exploration database within GEMS SQL. This should eliminate the requirement to combine databases for future estimations.

In addition to the drill hole data, other data such as cross-sectional geological interpretation polylines, section and level plan definitions, 3-D geological and excavation solids, point area data of assays and composites, as well as the block model, are stored within the GEMS database.

16.2.3 Mineral Resource Intercepts

LSG has calculated length weighted average grades for the mineralized intersections used for the grade estimation process. Where intersections are less than the determined minimum length of 1.5 m, the intersections were extended to that minimum length, over partial sample intervals, through weighted averaging with zero grade for the additional interval used.

The approach to compositing (intersection calculation) utilized by LSG is consistent with the polygonal estimation procedure and the results are consistent with the input database. Alternative compositing methods could be used in the future that are related to different approaches and methods of Mineral Resource estimation at the Timmins Mine Project.

16.2.4 Grade Capping

Grade capping, also sometimes referred to as top cutting, assay grades is commonly used in the Mineral Resource estimation process to limit the effect (risk) associated with extremely high assay values. Philosophies or approaches to establishing and using grade capping are variable across the industry and include not using grade caps at all, arbitrarily setting all assay grades greater than 1 oz/ton to 1 oz/ton, choosing the grade cap value to correspond to the 95th percentile in a cumulative distribution, or evaluation of the shape and values of histograms and/or probability plots to identify an outlier population. LSG has utilized grade capping in its estimation of the Timmins Mine Project by setting the upper value to be used for estimation of the UM Zone and Vein Zone at 50 g/t Au and the upper value for the FW Zone at 30 g/t Au for both the polygonal and block model Mineral Resource. For further details refer to the 2007 Technical Report.

16.2.5 Block Model Assay Compositing and Statistical Analysis

Each 3-D solid was assigned a unique numeric rock code which was used to back code a rock code into all drill hole solid intersections. This solid intersection table was used to generate a set of equal length composites of 1m length within the limits of the 3-D solid. A raw set of assay values was also generated from within the 3-D solids. Both the 1m composites and the raw gold assay values were stored in a GEMS point area workspace and basic statistics performed on this data.

Table 16-4 shows the basic statistics of the original (uncomposited) samples within the limits of each 3-D solid used for the Mineral Resource estimate, above the 120m Elevation.

Table 16-4: Basic Statistics of Raw Au Assays

Zone	# Samples	Minimum (gpt Au)	Maximum (gpt Au)	Mean (gpt Au)	95^{th} Percentile	Coefficient of Variation
Vein 1	131	0.00	97.50	7.37	57.74	2.71
Vein 2	39	0.00	334.18	18.14	53.35	2.94
Main Zone 1	54	0.01	58.61	6.94	33.80	1.68
Main Zone 2,2a	103	0.00	70.46	3.74	13.60	2.21
Main Zone 3	63	0.00	49.77	2.88	12.49	2.35
Main Zone 4	49	0.00	22.60	2.94	16.62	1.69
All Zones	439	0.00	334.18	6.28	30.05	3.30

A review of the samples used for the block model estimate for the vein and main zones showed appropriate cut- off values varying between 13.6 to 57.74 gpt Au, based on the 95th percentile of the cumulative distribution. The limited number of samples does not warrant the adjustment of the original cut-off value of 50 gpt Au used for the polygonal resource estimation. The 50 gpt Au cut-off would represent the 98th percentile within the cumulative frequency distribution. Individual assay values were cut to 50 gpt Au and a separate Au50 field was used for calculating cut composite values for Mineral Resource estimation.

A total of 439 assay samples were utilized in the block model resource estimate above the 120m Elevation and of these samples, 13 were found to be above the 50 gpt Au cut-off. Similar to the 2007 polygonal estimation, the net indicated contained Au estimated utilizing the top cut is approximately 30% less than in the estimate with uncut assay values for the total Mineral Resource and approximately 12% for the Mineral Resource above the 120m Elevation.

The number of drill holes that have been used to define the block model resource above the 120m Level are shown in Table 16-5. The holes have been divided into surface and underground on the basis of where the hole was collared. Surface data relates to drilling included in the 2007 Estimate, while underground data refers to recent validation drilling that was completed after the 2007 Estimate. A total of 86 new underground holes were used to update the resources above the 120m Elevation.

Table 16-5: Drilling Per Resource Zone

Zone	Surface DDH	Underground DDH	Total DDH
Vein 1	12	31	43
Vein 2	4	5	9
Main Zone 1	10	7	17
Main Zone 2,2a	12	21	33
Main Zone 3	9	15	24
Main Zone 4	10	7	17
All Zones	57	86	143

Basic statistics were compiled on the basis of the three main mineralized zones consisting of 118 composites for Vein 1, 28 composites for Vein 2 and 253 composites for the combined Main Zone. Table 16-6 summarizes the statistics of the uncapped composites inside each zone.

Table 16-6: Basic Statistics of 1m Composite Gold Values

Zone	# Samples	Minimum (gpt Au)	Maximum (gpt Au)	Mean (gpt Au)	95^{th} Percentile	Coefficient of Variation
Vein 1	118	0.00	92.76	5.03	24.53	2.66
Vein 2	28	0.00	86.76	13.96	82.81	1.64
Main Zone	253	0.00	70.46	4.19	15.96	2.06
All Zones	399	0.00	92.76	5.13	21.09	2.33

The 50 gpt Au cut that was used for the Mineral Resource estimate corresponds to roughly the 98th percentile in a cumulative frequency distribution (Figure 13). Further work to establish proper capping levels should be undertaken once sufficient sampling of individual zones is available and mining history of the zones has progressed so that reconciliation of data between predicted and actual grades can be assessed.

16.3 Specific Gravity

Specific gravity ("SG") was determined on selected samples from the Deposit from each of the mineralized zones in conjunction with metallurgical testing at two testing facilities. LSG therefore reports a total of six SG values. For each of the three-mineralization types, LSG has used an average of the two SG determinations (Table 16-7).

Table 16-7: Specific Gravity Determination for the TM Zones

Specific Gravity Determination for the TM Zones			
	SGS Lakefield	**RPC**	**Average**
Vein Zones	2.79 and 2.80	2.84	2.81
Footwall Zones	2.87	2.89	2.88
Ultramafic Zones	2.99	2.85	2.92

The values are consistent with style of mineralization and mineralogy, and the variance between zones is consistent with reported sulphide abundance and host rock type. The SG determinations are from bulk samples and therefore represent the average of several intersections, and as such, are reasonable representations of the zones and according to WGM's 2007 audit, are considered to be valid for the purposes of a Mineral Resource estimate. However, possible significant variance exists, especially with the UM Zone determinations, and WGM recommended that LSG complete a number of additional SG determinations on core samples with the objective of more completely characterizing the possible range of SGs within the zones. An attempt should be made to correlate these SGs with mineralogy to potentially establish sub-types of mineralization that may be used in future Mineral Resource estimates. Insufficient new data is available for the UM Zone to be able to update sub-types of this mineralization or update the overall specific gravity of the zone.

The block model resource estimate used a default density value of 2.81 for all of the Vein Zones. The consistent nature of the mineralization, with limited sulphides, supports the use of a single density value.

16.4 Polygonal Mineral Resource Modelling

For Mineral Resources below the 120m Elevation, LSG interpreted polygons on cross sections, drawn as approximately rectangular polygons consistent with the interpreted geological dip of the mineralization. A polygon was interpreted for each intersection and was extended halfway to adjacent holes on section. Reasonable limits were used, typically half the drill spacing, where there was no constraining drill hole (e.g., deepest hole on a section). As part of WGM's 2007 Mineral Resource audit, a section-by-section inspection was completed of the polygons and drill hole intercepts to ensure that polygon widths were consistent with intersection lengths. Polygon construction was generally orderly with only minor gaps and/or overlaps identified.

LSG calculated polygon volume in a spreadsheet utilizing polygon width, dip length (half the distance to adjacent holes scaled on section), and strike length (distance between cross sections). Polygon width (true width) was calculated trigonometrically utilizing the drill hole orientation and the interpreted dip orientation of the mineralization. The dip length was manually measured from the polygon on section and the strike length was the sum of the ½ distances to the adjacent sections.

LSG estimated the polygon tonnage as the multiplication of the volume (as determined above) and density. The volume of a number of polygons was calculated by WGM utilizing GEMS, the sectional polygon outlines (strings) supplied by LSG, and the section definitions and limits defined by LSG. WGM determined tonnage through the multiplication of the calculated volumes with the appropriate zone density for five cross sections. The results were audited by WGM and indicated an acceptable agreement between the two methods of estimation. Please refer to Section 16.2.6 of the 2007 Technical Report for more detailed explanation and for polygon tonnage comparison plots.

16.5 Block Model Mineral Resource Modelling

16.5.1 General

The grade of the Mineral Resources above the 120m Level was estimated by using the ID² interpolation method. This method interpolates the grade of a block from several composites within a defined distance range from the block. The estimation uses the inverse of the distance between a composite and the block as the weighting factor to determine grade.

16.5.2 Block Model Grid Parameters

The Mineral Resources have been estimated using a single grid of regular blocks. A check of all 3-D solids used for the resource estimate was conducted to insure that no blocks would report to more than one solid. A single resource folder was defined in GEMS and all of the 3-D solids for the Mineral Resource estimate were contained in this folder. A summary of the block model grid parameters are shown in Table 16-8.

Table 16-8: Block Model Grid Parameters

Model Origin	Grid	Model Dimension		Block Dimension	
X	4800 E	Rows	300	Row width	1.0 m
Y	7700 N	Columns	250	Column width	1.0 m
Z	10000 el	Levels	200	Level height	1.0 m
		Orientation	No rotation		

16.5.3 Grade Interpolation

The geology and overall drill hole spacing were used to determine the appropriate search ellipse for the various Mineral Resource categories. The variography of the zones was checked, but the limited number of samples did not allow for any further refinements to the search ellipse. A single search ellipse was determined to adequately model the various zones, as the strike and dip of the zones remained relatively constant. The search parameters and criteria for grade interpolation are summarized in Table 16-9.

Table 16-9: Search Ellipse Parameters

	Indicated Mineral Resources	Inferred Mineral Resources
Search Ellipsoid Dimension	20 m X, 20 m Y, 10m Z	40 m X, 40 m Y, 20m Z
Search Ellipsoid Rotation	ZYZ: 5°, -65°, 0°	ZYZ: 5°, -65°, 0°
Min # samples to estimate block grade	3	3
Max # samples to estimate block grade	5	5
Max # samples from a single hole	2	2
Min # of DDHs to estimate block grade	2	2

One metre cubes formed the individual blocks within the block model of the upper vein Mineral Resources. The volumes and tonnes are calculated by GEMS using the blocks falling within the limits of the 3-D wireframes, with partial blocks being assigned volume and tonnage according to the percentage of the block within each 3-D solid. These blocks are then accumulated and summarized in a table in Excel format. Only those blocks above the lower cut-off of 3.0 gpt Au are included in the Mineral Resource estimate.

16.6 Mineral Resource Classification, Conclusion and General Comments

16.6.1 Polygonal Mineral Resource Below 120m Elevation

The basis of the Mineral Resource classification (Indicated and Inferred) for the TM Deposit below the 120m Elevation is interpreted geological and grade continuity. Where two or more polygons are interpreted to be continuous on cross section (dip), across section (strike), or a combination of both (plunge), these blocks have been classified by LSG as Indicated. Those polygons which do not have interpreted continuity or are isolated polygons due to lower drill density have been classified by LSG as Inferred. The Indicated blocks form a relatively consistent shape with a well-defined moderate to steep plunge to the west. This orientation is consistent with the orientation of regional scale structures and smaller scale parasitic (?) fold axis plunge.

In the 2007 Technical Report, WGM concluded that the economic parameters used by LSG to define potentially economic intersections were reasonable for this stage of a Mineral Resource estimate. The estimation method and parameters have been correctly and consistently applied. The estimation results, i.e., tonnage, grade, and contained gold, are consistent with the input data. The criteria used for the classification of the Mineral Resources as Indicated and Inferred is consistent with NI 43-101 and the CIM Standards, and the resulting Indicated and Inferred tonnages are consistent with the defined Mineral Resource blocks.

16.6.2 Block Model Mineral Resource Above 120m Elevation

LSG has classified the Timmins Mine Resource estimate above the 120m Elevation as Indicated and Inferred Mineral Resources. The resources show sensitivity to cut-off grade and are summarized in Table 16-10. The higher grade Vein 1 and 2 shows less sensitivity to cut-off while the lower grade Main Zone is more sensitive.

The 3.0 gpt lower cut-off was used for the resource estimation based on cost parameters and consistency with previous reporting of the zone. At higher cut-off values, the zones tend to become more broken up and disjointed, which would make mining of the zones at the resource grade difficult to achieve.

Two interpolation passes were used to establish grade and Mineral Resource categories. The Indicated Mineral Resources used a smaller search ellipse (20 m along strike, 20 m down dip and 10 m in height) and a larger ellipse size (40 m along strike, 40 m down dip and 20 m in height) was used to define Inferred Mineral Resources. A minimum of three composite samples were required for interpolation, with no more than two from a single drill hole. To establish geological continuity, a minimum of two drill holes were required for grade interpolation. Due to the relatively dense drilling completed in the area above the 120 m Elevation, the bulk of the Mineral Resources fall within the Indicated category.

Table 16-10: Mineral Resource above 120m Level Showing Sensitivity to Cut-off Grade

Cut-off Grade (gpt Au)	Indicated Mineral Resource			Inferred Mineral Resource		
	Tonnes *	Grade (gpt Au)	Au (oz) *	Tonnes *	Grade (gpt Au)	Au (oz) *
Vein 1						
1.0	11,200	7.11	2,600	3,700	6.93	800
2.0	8,800	8.67	2,500	3,400	7.41	800
3.0	**7,800**	**9.45**	**2,400**	**3,100**	**7.84**	**800**
4.0	7,100	10.06	2,300	2,900	8.19	800
5.0	6,500	10.62	2,200	2,700	8.51	700
Vein 2						
1.0	4,600	11.00	1,600	200	9.70	100
2.0	4,200	12.04	1,600	200	9.70	100
3.0	**3,700**	**13.14**	**1,600**	**200**	**9.70**	**100**
4.0	3,400	14.02	1,500	200	9.70	100
5.0	3,100	14.99	1,500	100	16.77	100
Main Zone						
1.0	40,000	4.82	6,200	4,600	5.91	900
2.0	31,000	5.78	5,800	4,300	6.14	900
3.0	**22,700**	**6.99**	**5,100**	**3,100**	**7.66**	**800**
4.0	17,200	8.13	4,500	2,900	7.96	700
5.0	13,800	9.04	4,000	2,600	8.29	700

* All tonnage and Au (oz) figures rounded to the nearest hundred.

To verify the block interpolation parameters, the drill hole composite intervals were visually compared with block grades on both vertical and longitudinal sections. The block grade interpolations of the various zones are shown in Figure 14 and individual longitudinal of the various zones are shown in Figures 15 to 20. Zone 2a, a subset of Zone 2, due to its limited size, has not been reproduced in this report.

A summary of the zones that comprise the Indicated and Inferred Mineral Resources are tabulated in Table 16-11 and Table 16-12. Both the upper cut (50 gpt Au) and the uncut gold values are shown. This summary is based on a lower cut-off grade of 3.0 gpt Au. The tonnes and grade of development work that has been extracted is also shown, in order to compare the 2007 polygonal resource estimation for this area. Development tonnes and grade were estimated using 3D wireframes and block model data in GEMS, and not actual production data.

Table 16-11: Indicated Mineral Resources above 120m Elevation at 3.0 gpt Cut-off

Zone	Indicated Mineral Resource				
	Tonnes *	**Grade (gpt Au)**	**Au (oz) ***	**Uncut (gpt Au)**	**Uncut Au (oz) ***
Vein 1	7,800	9.45	2,400	12.49	3,100
Vein 2	3,700	13.14	1,600	15.87	1,900
MZ	22,700	6.99	5,100	7.23	5,300
Subtotal	**34,300**	**8.22**	**9,100**	**9.37**	**10,300**
Development	4,700	8.52	1,300	8.93	1,300
Total	**38,900**	**8.26**	**10,300**	**9.32**	**11,700**

* All tonnage and Au (oz) figures rounded to the nearest hundred. Totals may not add up due to rounding.

Table 16-12: Inferred Mineral Resources above 120m Elevation at 3.0 gpt Cut-off

Zone	Inferred Mineral Resource				
	Tonnes *	**Grade (gpt Au)**	**Au (oz) ***	**Uncut (gpt Au)**	**Uncut Au (oz) ***
Vein 1	3,100	7.84	800	7.84	800
Vein 2	200	9.70	100	10.95	100
MZ	3,100	7.66	800	7.97	800
Total	**6,400**	**7.82**	**1,600**	**8.01**	**1,700**

* All tonnage and Au (oz) figures rounded to the nearest hundred. Totals may not add up due to rounding.

A comparison of the 2007 polygonal Indicated Mineral Resource to the current block model Indicated Mineral Resource is summarized in Table 16-13. The net effect is a reduction of 18,200 tonnes, a decrease in grade of 3.50 gpt Au and a reduction of ounces by 11,300 ounces to the Indicated Mineral Resource. The more detailed underground drilling confirmed the general location of the polygonal resource, although the drilling tended to limit its extent and to reduce the effect of individual high grade assays, which are projected out the entire extent of the polygon blocks. . The current estimation resulted in a reduction of tonnes by approximately 32% and in ounces by approximately 52%. Further work would be required to determine if this is a local effect or to what extent this could be applied to the remaining Mineral Resources within the Vein Zones. The area above the 120m Elevation represented approximately 16% of the Indicated Vein Zone resources.

The impact of a similar reduction to the remaining resources in the Vein Zone would result in a decrease of 92,000 tonnes and 46,300 ounces to the Indicated Mineral Resources.

Mineralization within the Footwall and Ultramafic Zones demonstrates better grade continuity, due to the nature of the mineralization and it would not be expected to see similar reductions upon more detailed drilling. Additional drilling and bulk sampling of the zones is required to determine if this is indeed the case.

Achieving the predicted tonnes and grade will require selective extraction from within the current geological models over narrower widths than the current mining. A 3D view of the 3 gpt Au isoshells generated by GEMS within the limits of the geological model are illustrated in Figure 21.

A comparison of the 2007 polygonal Inferred Mineral Resource to the current block model Inferred Mineral Resource is summarized in Table 16-13. The net effect is a reduction of 73,000 tonnes, an increase in grade by 3.39 gpt Au and a decrease in ounces of 9,700 to the Inferred Mineral Resource. Whereas single drill hole intersections were modeled and included in the polygonal estimate, a minimum of three composites from two different drill holes were required for consideration in the current block model grade interpolation. A number of isolated single intersection polygon intersections which were not included in the current Mineral Resource may provide exploration drill targets that could be brought into future Mineral Resources with additional drilling. The area above the 120m Elevation represented approximately 15% of the Inferred Vein Zone Resources.

The net effect to the Inferred Mineral Resources above the 120m Elevation is a reduction in tonnes by approximately 92% and in ounces by approximately 86%. The impact of a similar reduction, if applied to the remaining Inferred Vein Zone Resource, would result in a reduction of tonnes by 426,000 and in ounces by 75,200. Additional drilling and modeling would be required to determine the extent of the reduction to these Inferred Mineral Resources.

Table 16-13: Comparison of Polygon and Block Model Mineral Resources Above 120m Elevation

Resource	Indicated Mineral Resource (3gpt Au lower cut)				
	Tonnes *	Grade (gpt Au)	Au (oz) *	Uncut (gpt Au)	Uncut Au (oz) *
Block Model	38,900	8.26	10,300	9.32	11,700
Polygon	57,200	11.76	21,600	13.35	24,500
Net difference	**-18,200**	**-3.50**	**-11,300**	**-4.03**	**-12,900**

Resource	Inferred Mineral Resource (3gpt Au lower cut)				
	Tonnes *	Grade (gpt Au)	Au (oz) *	Uncut (gpt Au)	Uncut Au (oz) *
Block Model	6,400	7.82	1,600	8.01	1,700
Polygon	79,400	4.43	11,300	4.43	11,300
Net difference	**-73,000**	**3.39**	**-9,700**	**3.58**	**-9,700**

* All tonnage and Au (oz) figures rounded to the nearest hundred. Net difference may not add up due to rounding.

A further validation of the block model Mineral Resource grade was undertaken to compare the interpolated grade versus the chip and muck sample grades obtained from development work on selected zones and levels. The chip sample data is currently tabulated in a master production tracking sheet in Excel format, on a round by round basis and illustrated in AutoCAD as a series of level plans. The current system does not readily allow for inclusion of these samples into Mineral Resource modeling and it is suggested that LSG look into a methodology to utilize these samples to augment the drill hole assay information or at least provide another measure to assess the validity of the drill hole resource model.

Three zones from various levels were used as test cases to reconcile block model grade versus chip and muck sample data. The selected zones were V2 on the 50 m Level, Main Zone 1 on the 80 m Level and Main Zone 2 on the 110 m Level. The three levels are shown in Figures 22 to 24 and illustrate the extent and grade of the resource block model relative to the chip and muck sample grades from the same area. Good correlation exists between the strike extent of where the model predicted grade and the actual limits determined from development on the zones. A summary of the results from these three zones are summarized in Table 16-14. The limited sample size due to the relatively short strike lengths of the zones tends to over exaggerate differences between the estimations. To estimate the Mineral Resource grade the geological model had to be expanded out to the development width with dilution at zero grade. This dilution amounted to an average of approximately 27% on the three levels.

Table 16-14: Comparison of Chip and Muck Grades Versus Resource Grades

Level	Zone	Tonnes	Chips (gpt Au)	Mucks (gpt Au)	Model (gpt Au)
50	Vein 2	1,582	1.64	na	6.23
80	Main Zone 1	1,063	4.25	3.87	5.57
110	Main Zone 2	760	9.11	3.40	3.73
Totals		**3,405**	**4.12**	**3.67**	**5.47**

The totals tend to reduce local variation and reasonable correlation is evident between the chip, muck and model grades. Additional work is required to establish trends between the various sample types and to reconcile these values to mill production.

The limited processing of the Vein material and the mixed nature of the mineralization sent to the mill, did not allow for any direct reconciliation of extracted material versus predicted Mineral Resource grade. LSG should establish tracking mechanisms, so that a proper reconciliation of resource grade versus extracted grade versus mill production grade can be assessed. This will also aid in setting more definitive capping levels for future Mineral Resource estimates.

LSG concludes that the economic parameters used to define potentially economic intersections are reasonable for this stage of a Mineral Resource estimate. The estimation method and parameters have been correctly and consistently applied. The estimation results, i.e., tonnage, grade, and contained gold, are consistent with the input data. The criteria used for the classification of the Mineral Resources as Indicated and Inferred is consistent with NI 43-101 and the CIM Standards, and the resulting Indicated and Inferred tonnages are consistent with the defined Mineral Resource blocks.

The TM Project Mineral Resources are unlikely to be materially affected by any presently known environmental, permitting, legal, title, taxation, socio-economic, marketing or political issues.

The extent to which the TM Deposit Mineral Resources may be materially affected by mining, metallurgical and infrastructure factors has been adequately defined elsewhere in this report.

16.6.3 Mineral Resource Sensitivity

The results of the sensitivity analysis indicate that the Project is most sensitive to grade and gold price. Further changes in gold price could trigger a change in cut-off grade and in that case, the Mineral Resources should be re-estimated.

16.7 Mineral Reserve Estimation

The mineral reserve for the TM deposit were estimated with an effective date of August 28, 2009 based on the mineral resources estimate generated by LSG (WGM audited the polygonal resource model). For the purpose of mineral reserves estimation, the qualified person is George B. Darling, P. Eng.

The mineral reserves estimate was based on the following parameters developed by the QP as part of the mine plan:

- Mining recoveries from 86 to 92%;
- Dilution of 12% for cut and fill vein mining and 27% for long-hole stoping, using dilution grades of 0 g/t Au, except for the Ultramafic Zone (1 g/t Au diluted grade);
- Minimum mining width of 2 (cut & fill) to 3.5 m (open stoping);
- Mill recovery: 95%;
- Total operating cost of $95.08/tonne.

The mineral reserves have been classified according to the definitions of National Instrument NI 43-101 and the CIM Standards. The definitions are contained in the Glossary at the end of this report.

16.8 Conversion of Mineral Resources to Mineral Reserves

Probable mineral reserves were estimated at 3,358,000 tonnes grading an average 7.52 g/t Au, containing 812,006 oz of gold (24.8Mg). Please refer to

Table 16-15.

Table 16-15: Mineral Reserve Estimate (cut grades):

Mineral Reserve	Tonnes	Gold g/t	Contained
Category			Gold (ounces)
Probable	3,359,760	7.52	812,006

The distribution of Mineral Reserves by mine level is shown in Table 16-16. The distribution of contained gold by mine level is shown in Table 16-17.

Table 16-16: Mineral Reserves Tonnage by Level

Level	Tonnage	%
200	202,565	6
400	234,554	6.9
600	653,432	19.3
800	1,000,202	29.5
1000	774,285	22.9
1200	521,553	15.4
1400	0	0
Total	3,385,760	100.0

Table 16-17: Mineral Reserves Contained Ounces by Level

Level	Mined Ounces CUT	%
200	42,500	5.23
400	53,122	6.54
600	168,456	20.75
800	229,352	28.25
1000	203,015	25.00
1200	115,561	14.23
1400	0	0
Total	812,006	100.00

Economic analysis and costs sections of this report are based on estimated mineral reserves only.

Modification of any of the economics parameters used may change the break-even cut-off grade and the mineral resources and mineral reserves estimate.

The qualified person does not anticipate any material changes to the mineral reserves estimate due to mining, metallurgical, infrastructure or other known factors based upon the polygonal resource as provided by LSG. The mineral reserves have established economic viability as demonstrated in Section 18 of this report.

17.0 OTHER RELEVANT DATA AND INFORMATION (ITEM 20)

To the best of the authors' knowledge, there is no other relevant data pertaining to the TM Project that is not covered in this report.

18.0 ADDITIONAL REQUIREMENTS FOR DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES (ITEM 25)

18.1 Mining Operations

The underground mineral reserves were based on the LSG's polygonal resource estimate audited by WGM. The Veins and Main Zone lenses located above 120 m Elevation were recently modeled using block modeling techniques. Both models were converted to a series of wireframes and modeled in Mine2-4D environment. Wireframes were created by joining lines together that were generated to define 2D mineralization outlines (on cross sections or level plans) and to produce a visual 3D view. Mineral reserves were validated by cross-referencing to the mineral resources estimate.

The mine plan utilizes the advanced exploration parameters (AEP) with the addition of ramp access from surface. It formed the basis for shaft access and the main capital development. The mining was based on a strategy of sinking the shaft to access the 650 Level and collecting a bulk sample. The 650 Level was deemed the optimum level by LSG where the Ultramafic and Footwall Zones can be accessed. The mining works completed during the AEP phase will provide the basis to later proceed with pre-production mining and also to extend the shaft to the 710 m Elevation.

The mine planning software Mine2-4D was used to create a three dimensional model of the underground mine workings. Wireframes of the zones that make up the deposit were provided by LSG. In the QP's opinion Mine2-4D software provides a good planning feature in that mine development shown in the 3D model is linked directly to the development schedule. Changes made in the 3D model are automatically used to update the schedule spreadsheet.

Figure 6 shows the general 3D view of the mine model.

During the AEP work the 525 Level has been partially developed in order to access the exploration drill platforms. The 650 Level will be open for basic infrastructure and to extract the bulk sample. This development will help reduce the duration of the pre-production period.

The production schedule is shown in Table 18-1.

Table 18-1: Production Schedule

ITEM	UNIT	2010	2011	2012	2013	2014	2015	2016	2017	Total
Working Days	d	355	355	355	355	355	355	355	125	
Production Rate	t/d	1,230	1,562	1,566	1,268	1,392	1,138	1,000	800	
	1300t/a	440	554	556	450	494	404	355	105	3,359
Grade in Waste	g/t	0.94	0.98	0.91	0.86	0.94	0.94	0.8	0.8	
Weighted Dilution Rate	%	16.56	22.11	16.94	21.46	17.82	18.30	18.03	16.95	
Diluted Grade	g/t	7.07	8.06	6.68	6.76	6.70	7.74	9.74	9.74	7.52
Contained Au	kg	3,107	4,470	3,714	3,044	3,312	3,125	3,457	1,025	25,256

Table notes:

1) The LOM averages for mined gold grade and external dilution are 7.52 g/t Au and 14.53% respectively.
2) Total scheduled production is 200,000 tonnes less than total Mineral Reserves because of the bulk sample during AEP.

The mine work force has been estimated to include the functions of development, drilling, blasting, ground support, production mucking and trucking, backfilling, maintenance, technical services, and supervision personnel. The mine is planned as owner operated.

Table 18-2 shows the planned mine work force at the TM Project. Labour productivities have been benchmarked against similar operations. The working shifts have been set at 10.5 hours with a 4 on 4 off shift rotation.

Since Timmins is a large mining camp, experienced miners are locally available and no provisions were made to build camp infrastructure. The bonus rates included in the operating costs are expected to attract local miners.

Table 18-2: Staffing Schedule Full Production

Position(s)	Pre-Production	Production
VP Operations	1	1
Director Technical Services	1	1
Director Human Resources	1	1
Controller	1	1
Administration Staff	5	5
Security Officer	1	1
Environmental Group	2	2
Information Support	1	1
MINE SITE:		
Mine Manager	1	1
Superintendents	3	3
Foremen	4	6
Mine Technical Staff	8	8
Administration Staff	3	3
Miners	59	77
Shaft Operation	12	12
Mine Maintenance	22	22
BC SITE:		
Site General Staff	1	1
Mill Staff	9	9
Mill Operators/Maintenance	24	24
Total Payroll	**157**	**175**

18.2 Mining Method

Timmins Mine is currently planned with shaft and ramp access from surface. The ramp access area is driven down to the 200 Level and is in the process of being connected with the shaft. The AEP shaft level intervals are at 200 Level (200 m below surface), 400 Level, 525 Level and 650 Level with the shaft bottom being at the 710 m Elevation. Once commercial production decision is approved, all of these levels will be connected by ramps. This will allow the opportunity to use ramp access for sublevels and cut and fills stoping.

The mining of the ore bodies below shaft bottom will be carried out by ramp and truck haulage.

The two main mining methods selected by the QP are open stoping and mechanized cut and fill. Shrinkage stoping is also an option but a more conservative approach was used in this report. Open stoping was selected where the deposit geometry appeared to be regular between 25 m sublevels. Cut and fill is more selective and it is planned for areas that are more irregular and thinner. It is commonly used for this type of deposit in the Timmins area. During the AEP bulk sample via the ramp shrinkage mining has taken place and this strategy may continue throughout the deposit. All of these methods will use rockfill.

The open stoping (longhole sublevel and uppers mining) will utilize a blast hole diameter of less than 100 mm to help control wall dilution from blast damage.

The pie graphs in Chart 18-1 and Chart 18-2 show the relative distribution of gold produced and tonnes mined by mining method.

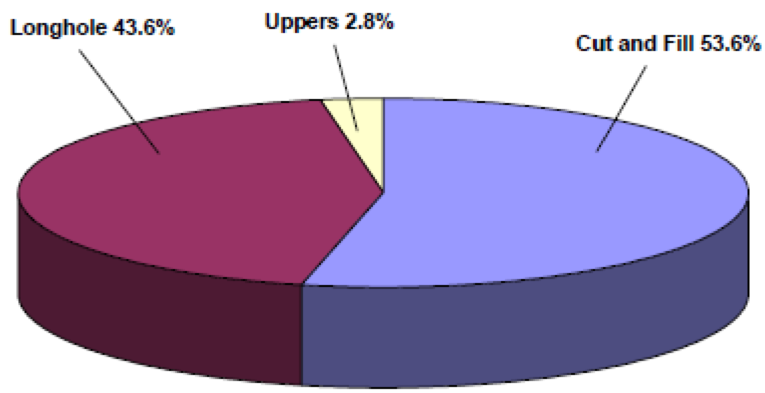


Chart 18-1: Mined Au Percent by Mining Method

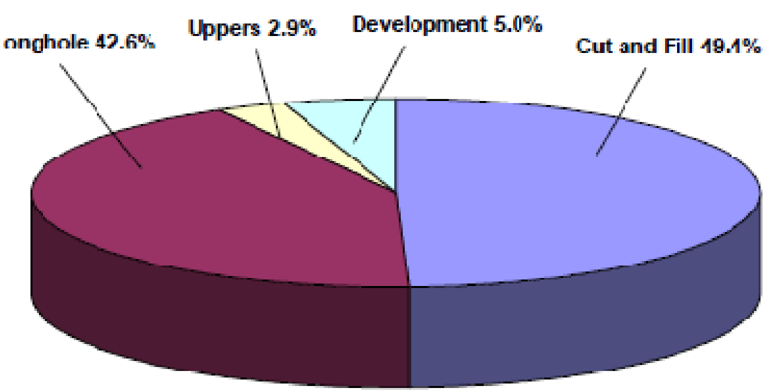


Chart 18-2: Mined Tonnage Percent by Mining Method

Ideally, the use of jackleg mining would be of benefit in certain veins but the availability of miners with jackleg mining skills may be limited.

Thirty percent of the waste rock generated by underground development will be placed back in the mined out stopes to avoid hoisting or trucking it to surface. Waste storage availability (determined by stopes filling cycles) is the critical parameter that determines the ratio of total waste that can be retained underground.

Environmentally it is beneficial to store the waste underground. Further studies may include the use of pastefill.

Table 18-3 shows the stope tonnes broken down by zone and by mining method.

Table 18-3: Tonnes by Zone and Mining Methods

Zone	Total Tonnes	Cut and Fill	Longhole	Uppers
UMU	1,082,819	197,594	878,591	6,634
SUM	198,833	35,396	150,794	12,644
NUM	500,500	330,715	167,694	2,091
CT	55,995	54,213	0	1,782
UFW	413,685	136,516	218,929	58,240
FW1	722,436	696,182	23,559	2,696
FW2	67,032	42,711	14,946	9,375
MZ	100,312	44,267	51,816	4,229
V1	25,758	22,952	0	2,806
V2	108,133	105,601	0	2,532
V3	83,256	82,084	0	1,172
Total	3,358,760	1,748,231	1,506,329	104,201

The planned mine equipment fleet is based on an averaged 1,300 tpd over life of mine operation. Since the mine is a mixture of narrow vein ore bodies and larger open stopes, varying sizes of equipment are specified. Budget quotes and specifications were obtained from local equipment manufactures. Equipment selection decisions were intended to minimize excavation requirements, and considered the extra clearance needed to accommodate auxiliary ventilation in the headings.

A list of AEP, pre-production and production equipment is shown in Table 18-4. A total of 46 units will be required, 32 of them in the AEP stage. Productivity estimates based on haulage distances were used to estimate the number of LHDs and mine trucks required. Since the orepasses will be close to the orebody and the shaft, the planned productivities are relatively high.

Table 18-4: Mine Mobile Equipment

Equipment Fleet	Advance Exploration		Pre-Production		Total Fleet	
Underground Units	Fleet Number	HP	Fleet Number	HP	Fleet Number	HP
Development:						
Stope Jumbo (Production and development)	1	142	1	142	2	284
Ramp Jumbo	2	160	1	160	3	480
Scoop (6.5 yard) Dev't and Production	5	250		250	5	1250
Scoop (3.5 yard) Dev't Shaft 650	1				1	
Scissor Lift – Bolting	3	88	2	88	5	440
Production:						
Truck (30 tonnes)			2	300	2	600
Truck (20 tonne) MT 2010EJC 530	2	300			2	600
Cat 6 Wheel Truck	1	250			1	250
Mine Services:						
Utility Scoop (2yard)	3	250		250	3	750
Boom Truck - Construction, Nipping	1	88	1	88	2	166
Scissor Lift - Utility/Rehab	1	88	1	88	2	166
Grader			1	140	1	140
Personnel Carrier	2	88	1	88	3	264
Forklift	2	49	1	49	3	147
Supervision:						
Tractor Supervision, Survey	3	49	2	49	5	245
Maintenance:						
Lube Truck			1	100	1	100
Totals - Underground	27		14		41	
Surface Units						
980 c/w Forks	2	200		2	2	
Pick Up Truck	3	49			3	
Total Surface	**5**				**5**	
Total Units	**32**		**14**		**46**	

18.3 Mine Site Services

18.3.1 Ore and Waste Handling

The TM ore will be dumped from the shaft onto a low-permeability pad adjacent to the shaft house. Ore will be loaded onto highway trucks for transport to the Bell Creek Mill, approximately 40 km east of the mine site.

The waste rock from underground development will also be skipped or trucked to surface and dumped onto a low-permeability pad. This waste will be loaded onto mine trucks and transported to the waste rock dump on the south section of the site via internal roads as shown in Figure 25.

18.3.2 Power Supply

Preliminary estimates indicated a production load of 8.7 MVA plus 25% or 10 MVA will be required. Discussions with Hydro One along with the Independent Electricity System Operator ("IESO") were initiated. A positive response was received from both IESO and the Ontario Energy Board (OEB) that allowed connection of the mine load to the 115kV feeders.

The 115kV line was tapped from the Hydro One corridor south of Highway 101. The line was extended to the TM site, approximately 800 m, to a new 10 MVA substation. A 10 MVA substation was constructed to receive 115kV power from Hydro One feeder T61S and step down the power to 4,160/2,400 Volts. The transformer has provision for future fan cooling to increase its capacity to 13 MVA if required. The transformer is equipped with automatic on line top changer to provide constant voltage. The transformer secondary will feeds into 4,160/2,400 Volt switchgear located in the hoist building.

A diesel powered emergency backup electrical generator is located immediately south of the hoistroom.

18.3.3 Water Supply

Personal wash water is supplied from a water-well located on the west side of the property. The water is treated for supply to showers, toilets, and sinks with an ultraviolet (UV) disinfection system located inside the change house building.

The mine process water is primarily supplied from the reclaim water pond on the south side of the property (Figure 26).

18.3.4 Drainage and Water Discharge Control

Clean surface runoff water on the project site that does not have contact with process materials (e.g. ore) is collected and diverted to Thunder Creek via the internal road culvert, as shown in Figure 25. This practice facilitate monitoring and management of storm water.

Existing riparian buffers are maintained during the life of the project, in general accordance with the Timber Management Guidelines for the Protection of Fish Habitat (MNR, 1988). Conventional erosion control structures deployed around the perimeter of the project site, which prevents the release of sediment into surrounding watercourses. Control structures consisting of non-woven silt fence dug into the soil, as per Environmental Guidelines for Access Roads and Water Crossings (MNR, 1995), with straw bales staked into place on the upstream side of the silt fence. These control structures are maintained until the project site is re-vegetated and is physically stable, post closure. Spoils and grubbed overburden will be stockpiled away from watercourses and seeded to minimize erosion.

Mine water, as well as runoff from the shaft dump pocket and the (low-permeability) Containment Pad will be directed to the Containment Pond. This water is recycled for use in the underground mine. Excess water is treated and discharged to the Tatachikapika River.

18.3.5 Fire Protection

Fire protection for the offices and the shops is provided from a wet-well equipped with fire pumps adjacent to the recycle pond. The water supply for the fire protection is from the recycle pond. All fire protection systems for the buildings meet the requirements of the Ontario Building Code.

Fire extinguishers are conveniently located throughout the buildings, property and on all mobile equipment as required by safety standards and maintained certified at all times.

18.3.6 Sewage Disposal

Sewage is collected into a weeping system that is installed on the west side of the access road on the north side of the property (Figure 25). Sewage pumping maintenance services for these weeping systems is contracted out to a local provider.

18.3.7 Communications

Communications to and from the TM Project is via fibre optic (voice, facsimile, and internet). Inter-site communications for operations and security uses RF short wave radio transmission. A total of four radio licenses are active and in compliance (license #'s 5065774, 5065778, 5065898 and 5065899).

18.3.8 Buildings and Ancillary Facilities

The internal road accesses the infrastructure at the site. This includes the ramp area, the shaft, hoist room, storage yard, septic bed, recycle pond, and the waste stockpile. The main access road is branched to the west accessing the shaft, hoist room, warehouse and administration offices. The north branch accesses the water treatment pond, the stockpile yard and the explosives storage magazines. The branch to the east accesses the ramp portal area with related yard and buildings (Figure 25).

The main security office and gatehouse is located at the south end of the main parking lot. This is the only entry point to the mine site.

The administration building is located on the south side of the mine headframe. This building is portable trailer-type structure consisting of engineering/geology and drafting offices, change house dry, and other administrative offices. The maintenance shop is a Quonset hut-type building consisting of a small overhead crane and other maintenance facilities for equipment maintenance. The warehouse building is a butler style structure erected on a cement pad adjacent to the hoistroom. All of these building facilities are owned by LSG.

The change house consists of a change area that is equipped with benches, personal lockers, and dry baskets. This area also consists of two shower areas with enough showerheads to support the required work force at the property. There is also a single, segregated change house room within the main change house used by women.

Gasoline and diesel fuels for mobile equipment and mine vehicle use are stored on site within the Sat-Stat system of storage tanks. These tanks are surrounded by a containment berm.

Reagents and other chemicals are stored in their original packing in the warehouse.

Regular preventative and required maintenance and repairs of buildings, internal roads, civil work, process equipment, utility and mobile equipment is performed on site by LSG personnel.

A contracted maintenance services is used to supplement the LSG maintenance personnel as required.

18.3.9 Waste Disposal

All of the waste rock on 650 Level is currently hoisted along with the waste rock hauled to surface from the ramp area. This waste rock is stockpiled on surface.

Later during production when empty stopes are available all waste rock generated from development can be retained underground and used as backfill.

The surface waste stockpile is used to construct roads and pads for storage facilities. No waste from surface will be transferred back underground. At this stage, placing rock back underground through waste passes has not been considered and should be evaluated at the feasibility stage (Figure 25).

18.4 Processing

The design capacity of the BC Mill is approximately 500,000 dry tonnes of ore per year.

The original mill was expanded a number of times until 1,500 tpd capacity was reached in 2001. The main equipment includes two grinding mills, 1,400 and 400 hp, five leach tanks, eight CIP tanks, and a carbon - pressure strip based gold recovery circuit. Ore from the Hoyle Pond Mine was directed to the Bell Creek Mill for processing with the tailings being discharged at the Bell Creek Tailings Facility (BCTF) until July 2002 when the mill was shut-down and mothballed. No tailings were discharged at BCTF after July 2002. The mill was under care and maintenance from 2002 until start-up in May 2009.

The BC Mill also includes an adjacent conventional tailings disposal facility that is expandable to the present total Timmins Mineral Reserve and beyond. Expansion of the tailings facility to ultimately accommodate about 5M additional tonnes has been permitted. Staged expansion has been estimated by AMEC to cost $2.4M over a 10-year mine life for the 3 million tonnes of tailings expected from planned TM ore. The current tailings facility covers approximately 75 ha (185 acres). The site contains four basins as follows:

- Phases 1 and 2 combined = 24 ha (60 acres);
- Phase 3 = 31 ha (77 acres);
- Clearwater pond = 18 ha (45 acres); and
- Treated affluent pond = 2 ha (5 acres).

The author considers the TM deposit to be amenable to the conventional cyanidation processing of the Bell Creek Mill at a reasonable cost and will yield a gold recovery of approximately 95% with no abnormal environmental implications.

18.5 Markets

Gold prices have steadily increased over the last five years from US$350/oz to US$1,000/oz at the present time. Gold market forecasts in the industry have generally been bullish and increased price forecasts over the near term outnumber those that forecast lower prices. For the economic analysis, two year historical averages have been used to select the gold price and exchange rate. The base case values are US$ 950/oz Au and an exchange rate of US$1.15 /CA$. Refer to Figure 28.

18.6 Contracts

LSG has obtained the quotes from Northwest Transport to haul ore from the TM site to the BC Mill. The route that is being utilized is via Highway 101 to Halnor Road onto Bell Creek Mill. The hauling cost has been negotiated at $5.44/t.

Transportation of gold from the BC Mill site is handled by the Royal Canadian Mint.

LSG has obtained a parts contract agreement with Atlas Copco for their underground equipment.

The Advanced Exploration Program via shaft access and the 650 m level development has been successfully negotiated with Dumas Contracting of Timmins.

The rest of the operations related to mining, concentrating, smelting and refining will be done by LSG employees.

18.7 Environmental Considerations

LSG's corporate goal is to lay the foundation for the development, operation, and rehabilitation of profitable mines in a manner that respects and responds to the social, environmental, and economic needs of present generations and anticipates those of future generations in the communities where LSG works.

18.7.1 Reclamation

Mine closure is the orderly and safe environmental conversion of an operating mine to a 'closed-out' state.

At the conclusion of the mine life, the closeout rehabilitation measures summarized below will be implemented:

- Removal of surface buildings and associated infrastructure;
- Dewater the Containment Pond by pumping pre-treated water into the underground workings, or treat and release any remaining water;
- Deposition of rock from the containment pad into the underground workings;
- Excavation of the containment pad and pond and placement of the removed material in the underground workings;
- Flood the underground mine workings with lime treated water to above the elevation of backfilled materials to prevent oxidation;
- Secure mine openings in accordance with regulatory requirements;
- Contour any piles of rock that do not pose a metal leaching risk; and
- Contour, cover, and re-vegetate disturbed areas using available overburden.

All infrastructures will be removed from site and other disturbed areas associated with the project will be re-vegetated, mainly through natural regeneration using seed banks in the overburden stored on site.

The above procedure was developed in compliance with the requirements and standards of the Mine Rehabilitation Code of the Mining Act.

LSG plans to maintain the BC Mill facilities in an operational mode after the end of the currently planned mine life; therefore the mill closure planning has not been finalized or included in this report.

LSG estimated mine closure costs at $624,706.

18.7.2 Bond Posting

LSG has posted $533,000 in a Letter of Credit with the Ministry of Northern Development and Mines (MNDM) to cover the costs of the AEP reclamation.

An additional Letter of Credit will be required upon change of Project status from AEP to production.

The Closure Plan was developed within the requirements and standards of the Mine Rehabilitation Code of the Mining Act.

18.8 Taxes and Royalties

Property tax and income tax will be paid according to the standard of Canada Customs and Revenue Agency (CCRA) requirements.

If required, as described in section 3.4 royalties will be paid to Mr. Lebrash; however, no mineral reserve have been declared on the claim subject to this royalty (refer to Section 3.4 of this report).

18.9 Capital Costs

The total pre-production expenditures (including advanced exploration) were determined to be $140M with an overall accuracy of "±20%" and they include the expenditures for the AEP. These are summarized in Table 18-5.

The total investment to be expended in order to reach the gold production phase starts with the point of authorization of the AEP, the re-commissioning of the BC Mill and pre-production to a rate of 1,500 tpd for a span of 2.5 years.

Costs were "actual" when available or estimated by the QP responsible for each of the main areas, with input from LSG as follows:

AEP at the TM Property;
The site preparation, support facilities and infrastructure, headframe, shaft sinking, hoisting and the ramp development down to the 200 m level was provided by LSG.

TM Production Mine;

- U/G Development and Mining: LSG actual cost and Consultant future estimated costs
- Mine Closure: Golder, Sudbury and LSG closure plan cost

BC Mill Re-Commissioning;
BC Mill equipment re-commissioning and operating costs. These are summarized in Table 18-6.

Table 18-5: Capital Expenditures Summary

	Advanced Exploration $M	Pre-Production $M		Production Incl. Closure $M
Advanced Exploration Program	140			
Mining		33		29
Total Project Investment	**140**	**33**		**29**
Total Project Including AEP		**202**		
Total Production CAPEX		**29**		
Working Capital, one month OPEX (not used, for reference only)		5.3		
BC Acquisition (Sunk Cost for Future Financial Evaluations):		7.5		
Project Sunk Costs Pool for Financial Evaluations:		52		
AE Costs for Sunk Costs in Production Phase financials:		140		

The capital cost estimate was divided into, ‘Pre-Production’ Capital and ‘Production’ Capital. Pre-Production capital includes all mine equipment and development expenditures from January 1, 2010 to September 30, 2101. ‘Production Capital’ is defined as further similar expenditures on-going after start-up and includes a tax related period of Post-Production Development Capital, starting October 1, 2010 when the mine is up to a steady state production of 75% capacity. A summary of Pre-Production and Production Capital Costs is shown in Table 18-5.

18.9.1 Payback

The payback period is 3 years based on the assumptions specified in the Section 18.11.2 of this report.

18.10 Update

The operating cost estimate for Bell Creek Mill is estimated at $17.62. The cost for ore haulage from the TM is contracted at $5.44/t. The following Table breaks down the operating costs:

Table 18-6: Mill Operating Costs

ITEM	COST/IN $	TOTAL
Safety	0.30	147,000
Salary Labour	1.65	809,580
Operations Labour	4.73	2,318,132
Electrical	4.13	2,025,639
Reagents and supplies	5.24	2,568,614
Misc. Consumables	0.51	249,900
Assay Lab Consumables	0.40	196,000
Outside Services	0.35	171,500
Security	0.30	147,000
Total Cost (Can$)	**$17.62**	**$8,633,365**

Processing Cost

Total unit and annual costs were derived from the nominal capacity of 1,500 tpd for 350 days per year and include both fixed and variable costs.

18.11 Economic Analysis

18.11.1 Introduction

The capital and operating costs used in the economic analysis are shown in Table 18-7 and

Table 18-8. The model was created to facilitate sensitivity analyses for gold prices, operating and capital expenses, production rate, and foreign exchange rate.

Table 18-7: Capital Cost Basis

	Advanced Exploration $M	Pre-Production $M	Production and Closure $M
Advanced Exploration Program net	140		
Mining		33	29
Total Project CAPEX	**140**	**33**	**29**
Total Production Phase		**62**	
Total Project Related Expenditures		**202**	

Table 18-8: Operating Cost Basis

	Unit Cost $/t	Annual $M
Mining	69.66	56.0
Trucking to Mill	5.44	2.6
Processing at BC Mill , Variable, Fixed and CAPEX Amortization	17.62	9.6
G & A	2.36	2.5
Total Project OPEX	95.08	70.7
Cash Cost:	$US369/oz	

18.11.2 Input and Assumptions

For this analysis, a base gold price of US$950/oz and 1.15 US$/CA$ exchange rate was used based on marketing analyst data representing. The cash flow is based on the LSG assumption that the AEP project flows into pre-production period. During this time AE drilling program would be completed and commercial production would start 9 months after the start of pre-production.

The costs of additional facilities, mine development, and the BC Mill commissioning, have been added to the overall Project economics. The AEP underground drilling program as outlined in the Cook/Dumas report was started in the second quarter (Q2) of 2009.

Project taxes were included in the operating costs calculation; however, they are excluded from the capital budget. The total pre-production costs were determined with an overall accuracy of 20%.

The main economic parameters developed in the PFS and used in the cash flows are shown in Table 18-9.

Table 18-9: Base Case Economic Parameters

Variable Processing Cost and Fixed Cost	$17.62/tonne
Mining Costs (Underground)	$69.66/tonne
GA Costs and other Fixed Costs	0.94M/year
Truck Haulage	$5.44/t
Gold Price	$US 950/troy oz
Exchange Rate	US$0.86/CA$
Gold Recovery	95%
Pay Factor/Refining/Transport Charges Gold Bullion	$3/oz
Equity/Debt Ratio	100%
Pre-Production Capitalized Expenditures	$33M
"Mine" Life	7.3 years
Post Production	$29M
TM Mine Closure	included

18.11.3 Model Results

The results of the analysis indicated an Internal Rate of Return ("IRR") of 28% at US$950/oz gold and 35% at US$1,050/oz gold.

Table 18-10: Base Case Analysis

Annual	**Net Present Value ($M's)**
Discount Rate	**Pre-Tax**
0%	300M
5%	184M
8%	135M
15%	62M
IRR	**28%**

The results indicate a Pre-Tax IRR of 28% and a net present value of $184M at a discount rate of 5%.

18.11.4 Sensitivity Analysis

In order to demonstrate the effect of changes in parameters and variables on the Project economics, a number of cash flow sensitivity runs were conducted. These results are summarized in Table 18-11.

Table 18-11: Sensitivities

Sensitivity	Variance	Value	NPV: $Mill				IRR
			0%	5%	8%	15%	
Gold Price	-5%	902.5 (US$/oz)	259	154	111	46	25.0%
	Base Case	950.0 (US$/oz)	300	184	135	62	28.2%
	+5%	997.5 (US$/oz)	341	213	160	78	31.3%
Total Capital	-5%	192.4 ($M)	305	188	139	65	29.1%
	Base Case	202.6 ($M)	300	183	135	62	28.2%
	+5%	212.7 ($M)	295	179	131	58	27.3%
Mining Cost	-5%	66.18 ($/t)	312	192	142	66	29.1%
	Base Case	69.66 ($/t)	300	184	135	62	28.2%
	+5%	73.14 ($/t)	288	175	128	57	27.3%
Exchange Rate	-5%	0.82 ($US/$CDN)	340	212	159	77	31.2%
	Base Case	0.86 ($US/$CDN)	300	183	135	62	28.2%
	+5%	0.90 ($US/$CDN)	264	158	114	47	25.4%
Labour Cost	-5%	94 ($M)	305	187	138	64	28.6%
	Base Case	99 ($M)	300	184	135	62	28.2%
	+5%	104 ($M)	295	180	132	60	27.8%
Mill Feed from Reserve	-5%	3203 (000's tonnes)	268	161	117	49	25.8%
	Base Case	3371 (000's tonnes)	300	183	135	62	28.2%
	+5%	3540 (000's tonnes)	332	206	154	74	30.6%
Truck Haulage	-5%	5.17 ($/t)	301	184	136	62	28.2%
	Base Case	5.44 ($/t)	300	184	135	62	28.2%
	+5%	5.71 ($/t)	299	183	135	61	28.0%
Custom Milling	-5%	16.74 ($/t)	303	186	137	63	28.4%
	Base Case	17.62 ($/t)	300	184	135	62	28.2%
	+5%	18.50 ($/t)	297	181	134	61	28.0%

19.0 INTERPRETATION AND CONCLUSIONS (ITEM 21)

Based on this report as well as the supporting reference documents, The Authors arrived at the following conclusions:

19.1 Exploration

- WGM considers that the LSG exploration stage has met industry standard; however WGM recommends check assaying be carried out on a regular and more frequent basis, so corrective measures can be undertaken in a timely manner in the unlikely event that a problem is detected.

19.2 Infrastructure

- The TM Property is easily accessible by highway;
- The Property is in close proximity to water sources and power grids; and
- The local Timmins mining oriented workforce will benefit the Project infrastructure

19.3 Mineral Resources and Production

- The mineral reserve estimate is compliant with NI43-101 requirements and CIM definitions;
- Probable mineral reserves were estimated at 3.4 Mt with an average grade 7.52 g/t Au;
- Up to 95% metallurgical gold recovery can be achieved from the TM ore;
- The TM Mine can reasonably achieve an average production rate of 1,300 t/d over the life of mine; and
- The BC Mill has been modified for a production rate of 1,500 tpd and it is suitable for TM ore treatment.

19.4 Project Economics

- The QP's mining plan and economic results meet LSG requirements; and
- The results of the economic analysis indicated IRR will be in the range of 28% to 35% at gold prices of US$950 and US$1,050/oz respectively. This positive economic result supports the continuing decision to implement of the AEP.

19.5 Environmental

- No environmental risks have been identified at the TM Project as a result of planned mining or mining related activities.

19.6 Risk and Opportunities

19.6.1 Project Risks

It is the QP's opinion that the most significant risks of the TM Project are related to:

- The use of the polygonal model. It is highly recommended that a block model be developed.
- External dilution could be higher than planned. Minimal geotechnical work has been completed and there is no mining history on the deposit demonstrating the ground conditions. Development of the zones wider than the geological model may be required to insure mining of the Mineral Resource ounces resulting in a lower grade.
- The continuity of the vein structures via ramp access from surface to 200 Level has demonstrated that the veins are not as continuous as previously planned. The mineral resources for the veins are approximately 25% less than previously reported.
- The AEP development on the 650 Level has demonstrated to be a little better than reported. The completion of the bulk sample will validate the results.
- There is some uncertainty in the grade estimation of the TM deposit. The Authors state in the 2007 Technical Review "Bulk sampling would be necessary to determine the grade of these structures reliably". The AEP will obtain a bulk sample;
- The Project schedule is aggressive. There is a risk of falling behind which would have an effect on the Project economics.
- Geotechnical considerations. A geotechnical study needs to be carried out to determine the mining sequence relationship of orebodies.

19.6.2 Project Opportunities

It is the Author's opinion that the most significant opportunities for the TM Project are related to:

- Current exploration drilling indicates the mineralization in most zones is open to depth, with some very wide zones with significant grade occurring on the lowest levels explored. This combined with a large area of pyroxenite to the south of the deposit on the adjoining Property also being explored by LSG demonstrates that significant potential exists to expand the known mineralization and add additional resources for the Project;

- There is potential for increased gold grades when grade capping is re-evaluated based on the results of the bulk sample planned during the AEP. To put this into perspective, the Author estimated that the mineral reserves grade would be approximately 35% higher if there was no grade capping applied;
- The bulk sample and geologic mapping program may provide a better understanding of mineralization controls and possibly an understanding of structural kinematics that may assist in predicting and identifying higher grade gold mineralization. The bulk sample may also provide the basis from which to quantify the effect of coarse gold mineralization that may not be accounted for in grade estimates based on drill hole samples only;
- The current gold price expressed in Canadian currency is higher than the base case economic assumptions. There is potential for increased revenue due to a positive variance in the gold price.

19.7 Project Conclusion

Exploration by LSG from 2003 to 2007 has defined a NI 43-101 Indicated mineral resources of 3.2 Mt with an average gold grade of 8.6 g/t Au. The QP has developed a mine plan and estimated Probable mineral reserves of 3.4 Mt grading 7.56 g/t Au.

The economic analysis indicates that the TM Project is most sensitive to gold price. Taking into consideration the present price of gold and the highly developed local infrastructure supporting the Project, it is considered to be an acceptable risk Project with a good pre-tax base case rate of return, and potential for a higher rate of return.

20.0 RECOMMENDATIONS (ITEM 22)

20.1 Recommended Work Programs

Geological

1) The application of grade capping should be further investigated.
2) The block model should be updated from a polygonal model using more modern modeling and estimation techniques for future mineral resources estimates. It should also be updated with the results obtained from the AEP.
3) Additional exploration targets near the proposed TM AEP shaft should be followed up.
4) The underground AEP is warranted and the updated mineral resource estimate should take the form of a 3D block model.

Mining

1) Re-evaluate the mining method (increase use of shrinkage mining) in conjunction with the AEP to minimize dilution.
2) Multiple mining fronts are required in order to maintain the increased production.
3) More geotechnical evaluations should be carried out to evaluate mining sequencing.

20.2 Budget

Based on the actual cost for LSG the estimated cost of the AEP is $140M. (Please refer to Table 20-1)

Table 20-1: AEP Capital Expenditures

AEP Capital Expenditures	$'000,s
Pre 2009 costs	71,870
2009 Budget Costs	66,601
Additional Mill Upgrades	1.561
Total Capitalized AEP Costs	**140,032**

The QP considers that additional costs of the above recommendations would be estimated at:

- Approximately $110,000 for AEP additional engineering studies;
- Approximately $500,000 for additional exploration on the Property; and
- The total budget would be $141M for the above programs if the recommendations are implemented.

21.0 REFERENCES (ITEM 23)

1) Gilders, R., Cheung, L. (RPC, Fredericton, BN, Canada), 2006, LSG Project, Preliminary Beneficiation Tests.
2) L.D.S. Winter, NI43-101 Technical Report for LSG Corp. on the Timmins Gold Project;
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4) John R. Sullivan, James G. Lavigne, Michael W. Kociumbas, (WGM, Toronto, ON, Canada), 2007, A technical review of the Timmins West Gold Project in Bristol Township, Timmins area, Ontario, Canada for LSG Gold Corp
5) Cook/Dumas, (Thunder Bay, ON, Canada), 2007, LSG Project AE study Hoisting Plan and Shaft, reference Y6364-01-R002rev4;
6) AMEC, (Fredericton, NB, Canada), 2007, BC Mill tailings facility study, tailings disposal and treated effluent storage;
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8) BHM Consultants, (Timmins, ON, Canada), 2007, LSG Gold Corp. BC Mill electrical and structural due diligence review, reference 07-045;
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10) G. Darling, (SRK, Sudbury, Ontario, Canada), 2007, LSG Gold Timmins West Pre-feasibility study report;
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12) Canadian Securities Administrators, 2005, National Instrument NI43-101, Standards of Disclosure for Mineral Projects;
13) Canadian Securities Administrators, 2005, National Instrument NI43-101, Form 43-101F1 Technical Reports for National Instrument NI43-101

22.0 SIGNATURE PAGE (ITEM 24)

The effective date of this Technical report, entitled "Updated NI 43-101 Technical report on the Timmins Mine Property, Ontario, Canada" and addressed to Lake Shore Gold Corp., is August 28, 2009.

"Signed and Sealed"

George Darling, P. Eng.

October 1, 2009

"Signed and Sealed"

Robert Kusins, P. Geo.

October 1, 2009

"Signed and Sealed"

Jacques Samson, P. Geo.

October 1, 2009

"Signed and Sealed"

Heather Miree, P. Geo

October 1, 2009

"Signed and Sealed"

Todd Fayram, Q.P. (MMSA)

October 1, 2009

23.0 GLOSSARY

23.1 Mineral Resources and Reserves

23.1.1 Mineral Resources

The Mineral Resources and mineral reserves have been classified according to the "CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines" (November 2005). Accordingly, the Resources have been classified as Measured, Indicated or Inferred, the Reserves have been classified as Proven, and Probable based on the Measured and Indicated Resources as defined below.

A Mineral Resource is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes.

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough to confirm both geological and grade continuity.

23.1.2 Mineral Reserves

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, and economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

A 'Probable Mineral Reserve' is the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

23.2 Glossary

Table 21.2.1: Glossary

Term	Definition
Assay:	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure:	All other expenditures not classified as operating costs.
Composite:	Combining more than one sample result to give an average result over a larger distance.
Concentrate:	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
Crushing:	Initial process of reducing ore particle size to render it more amenable for further processing.
Cut-off Grade (CoG):	The grade of mineralized rock, which determines as to whether or not it is economic to recover its gold content by further concentration.
Dilution:	Unwanted waste, which is mined with ore.
Dip:	Angle of inclination of a geological feature/rock from the horizontal.
Fault:	The surface of a fracture along which movement has occurred.
Footwall:	The underlying side of an orebody or stope.
Gangue:	Non-valuable components of the ore.
Grade:	The measure of concentration of gold within mineralized rock.
Hangingwall:	The overlying side of an orebody or slope.
Haulage:	A horizontal underground excavation which is used to transport mined ore.
Igneous:	Primary crystalline rock formed by the solidification of magma.
Level:	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Lithological:	Geological description pertaining to different rock types.
LoM Plans:	Life-of-Mine plans.
Material Properties:	Mine properties.
Milling:	A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease:	A lease area for which mineral rights are held.
Mining Assets:	The Material Properties and Significant Exploration Properties.
Ongoing Capital:	Capital estimates of a routine nature, which is necessary for sustaining operations.
Ore Reserve:	See Mineral Reserve.
Pillar:	Rock left behind to help support the excavations in an underground mine.
RoM:	Run-of-Mine.
Sedimentary:	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Shaft:	An opening cut downwards from the surface for transporting personnel, equipment, supplies, ore and waste.
Smelting:	A high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase.
Stope:	Underground void created by mining.

Stratigraphy:	The study of stratified rocks in terms of time and space.
Strike:	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulphide:	A sulphur bearing mineral.
Tailings:	Finely ground waste rock from which valuable minerals or metals have been extracted.
Thickening:	The process of concentrating solid particles in suspension.
Total Expenditure:	All expenditures including those of an operating and capital nature.

23.2.1 Abbreviations

The metric system has been used throughout this report unless otherwise stated. All currency is in Canadian Dollars unless stated otherwise.. Market prices are reported in US$ per troy oz of gold and silver. Tonnes are metric of 1,000 kg, or 2,204.6lbs. The following abbreviations are used in this report.

Table 21.2.1: Abbreviations

Abbreviation	Unit or Term
AA	atomic absorption
Au	gold
AuEq	gold equivalent grade
°C	degrees Centigrade
CIL	carbon-in-leach
CoG	Cut-off-Grade
cm	centimetre
cm2	square centimetre
cm3	cubic centimetre
°	degree (degrees)
ft	foot (feet)
ft2	square foot (feet)
ft3	cubic foot (feet)
g	gram
gpt	grams per tonne
ha	hectares
HDPE	Height Density Polyethylene
hp	horsepower
kg	kilograms
km	kilometre
km2	square kilometre
kV	kilovolt
l	litter
LoM	Life-of-Mine
m	meter

m2	square meter
m3	cubic meter
mm	millimetre
mm2	square millimetre
mm3	cubic millimetre
Moz	million troy ounces
Mt	million tonnes
MW	million watts
m.y.	million years
NI 43-101	Canadian National Instrument 43-101
oz	troy ounce
%	percent
ppb	parts per billion
ppm	parts per million
QA/QC	Quality Assurance/Quality Control
RC	rotary circulation drilling
RoM	Run-of-Mine
RQD	Rock Quality Description
s	second
SG	specific gravity
st	short ton (2,000 pounds)
t	tonne (metric ton) (2,204.6 pounds)
tph	tonnes per hour
t/a	tonnes per annum
t/d	tonnes per day
tpy	tonnes per year
TSF	tailings storage facility
µ	micron or microns
V	volts
W	watt
yr	year

Appendix A

Report Figures

Report Figures

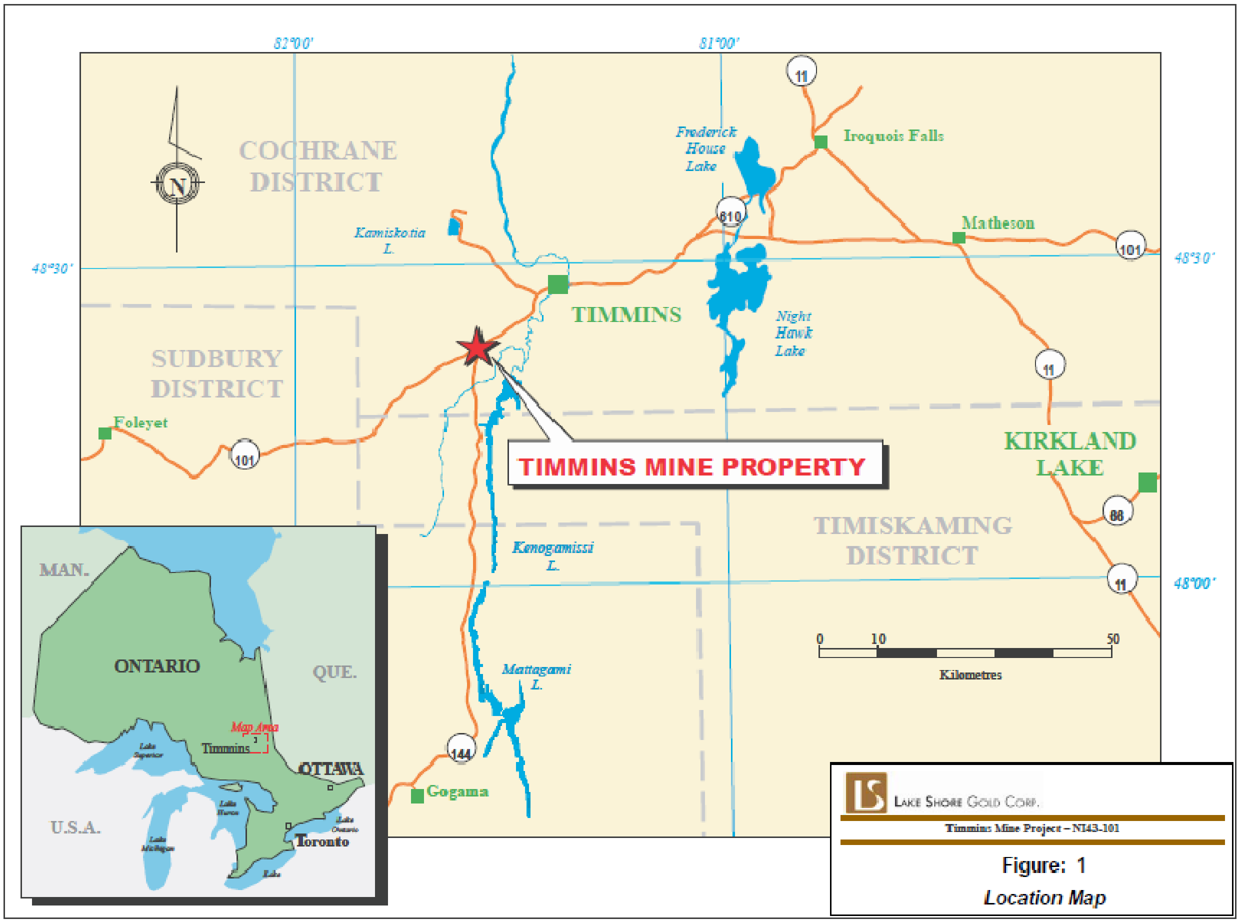

Figure 1: Location Map

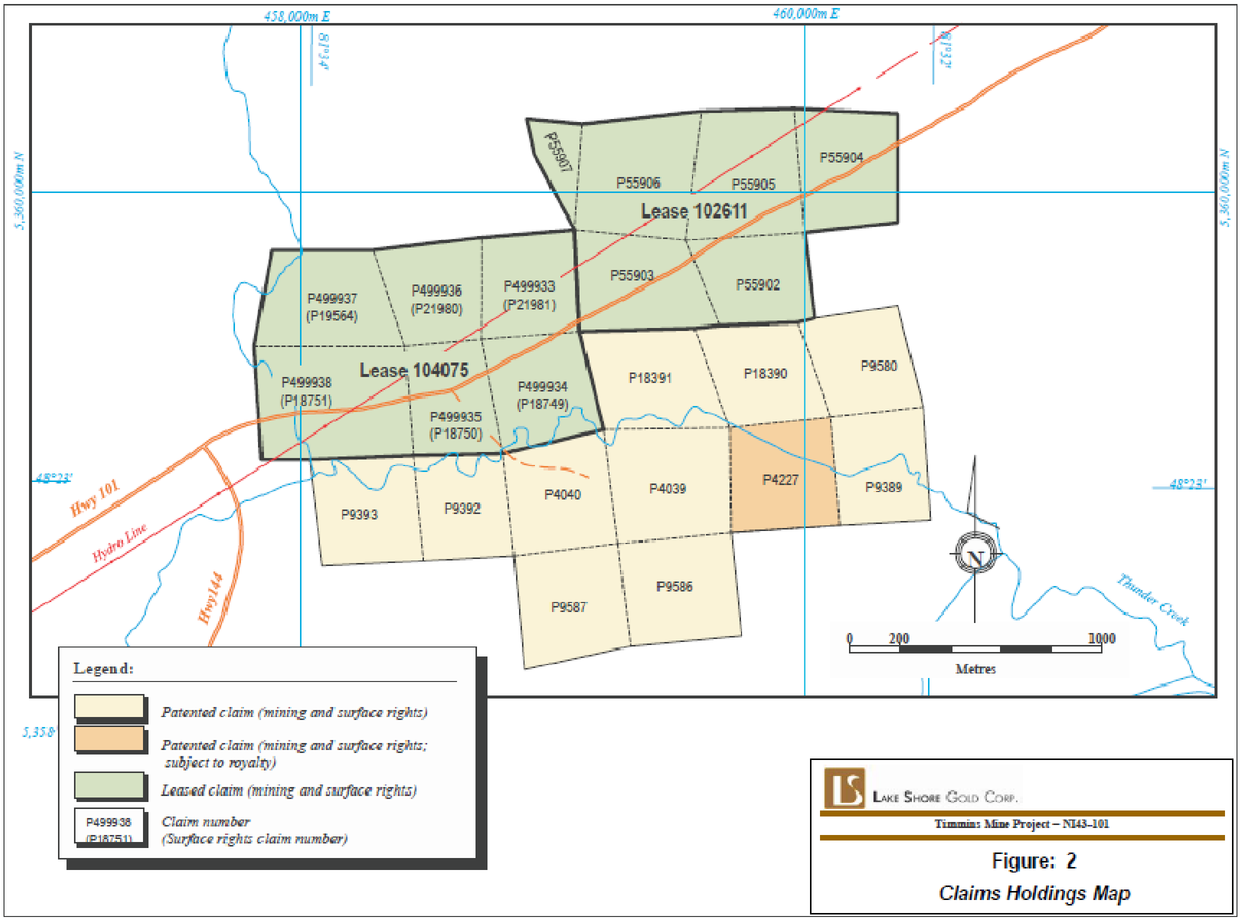

Figure 2: Claims Map

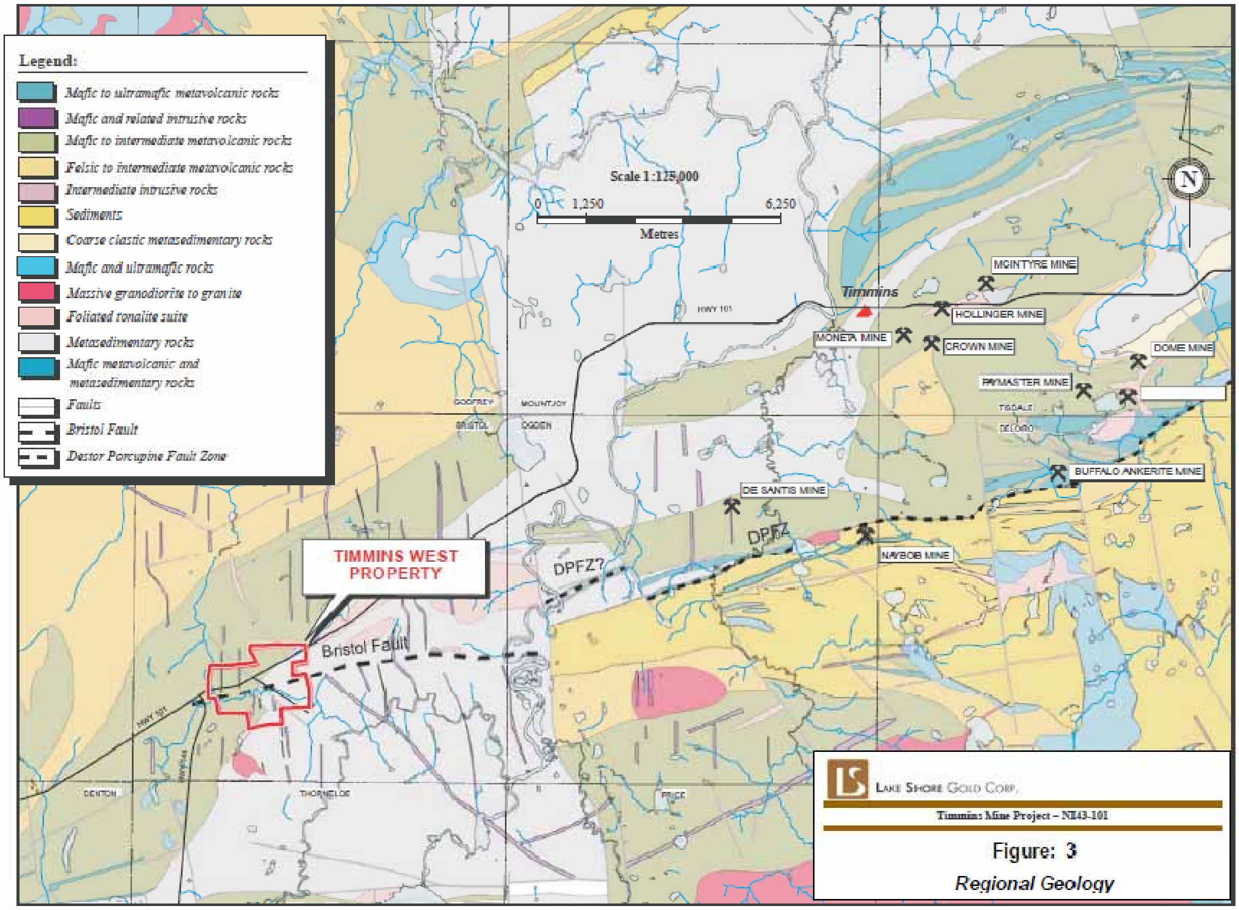

Figure 3: Regional Geology

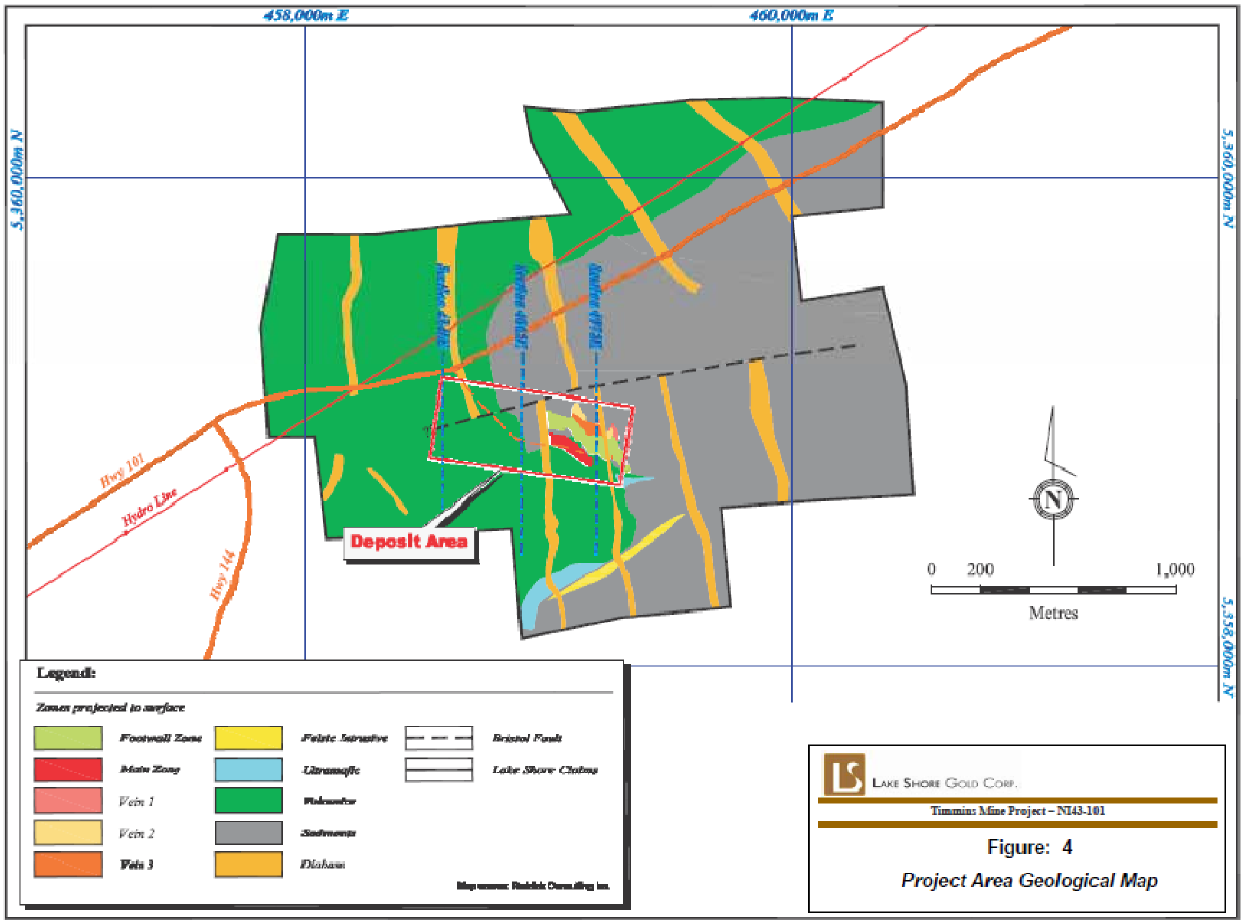

Figure 4: Project Area Geological Map

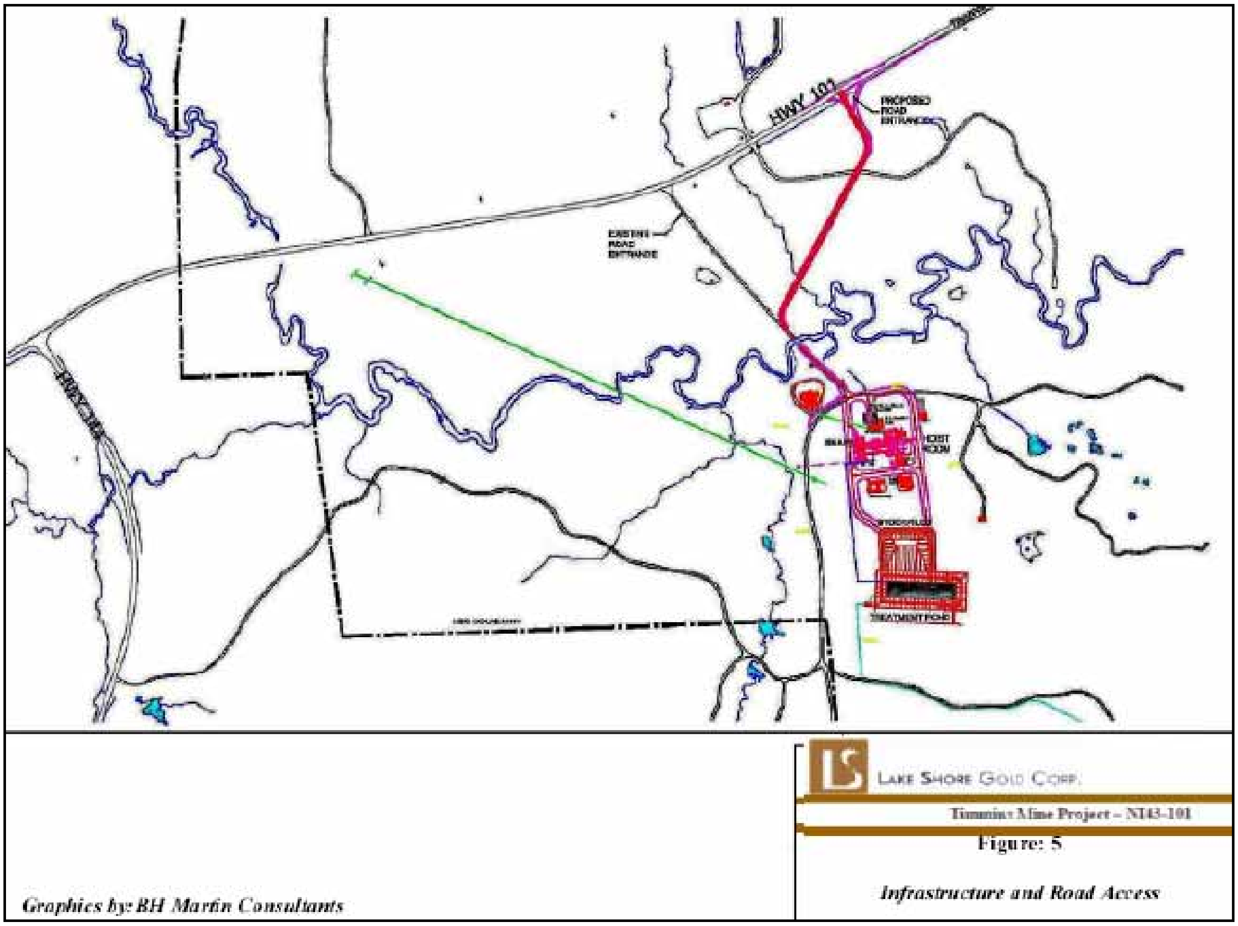

Figure 5: Infrastructure and Road Access

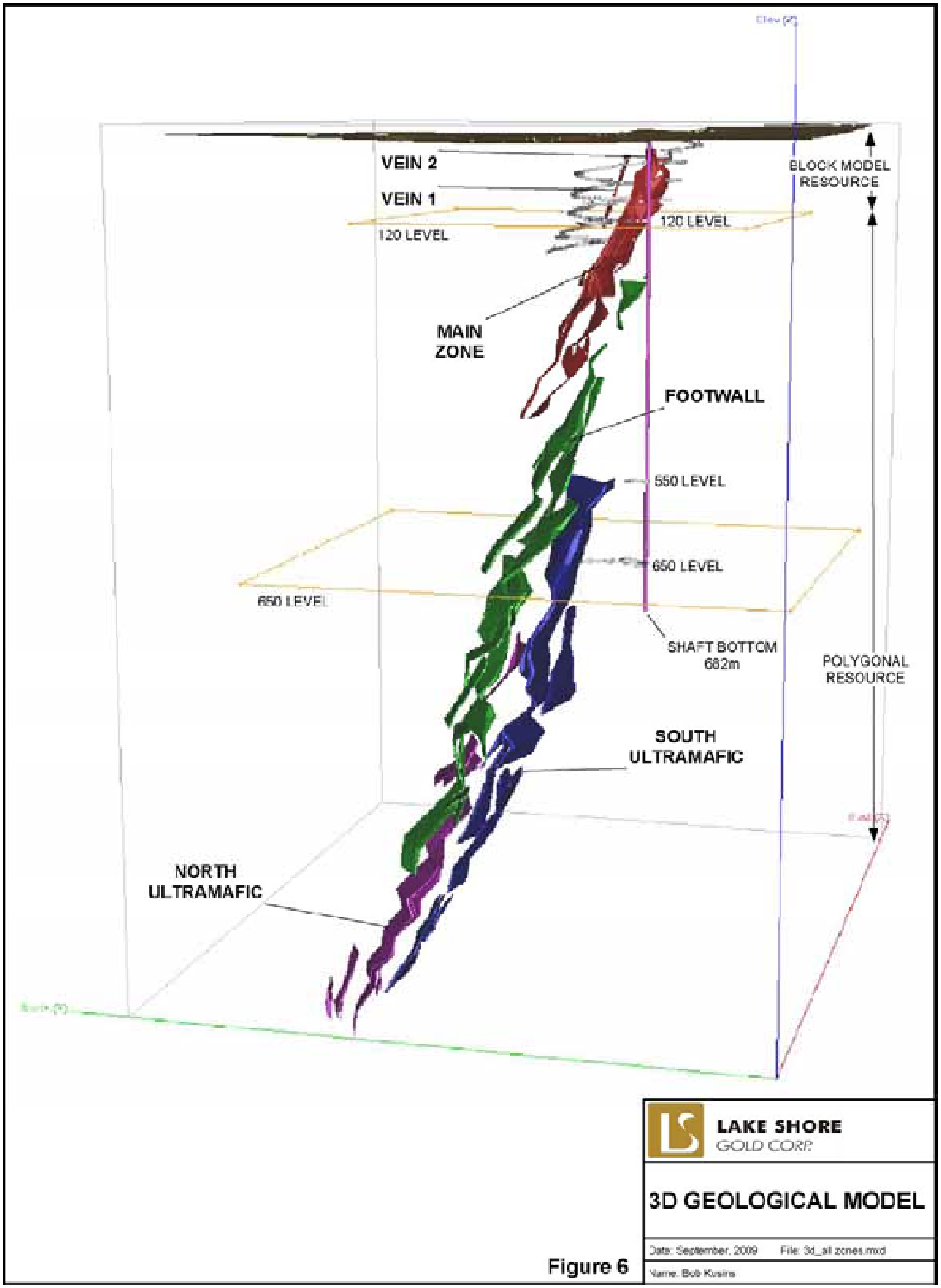

Fig 6: 3D geological Model

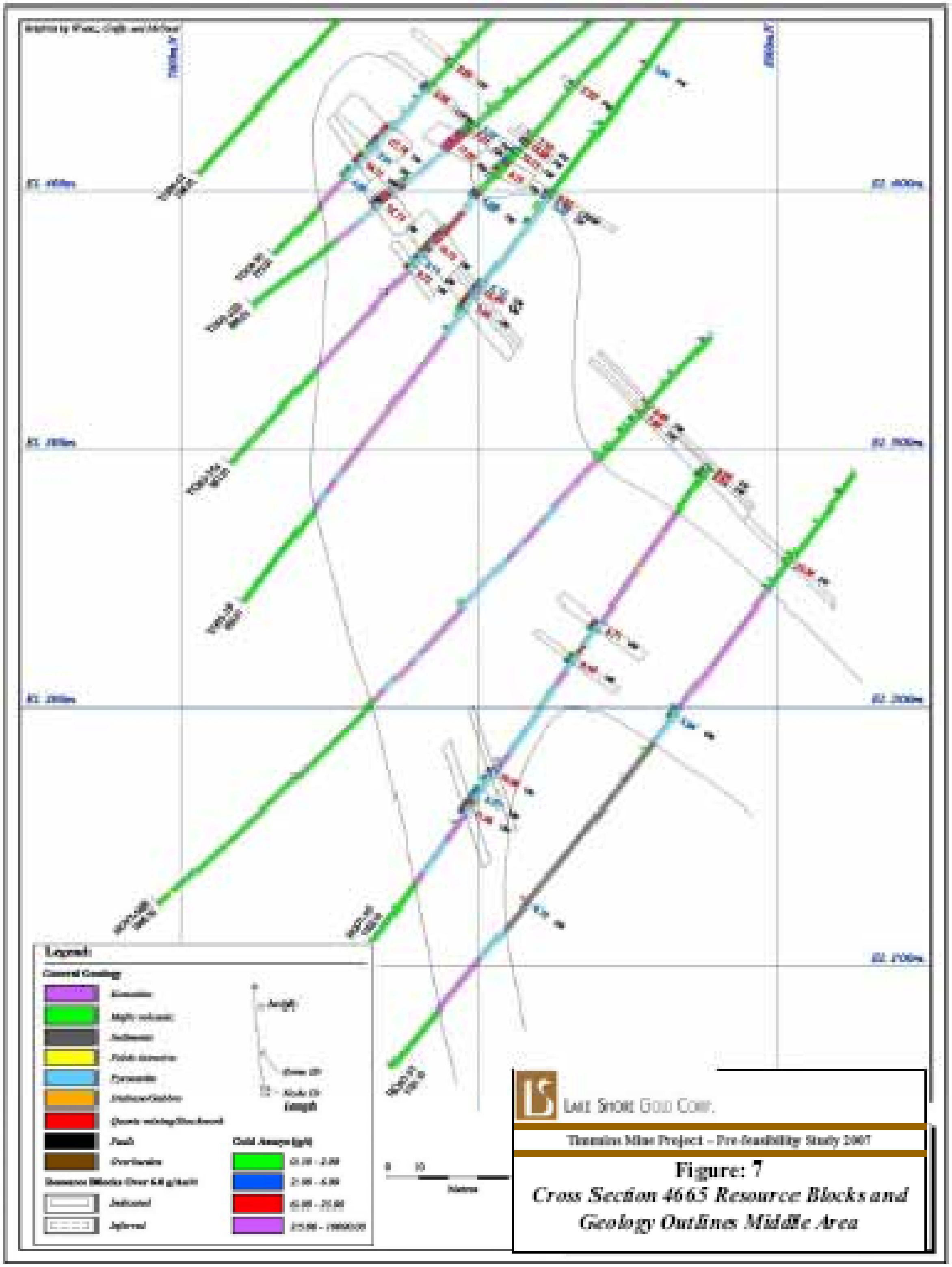

Figure 7: Cross Section 4665 Resource Blocks and Geology Outlines Middle Area

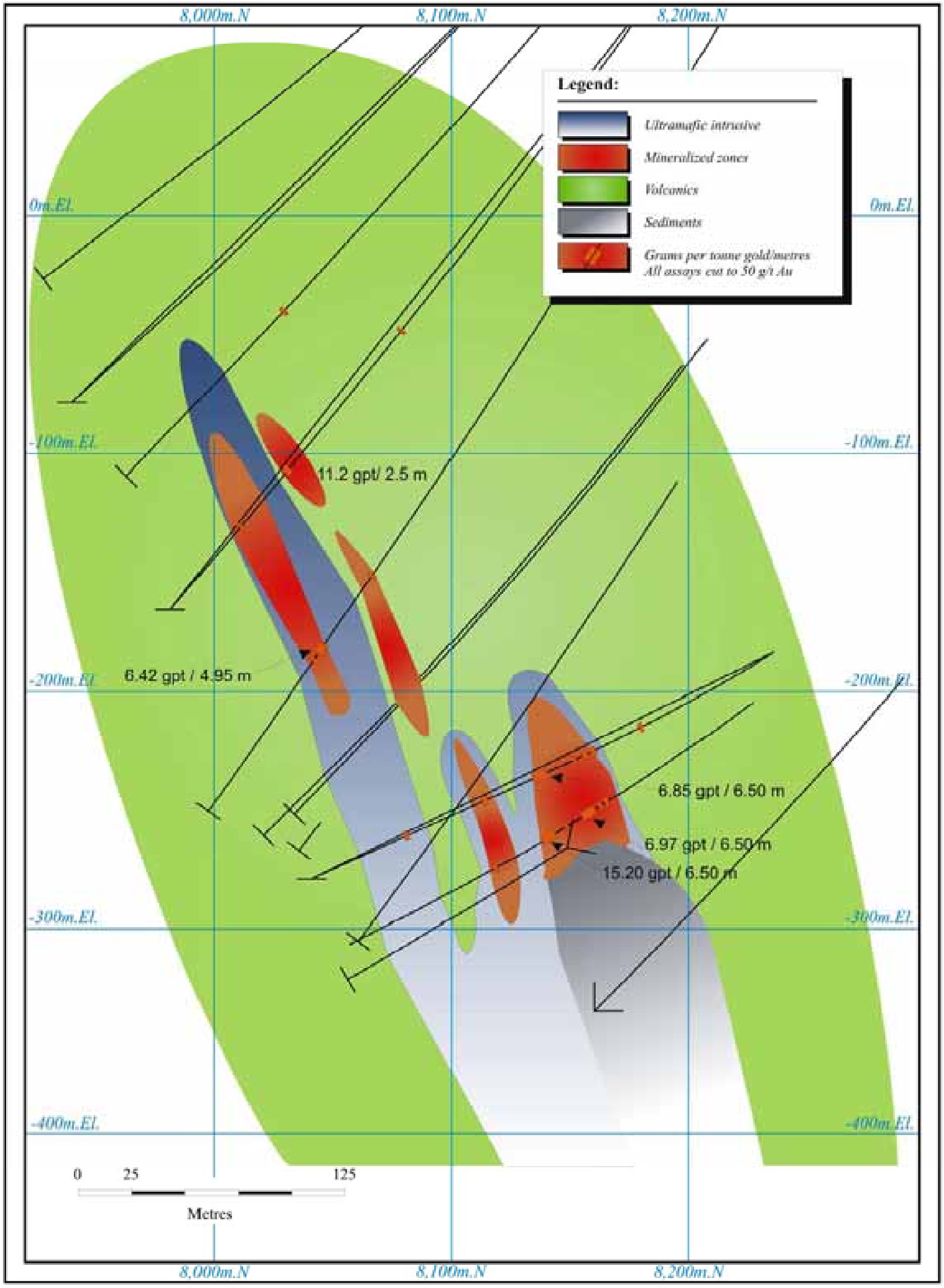

Figure 8: Cross Section 4340E Showing Geological interpretation Lower Area

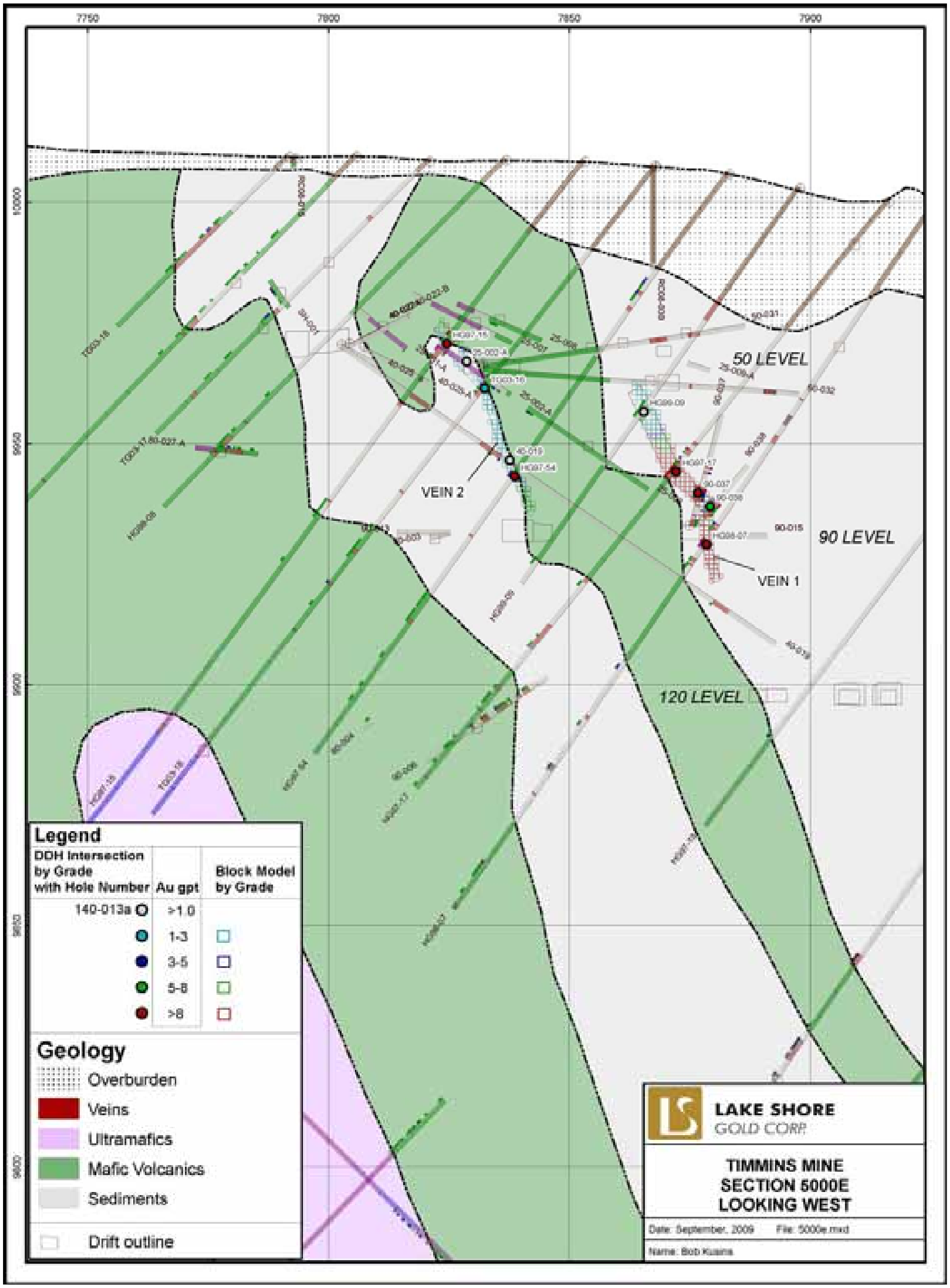

Figure 9: Cross Section 5000E Geological Interpretation and Resource Blocks Upper Area.

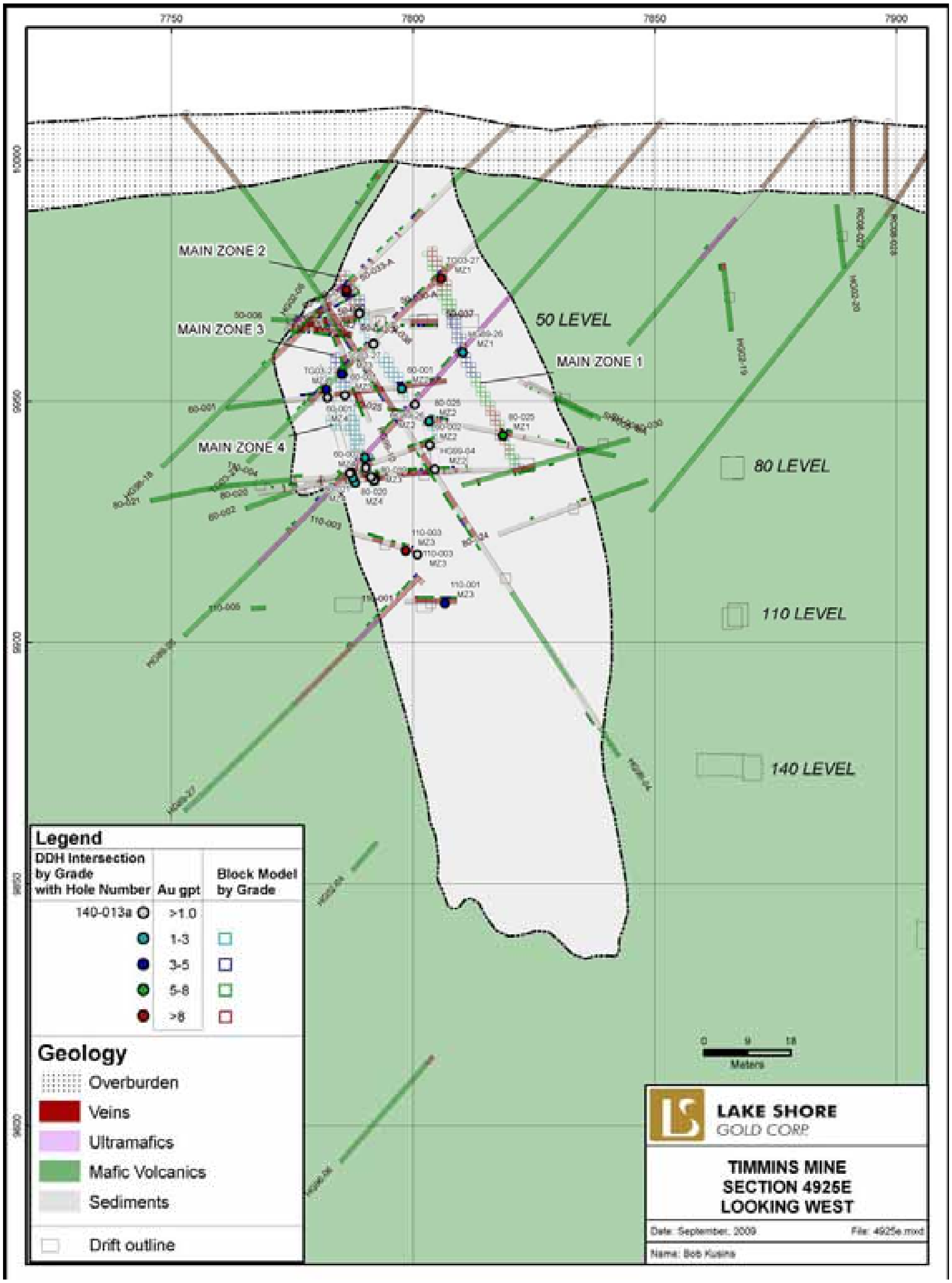

Figure 10: Cross Section 4925E Geological Interpretation and Resource Blocks Upper Area.

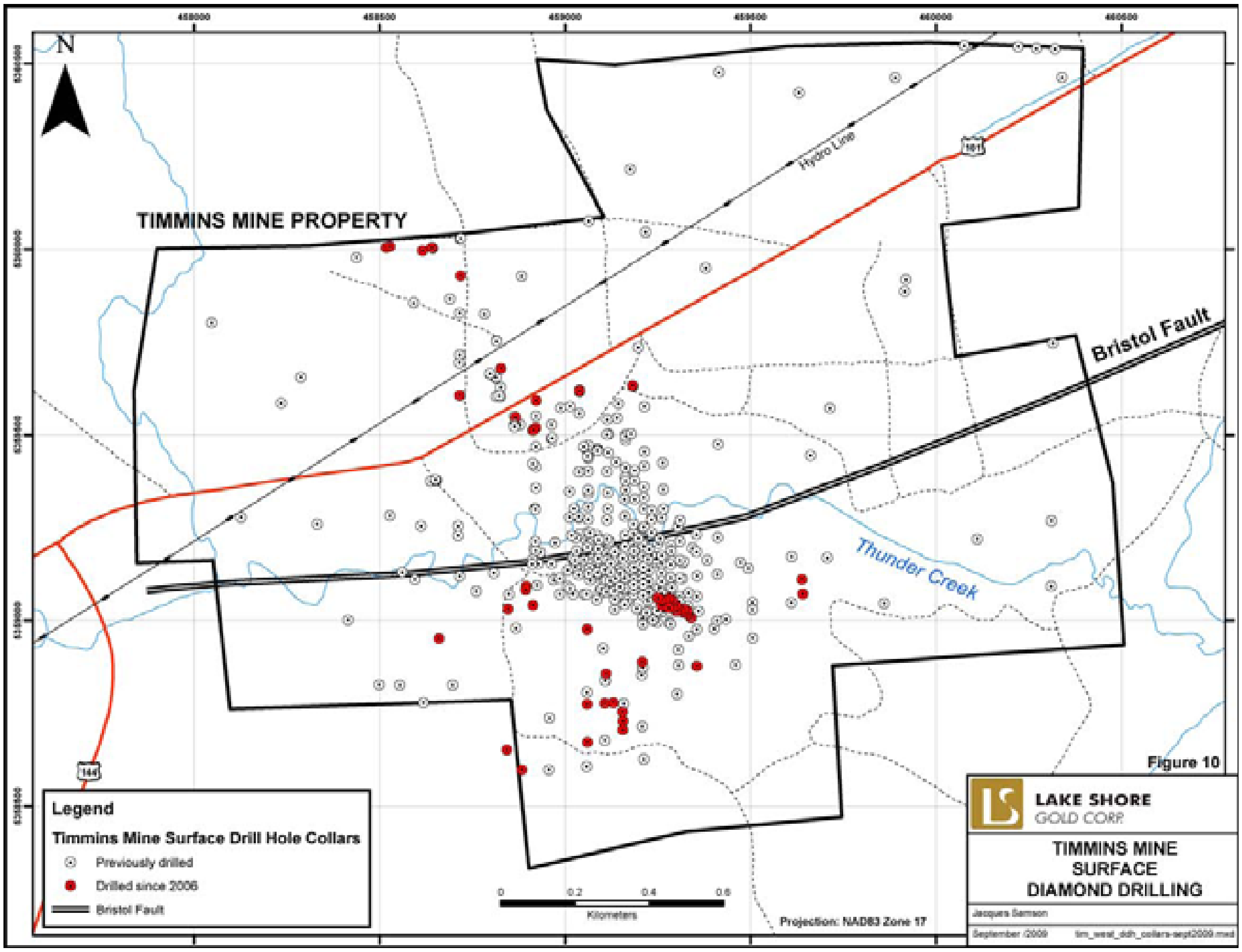

Figure 11: TM Diamond Drill Surface Facilities

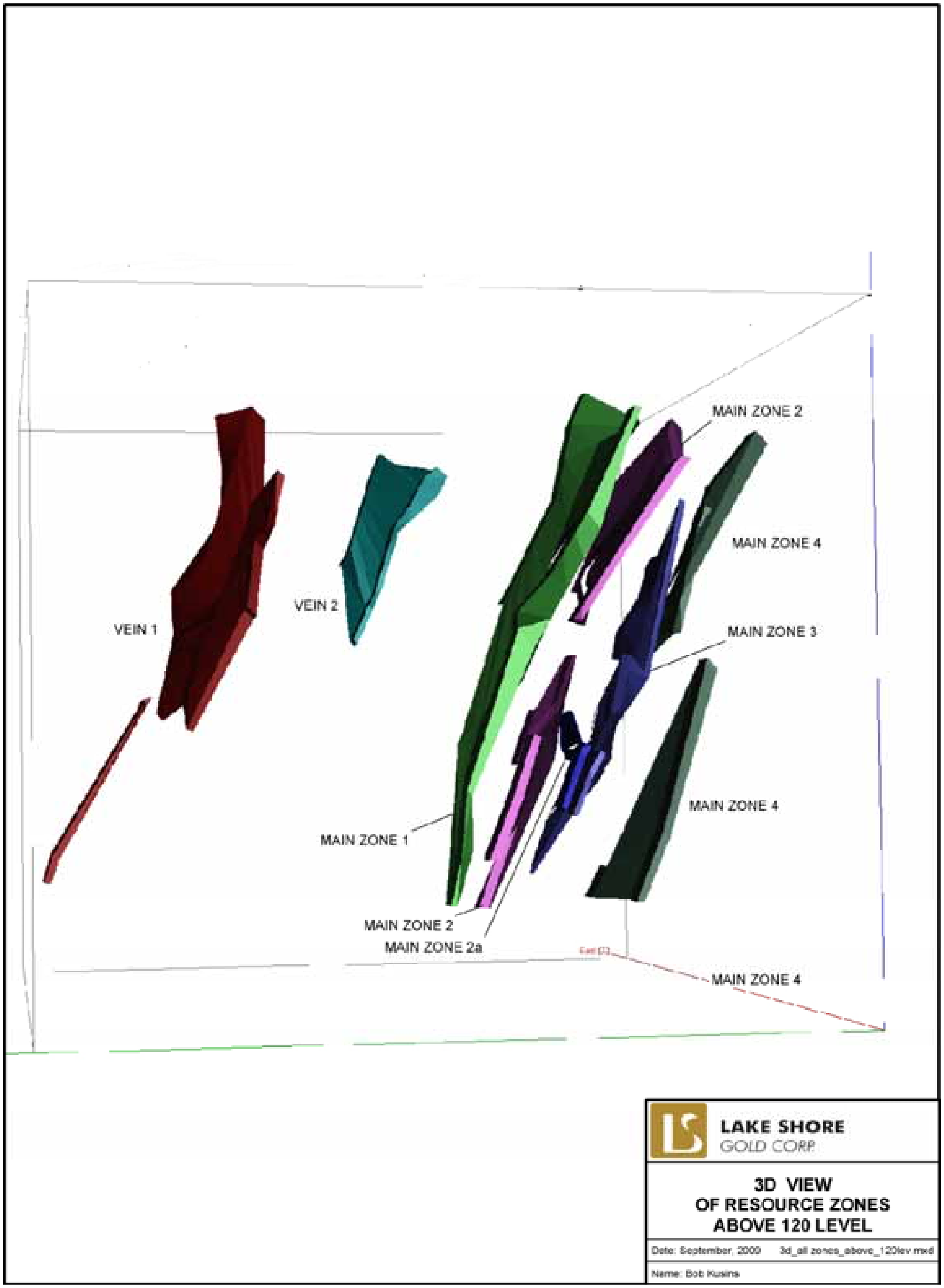

Figure 12: 3D View of Resource Zones Above 120 m Elevation.

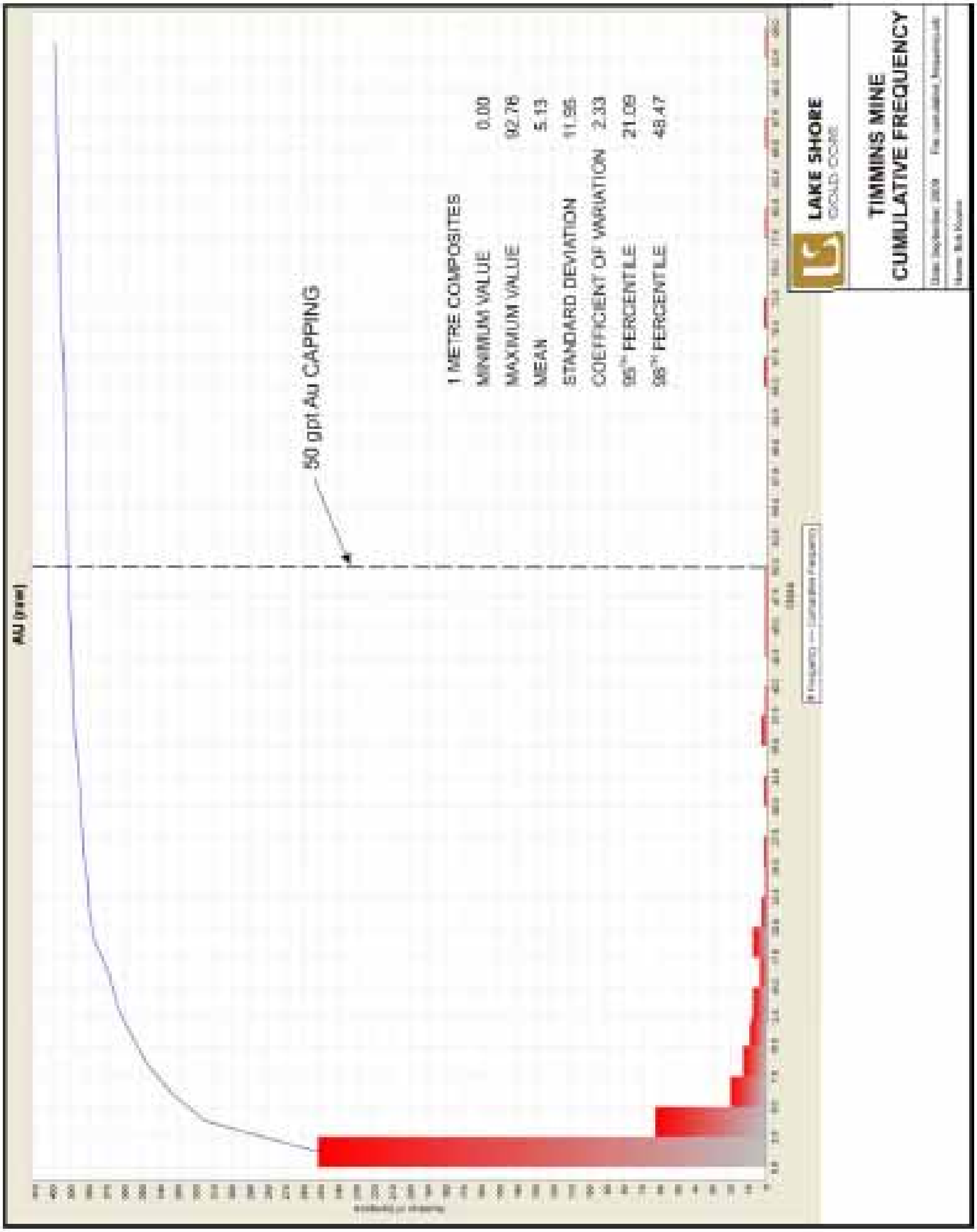

Figure 13: Cumulative Frequency of Composite Values.

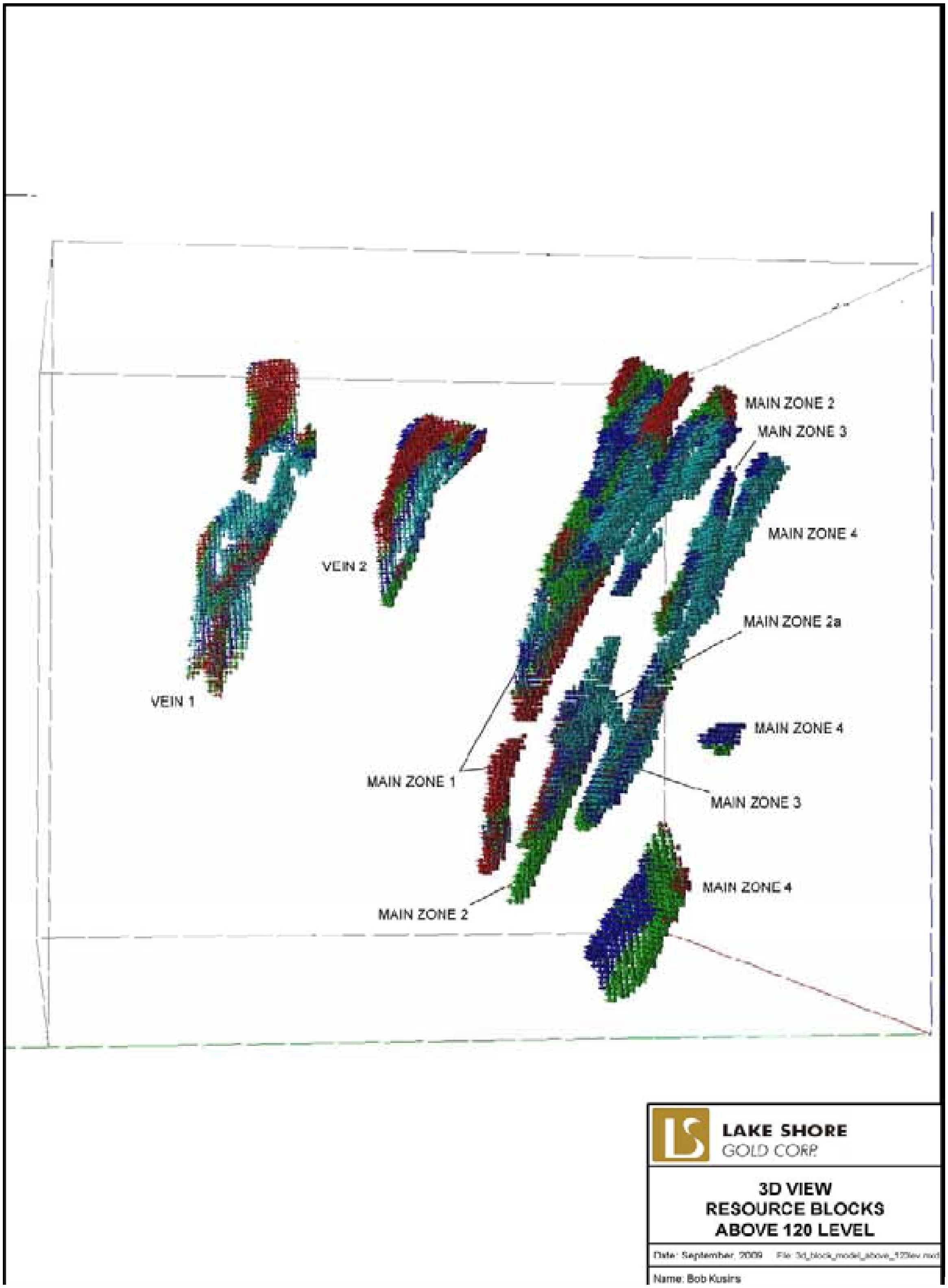

Figure 14: 3D View of Resource Blocks Above 120 Elevation.

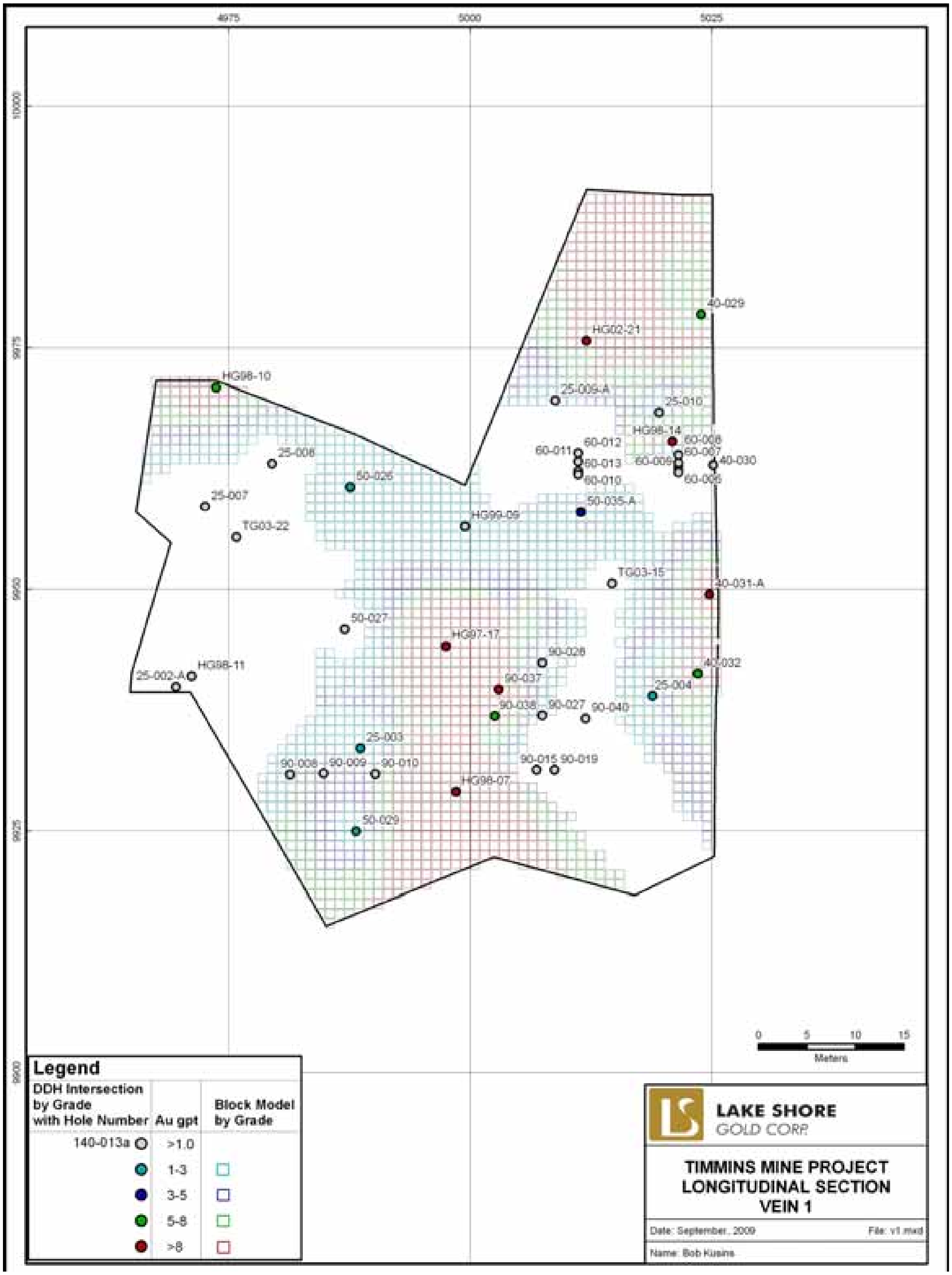

Figure 15: Vein 1 Longitudinal Section Grade Blocks

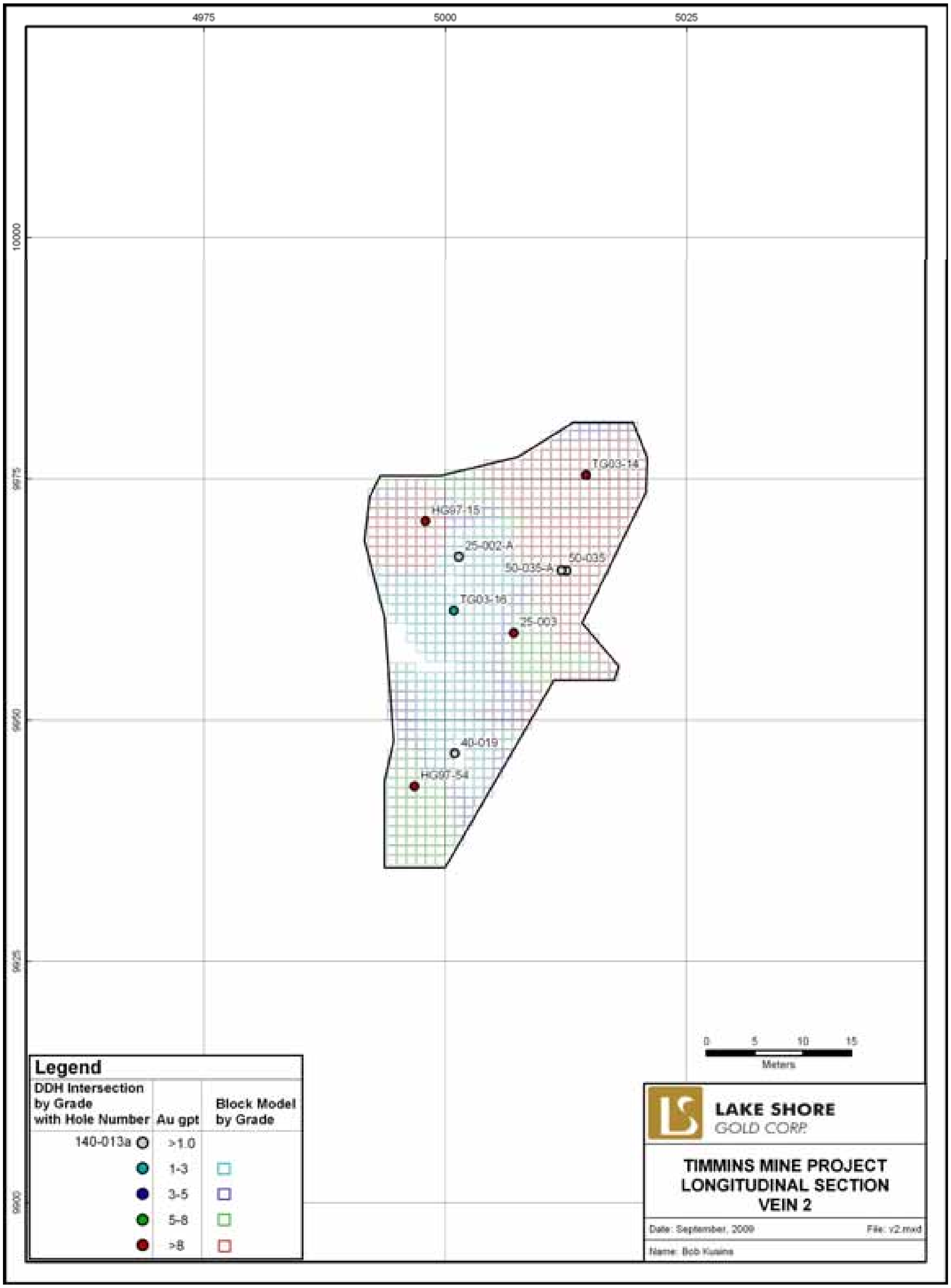

Figure 16: Vein 2 Longitudinal Section Grade Blocks.

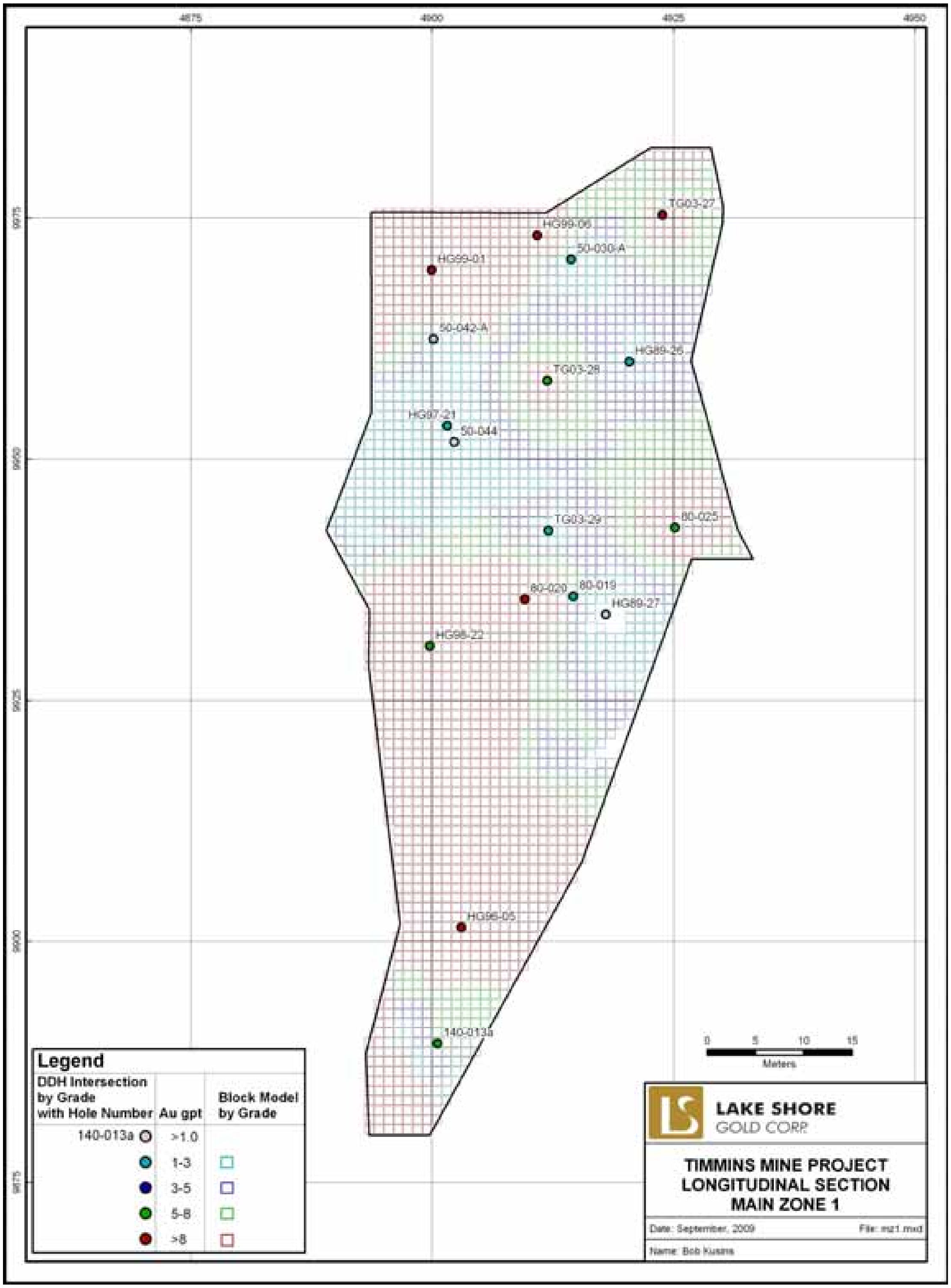

Figure 17: Main Zone 1 Longitudinal Section Grade Blocks.

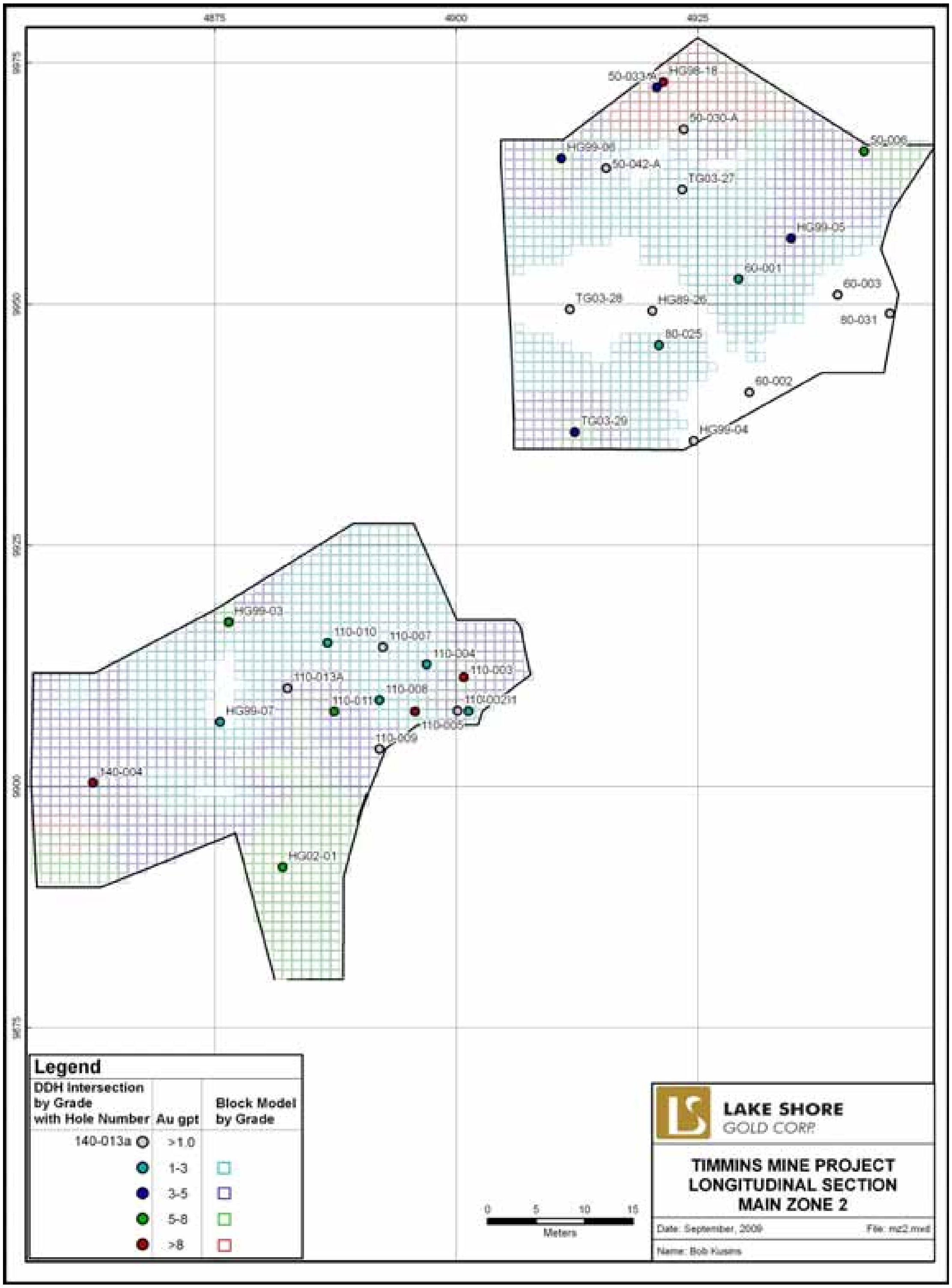

Figure 18: Main Zone 2 Longitudinal Section Grade Blocks

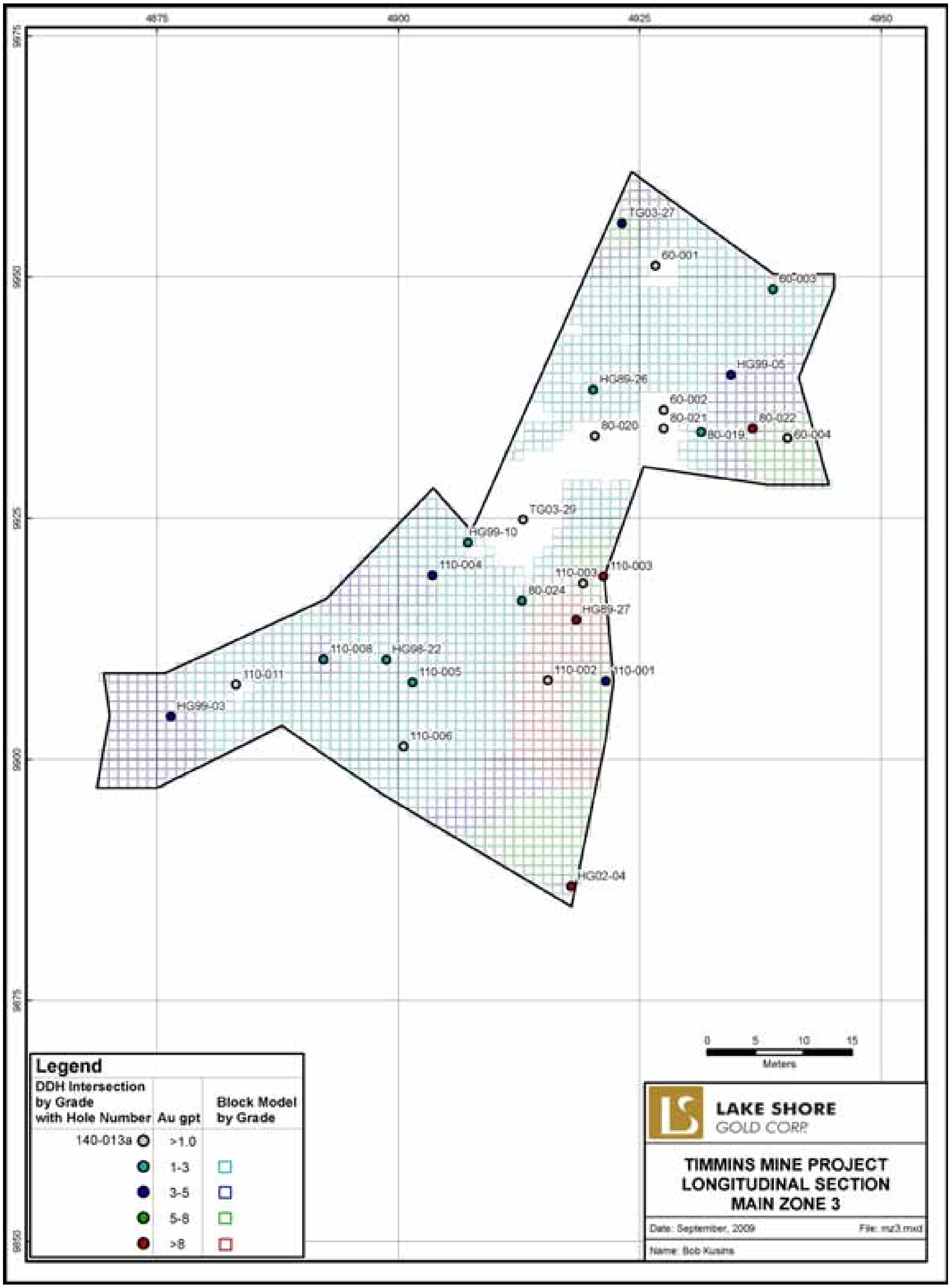

Figure 19: Main Zone 3 Longitudinal Section Grade Blocks.

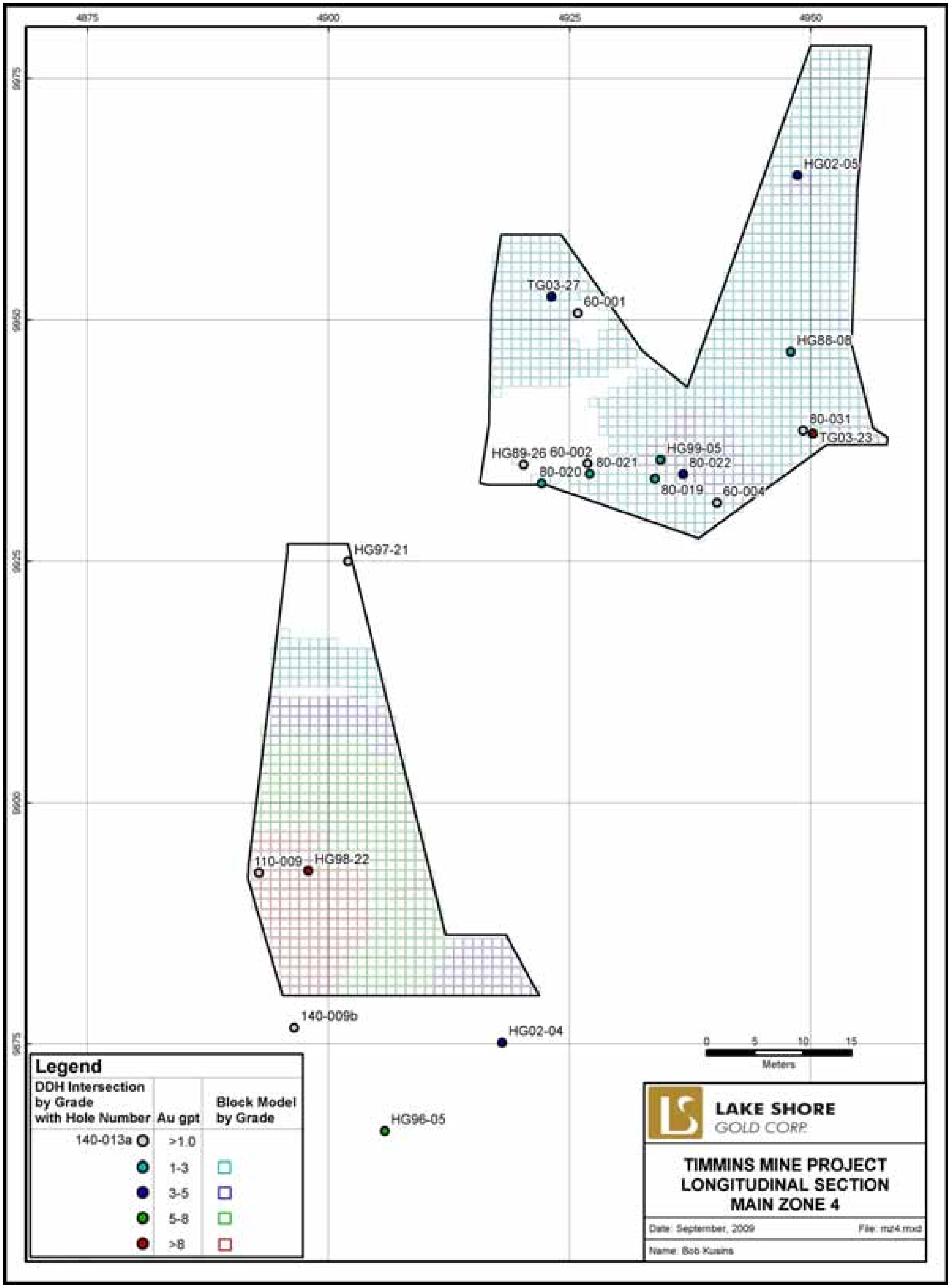

Figure 20: Main Zone 4 Longitudinal Section Grade Blocks.

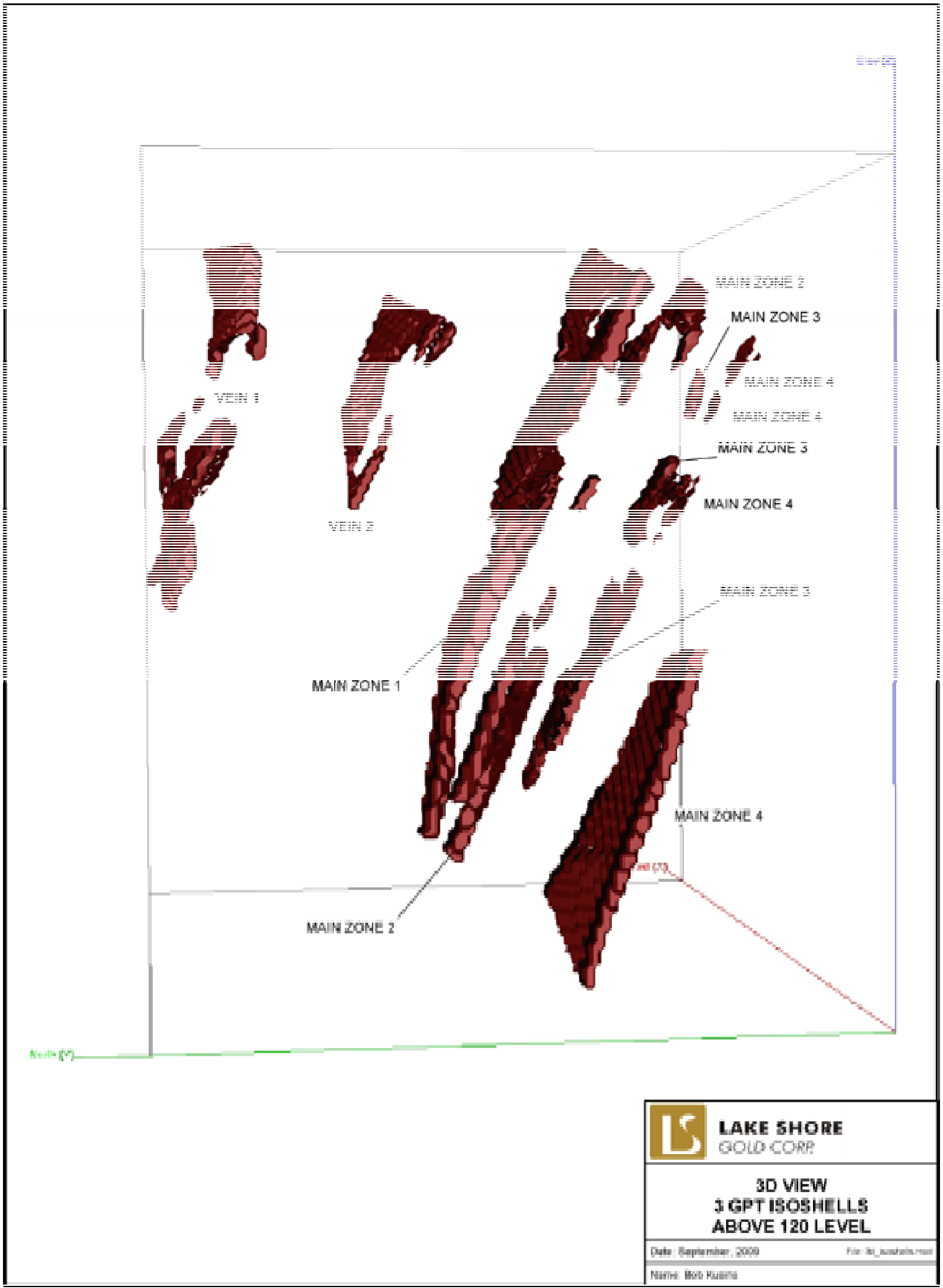

Figure 21: 3D View of 3 gpt Isoshells Above 120 Elevation.

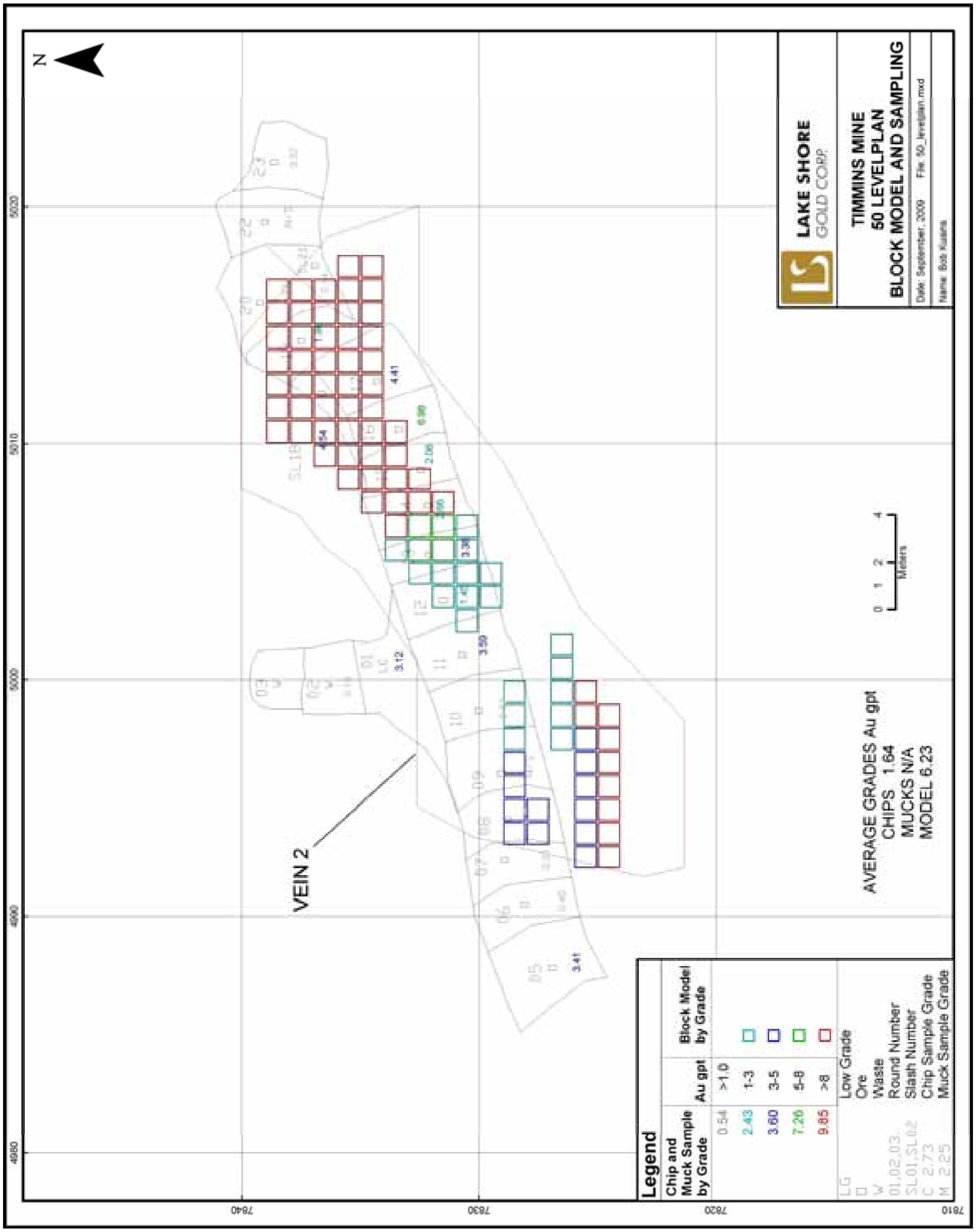

Figure 22: 50m Elevation Vein 2 Grade Blocks and Development.

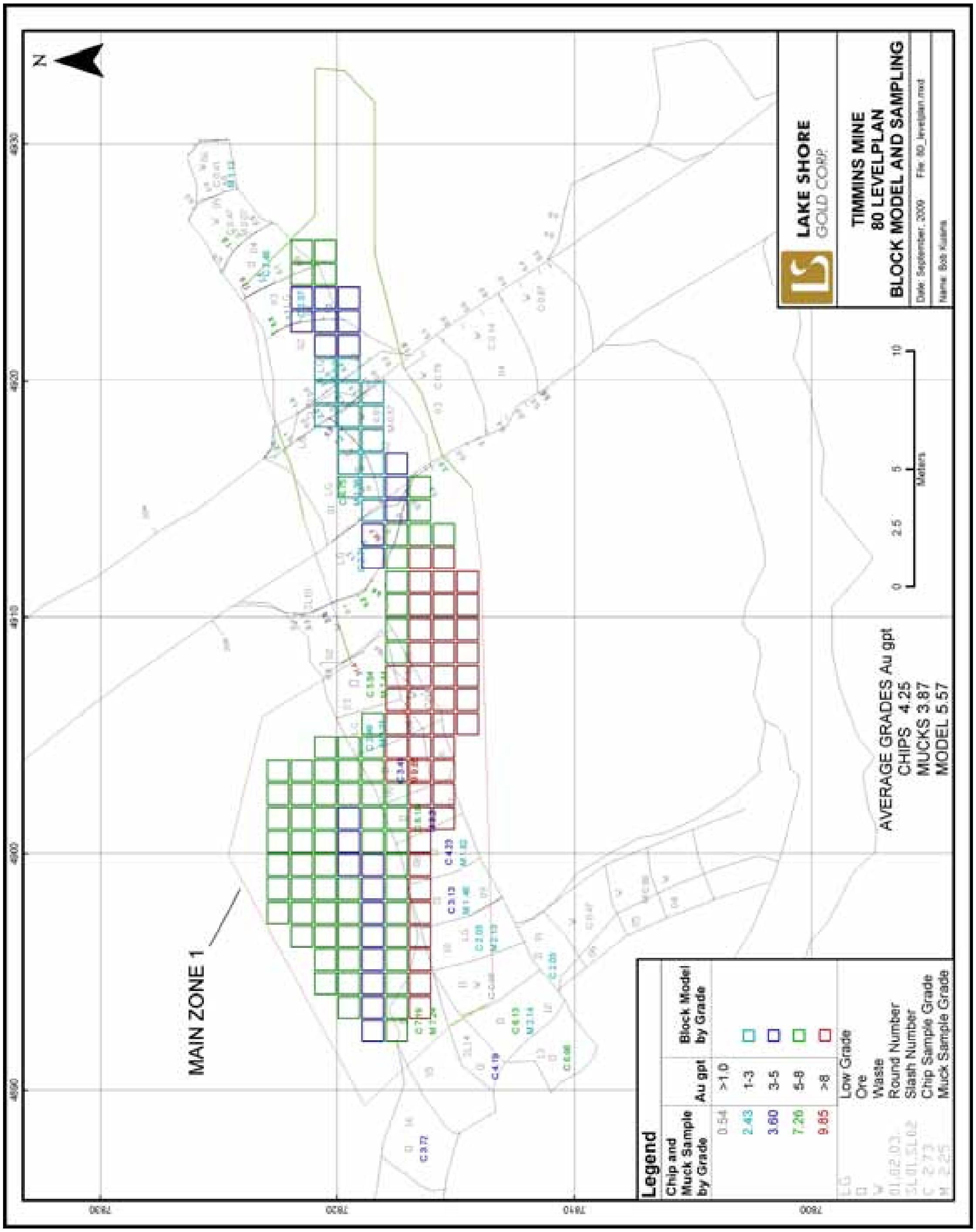

Figure 23: 80m Elevation Main Zone 1 Grade Blocks and Development

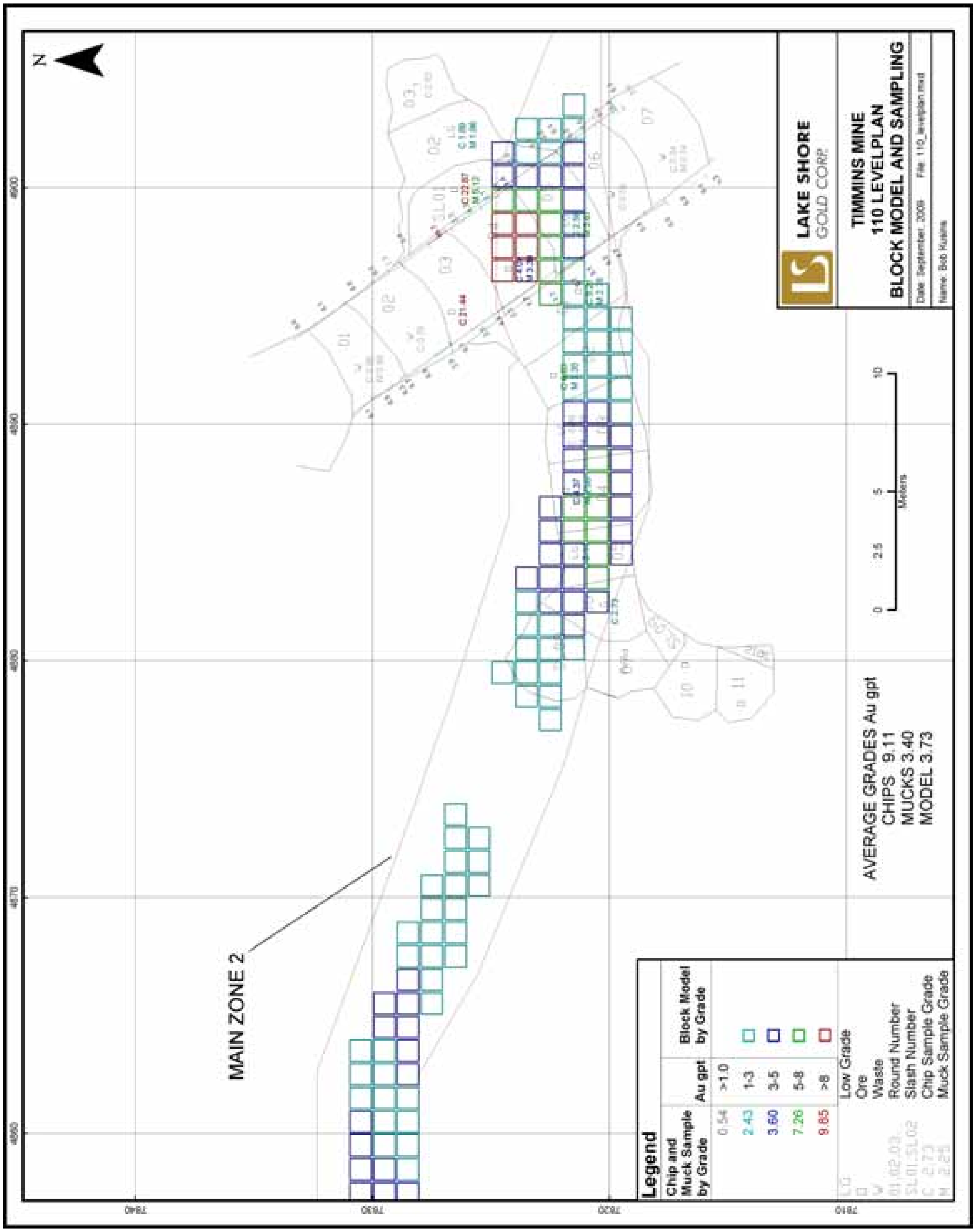

Figure 24: 110 m Elevation Main Zone 2 Grade Blocks and Development

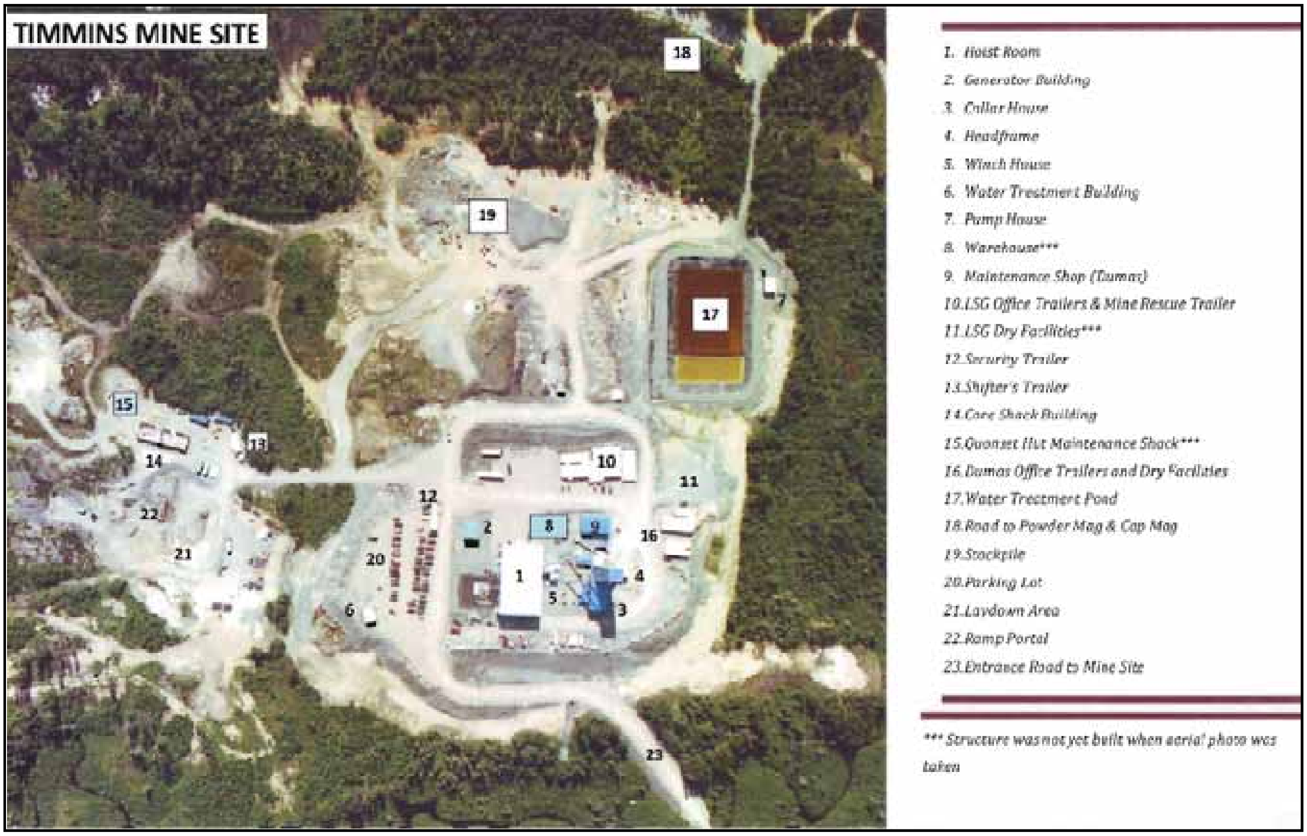

Figure 25: TM Surface facilities for AE

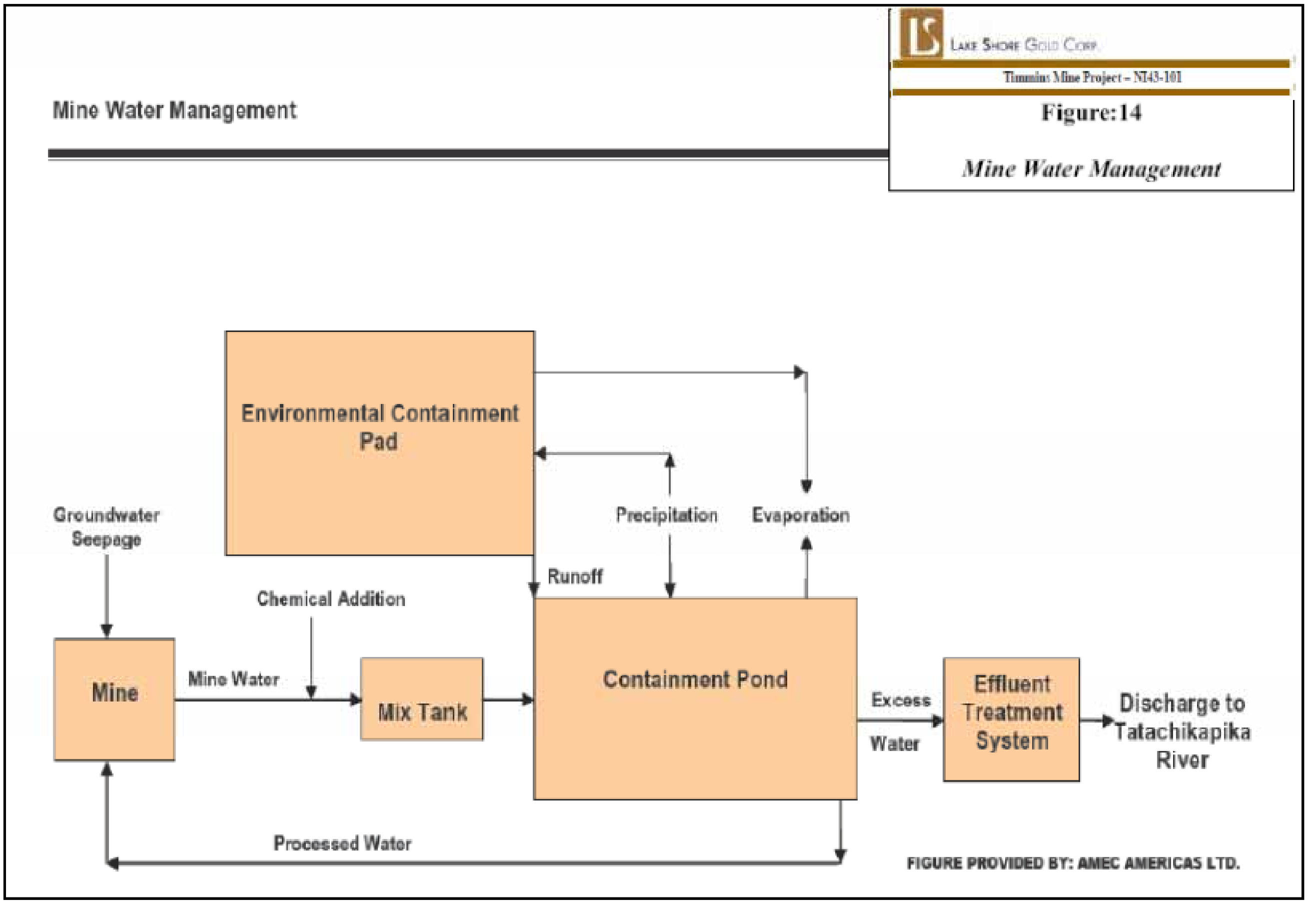

Figure 26: TM Water Management Flowchart

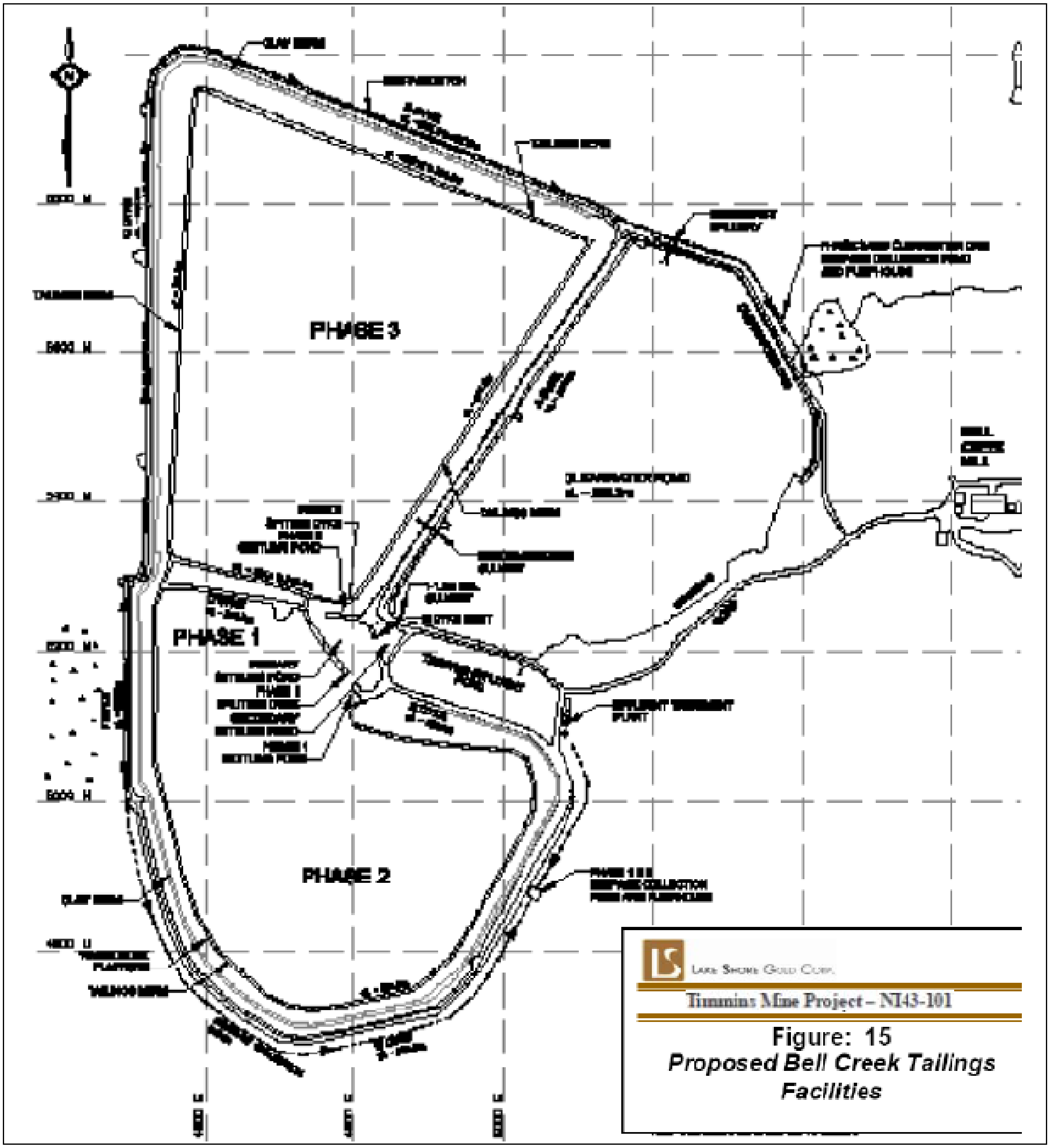

Figure 27: Proposed BC Tailings Facilities

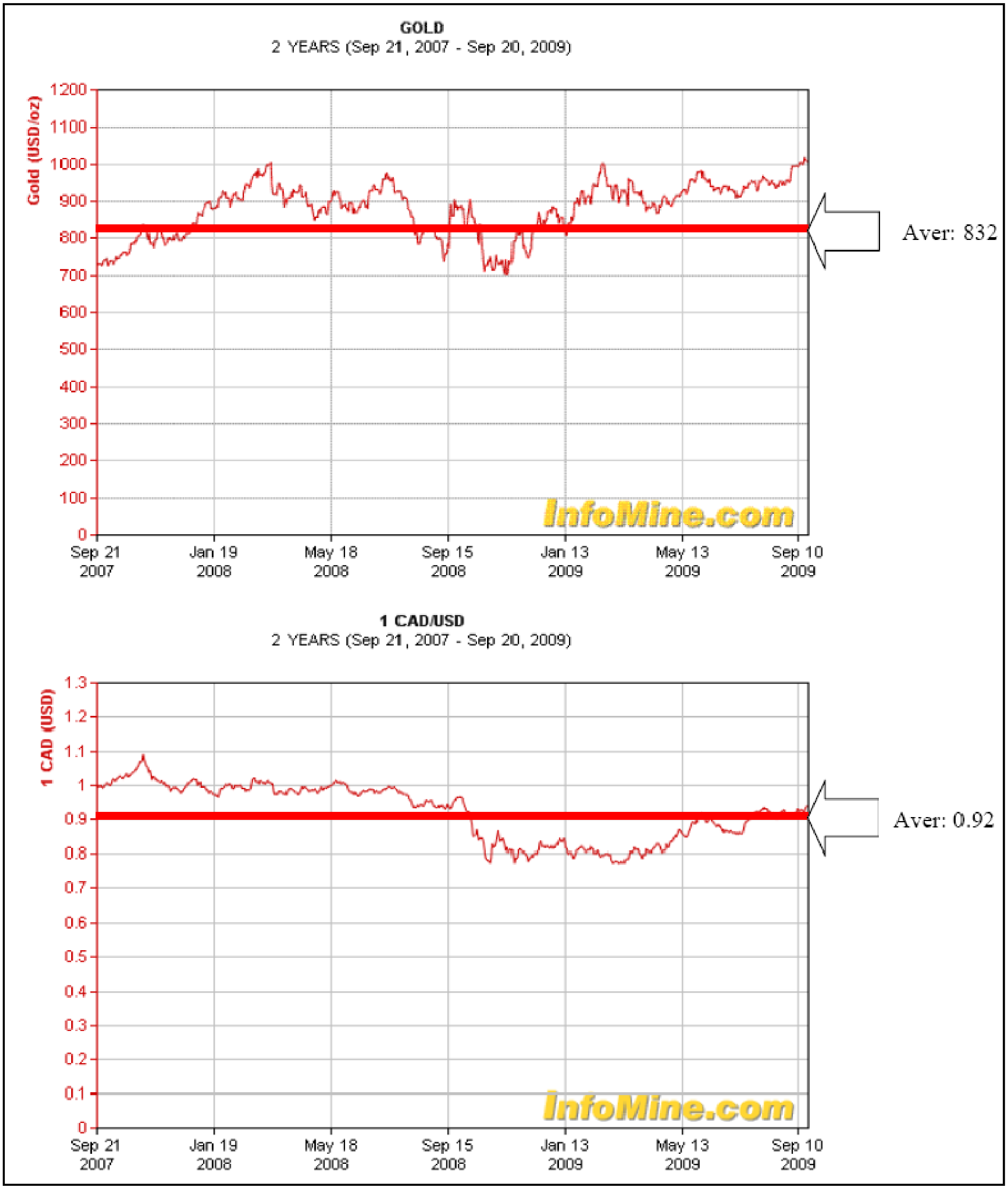

Figure 28: 2 Year $CDN/US Exchange Rate and Gold Prices

Appendix B

Certificates of Qualified Persons

George B. Darling
Stantec Consulting
1760 Regent Street, Sudbury, Ontario P3E 3Z8

Telephone: 705 566 6891
Fax: 705 566 5589
Email: george.darling@stantec.com

CERTIFICATE of AUTHOR

I, George B. Darling, *P.Eng.*, do hereby certify that:

1. I am Principal Engineer of: Stantec Consulting
   1760 Regent Street
   Sudbury, Ontario, Canada P3E 3Z8

2. I graduated with a degree in Mining Engineering from Queen's University in 1976. In addition, I obtained a B.Sc. from Sydney University in Geology and Geophysics in 1970.

3. I am a member of the Professional Engineers of Ontario.

4. I have worked as a Mining Engineer for 33 years since graduation from university.

5. 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 and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of the following sections: Executive Summary (partially with PRS), 1, 2, 3 (partially with WGM and AMEC), 4 (partially with WGM), 16.5.1, 16.6, 16.7, 18.1, 18.2, 18.6, 18.9 (partially with PRS), 18.10 (partially with PRS), 19 (partially with WGM) and 20 (partially with WGM) of the technical report titled "NI 43-1-1 Technical Report, Lake Shore Gold Corp. Timmins Mine, Timmins Ontario", dated October 1, 2009 (the "Technical Report") relating to the Timmins Mine. I visited the Timmins Mine Property on September 11th, 2009 for one day.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-1-1 and Fore 43-1-1F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 1st Day of October 2009

MINING
GEORGE B. DARLING
PROVINCE OF ONTARIO

Stamp

I Todd S. Fayram, QP do hereby certify that:

1. I am a United States citizen residing at 1300 West Copper Street, Butte, Montana, 59701 USA.
2. I graduated with a Bachelor of Science degree in Mineral Processing Engineering from the Montana College of Mineral Sciences and Technology (Montana Tech) in 1984.
3. I am a registered Qualified Person in Metallurgy with the Mining and Metallurgical Society of America (QP# 01300QP).
4. I have experience in my profession since 1984 in the field of mineral processing, metallurgy, project development, and mine operations to include operations general management and engineering in both producing, developing and exploration projects in precious metals, base metals, and industrial minerals. Applicable employment includes Asarco (1980-1984, 1987-1988), Hecla Mining Company (1989-1994), Rea Mining Corp (1994-1996), Wharf Resources (1996-1997), Unifield Engineering (1997-2000), Pasminco Zinc (2000-2003), Consulting (2003 to present) including (Elkhorn Goldfields 2003-2004), Dyantec Mining Services (2007-2008), Lake Shore Gold (2007-2009), Apex Silver (2008), Minefinders Corp (2007-2009), and Getty Copper (2008-2009). I have been involved in all aspects of mine and plant design.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
6. The Technical Report is titled "NI 43-101 Technical Report, Lake Shore Gold Corp., Timmins Project, Timmins, Ontario" dated September 15, 2009 and I was the principal author Section 15, 18.4, 18.10 (Process Costs), and the Processing Section of the Executive Summary. I visited the property on January 19-22, 2009.
7. I have not had any prior involvement with the property or the company that is the subject of the Technical Report.
8. As of the date of this certificate, to the best of my knowledge, information, and belief, the Technical Report contains all the technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of Lake Shore Gold, Corp. applying the tests in Section 1.4 of National Instrument 43-101.
10. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report that I have reviewed have been prepared in compliance with these standards.
11. I consent to the filing of the sections of the Technical Report that I have reviewed with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated September 29, 2009

Todd S. Fayram, MMSA – 01300QP

Robert Kusins P. Geo.
126 Forest Place
Timmins, Ontario P4N 8K1

Tel.: 705-269-4344
Fax: 705-268-1794
E-mail: bkusins@lsgold.com

CERTIFICATE of AUTHOR

I, Robert Kusins, P. Geo., do hereby certify that:

1. I reside at 126 Forest Place, Timmins, Ontario, P4N 8K1.

2. I graduated with a B.Sc. Degree in Geology from McMaster University in 1978.

3. I am a member of the Association of Professional Geoscientists of Ontario (Registration Number 0196).

4. I have worked continuously as a geologist for a total of 31 years since my graduation from university.

5. 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.

6. I am responsible for sections 6 to 11, 13 to 16.6 and parts of section 19 and 20 of the report.

7. I visited the property on September 30, 2009 and on numerous prior occasions.

8. 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.

9. I am currently employed by Lake Shore Gold as Chief Resource Geologist. I currently do not hold any securities of Lake Shore Gold other than options under the Lake Shore Gold's employee stock option plan.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 30th Day of September 2009

R. Kusins, P. Geo., B.Sc.

PROFESSIONAL GEOSCIENTIST
ROBERT KUSINS
PRACTISING MEMBER
0196
ONTARIO

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report titled:
"Updated NI 43-101 Report on the Timmins Mine Property, Ontario, Canada",
dated August 28, 2009

I, Heather Miree, do hereby certify that:

1. I reside at 1820 Mahoney Road North, Timmins, Ontario, P4N 7C3.
2. I am a graduate of the University of Waterloo, Waterloo, Ontario with a B. Sc. (Hons.) in Applied Earth Science (1986).
3. I have practiced my profession continuously since 1986. I have been directly involved in the exploration and production of many mineral commodities, in many areas of Canada and the United States of America, including gold and base metals.
4. I am registered as a Licensee of the Association of Professional Geologists of Ontario (Membership Number 1046) and a Licensee of the Association of Professional Engineers, Geologists and Geophysicists of Alberta (Membership Number 71291).
5. I am employed as Chief Mine Geologist with Lake Shore Gold Corp. at the Timmins Project, which is the subject of this report.
6. I have knowledge and experience of the geology, core logging, sampling, and security practices of the Timmins Project, particularly in reference to the underground portion of the program and have experience in the preparation of technical studies.
7. I have read the definition of "qualified person" set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
8. I commenced work at the Timmins Project under the employ of Lake Shore Gold Corp., commencing in January 2009, supervising the geological aspects of the project including supervision of underground geological activities, core logging, sampling and data compilation since that time.
9. I share responsibility for section 12 with co-author Jacques Samson.
10. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance with that instrument and form.
11. As of the date of this certificate, to the best of my knowledge and belief, this technical report contains all scientific and technical information that is required to be disclosed to make this technical report not misleading.
12. I consent to the filing of this Technical Report with any stock exchange and other regulatory authority and any publication of this Technical Report by them for regulatory purposes,

CERTIFICATE OF QUALIFIED PERSON

Jacques Samson, P.Geo.
Lake Shore Gold Corp.
1515 Government Road
Timmins, ON, P4N 7W7
Tel: (705) 269-4344

I, Jacques Samson, do hereby certify that:

1. I reside at 806 Denise Street, Timmins, Ontario, P4N 7N8.
2. I hold a Bachelor of Science (Honours) Degree in Geology (1986) from the University of Ottawa, Ottawa, Ontario.
3. I have been practising my profession since 1986, and have experience with regards to the planning and supervision of mineral exploration programs.
4. I am a registered practicing member of the Association of Professional Geoscientists of Ontario (APGO member 0421).
5. As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).
6. This certificate applies to the technical report titled "Updated NI 43-101 Technical Report on the Timmins Mine Property, Ontario, Canada, dated August 28 2009.
7. I am jointly responsible for section 12 of the report with co-author Heather Miree.
8. I am currently employed by Lake Shore Gold Corp. As Project Geologist, I have been directly involved in the supervision of the surface diamond drilling program carried out by the company since 2003 on the Timmins Mine Property.
9. 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.
10. I have read National Instrument 43–101 and Form 43-101, and the technical Report has been prepared in compliance with that instrument and form.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated 30th day of September 2009

Jacques Samson, P.Geo.

PROFESSIONAL GEOSCIENTIST
JACQUES SAMSON
PRACTISING MEMBER
0421
ONTARIO