REPORT OF PROPERTY REVIEW AND SAMPLING PROJECT
TREASURE MOUNTAIN PROPERTY
TULAMEEN RIVER AREA, B.C.
CANADA

NTS 92H/6E
UTM NAD83 Zone 10, 641000E, 5476000N
Lat. 49°25'00"N, Long. 121°03'40"W

Report Prepared for Huldra Silver Inc.
3475 West 34th Avenue,
Vancouver, B.C., V6N 2K5.

Report Prepared by Erik Ostensoe, P. Geo.
4306 West 3rd Avenue,
Vancouver, B.C., V6R 1M7.

and

Farshad Shirvani, M.Sc.
1405 - 675 West Hastings Street,
Vancouver, B.C., V6B 1N2.

Date of Field Work: July 13th to 19th, 2007

Date of Report: July 30th, 2008

30429
Vol. 2

HULDRA SILVER INC.

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PROFESSIONAL PROVINCE OF E. A. OSTENSOE BRITISH COLUMBIA GEOSCIENTIST

and

Farshad Shirvani, M.Sc.
1405 - 675 West Hastings Street,
Vancouver, B. C., V6B 1N3.

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SUMMARY

Huldra Silver Inc. recently entered the Draft Permit Application process stage with the Province of British Columbia's Ministry of Energy and Mines for the development and operation of an underground silver-lead-zinc mine at Treasure Mountain, 29 km northeast of Hope, B. C. The historic mine has been explored and developed in several episodes since 1892 and now comprises an 1850 hectare tenured area with underground workings over a 295 metre vertical distance. Two small mills were built during the period 1930 to 1956 but production was very limited. Huldra profitably shipped 407 tonnes of raw "ore" to smelters in 1987 and reclaimed and expanded historic workings, completed technical surveys and built a data base of survey and sampling information. Progress toward entering a production stage was halted in 1989 due to difficult metal and financial market conditions.

A resource calculation prepared in 1989 by Livgard Consultants Ltd. reported a **non-National Instrument 43-101 compliant** resource of 61,635 tonnes "proven" with 27.96 oz./tonne silver, 4.53% lead and 5.29% zinc, and 71,402 tonnes "probable" with 28.14 oz/tonne silver, 4.11% lead and 4.71% zinc. A "possible" resource of 148,000 tonnes was not given a metal content estimate.

Despite the challenge of low metal prices and depressed market conditions that prevailed during the 1990s and until 2006, the property owner, Huldra Silver Inc., maintained its Treasure Mountain property and periodically completed limited exploration programs in the vicinity of the mine. Management took the opportunity to bring property maps, particularly mine plans, up to date and conducted small programs of rotary drilling to investigate geologically enigmatic parts of the property where evidence of significant mineralization had been reported. In summer, 2007, the mine workings were re-entered on two levels and a limited amount of check sampling was done in order to either verify and bring to NI 43-101 compliant standards the 1989 Livgard resource calculations or to enable a re-calculation of resources. Seventy-eight chip samples were analysed by induced coupled plasma analytical methods for 30 elements. Ten samples were then re-analysed as a means of checking the accuracy of the laboratory, which proved to be satisfactory. Samples that reported high silver, lead and/or zinc values were then assayed in order to obtain more precise values for those elements.

Silver analyses, when the 1988-era samples were compared to 2007 samples, were found to be somewhat disparate, with differences as great as 100% and more. Several possible explanations have been identified.

A resource calculation compliant with National Instrument 43-101 and Form 101F was prepared on the basis of digitized versions of the Treasure Mountain survey and assay data. That data was tested to ensure that it is of acceptable quality for purposes of resource calculations and was judged to be useable. The unsatisfactory correlation of historic and recent (2007) silver analyses precluded categorizing any resources as "measured".

Mine permitting and planning initiatives are currently in progress with the objective of mining and processing much of the identified resource. Conceptually, a seasonal, eight-month, mine and mill operation will operate at a nominal rate of 135 tonnes daily. A mine plan developed for Huldra by A. J. Beaton Mining Ltd. proposes extracting the vein

by conventional small mine methods: working from raises, miners will deliver broken ore to draw points, initially on Level 3 or on a yet to be driven level intermediate between present Levels 3 and 4, and, using track haulage, move it to surface and then by truck to the mill which will be located east of the portal of Level 4.

The mill cycle will include crushing, milling (ball mill), flotation and production of lead and zinc concentrates that will be marketed. The bulk of the silver will report to the lead concentrate.

Entech Environmental Consultants Ltd. has prepared an Environmental Impact Assessment of the Tulameen River Drainage Basin and has identified certain topics that will need special attention when proposed mining plans are implemented. Further studies and on-going monitoring requirements were discussed but no serious impediments to mine development were identified.

Terracad Ltd. using CIM Definition Standards for Mineral Resources and Mineral Reserves and all available assay and survey data, conducted modeling studies and identified a NI-43-101 compliant resource that occurs in narrow, sharply defined veins. Following the advice of the consulting mining engineer, the resource was diluted for practical purposes to a 1.2 metre mining width. The resource comprises, in the hangingwall domain 89,105 tonnes in the "indicated" and "inferred" categories above a calculated cut-off grade of 8.46 oz./tonne silver,1.59% lead and 1.38% zinc and in the footwall domain, 17,478 tonnes in the inferred category above a calculated cut-off grade of 15.05 oz/tonne silver, 0.29% lead and 4.33% zinc.

[Note that these estimates are more fully discussed with cautionary language in the main body of this report.]

The company's consulting mining engineer, a proficient operator of narrow vein mines, forecast in 2006 that total operating costs will be about $150/ton (equivalent to $165/tonne) and has more recently confirmed that figure (A. J. Beaton, P. Eng., 2008, pers. comm.). That figure will be addressed in the Economic Evaluation. This report includes resource estimates on the basis of the following gross value cut-offs: >$150, >$165, >$180 and >$200.

The metal values assigned to the resource estimate are provisional to the extent that the 2007 program of re-sampling on Levels 1 and 2 of the mine returned silver, lead and zinc values that are at variance with earlier values. Silver is more at variance than are lead and zinc. Much of the difference in silver values could conceivably be ascribed to the erratic presence of native silver and silver-rich sulphosalt minerals that preclude duplication of sample values but other factors are undoubtedly at work. Randomness (i.e. "nugget effect") is unlikely to result in the observed variations and further sampling, with elaborate quality control measures, will have to be completed in order to more precisely determine the silver content. The resources that have been reported are sufficient in tonnage and apparent grade to proceed to an Economic Assessment that will, in turn, indicate whether a small underground mine is viable.

PROFESSIONAL
PROVINCE OF
E. A. OSTENSOE
BRITISH COLUMBIA
GEOSCIENTIST

Erik A. Ostensoe

1.0 INTRODUCTION

1.1 Introduction

The Treasure Mountain silver-zinc-lead-copper deposit, located 29 km east of Hope, British Columbia, Canada, (Figures 1, 2 and 3) was discovered in 1892 and has been the site of several episodes of exploration by means of surface and underground workings. Two mills have been constructed on the property but at present there are no facilities. The current owners have made shipments of high grade ores and there is a small stockpile of broken ore situated near the lowermost mine entrance.

Huldra Silver Inc., owner of the property since 1980, conducted prospecting programs on Treasure Mountain and discovered, or, possibly, re-discovered, and outlined in 1985 the surface expression of the "C" vein. That vein became the main focus of attention and was determined to be the same structure, up-dip from the mineralization on Level 1, an historic working. The company completed major programs of work on the property in the period 1987 to 1989: old mine workings from the 1910 - 1950s era on two levels were re-entered and extended and two new levels, several raises and a sub-level were established. All parts of the mine were sampled or re-sampled and a number of engineering and environmental studies were undertaken. The mine, due principally to low silver prices that prevailed, was mostly idle from 1989 through July, 2007, at which time parts of the mine were accessed for purposes of re-sampling certain sections of the main vein. Re-sampling was deemed necessary in order to verify the metal values calculated in 1989 by Livgard Consultants Ltd. for the "C" vein mineralization: resource calculations from surface to Level 4 were prepared before introduction of National Instrument 43-101 and related policies and for that reason were not acceptable without being redefined in terms of NI 43-101.

Work at the mine property commenced in mid-July 2007 under the supervision of the District Inspector of Mines, a licensed Mine Manager and a licensed Shift Boss: roads were cleared of debris and rockfalls, and portals were re-opened, Levels 1 was drained, and underground workings on Levels 1 and 2 were inspected for safety considerations. Particular attention was directed to air quality, and to portal areas where shoring and other timbers had deteriorated due to the passage of time and the influence of weather. The District Inspector of Mines, in consultation with the owner's personnel, refused entry to Levels 3 and 4 due to a number of concerns about loose rock, failing timbers and possible "bad" air conditions, and directed that those levels be rendered temporarily inaccessible. Overhanging brows and timbered entrances to those levels have deteriorated and air quality is uncertain but there is no suggestion that rock conditions in the internal mine workings are unsafe.

The writer, a consulting geologist, was engaged by Huldra Silver Inc. to conduct a property review and to design and execute a small program of underground sampling of part of the principal Treasure Mountain mineral zone, the "C" vein, and to prepare a NI 43-101-compliant technical report. Sample values obtained from previous sampling in the period 1987 to 1988, were to be compared with the newly generated assay data. Accordingly, from July 13 to July 18, 2007 inclusive, the writer with a small sampling crew prepared 78 chip samples, from Level 1 and Level 2 of the mine. He participated in the entire sampling program, took charge of all samples immediately upon creation and, at the end of the project, conveyed the samples to an accredited analytical laboratory.

Pulps from those samples were then used in a program of metallurgical testing for use in mill design studies.

Major element analyses from the July, 2007 sampling have been compared to their counterparts from the 1987-9 period of work. Farshad Shirvani, M. Sc., a principal of Terracad Ltd., conducted modeling studies using current standards and methods to determine a resource calculation based on data, old and new, from all parts of the mine, for use in an Economic Analysis. Such an analysis is a requirement and prerequisite to obtaining a permit to operate a small underground mine at Treasure Mountain.

1.2 Terms of Reference

This report was prepared for Huldra Silver Inc., a British Columbia corporation, in order to review the status of the Treasure Mountain property and to qualify certain analytical data from its underground and open pit mine site to a standard consistent with current guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum.

The CIMM Guidelines, adopted by CIM Standing Committee on Reserve Definitions, December 11, 2005, require that mineral reserve and mineral resource estimates be assigned to categories that reflect the level of confidence implicit in those categories. That level is a function of the geological information available, the quantity and quality of data available and an interpretation of that data and information. The authors of this report have referred to resource estimates that were prepared immediately upon completion of the most recent (1988) underground work. That work was supervised by E. Livgard, P. Eng., a not-at-arms length Qualified Person (as currently defined) with extensive experience in underground operations, both at producing mines and at exploration stage properties, who also prepared the resource estimates.

1.3 Reliance on Other Experts

The writer, in compiling the data presented in this report, was aided by Magnus Bratlien, President of Huldra Silver Inc., who has been intimately involved with all aspects of work on the Treasure Mountain mineral deposit since 1979. In addition to providing guidance and assistance in the field and underground, he made available historic base maps and original versions of assay certificates and technical reports that were essential to the project. A copy of a plan of underground workings dated July, 1952 and attributed to "F.W.H." and Silver Hill Mines Ltd., that is in Mr. Bratlien's possession provided the only available information concerning the Jensen tunnel: that information has been referred to in the text, along with cautionary comments.

Egil Livgard, P. Eng., consulting geologist, supervised all of the geological work related to Huldra Silver's work at Treasure Mountain, from the early days of that company's involvement until the present. Although he is considered "not-at-arms-length", he was generous with his help and advice and offered useful suggestions concerning treatment of both the historic and recently-acquired data.

A. J. Beaton, P. Eng., consulting mining engineer, prepared an evaluation report that is quoted in Section 1.6, History, of this report. That report, dated 2006, may not be current in terms of costs and other forecasts and is not a feasibility study, nor is it an Economic Evaluation as is required under current National Instrument 43-101 rules for reporting reserves and resources; Mr. Beaton, however, has confirmed recently that his projected operating cost of "about $150 per ton" (i.e. $165 per metric tonne) is still a valid figure (Beaton, 2008, personal comm.).

The author of this report is not qualified to comment on mine planning, construction scheduling and cost forecasting issues that are dealt with in the Beaton report and disclaims responsibility for items that are quoted from that source.

Jasman Yee & Associates Inc., consulting metallurgists, of Burnaby, B.C., in the past provided test work and advice to Huldra Silver Inc. and in 2006 prepared a complete conceptual flow sheet for a 150 tpd processing plant. Using material from the 2007 program of sampling, JYA conducted further tests and with few adjustments prepared a revised mill flow sheet that includes crushing, grinding, flotation and concentrate dewatering. Information from the JYA report dated February 15, 2008 has been included in this report and the flowsheet is reproduced.

The author of this report is not qualified to comment on metallurgical processes and disclaims responsibility for items that are taken from the JYA report.

McElhanney Consulting Services Ltd., a surveying, mapping and engineering company, digitized survey and analytical data locations that were essential parts of the modeling exercises.

Environmental assessments by Entech Environmental Consultants Ltd. were reviewed and although such studies are continuing, the topics are outside of the substance of this report. Apart from identification of possible avalanche hazards, few areas of environmental concern were disclosed.

Terracad Ltd., a provider of computer-based technical and graphic services to the mineral industry in British Columbia and elsewhere, was engaged by Huldra Silver Inc. to conduct modeling studies based on available survey and analytical data. Farshad Shirvani, M. Sc., the principal of that company, is almost entirely responsible for modeling the graphic representations of the data and for compiling the resource estimates that are included in this report. Mr. Shirvani, who is a co-author of this report, is not a Qualified Person as defined by National Instrument 43-101 but he is fully conversant with most computer-aided methods of modeling and measuring mineral deposits and, with guidance from senior mineral industry personnel, has conducted several resource studies. Mr. Shirvani is in the approval process of becoming registered with the Association of Professional Engineers and Geoscientists of British Columbia and a statement of his qualifications is included in this report.

Metal values quoted in this report were derived from assay and analytical data reported by Min-En Laboratories Ltd. until 1989 and international Plasma Labs Ltd. in 2007 and 2008. Original certificates were available for almost the entire body of data. Certain data has been transcribed from the certificates into computer-accessible files by McElhanney Consulting Services Ltd. and Terracad Ltd. personnel and then into modeling studies and resource calculations. International Plasma Labs Ltd. performed ICP-MS analyses of all the 2007 samples, cross-checked a number of silver analyses by fire assay and gravimetric methods and in view of disparities between the ICP-MS and FA/gravimetric determinations, analysed all samples with 500 ppm or greater silver content by the latter method. Samples that indicated lead and zinc values greater than 1% (10,000 ppm) by ICP-MS methods were assayed.

The figures for resources presented in this report are estimates and no assurance can be given that the anticipated level of grades of resources will be realized. Factors that may influence the realization of the anticipated grades include geological complexity, mining practicalities and mill performance. Due to mining requirements, small variances both positive and negative must be anticipated in the short term.

1.4 Property Description and Location

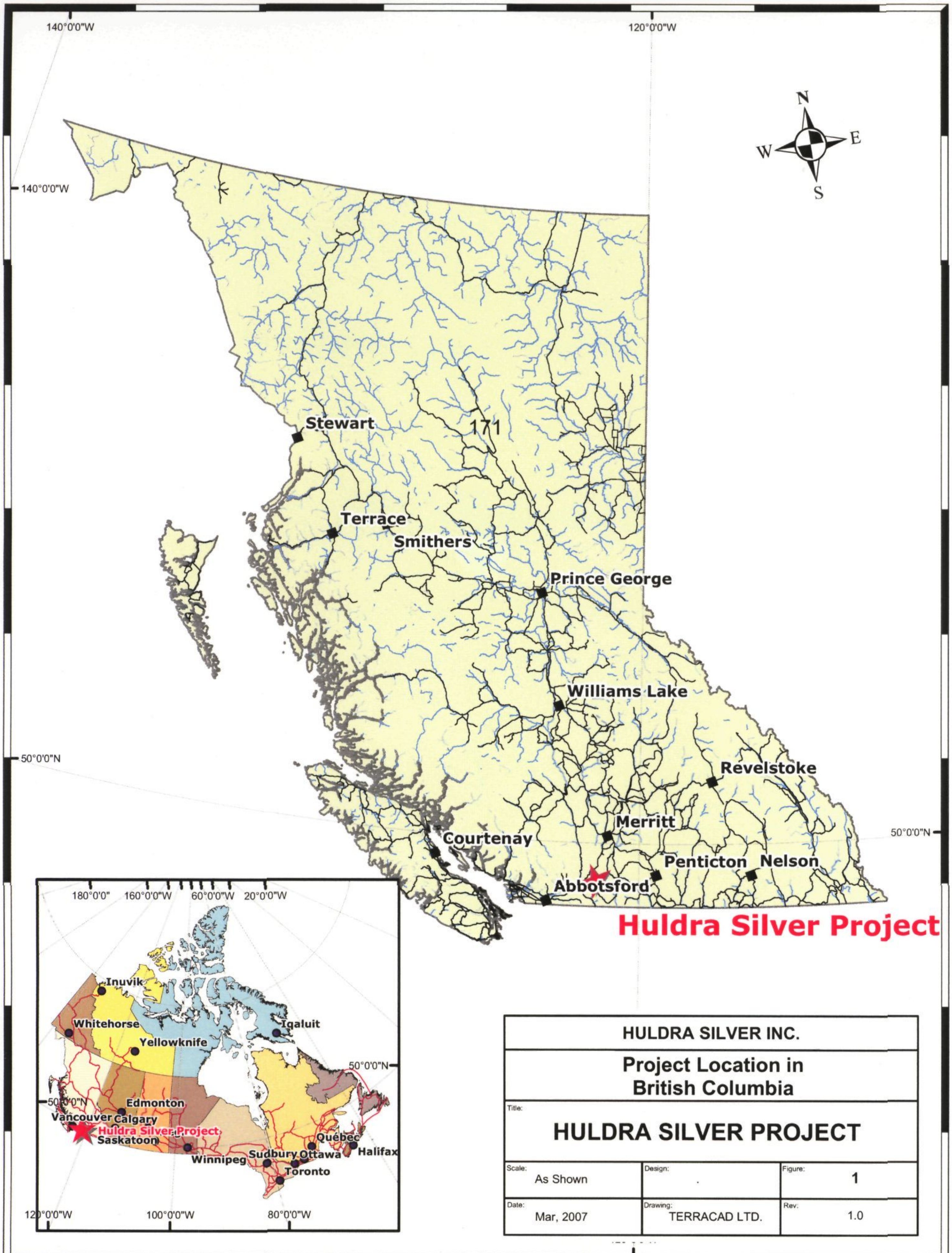
The Treasure Mountain mine property, owned in its entirety by Huldra Silver Inc., is situated in Similkameen Mining Division, British Columbia, and is approximately centered at UTM Zone 10, 641000 East, 4786000 North. Conventional geographic location is in NTS sheet 92H at latitude 49°25'00"N, longitude 121°03'20"W (Figures 1 - 3). The property comprises 51 mineral tenures with total area approximately 2,851.61 hectares (7,046.48 acres) and is configured approximately as shown in Figure 3 of this report. Table 1 is a complete list of the various tenures, all of which are in good standing until 2009 or later dates. A lease survey by McElhanney Consulting Services Ltd. of mineral tenures covering the Treasure Mountain vein deposit has been approved by the Surveyor General.

Surface rights to small parts of the mineral tenures are held by unrelated parties that have occupied the land for recreational and logging purposes. None of the Huldra Silver Inc. workings or proposed facilities are located in those areas.

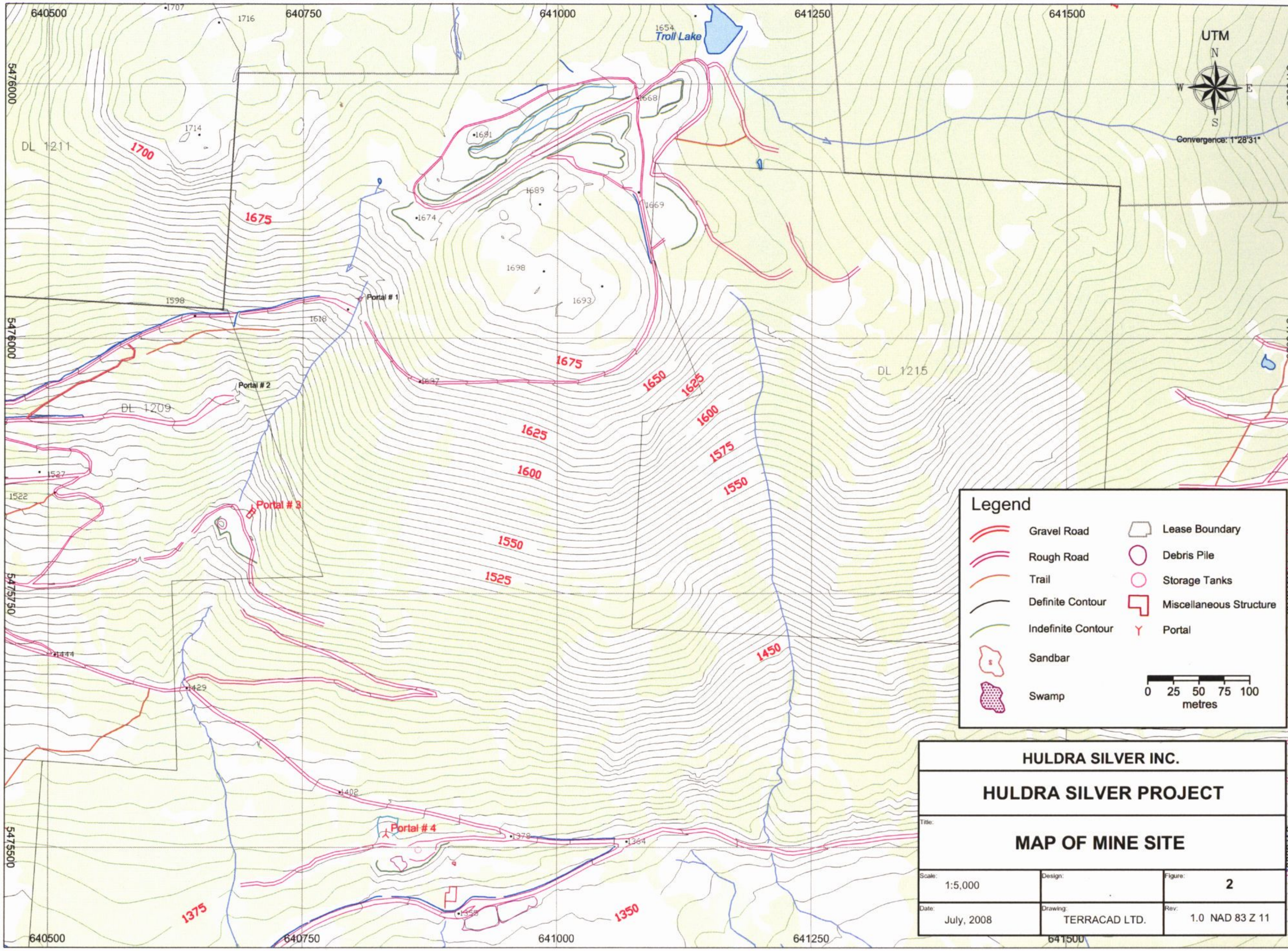
The Treasure Mountain mine is located in the Amberty Creek drainage of Vuich Creek, a tributary of Tulameen River, and is 34 km southwest of the village of Tulameen, British Columbia (Figures 1 - 3). Mine workings extend from 1382 to 1670 metres elevation a.s.l. and comprise four levels, numerous raises, some of which provide ventilation and also access between levels, and an open cut located near the top of Treasure Mountain, approximately 50 metres higher than Level 1 of the mine. There are no permanent structures on the property apart from the underground workings.

A small program of sampling was completed in July, 2007 in Levels 1 and 2 of the mine. Upon completion, in accordance with safety standards and at the request of the District Inspector of Mines, all mine access areas were equipped with drainages to avoid undue flooding of the workings with attendant hazards, and also were securely timbered to prevent access.

Treasure Mountain lies in the traditional territory of the Upper Similkameen Indian Band. The Band's Archaeology Department has undertaken a Preliminary Archaeological



140°0'0"W
120°0'0"W
140°0'0"W
50°0'0"N
50°0'0"N
N
S
E
W
Stewart
171
Terrace
Smithers
Prince George
Williams Lake
Revelstoke
Merritt
Courtenay
Penticton
Nelson
Abbotsford
Huldra Silver Project
180°0'0"
160°0'0"W
60°0'0"W
20°0'0"W
Inuvik
Whitehorse
Yellowknife
Iqaluit
50°0'0"N
Edmonton
Vancouver
Calgary
Huldra Silver Project
Saskatoon
Winnipeg
Sudbury
Ottawa
Québec
Halifax
Toronto
120°0'0"W
100°0'0"W
80°0'0"W
HULDRA SILVER INC.
Project Location in British Columbia
Title:
HULDRA SILVER PROJECT
Scale: As Shown
Design:
Figure: 1
Date: Mar, 2007
Drawing: TERRACAD LTD.
Rev: 1.0



HULDRA SILVER INC.
HULDRA SILVER PROJECT
Title:
MAP OF MINE SITE
Scale: 1:5,000
Design:
Figure: 2
Date: July, 2008
Drawing: TERRACAD LTD.
Rev: 1.0 NAD 83 Z 11
Legend
Gravel Road
Rough Road
Trail
Definite Contour
Indefinite Contour
Sandbar
Swamp
Lease Boundary
Debris Pile
Storage Tanks
Miscellaneous Structure
Portal
0 25 50 75 100
metres
UTM
N
S
E
W
Convergence: 1°28'31"
Troll Lake
Portal # 1
Portal # 2
Portal # 3
Portal # 4
DL 1211
DL 1209
DL 1215
640500
640750
641000
641250
641500
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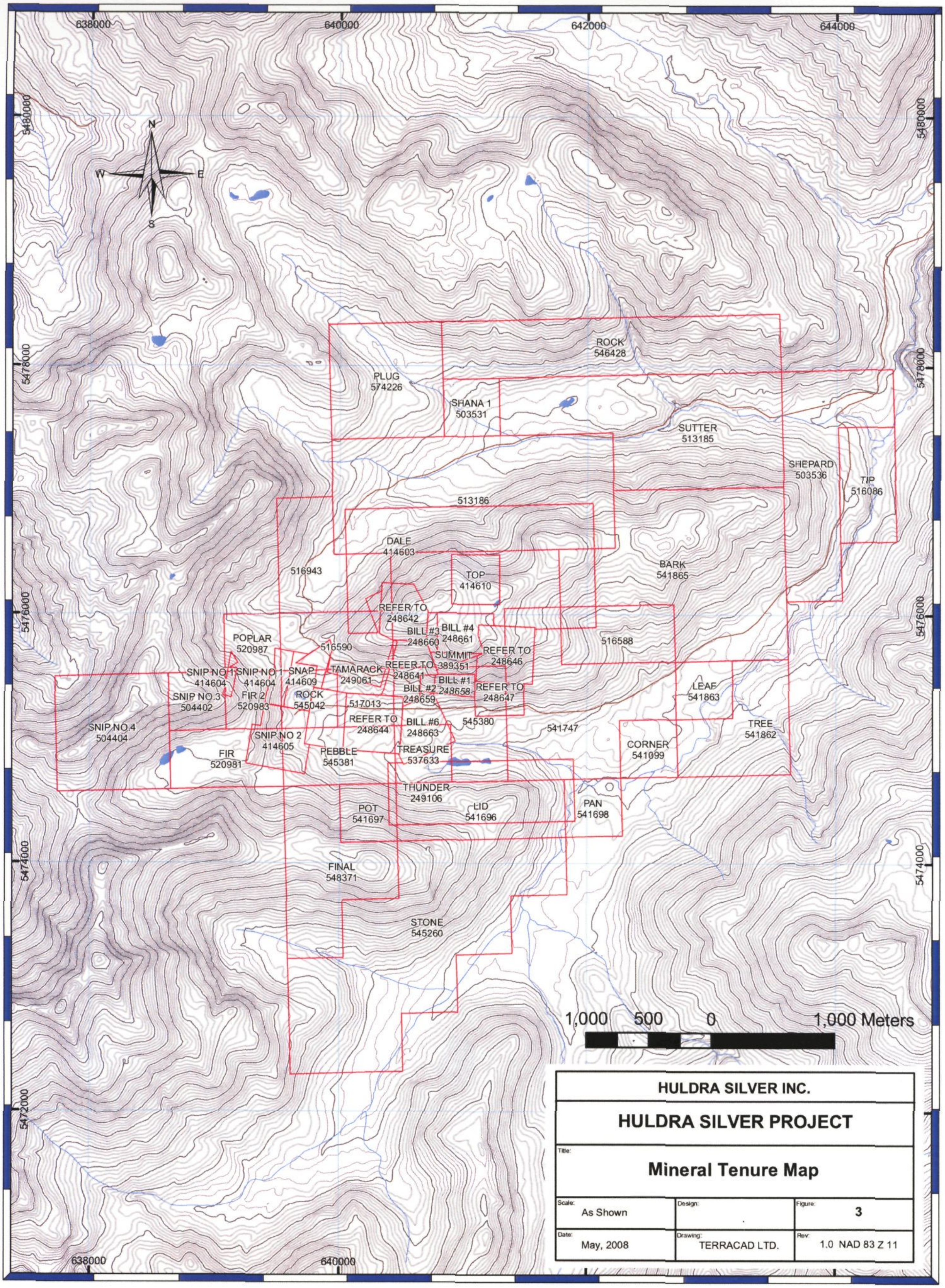
5475500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PLUG
574226
SHANA 1
503531
SUTTER
513185
SHEPARD
503536
TIP
516086
513186
DALE
414603
516943
TOP
414610
BARK
541865
REFER TO
248642
BILL #3
248660
BILL #4
248661
POPLAR
520987
516590
SUMMIT
389351
REFER TO
248646
516588
SNIP NO 1
414604
SNIP NO
414604
SNAP
414609
TAMARACK
249061
REFER TO
248641
BILL #1
248658
BILL #2
248659
REFER TO
248647
LEAF
541863
SNIP NO 3
504402
FIR 2
520983
ROCK
545042
517013
SNIP NO 4
504404
REFER TO
248644
BILL #6
248663
545380
541747
TREE
541862
SNIP NO 2
414605
FIR
520981
PEBBLE
545381
TREASURE
537633
CORNER
541099
THUNDER
249106
POT
541697
LID
541696
PAN
541698
FINAL
548371
STONE
545260
1,000
500
0
1,000 Meters
HULDRA SILVER INC.
HULDRA SILVER PROJECT
Title:
Mineral Tenure Map
Scale:
As Shown
Design:
Figure:
3
Date:
May, 2008
Drawing:
TERRACAD LTD.
Rev:
1.0 NAD 83 Z 11

Reconnaissance of the Treasure Mountain area (Upper Similkameen Indian Band, 2006). No important sites were found and it was determined that the archaeological potential of the area varies from low to moderate (AMEC, 2007). Sites and items of possible archaeological interest that in the future may be recognized by the mining company will be referred to the Band for further assessment.

1.5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

Access to Treasure Mountain is provided by an historic unpaved road from the village of Tulameen and by a similar but newer and better maintained BC Forest Service road that leaves Highway 5 immediately north of the Coquihalla Toll Booth Plaza. The former distance is about 34 km, the latter, 38 km. Both routes may be useable on a year 'round basis but in winter, only if maintenance is provided by logging companies. The mining company has indicated that their mining plan will provide for a period of idleness during certain winter months in order to avoid hazards, including avalanche danger, associated with winter operations.

Treasure Mountain is situated in rugged mountainous terrain that forms the westernmost extent of the Okanagan Highlands, transitional to the Cascade Mountains: consequently it experiences pleasant summers with occasional thunderstorms and moderate temperatures, and cold winters, with deep snow accumulations. Local roads that service active forestry operations are commonly kept open during the winter months for log haulage but are subject to closure as conditions and log markets dictate. Temperatures recorded at closest observation stations indicate that Treasure Mountain experiences winter lows of about -40°C and summer highs of about 30°C. Winter 2007-2008 featured exceptionally heavy snowfalls that resulted in closures of short duration of nearby provincial Highway 5, the Coquihalla route.

The May 2007 and April 2008 up-dated Treasure Mountain Draft Permit Application prepared for Huldra Silver Inc. by AMEC Earth & Environmental for submission to the provincial ministries of Energy and Mines and the Environment includes discussions of the sufficiency of surface rights for mining operations, the availability and sources of power, availability and quality of water, fisheries and aquatic resources, terrain and soils, natural hazards, wildlife, mine and processing plans, potential tailings storage sites, ARD, reclamation and final closure, and employment Although further study and planning are required before final decisions are made, no obstacles to further mine development entirely within property boundaries were identified.

Table 1: Mineral Tenures

Tenure Number	Claim Name	Good to Date	Area (hectares)
248641		2009/feb/13	25.0
248642		2017/may/10	25.0
248643		2009/feb/13	25.0
248644		2009/feb/13	25.0
248645		2009/feb/13	25.0
248646		2009/feb/13	25.0
248647		2009/feb/13	25.0
248658	Bill #1	2009/feb/13	25.0
248659	Bill #2	2009/feb/13	25.0
248660	Bill #3	2009/feb/13	25.0
248661	Bill #4	2009/feb/13	25.0
248663	Bill #6	2009/feb/13	25.0
249061	Tamarack Fr.	2009/feb/13	25.0
249106	Thunder	2017/may/10	75.0
249108	Troll Fr.	2016/may/10	25.0
389351	Summit	2016/may/10	25.0
414603	Dale	2017/may/10	200.0
414604	Snip No. 1	2017/may/10	25.0
414605	Snip No. 2	2017/may/10	25.0
414609	Snap	2016/may/10	25.0
414610	Top	2009/sep/29	25.0
503531	Shana 1	2017/may/10	21.009
503536	Shepard	2017/may/10	105.05
504402	Snip No. 3	2017/may/10	21.107
504404	Snip No. 4	2017/may/10	84.071
513185	Sutter	2017/may/10	168.075
513186		2017/may/10	210.112
516086	Tip	2017/may/10	42.023
516588		2009/feb/13	42.031
516590		2017/may/10	42.031
516943		2017/may/10	63.04
517013		2017/may/10	21.017
520981	Fir	2017/may/10	42.037
520983	Fir 2	2017/may/10	21.017
520987	Poplar	2017/may/10	21.015
537633	Treasure Mountain	2009/feb/13	21.0186
541099	Corner	2009/feb/13	21.0184
541696	Lid	2017/may/10	63.0608
541697	Pot	2017/may/10	21.0204
541698	Pan	2017/may/10	21.0201
541747		2009/feb/13	126.1025
541862	Tree	2017/may/10	63.0531
541863	Leaf	2017/may/10	21.0165
541865	Bark	2017/may/10	189.124
545042	Rock	2017/may/10	21.017
545260	Stone	2017/may/10	231.2698
545380		2009/feb/13	63.0571
545381	Pebble	2017/feb/13	42.0374
546428	Rock	2017/may/10	126.042
548371	Final	2017/may/10	84.088
574226	Plug	2009/jan/21	84.031

Total area **2851.61 hectares**

Apart from road access, Treasure Mountain lacks all infrastructure and Huldra Silver Inc. has no permanent structures in place. Electrical power is present at a location a few kms west of Tulameen village, about 28 km east of the site. Princeton, located 100 km east and formerly an important mining town, can provide most of the services required by travelers and forest and mining industry operators, including hospital, schools, and accommodation. The current high level of activity resulting from initiatives to re-open the former Copper Mountain mine site, located 15 km south of Princeton, may exacerbate the prevailing local shortage of skilled miners and artisans who will be required if Treasure Mountain achieves production.

The Treasure Mountain area lies at the transition between the Okanagan Highlands of the Interior Plateau and the northern extent of the Cascade Mountains. Nearby mountain ranges rise to about 1850 metres and the plateau, to about 1500 metres. The mine workings extend from Level 4 (not accessible) at elevation 1380 metres, near Amberty Creek up the steep south-facing slope of Treasure Mountain to a surface open-cut at elevation 1675 metres near the mountain top.

1.6 History

Parts of the following section are based on detailed historical information contained in a 1987 report by J. J. McDougall & Associates and on a detailed and informative but less well documented anecdotal history by James Laird that was included in a popular "rock hound" magazine.

Mineral deposits in and near Treasure Mountain were first recognized in 1892. A small number of galena veins were prospected in subsequent years, including the "Silver Chief", "Mary E" and "Whynot #3" prospects, all of which later became part of the Treasure Mountain mine. The latter was incrementally developed in a series of initiatives that included drifting and raising on different parts of the mineralized system that included footwall and hangingwall strands. Several nearby prospects were investigated by trenching and short adits.

A milling operation at Treasure Mountain in the period 1930 through 1932 processed approximately 4,000 tons that yielded 39,558 oz. silver, 379,532 lb. lead and 88,455 lb. zinc, plus cadmium (source: McDougall 1987 report, quoting Turnbull, private report, 1948). References in historic documents to periodic small "ore" shipments cannot be verified.

Silver Hill Mines Ltd. in 1950 constructed a 50 tpd flotation mill that is reported to have been in place until at least 1956 but production is not recorded.

J. M. Black, in 1952 mapped the surface geology of the property for the B. C. Department of Mines (Black, 1952).

Copper Range Exploration Co. Inc. in 1971 using alidade and plane table control produced a map of the surface geology of the south slope of Treasure Mountain, but apparently did not recognize the surface expression of the "C" vein and did not continue their work. The geology map is similar to that of Dr. Black.

Magnus Bratlien acquired parts of the Treasure Mountain property in 1979, formed Huldra Silver Inc. in 1980, and subsequently added other claims to achieve the present configuration. Huldra then conducted soil surveys and EM 16-VLF electromagnetic surveys, followed in 1981 by 1700 feet of diamond drilling and in 1983, 2612 feet of diamond drilling. This drilling provided marginally interesting economic values, including silver values as high as 126.6 opt across 18 cm and 107.9 opt across 30 cm (McDougall report, 1987, p. 2), and also much important geological information that subsequently justified programs of backhoe trenching near the top of Treasure Mountain where the principal vein, sometimes referred to as the "C" vein, which was a new discovery, was exposed almost continuously for 250 metres. The vein was sampled in detail by James Laird, geologist, of Laird Exploration Ltd. and in 1987 become the site from which approximately 407 ton of raw and partially sorted "ore" were taken and shipped to smelters. Laird's sampling, totaling 240 samples, indicated average "C" vein width of 0.68 metres (2.2 feet) with "...64 oz. silver, 11% lead and 2% zinc plus a low antimony content" (McDougall, 1987, p. 17).

The following passage is quoted in its entirety from the McDougall report:

The 1985 assaying was performed by Chemex Labs Ltd. of Vancouver and the 1986 assaying was performed by Min-Ex (sic.) Laboratories Ltd., also of Vancouver. As sections of the same zone were assayed in both years and numerous additional samples have been assayed at various laboratories, a good sampling and assay check has been provided with no major discrepancies apparent (McDougall, 1987, p. 21).

The No. 1 Level adit was re-opened in 1986 and a 43 metre length of vein was sampled in the old workings. Major work programs were directed to the mine in the period 1987 through 1989. Levels 2 and 3, with final lengths 392 and 632 metres respectively, were driven and Levels 1 and 4 were extended. Raises were excavated to provide information concerning continuity of mineralization and, where they passed between levels, to provide additional access and ventilation. Mine workings, including crosscuts, drifts and raises, total approximately 2,800 metres (9,000 feet). The veins exposed in both historic and newly driven mine workings were channel sampled in 1988 under the supervision of E. Livgard, P. Eng. Samples were taken at one metre intervals and data from approximately 576 chip and channel samples of vein and wall rock taken from underground locations were analysed for major elements. The assay database also includes 238 surface samples and samples from 1153.5 m of diamond drill holes. As discussed in a later section of this report, 407 tons of development muck and stockpiled material, all of which came from a surface open cut, were shipped to smelters in Trail, B. C. and East Helena, Montana. Prior to shipping, the materials were in part machine sorted to remove lower grade materials and reduce the volume of the shipments. Subsequently, mine workings were surveyed and a reserve estimate (non-43-101 compliant) was calculated by Livgard Consulting Ltd., under the direction of E. Livgard, P. Eng. Several nearby areas prospective for silver and gold were investigated by trenching and sampling.

Coastech Research Inc. carried out preliminary metallurgical work on Treasure Mountain silver-lead-zinc materials prior to 1989 (details not available) that showed that the ore was "....free of contaminants and that 95% silver recovery could be retrieved through conventional concentrating" (AMEC, 2007, p. 5).

Huldra Silver Inc. in 1989 commissioned Orocon Inc. of North Vancouver, B. C., a firm specializing in mine evaluation, mill design and construction, to conduct a technical study of the Treasure Mountain project. That study incorporated metallurgical, geological, environmental and mining engineering components by various consultants and was an aid in determining "...the potential of the deposit to be profitably brought to production" (Orocon, 1989, p. 1). That review included ore reserve recalculations by Livgard Consulting Ltd., a metallurgical report by Bacon, Donaldson & Associates Ltd., and permitting information provided by Entech Environmental Consultants. A mining program, a mill flow sheet and a Cash Flow Schedule were developed on the basis of a 200 ton per day operation although the mill was designed to treat 300 tpd. The Bacon, Donaldson & Associates report on metallurgy confirmed the Coastech work that indicated "...recoveries for lead 94.2%, zinc 93.2%, and silver 94.6% with conventional flotation" (quoted in AMEC, 2007, p. 5) Cost to production was estimated at $9.0 million, including working capital. Operating costs were projected to be $92.25/ton.

Note that the above-quoted technical review was prepared prior to introduction of National Instrument 43-101 and CIMM Definition Standards for Mineral Reserves and Resources, is not a compliant Economic Assessment, is no longer current, and should not be used or relied upon in an evaluation of the Treasure Mountain deposit.

Huldra Silver Inc. in 1989 submitted a prospectus to the Mine Development Steering Committee with the objective of placing the Treasure Mountain property into production (Meyers and Hubner, 1989). Metal prices weakened substantially before the permitting process was completed and the Orocan recommendations were not implemented (Bratlien, 2007, personal communication). Underground work on the property ceased in 1989 but Huldra Silver Inc. in order to maintain the mineral tenures in good standing, completed several small programs of work, including soil surveys, some trenching, three surface and one underground drill programs, in the period 1990 - 2006.

Mr. A. J. Beaton, P. Eng., mining engineer and mining contractor, in 1998 was engaged to evaluate the feasibility of production and prepared an economic and production analysis of a 25,000 tons per year mine/mill operation. His analysis, which was **not NI 43-101 compliant**, concluded that a seasonal operation with mill capacity of 150 tons per day would achieve payback of capital within two years.

Work toward production again resumed in 2006 when the company engaged McElhanney Consulting Services Ltd. to prepare from current aerial photography a detailed surface map of the Treasure Mountain area as part of a renewed program to establish a small underground mine on the site. A. J. Beaton Mining Ltd., also in 2006, prepared a detailed production evaluation on the basis of available geological, metallurgical and environmental data (Beaton, 2006) and other engineering compilations and environmental studies were initiated.

The A. J. Beaton Mining Ltd. Evaluation Report of 2006 included a comprehensive review of the Treasure Mountain project as well as mining plans and other requirements for establishing a viable mining operation. The report included financial projections, permitting issues, infrastructure options and requirements, transportation and personnel, and discussed the need for further technical studies. Economic projections were based on silver prices of $8 to $15 USD per ounce and lead, $0.50 USD per pound and zinc, $1.50 USD per pound with an exchange rate of 1.10. The base case assumed $10 USD per ounce silver, 5.12% zinc and 4.57% lead. The unit operating cost was projected to be $149.89USD per ton (note: Imperial measure). A mining plan utilizing track haulage and both shrinkage and open stope mining methods was presented on the basis of mining 150,000 tons above the present Level 3 and included a study of the effect of doubling ore reserves. A pre-production schedule and an operating scenario, complete with cost forecasts utilizing 2006 costs, were included. The study also offered the opinion that the Treasure Mountain property "*...can be put into production as a viable, economic, small underground mine*" (Beaton, 2006). Capital requirements, including working capital, were forecast to be $9,715,000 and "*...the rate of return on this investment is over 50% at $10 USD per ounce silver*" (Beaton, op cit.). Beaton, et al., recognized a strong economic sensitivity to both the price of silver and to the possible development of additional resources that would result in a longer mine life.

Note that the A. J. Beaton Mining Ltd. evaluation report was prepared by a professional engineer with much experience in operating small narrow vein mines in British Columbia and elsewhere but it is not a feasibility study. The figures quoted above were based on 2006 data and were prepared for guidance of company management rather than for dissemination to the public and they should not be relied upon exclusively in any current appraisal of the deposit or company.

AMEC Earth & Environmental, a division of AMEC Americas Limited, in May, 2007 prepared a comprehensive document in support of a Draft Permit Application for the development and operation of a mine at the Treasure Mountain site. That report was predicated on a 135 metric tonnes per day operation in an eight month annual season and included a broad range of baseline topics, a mine plan, a discussion of processing, dams and waste emplacements, mine site infrastructure, water management, reclamation and closure. (AMEC, 2007). The AMEC report recommended a number of additional programs, including aquatic studies, avalanche survey, sampling and testing of waste rock, geotechnical evaluation of a tailings disposal site, metallurgical testing of "ore" to obtain representative tailings samples, an up-dated mineral resource assessment (under NI 43-101) and a detailed mine plan. Although the mineral resources that were calculated by Livgard Consulting Inc. were prepared prior to implementation of NI 43-101 reporting requirements and were at that time not adequately documented to qualify in accordance with current CIM Definition Standards for Mineral Resources and Reserves, they were considered valid for the AMEC study (AMEC, 2007, p. 2).

Huldra Silver Inc. in July, 2007, in response to the recommendations of AMEC, re-opened Level 1 and Level 2 for the purpose of re-sampling the vein and acquiring materials for metallurgical test work. Work was conducted under the supervision of the District Inspector of Mines who was concerned about the deteriorated condition of mine portals, possible hazards to the general public, and unknown conditions in the mine. Huldra engaged a professional geologist to direct the sampling work and a mining engineer and a licensed shift boss to supervise the rehabilitation work and to work closely underground with a small crew of samplers. Two experienced prospectors were on site to work with the geologist.

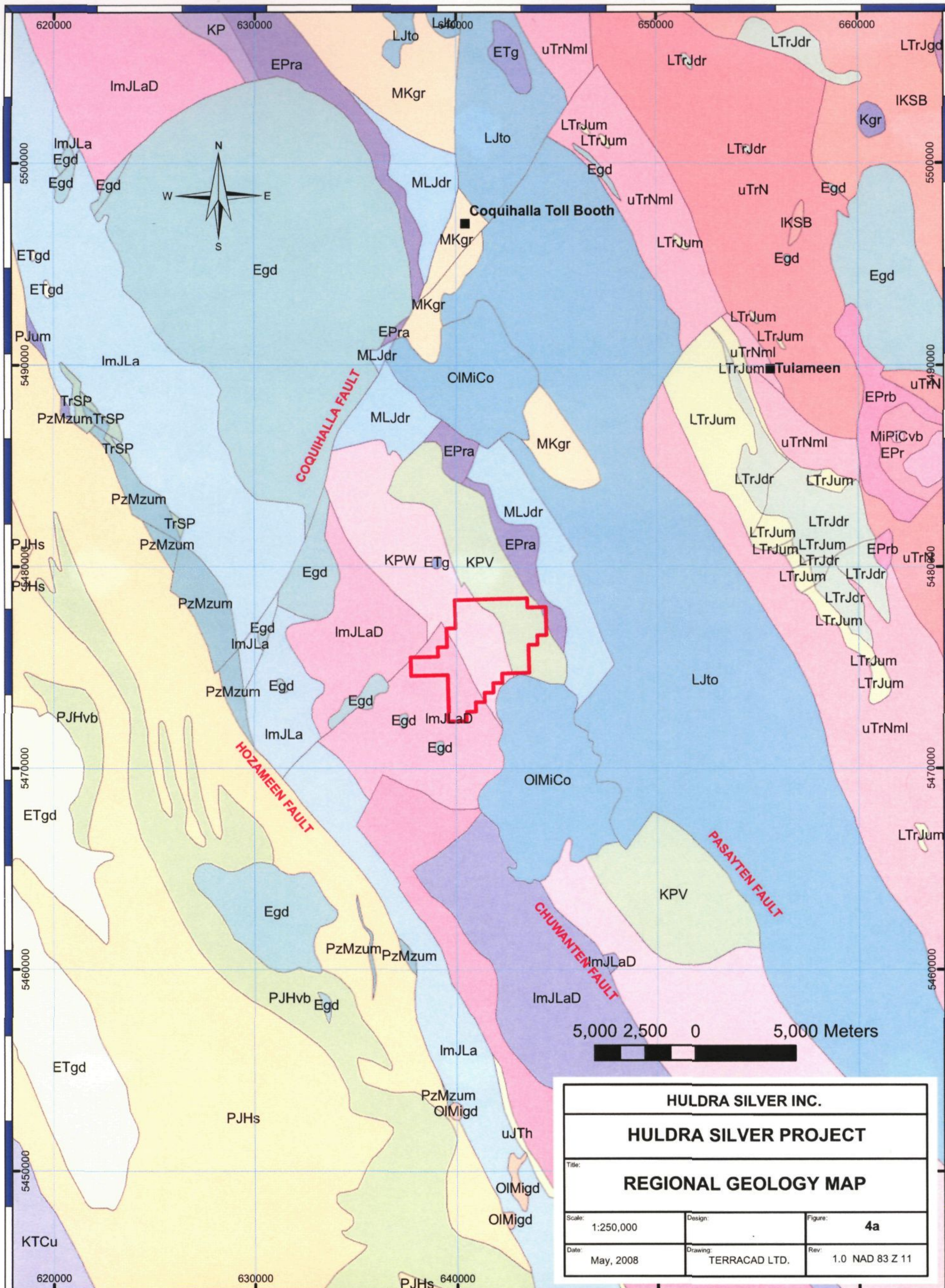
Seventy-eight rock samples were taken from sulphide-rich mineralization exposed in the underground workings on Levels 1 and 2. No samples were taken from the surface trench located near the top of Treasure Mountain that was the source of the raw ore shipments to the smelters nor from Levels 3 and 4, nor from any of the raises in the mine. Samples were taken from locations that were in part determined by accessibility, rock quality and the condition of the mine. Some areas in proximity to strongly sheared fracture zones were clearly unstable and in the interests of safety were avoided.

2.0 GEOLOGICAL SETTING

Treasure Mountain is situated in the northward continuation of the Cascade Mountains of Washington State. This system in Canada lies between the Fraser River to the west and the Okanagan valley to the east and is host to several important mines and mineral deposits.

> *"The belt contains sedimentary and volcanic rocks of Late Paleozoic to Cretaceous age plus younger intrusives and sediments. In B. C. it is characterized by subdivisions including the following, listed in order of decreasing age: Hozameen Gp, Nicola Gp, Ladner Gp, Dewdney Creek Gp, Jackass Mountain Gp and Pasayten Gp"* (McDougall, 1987, p. 8).

Monger in 1989 published Map 41-1989, Geology, Hope, British Columbia, a portion of which has been reproduced for inclusion in this report (Figure 4). The Monger report included a terrane map that places Treasure Mountain in the Tyaughton-Methow terrane: the mountain is transected by the northwesterly-trending Chuwanten thrust fault. The Pasayten fault lies east of and parallels the Chuwanten structure and separates Tyaughton-Methow terrane from Quesnellian terrane. Lithology at Treasure Mountain comprises Cretaceous Pasayten Group arkose, conglomerate, argillite, minor red beds and tuff (Monger, 1989, Sheet 1, Figure 2). The Eagle Plutonic Complex of Late Jurassic and Early Cretaceous age lies 3 km east and the Eocene Needle Peak Pluton of granodioritic composition is 10 km northwest. Small bodies of similar granodiorite, possibly shredded by faulting from the main body, are present in proximity to Treasure Mountain. A short distance to the south, about 2 km, Late Oligocene to Early Miocene felsic volcanic rocks, designated Coquihalla Formation, have overridden the area lying between the Chuwanten and Pasayten fault structures. A similar occurrence 10 km due north of Treasure Mountain lies wholly within the Eagle Complex, suggesting that the Coquihalla Formation represents a late stage of Eagle plutonism.



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HULDRA SILVER PROJECT
Title:
REGIONAL GEOLOGY MAP
Scale: 1:250,000
Design:
Figure: 4a
Date: May, 2008
Drawing: TERRACAD LTD.
Rev: 1.0 NAD 83 Z 11

Legend

Quaternary Unit

STRAT_AGEC, UNIT

1034, MiPiCvb - Cenozoic - Chilcotin Group basaltic volcanic rocks
1036, Migd - Cenozoic - Unnamed granodioritic intrusive rocks
1042, OlMiCo - Cenozoic - Coquihalla Formation calc-alkaline volcanic rocks
1042, OlMigd - Cenozoic - Unnamed granodioritic intrusive rocks
1050, ETg - Cenozoic - Unnamed intrusive rocks, undivided
1050, ETgd - Cenozoic - Unnamed granodioritic intrusive rocks
1054, EPr - Cenozoic - Princeton Group undivided sedimentary rocks
1054, EPra - Cenozoic - Princeton Group coarse clastic sedimentary rocks
1054, EPrb - Cenozoic - Princeton Group andesitic volcanic rocks
1054, Egd - Cenozoic - Unnamed granodioritic intrusive rocks
2010, KTCu - Mesozoic to Cenozoic - Custer Gneiss orthogneiss metamorphic rocks
2010, KTSl - Mesozoic to Cenozoic - Slollicum Schist greenstone, greenschist metamorphic rocks
2010, KTmm - Mesozoic to Cenozoic - Unnamed mid amphibolite/andalusite grade metamorphic rocks
2020, KP - Mesozoic - Pasayten Group undivided sedimentary rocks
2020, KPV - Mesozoic - Pasayten Group - Virginia Ridge Facies coarse clastic sedimentary rocks
2020, KPW - Mesozoic - Pasayten Group - Winthrop Facies coarse clastic sedimentary rocks
2020, Kgr - Mesozoic - Unnamed granite, alkali feldspar granite intrusive rocks
2020, Ks - Mesozoic - Unnamed undivided sedimentary rocks
2022, MKgd - Mesozoic - Unnamed granodioritic intrusive rocks
2022, MKgr - Mesozoic - Unnamed granite, alkali feldspar granite intrusive rocks
2022, MKqd - Mesozoic - Unnamed quartz dioritic intrusive rocks
2023, EKg - Mesozoic - Unnamed intrusive rocks, undivided
2023, lKGsv - Mesozoic - Gambier Group marine sedimentary and volcanic rocks
2023, lKJ - Mesozoic - Jackass Mountain Group undivided sedimentary rocks
2023, lKSB - Mesozoic - Spences Bridge Group undivided volcanic rocks
2051, LJto - Mesozoic - Unnamed tonalite intrusive rocks
2051, uJTh - Mesozoic - Thunder Lake Sequence coarse clastic sedimentary rocks
2052, MLJdr - Mesozoic - Unnamed dioritic intrusive rocks
2053, MJgr - Mesozoic - Unnamed granite, alkali feldspar granite intrusive rocks
2054, lmJLa - Mesozoic - Ladner Group mudstone, siltstone, shale fine clastic sedimentary rocks
2054, lmJLaD - Mesozoic - Dewdney Creek Formation coarse clastic sedimentary rocks
2054, lmJLaD - Mesozoic - Ladner Group - Dewdney Creek Formation coarse clastic sedimentary rocks
2055, lJHL - Mesozoic - Harrison Lake Formation andesitic volcanic rocks
2082, LTrJdr - Mesozoic - Unnamed dioritic intrusive rocks
2082, LTrJgd - Mesozoic - Unnamed granodioritic intrusive rocks
2082, LTrJum - Mesozoic - Unnamed ultramafic rocks
2082, uTrJCu - Mesozoic - Cultus Formation mudstone, siltstone, shale fine clastic sedimentary rocks
2090, TrSP - Mesozoic - Spider Peak Formation basaltic volcanic rocks
2090, Trog - Mesozoic - Unnamed orthogneiss metamorphic rocks
2091, uTrN - Mesozoic - Nicola Group undivided volcanic rocks
2091, uTrNC - Mesozoic - Nicola Group - Central Volcanic Facies andesitic volcanic rocks
2091, uTrNE - Mesozoic - Nicola Group - Eastern Volcanic Facies basaltic volcanic rocks
2091, uTrNml - Mesozoic - Nicola Group lower amphibolite/kyanite grade metamorphic rocks
2091, uTrNsf - Mesozoic - Nicola Group mudstone, siltstone, shale fine clastic sedimentary rocks
3010, PzMzCS - Paleozoic to Mesozoic - Cogburn Schist greenstone, greenschist metamorphic rocks
3010, PzMzum - Paleozoic to Mesozoic - Unnamed ultramafic rocks
3030, PJHs - Paleozoic to Mesozoic - Hozameen Complex undivided sedimentary rocks
3030, PJHvb - Paleozoic to Mesozoic - Hozameen Complex basaltic volcanic rocks
3030, PJum - Paleozoic to Mesozoic - Unnamed ultramafic rocks
3130, DPC - Paleozoic - Chilliwack Group undivided sedimentary rocks
4020, PrPzY - Proterozoic to Paleozo - Yellow Aster Complex dioritic intrusive rocks

Claim Outline

FAULT TYPE

Fault
Normal Fault
Thrust
Past Producer

HULDRA SILVER INC.		
HULDRA SILVER PROJECT		
Title: LEGEND TO ACCOMPANY REGIONAL GEOLOGY MAP		
Scale: 1:250,000	Design:	Figure: 4b
Date: May, 2008	Drawing: TERRACAD LTD.	Rev: 1.0 NAD 83 Z 11

Of primary interest in the immediate Treasure Mountain area are the Dewdney Creek and Pasayten Groups: the former comprises fragmental volcanic rocks with about 25% sedimentary members; the latter, which appears to be the principal host rock of the Treasure Mountain deposit, arkose, argillite and conglomerate. Both units trend northwesterly and are cut by sills and lamprophyre dykes and by dioritic to gabbroic intrusions of Tertiary age and both are transected by the dyke and fault-related mineralization.

> *"In the eastern property area a feldspar porphyry dyke crosses Treasure Mountain, striking east-westerly and dipping southerly. It occupies a major fault which cuts across both formations and at least one large sill. Thicknesses range from 21 m in the east to 1.5 m in the west. In the 1983 drill area the width of this dyke ranged from 2.4 to 3.6 m. Alteration, including carbonatization and chloritization, is common as the borders of this pre-mineral and highly sheared dyke appear to have been subjected to hydrothermal agencies accompanying mineralization. However, the dyke itself is apparently unmineralized"* (McDougall, op cit., p. 10).

The major fault referred to above has "*...possible displacement of 305 m (1000 feet) or more*" (McDougall, op cit. p. 10). Mineralization is located along the fault or closely related faults and on either wall of, and occasionally within, the dyke. The fault was mapped by Black (1952) as "*...having an arcuate trend.....with a severe flexure southward...*" (quoted by McDougall, op cit. p. 11) and "*...the indicated displacement being possibly several hundred feet*" (Black, 1952, p. A122) and noted formational offset "*....as much as several hundred feet to the left*" and "*...possibly there was vertical movement of several hundred feet* (ibid. p. A123).

Alteration occurs in proximity to the dyke and includes pyritization, carbonatization and chloritization. Metallic mineralization comprises sphalerite, galena, pyrite, arsenopyrite, tetrahedrite, stibnite, pyrrhotite, zinkenite, bournonite (McDougall, op cit., p. 11) and braunite (Jim Laird, 2007, personal comm.). Magnetite and hematite are also present. McDougall recognized native silver occurring within galena and zinkenite and speculated that it is also present in the tetrahedrite. Livgard (1989) refers to the importance of freibergite, a strongly argentiferous variety of tetrahedrite. Potentially valuable amounts of antimony and cadmium are reported in assays as are barium, mercury and gold. Gangue minerals include "comb" quartz and carbonates and manganiferous siderite (?). Historic data does not disclose metal values that may be present in the wallrocks adjacent to the "C" vein and although the 2007 sampling included several samples of such materials, the number of samples was insufficient to provide a meaningful indication of such values. For the purposes of this report and resource calculation, metal values in material that may dilute the "ore" are assumed to be "nil".

Several veins in addition to the "C" vein, are referred to in the J. J. McDougall and Associates (1987) report but details are few and there appears to be little certainty concerning their identity: some are splays from the principal "C" vein, others appear to be parallel structures. Vertical and lateral zoning of silver values is recognized, with silver to lead ratios apparently increasing from west to east, though elevations also increase in that direction. Silver to zinc ratios vary widely and Vulimiri ((1986, quoted by McDougall (1987, p.12)) suggested that proximity to the dyke influenced the silver ratio, "*...with a higher silver ratio away from the dyke*". Other veins, designated "A", "B" and

"D", have had very limited attention and prospectors have found areas on the north slope of Treasure Mountain near Sutter Creek with mineralization similar to the "C" vein.

Veins on the then John claim, now mineral tenures 516588 and 541747, located 1.1 km southeast of the mine workings, were discovered by prospecting an area where anomalous silver-in-soil geochemical responses were recorded. The occurrence, a.k.a. the "Ruby Zone", has been partially explored by trenching and diamond drill and reverse circulation drill holes and, although a porphyry dyke similar to the "Mine Dyke" is present, it is not possible to confirm that it is an extension of the dyke found in the mine area in proximity to the "C" vein and related mineral occurrences. Several samples from drill cuttings and trench samples with strong silver values and moderate lead and zinc values have been recorded and the area warrants further exploration.

3.0 DEPOSIT TYPES

Treasure Mountain mineral veins are classed as "fracture controlled", have little gangue and frequently feature central bands of massive mineralization with veinlets and disseminations distributed short distances outwards into the wallrocks. Sulphides and sulphosalts along with quartz, were introduced along fracture zones proximal to a single feldspar porphyry dyke that may be an off-shoot from granitic bodies that lie a short distance from the mine area.

The principal Treasure Mountain vein(s) occurs in proximity to the Treasure Mountain fault and a feldspar porphyry dyke that partially occupies the fault (Black, 1952) (Figure 5). The vein strikes northeasterly and dips 50° to 65° southeasterly. Ore shoots within the vein extend from 50 to 150 m in length and vary in thickness from 0.5 to 1.5 m and occasionally to more than 2.0 metres. The "C" vein has been explored from surface at 1680 m elevation to about 1390 m elevation, a dip distance of almost 350 m (Livgard, 2006).

3.0 MINERALIZATION

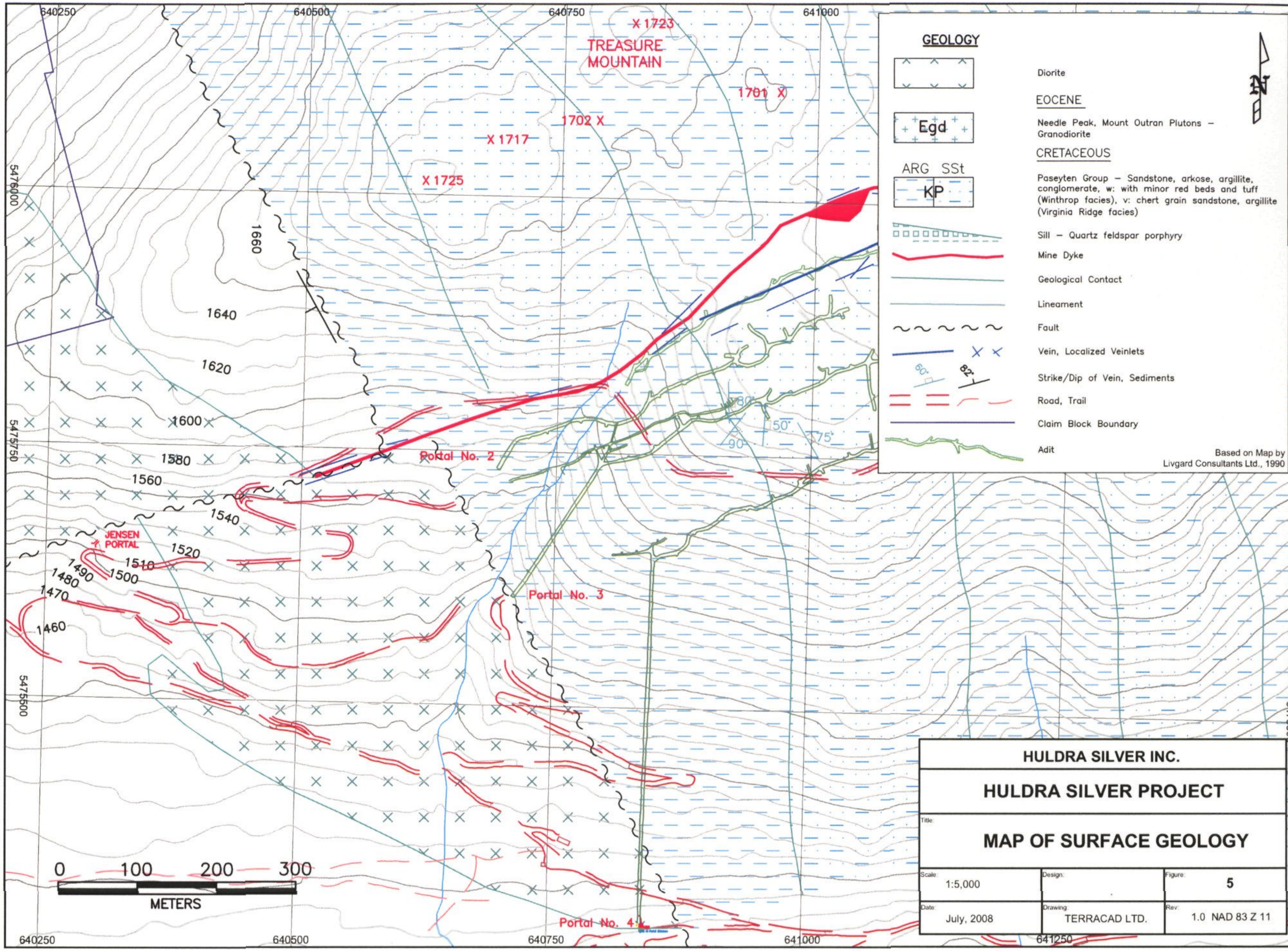
Treasure Mountain veins comprise as much as 90% sulphide and sulphosalt minerals with the remainder being varying quantities of quartz, carbonates (calcite, siderite, manganiferous siderite), chlorite and barite. The mineralization probably qualifies as mesothermal. Veins exhibit a banded or ribboned appearance with seams of massive sphalerite and/or cubic galena separated by narrow layers of gangue, largely quartz and/or carbonates. Pyrite in small quantities is ubiquitous, mostly as disseminated grains but also as irregular seams or layers. Resources have been measured on both the hangingwall and footwall of the Treasure Mountain dyke. Vein contacts with the enclosing dyke are sharply defined but small stringer veins occasionally penetrate the walls and (rarely) the main vein has been shown to be internal to the dyke.

Detailed descriptions by J. F. Harris, PhD., consulting petrographer, of polished sections of vein specimens and one polished thin section are included in Appendix 3 of this report. These samples were prepared for the purpose of aiding mill metallurgists in determining the distribution of silver-bearing and other components. Specimen 1-11 (West End) is characterized as a "...well banded vein" comprising sphalerite (60-70%), quartz (12-15%), boulangerite (7-8%), chalcopyrite (2-5%), tetrahedrite (2-3%), ankerite (2-3%), galena (1-2%), pyrite (0.3%), native silver (?) (minor), and covellite (trace). Specimen 1-17 (C Vein) is a "...banded vein" with 60-65% ankerite, 17-20% sphalerite, 5-7% quartz, 2-3% tetrahedrite, 2-3% boulangerite, 2-3% galena, 0.3% bournonite, 0.1% arsenopyrite and trace chalcopyrite. Specimen I-25 (C vein split?) comprises 60-65% quartz, 17-20% boulangerite, 15-17% sphalerite, 2-3% arsenopyrite and 0.2% galena. The polished thin-section was prepared from the reject portion of an atypical, low sulphide, high silver (50 - 70 opt Ag) style of mineralization in a portion of "C" vein. The sample contained an estimated 6% total sulphide content. The description is thorough and includes the observation that tetrahedrite, from scanning electron microscope analysis was confirmed as being strongly argentiferous (10% or more?). Ruby silver, probably pyrargyrite, was also present.

Livgard in an assessment report described the mineralization as follows:

The veins host silver, lead and zinc mineralization in a gangue of carbonate and quartz. The main silver mineral is freibergite, the lead mineral is silver rich galena and the zinc mineral is brown sphalerite darkening to black with depth. Lesser amounts of boulangerite, bournonite, chalcopyrite and magnetite have also been noted as well as minor pyrargyrite, stibnite, pyrrhotite and native silver. The grade of silver varies from nil up to 10000 grams per tonne silver and 10% lead-zinc. Near surface the mineralization is mainly carbonate, galena and freibergite. With increasing depth the quartz and sphalerite content increases and the carbonate, galena and freibergite content diminishes to the bottom level (#4 Level) about 300 metres below surface, where the vein hosts mostly quartz and black sphalerite. A raise from the bottom level encountered ruby silver mineralization in the main vein about 70 meters above the level. This type of mineralization would normally be higher in the vein system. It is believed that this was emplaced by a second mineralizing pulse" (Livgard, 2006, ARIS report #27944).

Modeling studies conducted by Terracad Ltd. demonstrate that a small discontinuity of the "ore" mineralization occurs in the vicinity of Level 2 of the underground workings. This feature was also recognized by Livgard (Livgard, op cit.) who attributed it to "...a second mineralizing pulse" (see above). The model confirms the presence of the apparent break but does not explain it.


GEOLOGY
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Egd
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CRETACEOUS
ARG SSt
KP
Paseyten Group – Sandstone, arkose, argillite, conglomerate, w: with minor red beds and tuff (Winthrop facies), v: chert grain sandstone, argillite (Virginia Ridge facies)
Sill – Quartz feldspar porphyry
Mine Dyke
Geological Contact
Lineament
Fault
Vein, Localized Veinlets
Strike/Dip of Vein, Sediments
Road, Trail
Claim Block Boundary
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Based on Map by Livgard Consultants Ltd., 1990
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July, 2008
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TERRACAD LTD.
Rev:
1.0 NAD 83 Z 11

As part of their metallurgical test work that is described in a later section of this report, Bacon, Donaldson and Associates submitted 12 samples of vein material to Vancouver Petrographics Ltd. for examination. John G. Payne (1989), consulting petrographer, reported that "*Major 'ore' minerals in veins are galena, sphalerite, tetrahedrite, and boulangerite. Minor 'ore' minerals are bournonite, and chalcopyrite, a trace of stibnite and native silver*", and recorded the following observations:

> *1) tetrahedrite and chalcopyrite decrease in abundance with depth*
> *2) boulangerite is very variable between samples, and is most abundant in Level 1*
> *3) bournonite is rare and is most abundant in Level 1*
> *4) pyrite generally is most abundant at depth*
> *5) pyrrhotite occurs only in one sample on Level 2*
> *6) native silver occurs only in one sample on Level 1*
> *7) stibnite occurs only in one sample on Level 2.*

Payne (op. cit. p. 2) also observed that:

> *"The distribution of silver cannot be readily explained in terms of mineral variations. Silver is present in native silver (one sample) and tetrahedrite, and probably also occurs in significant amounts in galena and boulangerite, particularly significant at lower levels, where the contents of tetrahedrite and native silver are low. The presence of significant silver in boulangerite and galena is suggested because in the sample containing native silver, all of that mineral occurs in exsolution (?) blebs in boulangerite and in galena".*

The above-cited observations by Livgard and the consulting petrographers have important implications with respect to the strong variability of silver values encountered in sampling the Treasure Mountain veins

4.0 EXPLORATION

Commencing in 1979, Magnus Bratlien and associates, and, following its founding in 1980, employees of Huldra Silver Inc., prospected the Treasure Mountain area in order to determine the locations of historic workings (i.e. from the 1892-era discovery, and from the 1932 - 1934 and early 1950's mining/milling operations) and to search for nearby areas that have geological potential to host similar occurrences and/or extensions of the Treasure Mountain dyke and vein(s). Details of that work are documented in company maps and records that show that reconnaissance, and in some cases more intensive work, was directed at least as far north as Sutter Creek, a distance of 2 km, and westerly to the upper slopes above Amberty Creek, a distance of 2 km.

The main Treasure Mountain vein system has been explored over a vertical distance of 295 metres by surface trenches and approximately 2,742 metres of underground workings, including 2,194 metres of drifts and crosscuts and 548 metres of raises. Earliest workers appear to have followed surface outcroppings of mineralization using hand tools and then mining techniques but missed what became the top of the principal "ore" zone.

Huldra Silver Inc. used a small backhoe in 1985 to expose galena-bearing arkosic rocks found on surface in proximity to a distinct naturally-occurring shallow trough-like depression about 50 metres higher in elevation than Level 1. Mr. Jim Laird sampled these uppermost surface workings over a distance of 250 metres. The sulphide-rich vein is variously reported to have "...averaged about 2194 grams per tonne silver and 12 per cent combined lead-zinc over a 0.68 metre width and along a vein length of 150 metres" (Meyers and Hubner, 1989) and "...averaged 35 oz/t silver and 7% lead-zinc combined, over a vein length of 820 feet across 4 feet (diluted) widths (Huldra Silver Inc., Progress Report, Feb. 1989). The 1987 J.J. McDougall & Associates Ltd. report quoted Vulimiri (1986) as reporting "...220 channel samples taken in 1986 plus 20 taken in 1985 along 250 m of "C" vein averaged 64 oz/ton silver, 11.1% lead and 2.0% zinc across a true width of 0.68 m (2.2 feet)". A bulk sample of "...about 2,400 tons of ore was mined from the C vein surface showings" (Huldra Silver, op. cit) of which 407 tons of mineralized rock, grading 98 opt silver, were shipped to smelters in Trail, B. C. and East Helena, Montana.

Level 1, which originally had length 65 metres and a small stope developed close to the portal, was re-entered by Mssrs. Bratlien and Laird in 1986 who then, , panel-sampled a mineralized structure that extended 30 m (+/-100 feet). This level was subsequently lengthened as part of the 1987-1988 development program.

Results from various exploration initiatives were the subject of a technical report prepared by J.J. McDougall and Associates, dated January 10, 1987, that recommended further geophysical and geochemical surveys, followed by trenching and drilling, to explore surface targets that had not yet been tested. They also recommended further exploratory drilling to prove up underground resources as well as shipments of open pittable mineralized rock. The technical report included a review report on the practicality of open pit mining the upper part of the "C" (i.e. main) vein.

The McDougall technical report was sufficiently positive in its recommendations to enable the company to finance the 1987-1989 programs of exploration and underground development that included work on all four levels of the mine and 1680 metres of underground drilling, and also surface exploration work that included grid preparation, prospecting, geochemical soil surveys, ground-based geophysical surveys, and trenching employing hand tools and an excavator.

The area immediately west of the present mine workings in the vicinity of the so-called Jensen adit, about 400 metres west of Level 3 portal, was subsequently trenched with a small bulldozer and in 1988 further explored by a small program of rotary drilling (see below). An historic plan from 1952 shows views of different parts of Treasure Mountain that were then controlled by Silver Hill Mines Ltd. including a sketch of the Jensen adit on which are plotted a series of assay samples with the notation that a mineral zone in the footwall of a dyke (not identified as the same dyke as is found in the main mine) assayed 29.25 opt silver, 18.2% lead and 15.4% zinc over width 0.8 feet (24 cm) and length 85 feet (25.9 m). Also on the same drawing is the notation
"*Shipments: 1926 - 23 tons sorted ore, Ag 49.5 oz, Pb 30%, Zn 12%*
1951 - 20.3 tons, Ag 23.65 oz, Pb 16.8%, Zn 14.6%"
(reference: FWH, July 1952). Another notation states "*Samples by Hill & Richmond*".

Caution: Note that the above-quoted figures are from an historic source and have not been verified by the writer. Although it is believed that the source map was drawn by Fred Hemsworth, P. Eng. and that the "Hill" refers to Henry Hill, P. Eng., both of whom at the time were Vancouver, B. C. based consulting engineers, there is no information concerning the sampling and assaying procedures and the data

(?) are included only in order to draw attention to an exploration target proximal to the proposed underground mine.

A small program of reverse circulation drilling, comprising 5 holes with total length 316.5 metres, was completed in July, 2005 in the vicinity of the "Jensen" workings, the probable western extension of the main vein at about the elevation of the No. 3 Level. The intention was to clarify a geologically complex area of mixed sedimentary and intrusive rock types and several occurrences of sulphide mineralization. Holes were inclined and directed northwesterly, approximately normal to regional structures. The following information is taken from assessment report #27944 (Livgard, 2006):

> Hole #HR03 intersected two narrow veins:
> at 61.0 m to 62.5 m - 1.5 m with 50 g/tonne Ag, 1596 ppm Pb and 2277 ppm Zn
> at 71.6 m to 73.15 m 1.5 m with 14.3 g/tonne Ag, 2538 ppm Pb and 7333 ppm Zn.
>
> Hole #HR04, closer to the "Jensen Adit", an historic working from which shipments of high grade silver-lead-zinc mineralization are reported, intersected from 22.85 m to 24.38 m - 1.53 m with 50 g/tonne Ag, 1.31% Pb and 1.74% Zn.
>
> Hole #HR05, located 40 m south and 85 m east of the "Jensen" adit intersected a quartz and carbonate vein with 25% dyke of which 1.5 m assayed 309 g/tonne Ag, 3.54% Pb and 6517 ppm Zn.

Results of Huldra's exploration in the Jensen adit area were inconclusive in defining the location of the possible extension of the "main" zone and there is uncertainty whether any of five rotary drill holes from the 2005 work actually intersected the "C" zone vein. The mineralized portions of rotary drill holes lie along the projected location of the "C" zone vein but cannot with certainty be related to either mineralization found at the Jensen adit or in the Treasure Mountain mine. The results however strongly support the exploration concept that further silver-lead-zinc resources may be found in and near the Treasure Mountain fault and dyke system.

Exploration in 1987 - 1989 was directed to the newly discovered "Ruby Zone", an area 1.1 km east of the mine that was found as a result of the soil sampling program where samples geochemically anomalous in silver, gold and base metals were obtained. Approximately 1.5 line-km of bulldozer trenches and roads were excavated and ten rotary reverse circulation holes with total length 575.4 metres explored a zone that was interpreted as a probable extension of the Treasure Mountain dyke and vein. Modest success was reported from the area, with numerous rotary reverse circulation chip sample intervals assaying as much as 34.48 opt Ag, 15.2% Pb and 0.04% Zn over 20 feet (6.1 m) (Livgard, 1990). The values have yet to be confirmed by core drilling methods and true thicknesses have not been determined. The area will be explored further when property work resumes.

5.0 DRILLING

No drilling was undertaken as part of the limited program of sampling that is the subject of this report. Huldra Silver Inc. in 1981, 1983, 1986, 1988 and 2005 tested various parts of the Treasure Mountain property by diamond drilling with most holes directed to the "C" vein. The Jensen Adit area west of the mine workings was explored by diamond drilling in 1988 and by rotary drilling in 2005. The Ruby Vein part of the property, about

1.1 km east of the mine was drilled in 1988 using a rotary drill.. When mining was in progress a few short diamond drill holes were directed into the walls of parts of the underground workings to search for metal values. That drilling was only partially satisfactory: the vein could be identified but core recovery in the vein was poor (Bratlien, 2008, personal communication).

6.0 SAMPLING METHOD AND APPROACH

The objectives of the program of work completed in July, 2007, in the Treasure Mountain mine were twofold: to obtain sufficient samples from the mineral zone(s) to permit an evaluation of resource calculations prepared in 1989, prior to implementation of National Instrument 43-101 guidelines and CIMM Definition Standards for Mineral Resources and Mineral Reserves, adopted in December, 2005, and, as a further benefit, to obtain a quantity of material representative of the principal mineral structures for use in further metallurgical testing.

The field program involved preparatory work in accordance with the recommendations of the District Inspector of Mines, to repair access roads and to de-water and rehabilitate mine workings that had been closed since 1989. The portal area of Level 1 had been dammed by slough that had trapped as much as two metres of water. Many of the portal timbers had also been damaged and had to be cleared and replaced. Inside the mine, conditions had to be inspected, loose material scaled from the back and walls, air quality determined, and rotted planks removed and/or replaced. An excavator was brought in to the site to facilitate road repairs and to move timbers and pipes near portals before the sampling work began.

Sampling commenced when underground conditions were satisfactory. A three person sampling crew was assembled comprising a professional geologist (Ostensoe) and two helpers, one of whom (J. Laird) is a geologist, and the other (M. Bratlien), is a prospector who, despite a long history of involvement with the Treasure Mountain area and who possesses a good understanding of the objectives and requirements of the task, is an "insider" and thus was not available as a sampler or sample handler.

The mine samples that formed the basis of the Livgard resource estimate (Livgard, 1989, Appendix 1 of this report) were taken in the various drifts and raises at one metre intervals while mining was in progress or very soon after. The "historic" resource calculation also included data from a number of short test holes that were placed to intersect the vein(s) where it was situated in the drift walls and to check its location and character between levels of the mine. The database of mine samples totals about 800 and it was postulated that a re-sampling of about 10% of the original number, taken without particular reference to the database in order to avoid, or at least minimize, biases, would suffice to give credibility to that data. The various raises and Levels 3 and 4 of the mine workings were inaccessible and could not be included in the program of check sampling.

Sample sites were selected with the following criteria: the vein was identified by visual inspection on the basis of its appearance in contrast to the wall rocks, presence of base metal and other sulphides or products of their oxidization, presence of limiting fractures and/or shears, and distance of about 5 metres from another re-sample site. Samples were taken using standard sampling tools: chisels and hammers were wielded to produce a continuous chip sample with weight of 1 kg or more from the sample interval. Chips were transferred to new standard plastic sample bags that were then closed with a "zip" tie. Bags were identified by temporary numbers that were recorded along with details of location and any other pertinent information in a waterproof notebook. The samples were accumulated underground for part of a shift and then conveyed to surface, placed in a locked vehicle and later taken to a temporary campsite near the Level 4 portal where drier conditions prevailed and a proper numbered assay tag replaced the temporary one. That campsite was normally secure: either supervised by an associate or closed to "outsiders". Visitors, who were very few in number, were outdoor enthusiasts who were only mildly curious about the mining activity and none expressed any particular interest in the project, the company or the samples.

Sample quality was influenced by the ability of the samplers to obtain consistent quantities of rock across the full width of the designated sample. The majority of the samples were taken from the high "back" (ceiling) of the drifts and the miners and samplers moved various pieces of staging to provide a platform from which the samplers could work somewhat comfortably while chipping. The sampler who held the sampling chisel or moil in one hand and struck it with a two- or four-pound hammer often had difficulty controlling the size of the chip or the path of the resulting chip that ideally would fall on to a plastic sheet or into a goldpan held close to the chisel by his partner. Inevitably some chips landed in the ditch and were not retrieved and some parts of the overhead could not be included in the sample. Nonetheless, vein material, comprising brittle sulphides, carbonate minerals and talcose or gougy gangue, was reasonably easily chipped. Presumably, the original sampling crew worked with similar materials but they would have been sampling freshly exposed rock and probably had better lighting and access to a broader array of tools and stagings.

Samples were taken with a certain sense of urgency with concern to not prolong the program: crew members, particularly the miners, were very accommodating, but had other pressing obligations and due to their previous involvement (in the case of the mining engineer), special skills and familiarity with the property could not easily be replaced. Diligent efforts were forthcoming and it is believed that the samples taken were of good quality and were suitably representative of the vein(s). Figures 9a – 9d illustrate 1988 sampling of Level 1 – 4 with 2007 sampling of levels 1 and 2. (Refer to the CD-ROM version to see details.)

The main Treasure Mountain vein, "C" vein, occurs close to, and partially in, a distinctive orange to grey-green coloured, medium grained feldspar porphyry dyke of presumed Tertiary age (Black, 1952). The country rock comprises altered argillite, siltstone and minor arkose, a possible turbidite sequence. The vein is reasonably consistent in character but pinches and swells, possibly reflecting slight variations in the attitude of the nearby dyke. In some sections of the mine the vein is present as a mere knife-edge fracture and in other places it widens or is elaborately folded to exhibit widths of a metre or greater and in still other areas it is present as two, or even more, strands. The greater portion of the vein lies close to the hangingwall (i.e. south) surface of the dyke but there is also a footwall segment that contributes an important volume of potentially mineable material. Mineralization is similarly capricious and varies from dominantly massive sphalerite with little or negligible amounts of tetrahedrite and galena, to coarse galena. Textures vary from strongly banded to massive. Samples from deeper parts of the mine workings often carry high silver values and small to very small amounts of lead and zinc. Observations concerning the association of zinc to lead and silver are presented in a later section of this report.

The samplers were unable to reliably distinguish, or even guess at, relatively "higher" or "lower" grades within the vein, a factor that ensured objectivity in sampling. The presence of strongly coloured manganese-rich alteration and of braunite, a sphalerite look-alike mineral, would have made attempts to discriminate grades even more difficult. In some locations the vein was sampled in two or even three segments in an attempt to distinguish parts that appeared to be largely wallrock from obviously strongly mineralized parts. As a generality, the 2007 samples were taken across somewhat greater widths than were the original samples. Several samples were taken from sites for which there is no historic record of metal values: it is assumed that samples were taken and processed at the time of excavation and that details of the assay values have been lost or misplaced, or possibly the sites were simply overlooked by the original sampling crew.

7.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

Samples were taken from the mine to a temporary campsite near the Level 4 portal where they were given proper identification tags, sealed and placed in polyfibre bags (aka "rice bags"). Upon completion of the sampling program, samples were taken by the geologist by private vehicle to the analytical laboratory in Richmond, B. C. The laboratory was instructed to perform standard procedures of sample preparation and analysis by ICP-MS methods.

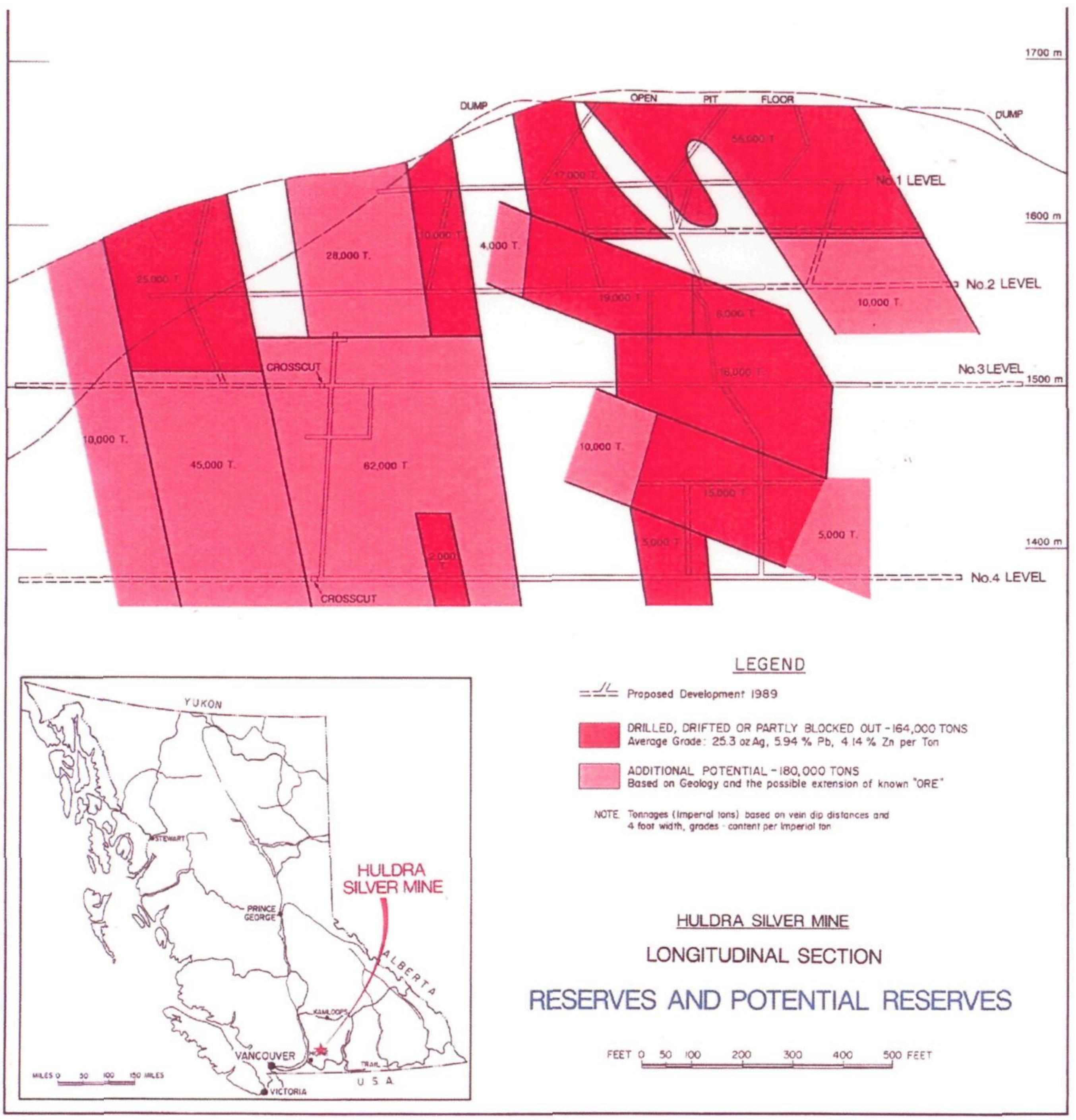


Fig. 6: Resource Blocks defined by Livgard (1988)

Following receipt of the ICP-MS analyses it was obvious that the metal contents, variously silver, lead and/or zinc, of certain samples exceeded levels that can accurately be determined by ICP methods. Of particular concern was the anecdotal and possibly faulty observation that at high concentrations silver tended to precipitate out of the solutions produced by the multi-acid digestion process. Samples with determinations that exceeded the upper detection limits for silver, lead and zinc, were re-analyzed by assay methods.

Samples were at all times until delivery to the laboratory in the care and custody of Erik Ostensoe, P. Geo., the Qualified Person who directed the sampling program at the Treasure Mountain site and personally delivered the samples to the analytical laboratory.

8.0 DATA VERIFICATION

Samples obtained from the 2007 program of work on the Treasure Mountain site were submitted to international Plasma Labs Ltd. in Richmond, B. C., a full service, ISO 9001:2000 certified company with many clients in the mineral exploration, mining and metallurgical fields. The author has toured the labs and observed facilities, procedures and personnel, all of which at the time of his inspection appeared satisfactory.

Seventy-nine chip samples were delivered to the laboratory. Samples were dried, weighed, and then crushed to pass through a one quarter inch screen. A 250 gram split was pulverized to 100% passing through a -150 mesh screen, and a 0.5 gram portion was digested in a multi-acid solution. The solution was then analysed for 30 elements by induced coupled plasma spectrometry. Results were certified by B. C. certified assayers. Reference: iPL Certificate #07G3198 in Appendix 2.

Analytical data, as expected, showed many samples with high values in silver, lead and zinc, the principal metals of interest, but many also reported high levels of manganese, aluminum, iron, antimony and cadmium. When ICP determinations showed greater than 500 ppm silver, a fire assay with gravimetric finish was also reported. Large discrepancies between the numbers reported by the two methods were attributed by the assayers to a tendency for silver if present in large quantities to precipitate out of the solute, with the result that the ICP analysis under-reports that metal. Ten sample "rejects" for samples with high silver contents, greater than the high detection limit for the procedure used, were re-analysed by fire assay with gravimetric finish method. Results consistently confirmed that the FA/gravimetric numbers were reproducible and that high silver values were being "low balled" by the ICP method.

Tables 2a and 2b illustrate the silver, lead and zinc analyses recorded for the original mine samples in the period 1987-1988, with the closest sample taken in 2007.

Table 2a: Sample Comparisons – Level 1

1988 sample and nearest 2007 sample	Location	Identity	Width (m)	Silver (opt)	Silver (g/tonne)	Lead %	Zinc %
4658	38 m from	C vein	0.5	56.73	1764.5	17.7	15.4
588059	portal		0.67		768.2	9.15	22
4663	42 m from	C vein	0.5	57.73	1795.6	17.2	14
588057	portal	HW wallrock	1		107.4	1.24	6.45
588058		FW C vein	0.38		180	2.59	9.38
4669	48 m from	C vein	1	6.36	197.8	1.23	11.3
588056	portal		0.48		6.9	0.08	0.38
4673	52 m from	C vein	0.5	16.48	512.6	4.33	11.8
588054	portal		0.8		251.2	1.38	11.02
588055			1		6	0.08	0.28
4675	56 m from	C vein	0.5	5.19	161.4	1.44	13.9
4676	portal	C vein	0.5	8.46	290.06	1.02	9.3
588053		Wallrock	0.94		9.5	0.04	0.3
588052		C vein	0.5		273.6	1.74	6.58
588051		Wallrock	0.5		26.8	0.58	0.88
34501	2 m SW of	C vein	0.6	0.65	20.2	0.54	0.24
588115	Sta. 1-14		0.5		623.5	6.49	1.62
34506	3 m North	C vein-split	1.14	102.08	3175	31.8	4.1
588114	of Sta. 1-14	C vein split	0.74		15	0.15	0.06
34511	2.5 m SW of	C vein+split	2	99.75	3102.6	23.8	10.8
588113	Sta. 1-15	C vein	1.65		864.3	13.48	2.16
588112		FW split	1.65		91.6	1	2.2
34517	2 m N of	C vein	0.9	94.5	2939.3	29.4	10.9
588111	Sta. 1-15	C vein split	0.5		1625.3	16.21	18.74
34520	4 m N of	C vein	0.4	18.67	580.7	5.56	7.85
588100	Sta. 1-15		0.4		6329.8	16.66	18.74
34525	9 m N of	C vein	0.3	50.17	1560.5	17.1	5.43
588099	Sta. 1-15		0.4		959.6	13.75	7.48
34530	14 m N of	C vein	0.42	29.75	925.3	6.02	16.8
588098	Sta. 1-15		0.5		200.3	1.26	6.19
34535	19 m N of	C vein	0.8	44.63	1388.1	12.7	14.2
588097	Sta. 1-15		0.7		20.6	0.2304	0.3716
34539	24 m N of	C vein	0.4	31.21	970.7	2.38	10.4
588096	Sta. 1-15		0.73		122.1	0.0682	2.25
33651	1.5m W of	C vein	1.1	18.08	562.3	2.23	5.11
588095	Sta. 1-16		0.52		15.1	0.1363	0.2482
33656	3 m NE of		1.1	11.84	368.3	2.89	5.98
588094	Sta. 1-16	HW	1.15		424.1	3.59	3.52
588093		FW	1.49		14.7	0.0873	0.0868
33666	8 m NE of	C vein	1	51.04	1587.5	10.1	5.32
588092	Sta. 1-16		1		986.4	15.02	3.76
34542	6 m N of	C vein	0.65	37.04	1152.1	1.98	25.2

1988 sample and nearest 2007 sample	Location	Identity	Width (m)	Silver (opt)	Silver (g/tonne)	Lead %	Zinc %
588091	Sta. 1-17		1.4		182.6	2.65	14.27
34547	2 m S of	C vein	0.28	16.33	507.9	4.35	11.95
588090	Sta. 1-18		0.55		317.9	3.48	0.5883
34802	3.5 m NW of	FW split	0.35	2.92	90.8	1.09	20.4
588089	Sta. 1-18		1		11.2	0.095	0.2337
34804	3 m NE of	C vein	0.3	22.75	707.6	12.7	5.02
588088	Sta. 1-18		1.6		169.3	1.2	8.57
34809	8.5 m NE of	C vein	1.6	85.46	2658.1	13.7	10.8
588087	Sta. 1-18	HW	0.55		183.9	5.68	4.45
588086		FW	1		32.4	0.2459	1.98
34814	14 m NE of	C vein	2.1	19.63	610.6	5.42	9.4
588085	Sta. 1-18		1.15		105.8	0.5761	6.38
588084			1.05		213.9	1.21	12.48
34818	Sta. 1-19	C vein	1.4	59.21	1841.6	16.2	14.6
588083			1.05		1679.6	13.56	4.35
588082			0.3		1380.3	10.04	16.18
34823	5 m NE of	C vein	0.2	59.21	1841.6	22	15.6
588081	Sta. 1-19		1.35		37.7	0.1898	0.3163
34826	9 m NE of	C vein	0.5	50.46	1569.5	19.4	4.12
	Sta. 1-19						
588080	11m NE of		0.6		209.6	8.73	0.6506
	Sta. 1-19						
34751	1 m N of	C vein	0.66	34.42	1070.6	12.2	5.9
	Sta. 1-20						
588079	2 m W of		1		6485.7	16.24	7.81
	Sta. 1-21						
34753	2 m N of	C vein	0.14	86.63	2694.5	41.6	5.55
588078	Sta. 1-21		0.4		33.3	0.333	0.2929
34758	7 m N of	C vein	0.4	67.96	2113.8	28.9	6.02
588077	Sta. 1-21		1.3		149.5	2.01	0.9756
34763	2 m SW of	C vein	0.34	51.63	1605.8	17.8	9.24
588076	Sta. 1-22		0.5		1348.2	11.29	2.11
34768	3 m NE of	C vein	0.2	55.42	1723.7	16.9	4.12
588075	Sta. 1-22		0.4		1489.5	11.85	8.86
34769	4.5 m N of	C vein	0.18	12.13	377.3	2.13	6.89
588074	Sta. 1-22		0.28		303.4	2.46	5.68
34774	Sta. 1-23	C vein	0.1	28.29	879.9	3.12	16.41
588073			0.1		510.4	2.05	0.9788
34778	5 m E of	C vein	0.1	12.1	376.3	2.7	0.84
588072	Sta. 1-23		0.28		4246.2	18.76	5.91
34787	10 m E of	C vein	0.2	25.96	807.4	9.85	1.03
588071	Sta. 1.23		0.42		1835.4	16.43	2.62
34786	5 m W of	C vein	0.1	111.42	3465.5	32.6	2.32
588070	Sta. 1-23		0.28		889.1	16.5	1.74

1988 sample and nearest 2007 sample	Location	Identity	Width (m)	Silver (opt)	Silver (g/tonne)	Lead %	Zinc %
588091	Sta. 1-17		1.4		182.6	2.65	14.27
34790 588069	2 m NE of Sta. 1-24	C vein	0.45 0.34	65.92	2050.3 8368.9	13 16.52	4.96 14.76
34795 588068	7 m NE of Sta. 1-24	C vein	0.35 0.4	123.96	3855.5 4675.4	51.5 15.64	1.41 1.36
34800 588067	12 m NE of Sta. 1-24	C vein	0.16 0.3	120.17	3737.6 3390.1	31.7 17.02	1.26 2.8
34654 588066	17 m N of Sta. 1-24	C vein	0.25 0.8	227.5	7075.9 32.4	14.7 0.25	8.16 1.88
36659 588065	4 m ENE of Sta. 1-25	C vein	0.75 0.76	60.67	1887 1765.2	18.85 11.85	8.09 4.86

Table 2b: Sample Comparisons – Level 2

1988 sample and nearest 2007 sample	Location	Identity	Width (m)	Silver (opt)	Silver (g/tonne)	Lead %	Zinc %
5013 589478	13 m E of Sta. 2-2	C vein	0.8 1.02	23.33	725.6 163.9	4.78 4.1	9.2 5.02
5021 589477	20.5 m E of Sta. 2-2	C vein	1.4 1.2	10.12 96.9	314.7 3322.3	1.61 1.77	14.7 6.54
5026 589476 assay	25.5 m E of Sta. 2-2	C vein	1.15 1.2	31.5	979.7 1136.4 1292.7	14.4 5.35	10.9 12
5031 589475	2 m W of Sta. 2-3	C vein	0.9 0.8	21.58	671.2 3009.7	5.5 2.01	15.5 10
5036 589474	2.5 m E of Sta. 2.3	C vein	0.65 0.64	19.75	614.3 145.1	6.8 1.3	8.25 9.05
5116 589473	4.5 m W of Sta. 2 - 8	C vein	0.32 0.33	5.27	163.9 129.9	0.94 1.25	6.8 37
5119 589472	2 m W of Sta. 2 - 8	C vein ?Wallrock?	0.45 0.55	66.5	2068.3 0.5	11.1 0.16	10.35 8.73
5126 589471	5 m NE of Sta. 2 - 8	C vein	0.8 0.93	5.42	168.6 254	2.14 0.78	12.8 23
5132 589470	11 m NE of Sta. 2 - 8	C vein	0.5 1.6	36.31	1129.3 76.2	2.86 0.23	15.8 8.52
5136 589469	2 m SW of Sta. 2 - 9	C vein	0.4 1	36.17	1125 1018.1	10.4 11	13.6 19
5141 589468 589467	2.5 m NE of Sta. 2 - 9	C vein FW HW	0.85 0.58 1.17	31.94	993.4 508.2 118.9	3.82 0.68 0.21	25 19 23
5145 589466	6.5 m NE of Sta. 2 - 9	C vein FW	0.8 0.5	3.11	96.73 169.8	0.51 0.61	3.2 7.71

1988 sample and nearest 2007 sample	Location	Identity	Width (m)	Silver (opt)	Silver (g/tonne)	Lead %	Zinc %
589465		HW	0.5		31.9	0.13	0.41
*TH - 10	near end of	C vein	2.44	48.42	1506	n/a	n/a
589463	stub	HW	1		29.8	0.04	5.42
589464		FW	1.6		971.2	0.09	18
*TH - 9	near entr. of	C vein	1.22	5.57	173.2	n/a	n/a
589462	stub		1.85		589.5	1.6	15.77
589461	in parallel drift	C vein	0.68		341.7	0.58	1.62
589460	end of drift	C vein	0.9		524.5	0.28	14.64
*TH - 7	proj'n of vein	C vein	1.22	5.43	168.9	n/a	n/a
589458	in crosscut	FW	0.74		2400.3	2.29	13.78
589459	west	HW	0.65		72.7	0.48	0.6035
23175	near end of	C vein	0.89	4.75	147.7	0.9	2.08
23174	crosscut		0.97	1.84	57.2	0.23	0.21
589455	west wall		1.36		52	0.7	0.3671
23173	near end of	C vein	0.58	24.5	1018	0.28	1.33
23172	crosscut		1.02	0.58	18	0.11	0.17
589456	east wall		1.75		19.8	0.24	0.0844
5450	main drift	C vein	1.3	23.01	715.7	7.4	1.3
5448	5 m west of		1.5	20.38	633.9	6.3	2.48
589453	Sta. 2 - 17		1.37		510	2.86	2.08
589454			1.3		80.2	0.99	0.7549
5463	1 m east of	C vein	1.5	5.86	182.2	2.41	0.29
589452	Sta. 2 - 20		1.48		333.1	5.13	0.1517
5482	3 m NE of	C vein	0.4	45.94	1428.9	18.4	1.68
589451	Sta. 2 - 23		0.78		423.6	8.26	1.79

Table 3a illustrates the comparative data for silver by ICP (Multi-Acid) method and Fire Assay with Gravimetric finish method. On average, for thirty-four samples the ICP (Multi-Acid digestion) silver values were 82% of FA/atomic absorption and 78% of FA/gravimetric silver values (see below). As shown in Table 3a, ICP values varied from 13% to 109.57% of the fire assays. Reference: iPL Certificate #07H3759 in Appendix 2.

iPL Laboratory report Certificate #07G3198 also highlighted the fact that many of the lead and zinc ICP analyses were in excess of the upper detection limits for that method, 10,000 ppm in each case. In order to obtain more precise values, the lab was then directed to re-analyse using assaying techniques all samples with lead and zinc values greater than 1%, as well as any with high silver values, greater than 500 ppm Ag, for which fire assay with gravimetric finish had not already been reported. Lead determinations were by multi-acid digestion and zinc, by wet assay and ICP spectrometry. Reference: iPL Certificate #08A0325 in Appendix 2.

Only small differences were reported between the lead and zinc values reported by ICP and wet assay/ICP methods. Table 3b, Lead Analyses-Assays, illustrates comparative data for lead determinations by ICP-MS and by assay methods. Lead values by ICP-MS are 99.47% of assay values. Table 3c, Zinc Analyses-Assays, illustrates comparative data for zinc determinations by ICP-MS and by assay methods: On average, ICP values for zinc are 102.27% of assay values.

Table 3a: SILVER ANALYSES AND ASSAYS - 2007 PROGRAM OF WORK
Comparison of Induced Coupled Plasma Method to Fire Assay with Atomic Adsorption Finish and Induced Coupled Plasma Method to Fire Assay with Gravimetric Finish

Spl No.	ICP (multi)	FA/AAS	RATIO ICP:FA/AAS	ICP (multi)	Fire Assay gravimetric	RATIO ICP:FA grav
588065	1607.1	1767.7	90.91%	1607.1	1765.2	91.04%
588067	1117.6	3387.4	32.99%	1117.6	3390.1	32.97%
588068	2075.6	4682.4	44.33%	2075.6	4675.4	44.39%
588069	1053.9	8370.3	12.59%	1053.9	8368.9	12.59%
588070	754.2	885.5	85.17%	754.2	889.1	84.83%
588071	1732	1831.8	94.55%	1732	1835.4	94.37%
588072	1583.4	4241.7	37.33%	1583.4	4246.2	37.29%
588073	490.7	506.7	96.84%	490.7	510.4	96.14%
588074	295.2	303.4	97.30%	295.2		
588075	1259.9	1478.6	85.21%	1259.9	1489.5	84.59%
588076	1311.3	1341.8	97.73%	1311.3	1348.2	97.26%
588079	983	6491.4	15.14%	983	6485.7	15.16%
588082	1482.9	1385.8	107.01%	1482.9	1380.3	107.43%
588083	1652.5	1698.7	97.28%	1652.5	1679.6	98.39%
588084	221	213.9	103.32%	221		
588090	293.8	317.9	92.42%	293.8		
588092	881.2	984.2	89.53%	881.2	986.4	89.33%
588094	378.1	424.1	89.15%	378.1		
588099	788.1	950	82.96%	788.1	959.6	82.13%
588100	1300	3626.3	35.85%	1300	6329.8	20.54%
588111	1418.2	1618.6	87.62%	1418.2	1625.3	87.26%
588113	854.6	850	100.54%	854.6	864.3	98.88%
588115	693.6	633	109.57%	693.6	623.5	111.24%
589453	508.2	510	99.65%	508.2	516.4	98.41%
589458	1841.8	2398.1	76.80%	1841.8	2400.3	76.73%
589460	436.7	524.5	83.26%	436.7	520.7	83.87%
589461	331.1	341.7	96.90%	331.1		

Spl No.	ICP (multi)	FA/AAS	RATIO ICP:FA/AAS	ICP (multi)	Fire Assay gravimetric	RATIO ICP:FA grav
589462	512.6	583.3	87.88%	512.6	589.5	86.96%
589464	895.1	971.2	92.16%	895.1	980	91.34%
589468	475.9	508.2	93.64%	475.9	501.3	94.93%
589469	935.9	1018	91.94%	935.9	1015.4	92.17%
589471	246.6	254	97.09%	246.6		
589475	2312.7	3009.7	76.84%	2312.7	3015.8	76.69%
589476	1176	1136.4	103.48%	1176	1142.5	102.93%

Table 3b: LEAD ANALYSES AND ASSAYS - 2007 PROGRAM OF WORK

Comparison of Induced Coupled Plasma Method to Fire Assay with Atomic Absorption Finish and with Gravimetric Finish - [samples >10,000 ppm lead]

Spl No.	% Lead ICP	% Lead AsyMuA	RATIO ICP/Assay	Spl No.	% Lead ICP	% Lead AsyMuA	RATIO ICP/Assay
588051	0.58			588097	0.23		
588052	1.75	1.74	100.57%	588098	1.24	1.26	98.41%
588053	0.04			588099	14	13.75	101.82%
588054	1.36	1.38	98.55%	588100	17	16.66	102.04%
588055	0.08			588111	16	16.21	98.70%
588056	0.08			588112	1		
588057	1.21	1.24	97.58%	588113	14	13.48	103.86%
588058	2.56	2.59	98.84%	588114	0.15		
588059	9.2	9.15	100.55%	588115	6.54	6.49	100.77%
588065	12	11.85	101.27%	589451	8.26	8.17	101.10%
588066	0.43			589452	5.13	5.02	102.19%
588067	17	17.02	99.88%	589453	2.86	2.88	99.31%
588068	16	15.64	102.30%	589457	2.09	2.1	99.52%
588069	17	16.52	102.91%	589458	2.24	2.29	97.82%
588070	16	16.5	96.97%	589459	0.48		
588071	16	16.43	97.38%	589460	0.28		
588072	19	18.76	101.28%	589461	0.58		
588073	2.04	2.05	99.51%	589462	1.59	1.6	99.38%
588074	2.44	2.46	99.19%	589463	0.04		
588075	12	11.85	101.27%	589464	0.09		
588076	11	11.29	97.43%	589465	0.13		

Spl No.	% Lead ICP	% Lead AsyMuA	RATIO ICP/Assay	Spl No.	% Lead ICP	% Lead AsyMuA	RATIO ICP/Assay
588077	1.97	2.01	98.01%	589466	0.61		
588078	0.33			589467	0.21		
588079	16	16.24	98.52%	589468	0.68		
588080	8.83	8.73	101.15%	589469	11	11.29	97.43%
588081	0.19			589470	0.23		
588082	10	10.04	99.60%	589471	0.78		
588083	14	13.56	103.24%	589472	0.16		
588084	1.16	1.21	95.87%	589473	1.25	1.28	97.66%
588085	0.58			589474	1.28	1.3	98.46%
588086	0.25			589475	2.01	2.11	95.26%
588087	5.71	5.68	100.53%	589476	5.35	5.3	100.94%
588088	1.17	1.2	97.50%	589477	1.77	1.78	99.44%
588089	0.1			589478	4.04	4.1	98.54%
588090	3.46	3.48	99.43%				
588091	2.58	2.65	97.36%	Average			
588092	15	15.02	99.87%				99.58%
588093	0.09						
588094	3.55	3.59	98.89%				
588095	0.14						
588096	0.07						

Table 3c. ZINC ANALYSES AND ASSAYS - 2007 PROGRAM OF WORK
Comparison of ICP Data to Fire Assay Data - [samples >10,000 ppm zinc]

Sample	% Zinc ICP	% Zinc Assay	RATIO ICP/Assay	Sample	% Zinc ICP	% Zinc Assay	RATIO ICP/Assay
588051	0.8771			588095	0.2482		
588052	6.69	6.58	101.7	588096	2.27	2.25	100.9
588053	0.3029			588097	0.3716		
588054	11	11.02	100	588098	6.24	6.19	100.8
588055	0.2775			588099	7.51	7.48	100.4
588056	0.3789			588100	21	21	100
588057	6.49	6.45	100.6	588111	19	18.74	101.4
588058	9.45	9.38	100.7	588112	2.17	2.2	98.6
588059	22	22	100	588113	2.11	2.16	97.7
588065	4.96	4.86	102	588114	0.0621		

Sample	% Zinc ICP	% Zinc Assay	RATIO ICP/Assay	Sample	% Zinc ICP	% Zinc Assay	RATIO ICP/Assay
588066	1.99	1.88	106	588115	1.6	1.62	98.7
588067	2.82	2.8	100.7	589451	1.79	1.83	97.8
588068	1.32	1.36	97	589452	0.1517		
588069	15	14.76	101.6	589453	2.08	2.1	99
588070	1.88	1.74	108	589454	0.7549		
588071	2.59	2.62	98.8	589455	0.3671		
588072	6	5.91	101.5	589456	0.0844		
588073	0.9788			589457	6.46	6.53	98.9
588074	5.72	5.68	100.7	589458	14	13.78	101.5
588075	9.03	8.86	101.9	589459	0.6035		
588076	2.04	2.11	96.7	589460	15	14.64	102.4
588077	0.9756			589461	1.6	1.62	98.7
588078	0.2929			589462	16	15.77	101.4
588079	7.79	7.81	99.7	589463	5.44	5.42	100.4
588080	0.6506			589464	18	18.35	98.1
588081	0.3163			589465	0.4122		
588082	16	16.18	98.9	589466	7.71	7.76	99.3
588083	4.41	4.35	101.4	589467	23	23	100
588084	13	12.48	104	589468	19	18.89	100.6
588085	6.42	6.38	100.6	589469	19	19.16	99.1
588086	2.05	1.98	104	589470	8.52	8.49	100.3
588087	4.5	4.45	101.1	589471	23	23	100
588088	8.63	8.57	100.7	589472	8.73	8.66	100.8
588089	0.2337			589473	37	35	105.7
588090	0.5883			589474	9.1	9.05	100.5
588091	14	14.27	98.1	589475	10	10.28	97.3
588092	3.73	3.76	99.2	589476	12	11.86	101.2
588093	0.0868			589477	6.54	6.49	100.8
588094	3.57	3.52	101.4	589478	5.08	5.02	101.2

Gold values for all samples were less than 0.4 g/metric ton.

The laboratory repeated the analysis of several samples and inserted standard reference samples and a blank sample into the batch of samples. This procedure is similar to that followed by all commercial laboratories and provides a comfort level concerning the reliability and reproducibility of data. The various duplicate and standard sample analyses are closely similar one to another and remove serious concerns about the laboratory procedures. Nonetheless, it is probable that the Treasure Mountain samples are chemically more complex than are reference samples and may react differently.

Data from the Livgard Consultants Ltd. sampling of the underground workings on Levels 1, 2, 3 and 4 of the Treasure Mountain mine were plotted on level plans at scale 1 cm to 2.5 metres. The Level 1 and Level 2 maps served as base maps on which were plotted the locations of the 2007 samples. The Livgard data, combined with a limited amount of diamond drill sample information and sampling in the various raises, in 1989 were the basis of a non-NI 43-101 compliant resource calculation: for calculation purposes the mineralized veins that averaged 0.6 metres in width were calculated to a minimum 1.22 metre width. The calculated resource was reported as 146,599 tons @ 25.37 opt silver, 4.53% lead and 5.29% zinc (Livgard, 1989) or, in metric terminology, 133,405 tonnes @ 27.91 opt (m) silver, 4.53% lead and 5.29% zinc The resource was characterized as 67,914 tons (61,740 tonnes) "proven" with 25.18 opt silver (27.70 opt(m)), 5.02% lead and 5.97% zinc, and 78,685 tons (71,532 tonnes) "probable" with 25.54 opt silver (28.06 opt(m)), 4.11% lead and 4.71% zinc. The Livgard Consultants Ltd. report is included as Appendix 1 of this report.

Note that the above-quoted resources were quoted in imperial measure and were calculated before implementation of National Instrument 43-101 and adoption of CIMM Definition Standards for Mineral Resources and Mineral Reserves. The figures by themselves should not be relied upon in an economic assessment of the Treasure Mountain mine.

The resources were calculated by an experienced registered professional engineer with many years experience in mining-related work, including mine geologist in underground gold and silver mines, and who was thoroughly familiar with procedures of resource and reserve calculation. Nonetheless, the author of this report has not conducted an independent assessment of the Livgard calculations and cannot verify the data, including knowledge of sampling procedures, analytical laboratory techniques and survey methods, that are the basis of those calculations. The author has, however, re-sampled portions of the underground workings of the Treasure Mountain mine and has reconciled his 2007 sampling with the earlier sampling. When both sets of data are calculated to the same 1.2 metre width and the individual values are compared, including by calculating the ratio of 1988 values to 2007 values, the spread of metal values, silver, lead and zinc, between samples taken in approximately the same locations is such that neither data set can be considered fully reliable (Table 3a, 3b, 3c).

The failure of the 2007 sampling to closely reproduce the original, ca. 1987-8, data may relate to (1) failure to correctly identify the locations of the original samples and to recognize the vein limits due to oxidation, accumulated mud or slime, (2) over- and under-representing while sampling certain portions of the mineralized structure due to unfamiliarity with the appearance of the vein, (3) the 2007 samples were chip samples whereas the earlier samples have been described as "channel" samples and, as a general rule, one may expect the latter to be more representative than the chip samples, (4) differences in laboratory preparation and analytical procedures between the earlier lab and the one that processed the 2007 samples, and (5) insufficient sampling over-all to accurately reflect the nature and variability of the somewhat complex vein structure and its metallic minerals. The latter factor is one that is commonly encountered in measurement of metal contents of narrow (and usually "high grade") mineral deposits.

Although it can for convenience be attributed to a "nugget" effect, it more accurately may be considered an inherent characteristic of such deposits. It is apparent that at Treasure Mountain silver occurs with complex chemistry and mineralogy that may result in wildly erratic distribution of values, in "native" form and in so-called silver minerals, including the "ruby silvers", proustite and pyrargyrite, the sulphosalts, including bournonite, boulangerite and tetrahedrite, (variety friebergite) and also lodges in the principal sulphide minerals, particularly galena.

Despite the above-cited *caveats*, and as detailed in the following section of this report, the Treasure Mountain mine is known to host a substantial quantity of "high-grade" silver-lead-zinc resources. Past mining-milling operations and more recent (1987) shipments of material from the surface opencut to smelters have confirmed the tenor of the deposit. A calculation of the extent of the mineralized vein on each level of the mine combined with the vertical continuity of the vein as demonstrated in the several raises between levels and those that extend from Level 1 to surface, are supportive of the Livgard resource figures in terms of volume. Modelling studies and more detailed, computer-driven calculations were performed as part of the program being reported and are included elsewhere in this report (section 12). Volume differences between the 1989 resource calculations and the current, 2008, resource calculations arise in part from the application of sophisticated computer-aided methods, more precise plotting of certain underground workings, use of different parameters, particularly metal values, more conservative projection of mineral zones due to guidelines of NI 43-101 and CIMM Standards, and the ability to be more objective in determining the distribution and limits of silver, lead and zinc values. The Terracad-generated block model incorporated almost all of the mineralization into "zones" without reference to vein width and consequently it shows a different morphology than did Livgard, et al. The Terracad resource measurement used different parameters, reflecting current assumed operating costs and metal prices, and applied techniques of variography to assign an averaged metal value to each block rather than the traditional system of polygons and "least squares".

9.0 ADJACENT PROPERTIES

Several historic silver-lead-zinc prospects are present in the upper Tulameen River valley area and formed what was once known as the "Summit Mining Camp" (Black, 1952, p. A119). Several of the prospects on the south slope of Treasure Mountain are included in the present Huldra Silver Inc. property and others are currently held by prospectors who perform annual labour. At present, Huldra is the only company active in the area, but, speculatively, further development of that company's property is highly likely to attract renewed prospecting and exploration to the whole Amberty Creek-Sutter Creek area.

Prospects in the Treasure Mountain area comprise narrow veins and sulphide stringers that are accompanied by varying amounts of quartz and carbonate. Host rocks are similar to those that host Huldra's "C" vein: thinly bedded argillite and tuff, and dykes have been reported. Veins vary in width but seldom exceed 50 cm. Low silver values, up to about 10 ounces per <u>ton</u>, have been recorded and lead and zinc are highly variable in the range of 1.0 to 15%. The importance of weakly developed faults has not been determined.

10.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Huldra Silver Inc. has amassed a quantity of mineral processing and metallurgical data. That material includes smelter shipments and several programs of preliminary laboratory scale tests. The company's mineral processing consultant, Jasman Yee and Associates Inc. (JYA) in 2006 reviewed previous work by Bacon Donaldson and Associates, metallurgical consultants, and in 2008 supervised additional test work by PRA Ltd. A schematic flow sheet has been developed to produce lead and zinc concentrates with metal recoveries in the mid-90% range (JYA, 2008). Much of the JYA report is included in this report (Appendix 4) and the entire comprehensive report, complete with test procedure details is retained by Huldra.

Huldra Silver Inc. in 1987 shipped a total of 15.09 tonnes of raw, partially cobbed "ore" to the East Helena, Montana, smelter:

Shipment no. 19870835, 3.28 dry tonnes, returned 0.01 opt gold, 153.85 opt silver, 55% lead, 0.40% copper, 8.00% zinc and 1.3% antimony.

Shipment no. 19870836, 11.81 dry tonnes, returned 0.01 opt gold, 200.55 opt silver, 69.4% lead, 0.40% copper, 4.00% zinc and 1.1% antimony.

Shipments totaling 390.73 dry tonnes of similar material were sent to the Trail, B. C. smelter and returned 0.017 opt gold, 96.55 opt silver, 31.3% lead, 0.47% copper, 6.9% zinc, 1.10% antimony and 0.15% arsenic.

The above-quoted tonnages and values are taken from original smelter-return sheets prepared by the respective smelters. The distinctly higher silver and lead values obtained from the shipments to the East Helena, Montana, smelter result from the shipped materials having been more carefully sorted than were those shipped to the Trail, B. C. smelter. Original smelter sheets were examined by the writers and are retained by Mr. Bratlien, president of Huldra Silver Inc.

Coastech Research Inc. in 1986(?) carried out preliminary metallurgical work but details and conclusions from that work were not available to the writer and have been, in any event, superceded by later testing work.

Bacon, Donaldson and Associates Ltd. in 1989 performed test work on four separate composite samples from different portions of the Treasure Mountain deposit. The results of their investigations were presented in a technical report titled "Investigation of Differential Lead and Zinc Flotation of Huldra Silver Composites" that was included as an appendix to a comprehensive report by Orocon Inc. dated May 26, 1989.

The Bacon, Donaldson and Associates Ltd. work was directed to determination of "...recoveries and grades of products produced by Pb-Zn differential flotation".

The samples were described as tabulated in Table 4:

Table 4: Composites

Composite number	Au opt	Ag opt	Cu %	Pb%	Zn%	Fe%	S%	Wt. lbs.	S.G.
1	0.002	20.178	0.15	6.40	11.52	9.60	8.86	50	3.27
2	0.010	31.266	0.12	6.60	3.32	8.00	4.60	91	3.16
3	0.011	18.312	0.19	4.80	14.72	11.60	10.92	143	3.40
4	0.008	22.770	0.10	1.00	0.46	11.60	2.53	32	2.99

A flotation procedure was developed using Composite 3, a high zinc product, that produced acceptable recoveries in marketable lead and zinc concentrates. Composites 1, 2 and 4 were tested with the same procedure. Results varied, with poorest performance being achieved from the lowest grade feed (Composite 4). The following table is from the Bacon, Donaldson report:

Table 5: Flotation Tests (after Bacon Donaldson and Associates Ltd. 1989)

Test No.	Composite No.	Product	Assays			Recovery %		
			Pb%	Zn%	Ag (g/tonne)	Pb	Zn	Ag
F5	3	Pb con	59.2	8.0	9107.7	85.7	3.2	83.9
		Zn con	0.7	51.9	213.3	4.8	94.0	8.8
		overall				90.5	97.2	92.7
F6	1	Pb con	72.0	1.4	5962.0	95.7	1.0	71.6
		Zn con	0.8	43.9	727.6	3.2	98.2	27.6
		overall				98.9	99.2	99.2
F7	2	Pb con	57.6	4.0	9639.5	96.2	13.5	97.5
		Zn con	0.9	32.3	164.5	96.2	13.5	97.5
		overall				97.4	97.1	98.8
F8	4	Pb con	44.8	0.7	9211.2	83.9	2.1	23.5
		Zn con	0.9	5.7	4611.7	7.9	87.2	55.9
		overall				91.8	89.3	79.4

The acid-producing potential of tailings from flotation test F1, Composite 3, was determined and found to be highly acid consuming.

The Bacon, Donaldson testing work is precisely detailed in their report. In their 'Conclusions and Recommendations' they state that "*A successful separation of lead from zinc by differential flotation was achieved with reasonable recoveries of lead, zinc and silver*" (BD & A, 1989, p 22).

Huldra Silver Inc. in 2006 commissioned a metallurgical and processing report from Jasman Yee and Associates Inc., ("JYA") consulting metallurgists, who, using the Bacon, Donaldson data, prepared a comprehensive schematic flow sheet on the basis of a processing plant with nominal capacity of 150 tons (135 tonnes) per 24 hour day, at 92% availability for 8 months of the year. That flow sheet was incorporated by AMEC Earth and Environmental ("AMEC") in the Draft Permit Applications for the Treasure Mountain mine. As part of their on-going work for Huldra Silver Inc., AMEC then engaged JYA to conduct test work on the 2007 batch of 78 newly collected samples from the mine. The objectives were to

> "*...duplicate the bench scale testing that had been used as a basis for flow sheet development and to generate samples of tailings for the following:*

Acid drainage potential testing
Tailings water quality determinations
Treatability assessment of the tailings water to meet CCME and BC discharge standards
*Solid-liquid separation testing to confirm that the tailings can be filtered for the dry stack"(*Yee, 2008, p. 2*).*

Results of the various tests of the tailings have been delivered to the Company's environmental consultant and are not part of this report.

JYA conducted confirmatory test work on materials obtained from the 2007 sampling program in order to check the suitability of milling and process metallurgy that had previously been recommended. The head grade of the master composite was gold - 0.16 g/mt, silver - 943.6g/mt, lead - 7.23%, zinc - 7.88%, and the silver:lead ratio was 4.2:1. Process Research Associates ("PRA") of Richmond, B. C. performed, under the guidance and supervision of JYA, a series of tests using the sample pulps from the 2007 program. The entire JYA report, dated February 15, 2008, without the appendices, is attached to this report as Appendix 4. In the 'Summary and Conclusions' section JYA indicate that the flow sheet presented in its 2006 report is viable. That flow sheet is reproduced as Figure 7 of this report and is included with the permission of Jasman Yee, P. Eng.

AMEC in their Draft Permit Applications discussed investigations concerning geology, resources, hydrology and water quality, wildlife and fisheries, soils, natural hazards, archaeology, mine plans, processing plans, tailings storage and impoundment, acid rock drainage (ARD), infrastructure including power supply, communications, accommodation, building requirements, water management, reclamation, closure and post closure maintenance. The Draft Application also included site drawings and an extensive water quality database. Much of the latter database was prepared by Huldra's environmental consultants, Entech Environmental Consultants Ltd. Ava Terra Services Inc. of Golden, B. C. reported on Snow Avalanche Exposure.

The Draft Permit Application was submitted in May, 2007 and re-submitted in enhanced form in April 2008.

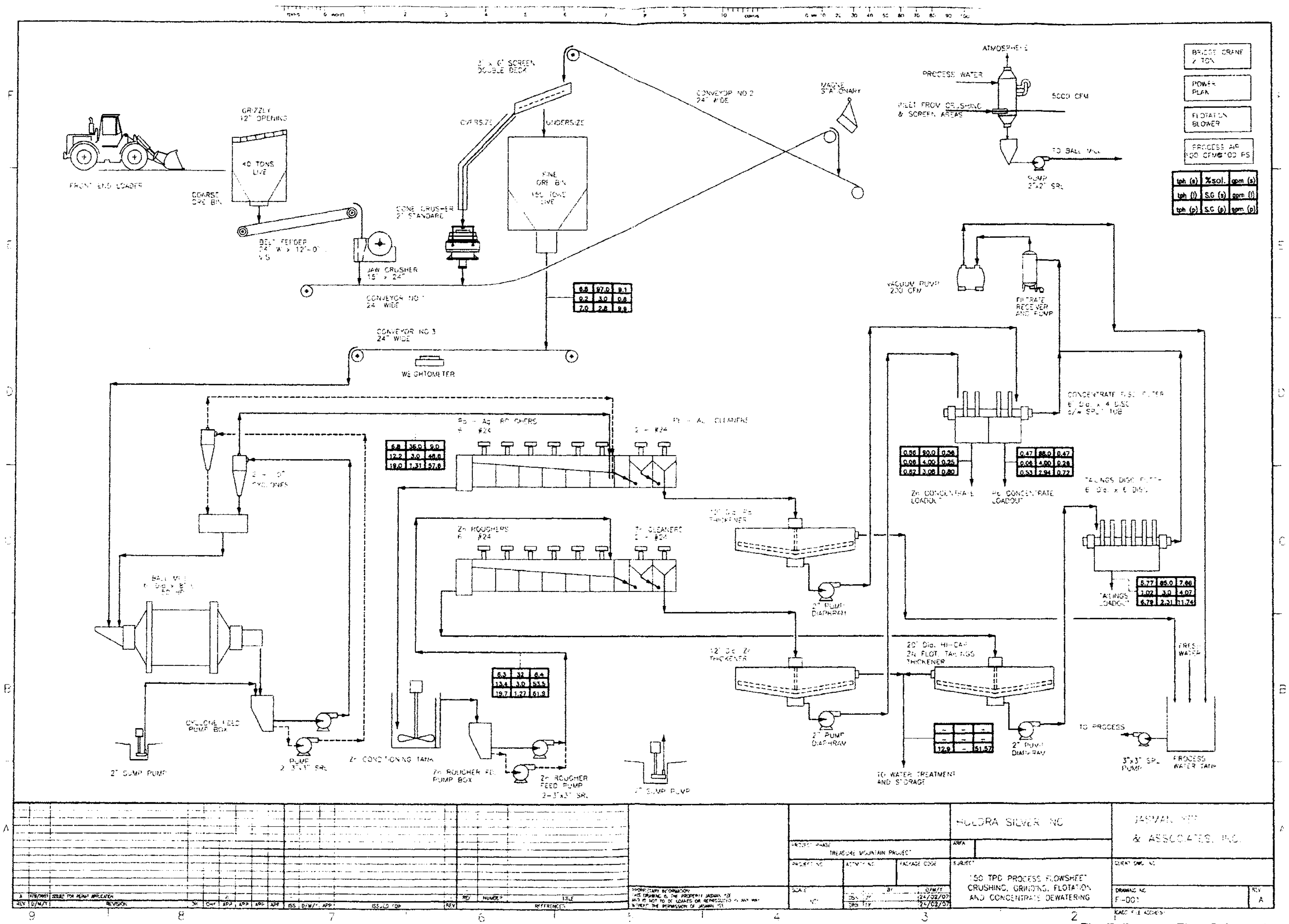


Fig. 7: Process Flow Schematic

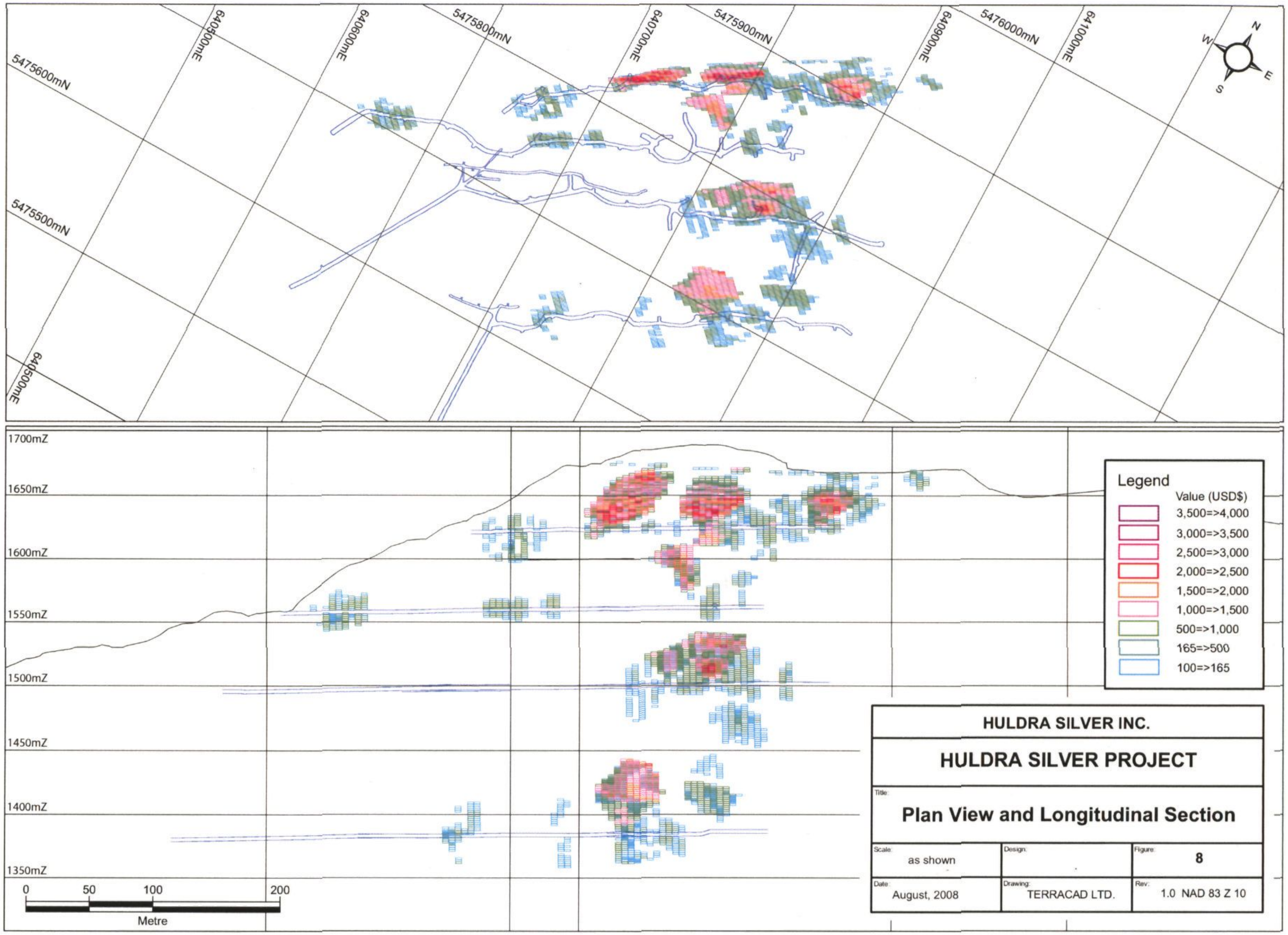


Figure 8: Hangingwall Vein Plan View and Longsection

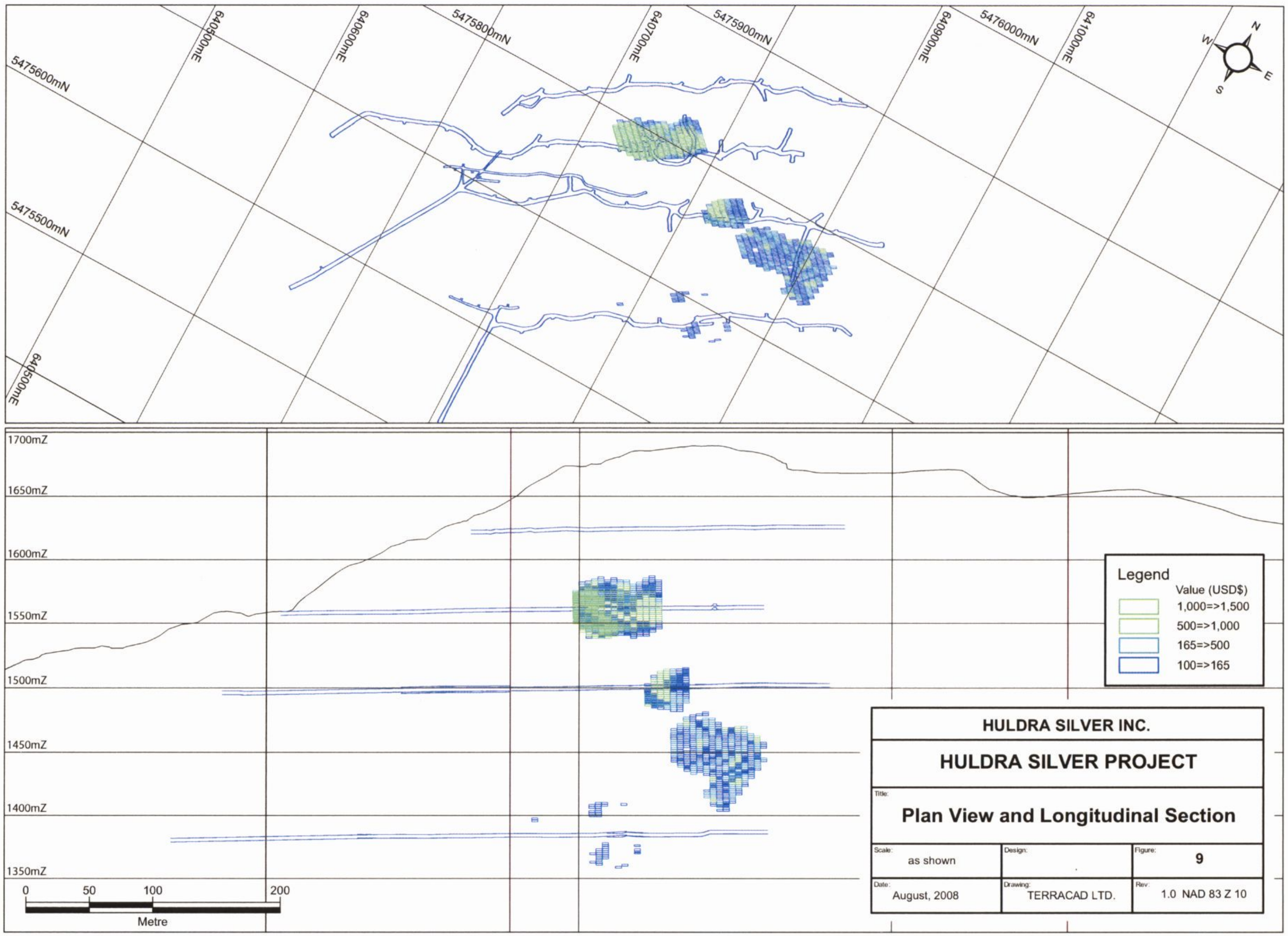


Figure 9: Footwall Vein Plan View and Longsection

11.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

11.1 Introduction

Livgard Consulting Ltd. in 1989 prepared a resource calculation for the Huldra Silver Inc. Treasure Mountain mine (Appendix 1). Available data and criteria and definitions then in general usage, prior to introduction of CIMM Definition Standards for Mineral Resources and Mineral Reserves, were used and a resource of "proven" and "probable" "reserves" totaling 133,037 metric tonnes grading 27.96 opt silver, 4.53% lead and 5.29% zinc was reported, along with "possible" resources of 148,000 tonnes. Those figures were prepared from sample data from 7,200 feet of drifts and crosscuts on four levels and over 1,800 feet of raises, plus 1153.5 feet of diamond drilling.

Note that the above-quoted resource figures are <u>not compliant</u> with current CIM Definition Standards for Mineral Resources and Mineral Reserves, nor with National Instrument 43-101 Definitions and although they were prepared by a qualified, experienced professional, should not be the sole basis on which to prepare an evaluation of resources of the Treasure Mountain mine. In particular, the category "possible" has a low level of geological confidence, has been virtually abandoned from current classifications of mineral resources, and is included only as part of the discussion of historic resource calculations.

With the senior author's supervision and participation, a limited program of sampling in the mine was completed in July, 2007, in order to evaluate the quality of the historic data and resulting estimates. Seventy-eight chip samples obtained from Level 1 and Level 2 were analysed and assayed by an accredited laboratory (Appendix 2).

11.2 Historic Resource Calculation

Livgard, et al., as detailed in the 1989 report that is included as Appendix 1, in calculating the Treasure Mountain mineral resources assumed a specific gravity of 3.5. Grade was calculated in troy ounces silver per metric tonne and lead and zinc, in percentages. "Ore" for the purposes of the calculation was defined as "...that tonne which contains ounces of silver plus percentage lead and zinc in excess of 15.0" and "Hence cut-off grade is ounces Ag + %Pb + %Zn of 15.0" (Livgard, 1989). The Livgard report also describes "proven", "probable" and "possible" ore in terms of then-acceptable definitions, none of which are in current usage, and it details certain assumptions concerning extensions of resource blocks. Areas that exceeded the cut-off grade but were based on unknown sampling and assaying reliability were reported only to the extent of 50% and were classified as "possible".

Drawings that illustrate the resource areas used in the Livgard calculation have been examined by the author (reference Figure 8). Livgard followed standard practices then in use in the mining and mineral exploration industries: resource blocks were drawn to extend assay information along geological/mineralogical trends and those extension distances were determined by the estimator's confidence in those trends and by proximity to assay information. A further consideration related to the practical aspects of "ore" extraction: stope development frequently demands inclusion of portions of a mineral zone that are of marginal or even negative value. If assay data were sufficiently closely spaced, projections were to the mid-point. If spacing was too great, values were extended 12.5 m from several adjoining "ore"-grade samples and classified as "proven", 12.5 m to 25.0 m was classified as "probable" and 25.0 m to 50.0 m was classified as "possible". Where only a small number of adjoining "ore"-grade samples were present or where the data comprised drill hole intersections as compared to drift samples, projections were from 12.5 m to 37.5 m. Blocks were identified by unique alpha-numeric designations and were quantified by x-y-z calculations. Tonnages were calculated by multiplying the x-y-z number in cubic metres by 3.5, the specific gravity. Grades were assigned by weighted averaging adjoining assays included in the block.

Note that the assumed specific gravity of 3.5 that in 1989 was assigned to Treasure Mountain resources may be greater than the actual specific gravity: a calculated figure, calculated using the 1989 assays for lead and zinc, but not taking account of other sulphide minerals known to be present, is 3.05, viz.

4.53% lead + 5.23% zinc + 86.88% gangue = (5.23% galena @ SG 7.5) + (7.89% sphalerite @ SG 4.0) + (86.88% gangue @ SG 2.70) = 3.05 SG.

Reference is, however, to the composite samples prepared for Bacon, Donaldson and Associates that were reported as having specific gravities of between 2.99 and 3.40 and a mean specific gravity of 3.2 (Table 4 of this report). If the latter gravity factor was applied to Treasure Mountain resources, the Livgard resource estimates would be reduced by about 8.6%. Further SG determinations are recommended in a later section of this report and the resource block model calculations resulted in a calculated SG for hangingwall zones, 3.30 and for footwall zones of 2.83

Experience gained from the 2007 program of check sampling and check assaying illustrates the difficulties that follow from the highly variable silver content of TM "ores". Other factors may have contributed to the discrepancies that were encountered.

11.3 2007 Program of Sampling

The senior author, in July, 2007 was engaged by Huldra Silver Inc. to complete a program of sampling in parts of the Treasure Mountain mine and to assemble certain data required to prepare a National Instrument 43-101 compliant report to be used in support of a Permit Application to operate a small scale underground mine coupled with an appropriate milling operation. Specifically, the report will be used as part of an Economic Evaluation, a necessary component of the Permit Application.

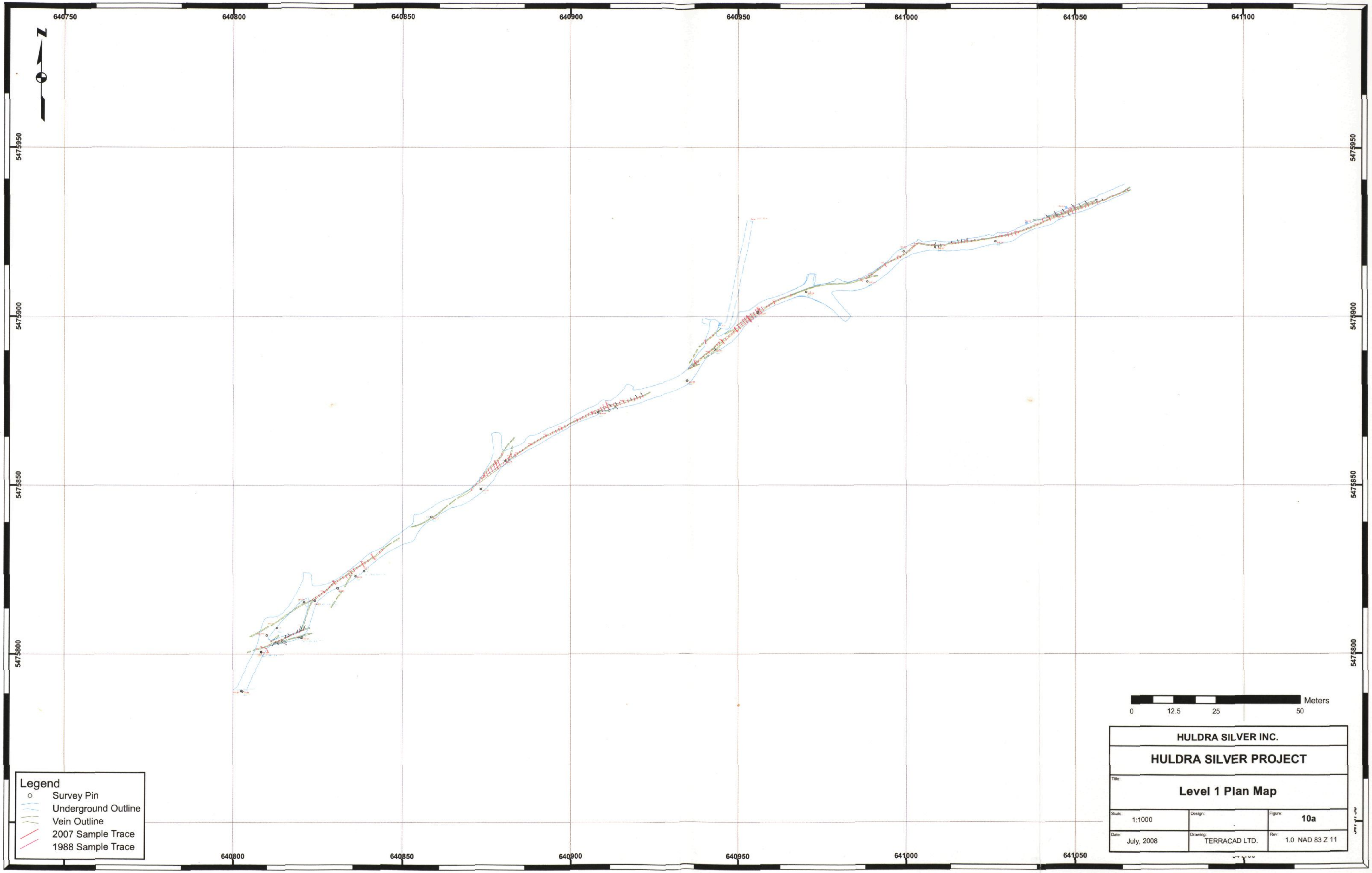
Field work was completed in the period July 13 to July 18, 2007 with a crew comprising a professional mining engineer, Al Beaton, P. Eng., a licenced shift boss with first aid ticket, Alec McPherson, and a consulting geologist, Erik Ostensoe, P. Geo. Magnus Bratlien, president of Huldra Silver Inc., and Jim Laird,geologist., also were part of the crew and assisted where needed. Samples were chipped and bagged by Ostensoe and by Laird, an experienced sampler who had previously worked on the Treasure Mountain property in 1986 - 1988. Mr. Bratlien, an "insider" by reason of being the president and a shareholder of Huldra Silver Inc., was not directly involved with any of the actual sampling but assisted the miners and samplers in moving and setting up staging platforms that supported the samplers and enabled them to comfortably reach overhead sites.

Levels 1 and 2 of the mine workings, in accordance with instructions from the District Inspector of Mines, were made accessible and safe by Mssrs. Beaton, mining engineer, and McPherson, registered shift boss. The portal area of Level 3 was deemed to be in an unsafe condition and remediation would have required more resources and time than were available. The principal concern with Level 4 was air quality: parts of the workings lacked natural ventilation and may have accumulated quantities of gases or simply have been depleted in oxygen due to water flow and/or absorption by moulds and other organisms. Levels 3 and 4, as part of the 2007 program, were rendered inaccessible by the installation of formidable timber structures. Assay samples were taken from Levels 1 and 2 from locations that were determined by the geologist (EAO) whose objective was to obtain between 75 and 100 samples from mineralized portions of the Treasure Mountain structure. Level plans at scale 1: 250, prepared subsequent to the 1987-1988 work, were used for reference to locations of samples and as a guide to assay information. Several samples were taken from sites for which there were no corresponding 1988 samples.

Seventy-eight samples, as detailed in Figures 9a (Level 1), 9b (Level 2), 9c (level 3), and 9d (4) and in Table 2, were obtained.
Note that 2007 sample locations are plotted on Figures 9a – 9d, Level Plans, that are included in the CD-ROM version of this report. An example of that information is reproduced in Figure 9.

Samples were chipped variously from the walls and "back" of the drifts using moils and chisels that were struck with a small (1.5 kg.) hammer. Chips and other fragments were collected in a gold pan held close to the sample site or were allowed to fall onto a clean plastic sheet and then were transferred into a clean (new) plastic sample bag. Bags were given an appropriate number and immediately closed with a temporary tie. Periodically the samplers would convey the accumulated samples to surface where they were placed in a locked vehicle. At the completion of the shift, samples were taken to the campsite at the Level 4 portal and in due course given an identifying numbered tag. Samples were taken, after completion of the field work, by private vehicle to the iPL Laboratories Ltd. laboratory in Richmond, B. C. Samples were in the control of the geologist at all times until delivery to the laboratory. One sample that inadvertently was received at the laboratory without adequate identification is believed to have comprised character specimens that were intended to be used for reference purposes.

iPL Laboratories Ltd., an ISO 9001:2000 certified company that offers a full range of analytical services, was instructed to analyse all samples for 30 elements using a standard inductively coupled argon plasma with optical mass spectrometry finish technique (commonly referred to ICP-MS). The method was effective within limits of detection but many of the samples contained metals, in particular silver, lead and zinc, in excess of the upper limits. Also, the initial analytical results were somewhat different from the corresponding data from the earlier, 1988, work, necessitating first a small amount of check assaying of ten samples for silver by fire assay and gravimetric methods, and then a larger amount of assaying of all samples that on initial analysis had returned values above certain limits: 500 ppm Ag, 1% lead and 1% zinc. It was speculated that in the digestion stage of the analytical process high silver contents could result in formation of insoluble silver compounds that then precipitated and were removed from the solution. That solution was depleted and the silver analyses by ICP-MS were grossly inaccurate, in many cases varying from the fire assays by more than 100%. Other factors undoubtedly contributed to the inconsistent analytical results.



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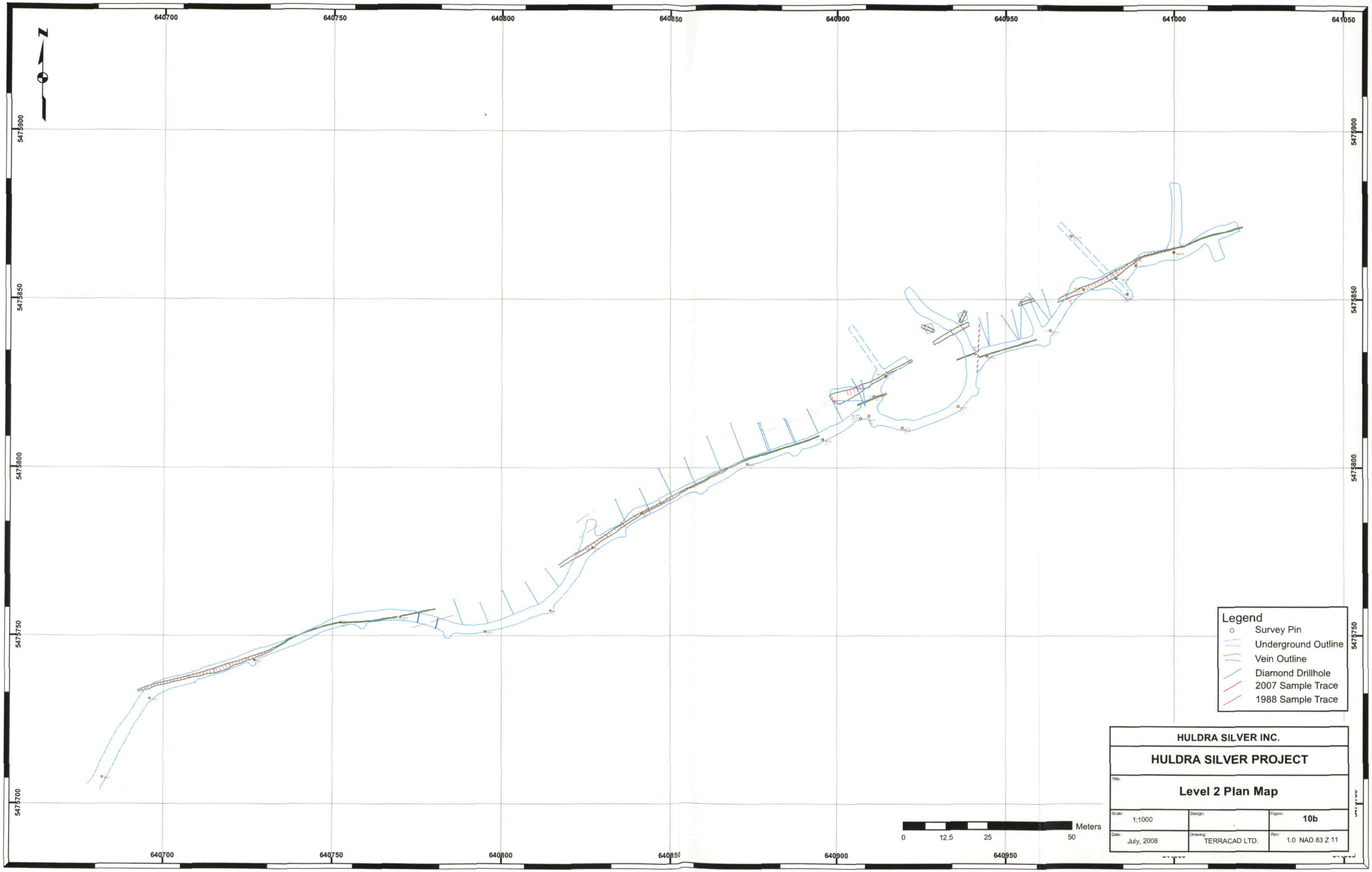

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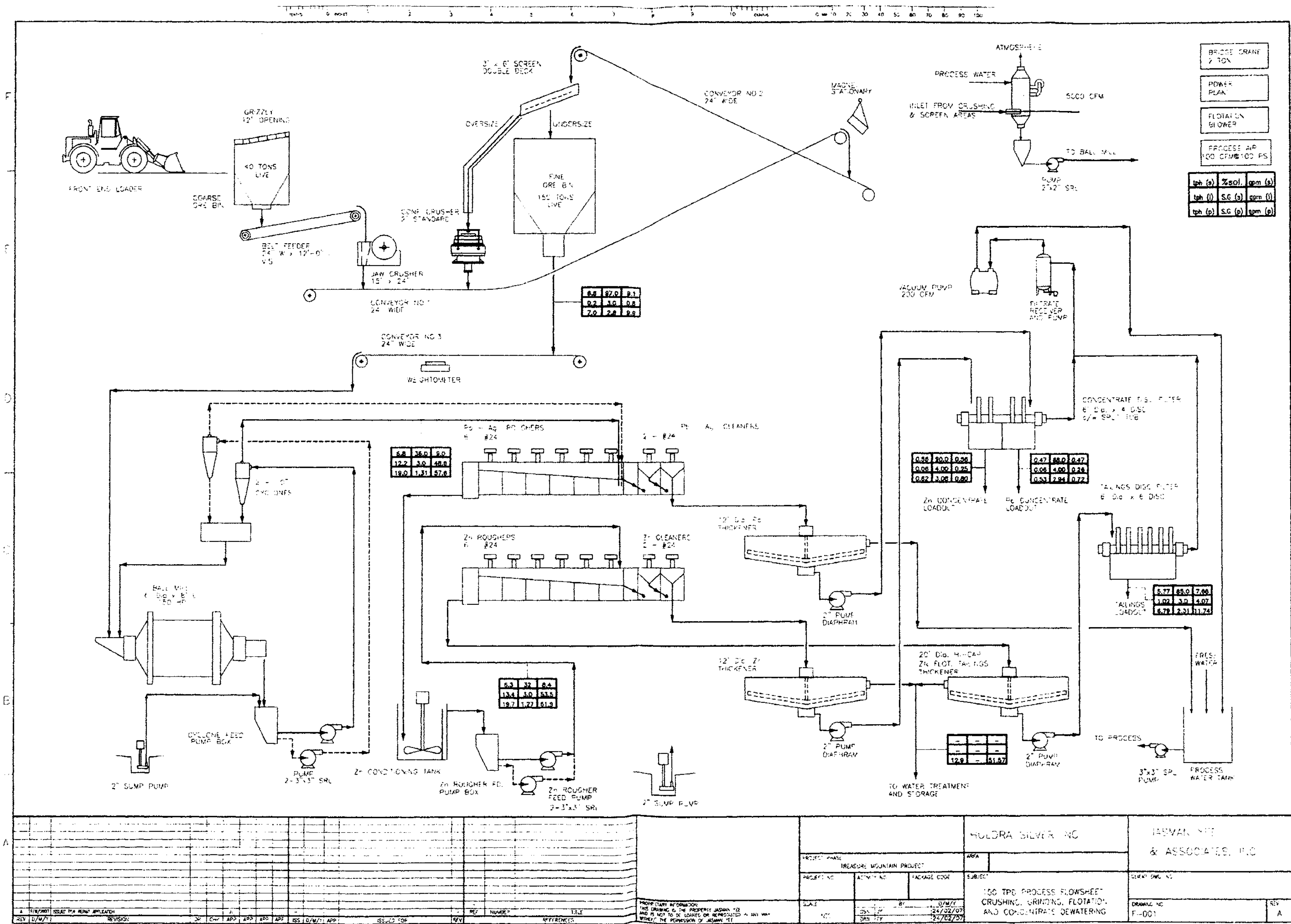


Fig. 7: Process Flow Schematic

The Treasure Mountain mine samples from the 2007 program were dried, crushed and a 250 gram split was pulverized and a 0.5 gram portion was digested in a multi-acid solution. That solution was aspirated into an inductively coupled argon plasma flame and an optical mass spectrometer measured the intensity of various spectra. The flame comprises spectral "windows" that are specific to each of 30 elements and the MS instrument is calibrated to "read" and record the amounts of each element in the solution. The laboratory routinely inserted blank, standard and duplicate samples into the stream of samples as a means of maintaining quality control.

The 2007 sampling program was not intended to fully duplicate similar sampling that had been completed in 1988 in conjunction with underground development. It was expected that by taking several samples from each of several separate mineral zones exposed in the drifts and comparing their silver, lead and zinc contents to the 1988 data, a level of confidence could be reached so that the detailed calculations by Livgard Consultants Inc. could be characterized as being acceptable for use in a preliminary economic assessment of a proposed small scale mining/milling operation.

The 2007 assays and the closest 1988 values are tabulated in Table 2 of this report. The 2007 metal values are frequently greatly at variance from the 1988 values. Even allowing for a number of "outliers" that might be expected, the divergence is difficult to reconcile. The variations in silver values may be attributed to the following factors: the presence of native silver, silver dissolved in other metallic mineral grains, particularly in galena, silver in tetrahedrite and other sulphosalt minerals, including freibergite, boulangerite, and bournonite, and the presence of grains of ruby silver minerals, particularly pyrargyrite, all of which cause or at the very least contribute to, a "nugget effect". The various minerals may contain varying amounts of silver and it is obvious that their presence in samples will influence greatly the assays. Less readily explained is the skewed distribution of lead and zinc values, with 1988 program samples generally having greater amounts. A sampling bias or a laboratory bias is suspected.

Four metallurgical samples prepared as composites from 2007 samples for use of Jasman Yee and Associates were calculated to contain from four to eleven opt silver to 1 per cent lead and from 1.17 to 22.9 opt silver to 1 per cent zinc (Table 6).

Table 6: (data from [1]Cominco and [2]Asarco smelter receipts (1988) and from [3]Jasman Yee and Associates, 2008)

Data source	tonnes	Silver oz./mt	% lead	% zinc	Ratio Silver:lead	Ratio Silver:zinc
Cominco[1]	390.73	87.77	31.3	6.9	2.84	12.72
ASARCO[2]	3.28	182	69.4	4	2.62	45.4
ASARCO[2]	11.81	139.8	55	8	2.54	17.5
Comp. 1[3]	n/a	107	20.15	4.67	5.3	22.9
Comp. 2[3]	n/a	24	5.5	6.18	4.36	3.88
Comp. 3[3]	n/a	18	1.64	7.11	11	2.5
Comp. 4[3]	n/a	17.4	2.12	14.92	8.2	1.17

The data shown in Table 6 illustrate the variations in metal values that characterize Treasure Mountain mineral zones.

The earliest samples of "C" vein zones, apparently numbering twenty, taken in 1985 from surface and trench exposures, were analysed by Chemex Labs. Ltd. of North Vancouver, B. C. (McDougall, 1987, p. 21). Samples taken in 1986 by Mr. Jim Laird from C vein following its exposure by trenching, were analysed by Min-En Laboratories Ltd., a company that no longer exists but that had a long history of providing satisfactory analytical services to the mining and other industries. The J. J. McDougall and Associates Ltd. report states, without supporting data, that: "As sections of the same zone were assayed in both years and numerous additional samples have been assayed at various laboratories, a good sampling and assay check has been provided with no major discrepancies apparent" (op cit. p. 21).

The 2007 samples were analysed by a modern, ISO certified laboratory that demonstrated good reproducibility of data from those samples and from standard and duplicate samples. There is, however, apart from the comparison of analyses of the 2007 samples that were taken from sites close to previously sampled (1987-1988) sites, no acceptable means of comparing the previous samples and/or laboratory to the 2007 work.

11.4 Audit of Tonnage Calculations

As part of his review of the Treasure Mountain mine data, the author examined the tonnage figures that were generated in 1989 by Livgard Consultants Ltd. (Appendix 1). Available data included a series of vertical sections that profile the vein, longitudinal sections that show the projections of "hangingwall" and "footwall" mineral zones between levels and to surface, and level plans that illustrate in detail the distribution of silver, lead and zinc values and that list in tabular form those values with corresponding assay sample numbers. All calculations were based on expanding the vein zone to an assumed mining width of 1.2 metres.

Livgard Consultants Ltd. presented their data in summary form in tables that are reproduced in Appendix 1 of this report. Drawings showing the outlines of various "ore" blocks were also prepared: the methodology.appears to have been acceptable and similar to that in general usage in the mining exploration industry at that time.

Note that Egil Livgard, P. Eng., the geological engineer who was responsible for the historic resource calculation that is included as Appendix 1 of this report, believes that, in part on the basis of his considerable experience in exploring, developing and extracting narrow vein silver-lead-zinc deposits, the NI 43-101 compliant block model and resource calculations presented in this report understate the potential dimensions of the Treasure Mountain deposit and do not adequately account for the realities of preparing and extracting such a resource.

Terracad Ltd. of Vancouver, B. C. was engaged by Huldra Silver Inc. to work with the senior author to prepare a three dimensional model of the Treasure Mountain mineral deposit, to perform such examinations and tests as are required to confirm metal contents, and to calculate mineral resources and mineral reserves in compliance with CIM Definition Standards for Mineral Resources and Mineral Reserves. McElhanney Consulting Services Ltd. provided digital files that included all relevant survey and analytical data from property work completed prior to July 2007. Mr. Bratlien, president of Huldra Silver Inc., provided original assay reports for essentially all Treasure Mountain property sampling in recent decades. None of the pre-1979 data was used in any of the resource studies and calculations.

The Terracad model of the Treasure Mountain "C" vein was developed by Farshad Shirvani, M. Sc., a geologist and GIS specialist who is familiar with most data management systems used in the mining industry, including Surpac, Gemcom and ArcInfo, and most aspects of model creation and resource measurement. Mr. Shirvani obtained technical training and an MSc degree at Shiraz University, Iran, has taken many courses in computer methods and programs and is thoroughly conversant with methods of managing data using computer techniques. He has been the manager and a principal owner of Terracad Ltd. for more than ten years and is in the process of applying for registration with the Association of Professional Engineers and Geoscientists of British Columbia. He worked closely with Erik Ostensoe, P. Geo., who is the Qualified Person for purposes of reporting Mineral Reserves and Mineral Resources for the Treasure Mountain project.

A Statement of Mr. Shirvani's Qualifications is included in this report.

<u>Definitions</u>

The designation "Mineral Reserves" cannot be applied to any of the Treasure Mountain mineralization: that category can be applied only to that portion of a mineral resource that is economically mineable as demonstrated "*...by at least a Preliminary Feasibility Study*" (CIM definition, p. 5). Although various engineering and economic studies have been directed to the Treasure Mountain project, no comprehensive economic and/or feasibility study has been completed and for that reason no part of the Treasure Mountain mineralization can be categorized as a "Mineral Reserve".

A "Mineral Resource" is a concentration of ...base and precious metals...*in such form and quantity and of such a grade or quantity that it has reasonable prospects for economic extraction*" (CIM definition, p. 4) and is further defined as "*an inventory of mineralization that under realistically assumed and justifiable technical and economic considerations might become economically extractable*" (ibid.). *Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Resource but has a lower level of confidence than a Measured Mineral Resource* (ibid. p. 3).

From the abundant evidence, including historic data obtained and compiled by Huldra Silver Inc., metallurgical studies, and smelter returns, coupled with the data provided by, principally, J. J. McDougall and Associates Ltd., and the senior author's personal familiarity with the upper portions of the Treasure Mountain "C" vein gained in the course of a program of limited re-sampling of parts of Levels 1 and 2 of the mine, the CIM Definitions Standards threshold definition of "Mineral Resource": "*an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions might become economically extractable*"; is, in the author's opinion, exceeded.

Due to the large and erratic discrepancies found in comparing historic and recent analytical data, particularly with respect to silver assays and analyses, no part of the

Treasure Mountain mineralization can be categorized as a "Measured Mineral Resource".

An "Indicated Mineral Resource" and an "Inferred Mineral Resource" have with justification been calculated with a level of confidence sufficient to allow mine planning and evaluation to proceed. The apparent overall spread of up to 20% between historic and recent silver analyses is greater than desired and should be reconciled at an early stage of any further technical and economic studies.

The Terracad modeling and resource calculation has incorporated virtually all currently available analytical data and has required certain assumptions concerning projections of data into areas for which no data are available. Projections have been based on available data relative to geologic trends, apparent continuities of mineral zones as observed on surface and in computer-generated models, on four levels and one sub-level plus several raises within the mine, and to a small degree, upon information from drill holes and from variography generated by computer-driven algorithms. Data have been diluted to a standard 1.2 metre width, the effective mining width as recommended by the consulting mining engineer, and for calculation purposes the wall rock dilution was assumed to be barren of metal values. The resulting resource calculations are believed to be conservative and, in part due to the author's lack of familiarity with the detailed geology and mineralogy of the mineral zones, there may be a tendency to under-report "Indicated" resources and to amplify the "Inferred" category. The likelihood of, in the future, raising a portion of "inferred" to "indicated" is judged to be "good".

None of the footwall mineral zone, due to the limited amount of data, was characterized as an “Indicated” resource.

11.5 Solid Models

A solid model of the Treasure Mountain deposit was developed from sampling and surveys of surface exposures (mainly the trench from which the test shipment was taken), four underground levels plus sub-levels and raises, and several drill holes. A data base was constructed from surveyed sites and existing drawings, with reference also to high resolution aerial photography that reveals topography and locations of mine entrances. Assay samples in this application are treated as "drill hole"-like intercepts with known coordinates for each end of the sample. Raise samples, even if they were taken in a horizontal cut, are treated as if each one extends half-way to the next sample. Having accurate spacial definition of each sample facilitates compositing of the values.

Two solid model domains were defined: hanging wall and footwall. Assay samples, unless otherwise described, are assumed to represent the entire width of the vein and to be oriented perpendicularly to the walls of the vein. In order to accommodate a recommended minimum 1.2 metre wide mining width it was necessary to extend, and hence dilute, many samples to 1.2 metres. The vein "envelope" was defined by placing a continuous surface onto each side of the assay sample array. Control was assured by visual inspection of the surface and obvious outliers that may have been taken for special reasons or that may have been mis-plotted were either corrected or rejected.

The block model comprises blocks with dimensions 5.0m by 2.0m by 1.5m and the model incorporates all mineralization for which data are available.

Four principal hangingwall zones, as illustrated in Figure 11, were defined by the boundaries and extensions of the deposit envelope and have been further identified as "A", "B", "C" and "D" zones. Similar zones had earlier been defined by Livgard Consultants Ltd. but with somewhat different dimensions and orientations. None of the pre-1979 assay data and none of the test hole nor sludge sample data were used in the model but the location of the walls of the principal vein in a critical section between Level 3 and Level 4 were in part deduced from drill hole intercepts. Although the limits of mineral zones were extended cautiously on the basis of data from test holes (non-core), vein contacts could not be determined with accuracy and the sludge samples were considered wholly unreliable with respect to determining metal values.

Figure 12 illustrates the footwall vein.

11.6 Sample Locations

Sample locations and data were provided in digital form from recent (1987-1988 data) mine plans that show locations of survey points and samples. Information is keyed to the UTM real coordinate mapping system.

11.7 Data Analysis

The database for the Treasure Mountain project comprises silver, lead and zinc assays from 1059 samples. Almost all samples were taken at one metre spacings from the vein at surface where it was exposed in 1986 by backhoe and hand trenching [that outcropping has since been removed and the trench deepened to as much as three metres below its original elevation] and from drifts, raises and sub-levels of the mine workings. 216 samples are from Level 1, of which 40 are from the near-portal historic workings that were reclaimed and sampled in 1986, and 176 were taken from the level which was extended by Huldra Silver Inc. in 1987-8. In general, samples were taken at one metre spacing wherever the "C" vein was identified. Two raises, 1-15 and 1-20, driven from Level 1 to surface to test the continuity and quality and tenor of the mineral zone, were sampled at one metre spacing as was a short blind raise, 1-23. Level 2 was also sampled at one metre intervals wherever the vein could be identified, along with a raise that connected Levels 1 and 2. The innermost 225 metre section of Level 3 also was sampled in detail although the western part of Level 3, which appears to have been stoped, is recorded in composites only. [The western part of Level 3 was not included in the present resource calculation.] Level 4 has five short sections that were sampled, of which two are footwall mineralization and three are hangingwall.

Basic statistical analyses of the Treasure Mountain assay data for silver, lead and zinc are tabulated in Table 7 and composite data frequency and cumulative curves for each metal are shown in Figures 15 (silver), Figure 16 (lead) and Figure 17 (zinc). Figure 18 illustrates the distribution of samples with the peaks representing the levels and surface locations and shows the anisotropic distribution of data.

Table 7: Hanging wall Composite Data: basic statistics

	Ag	Pb	Zn
Number of samples	1093	1095	1087
Minimum value	0.0014	0.0014	0.0011
Maximum value	420	67.5	36.4
Mean	28.07	5.44	4.46
Median	13.07	2.51	2.07
Geometric Mean	9.16	1.79	1.46
Variance	1731.97	61.62	35.50
Standard Deviation	41.62	7.85	5.96
Coefficient of variation	1.48	1.44	1.33
Skewness	3.33	2.98	2.21
Kurtosis	19.37	14.61	8.64
Natural Log Mean	2.22	0.58	0.38
Log Variance	3.60	3.57	3.56

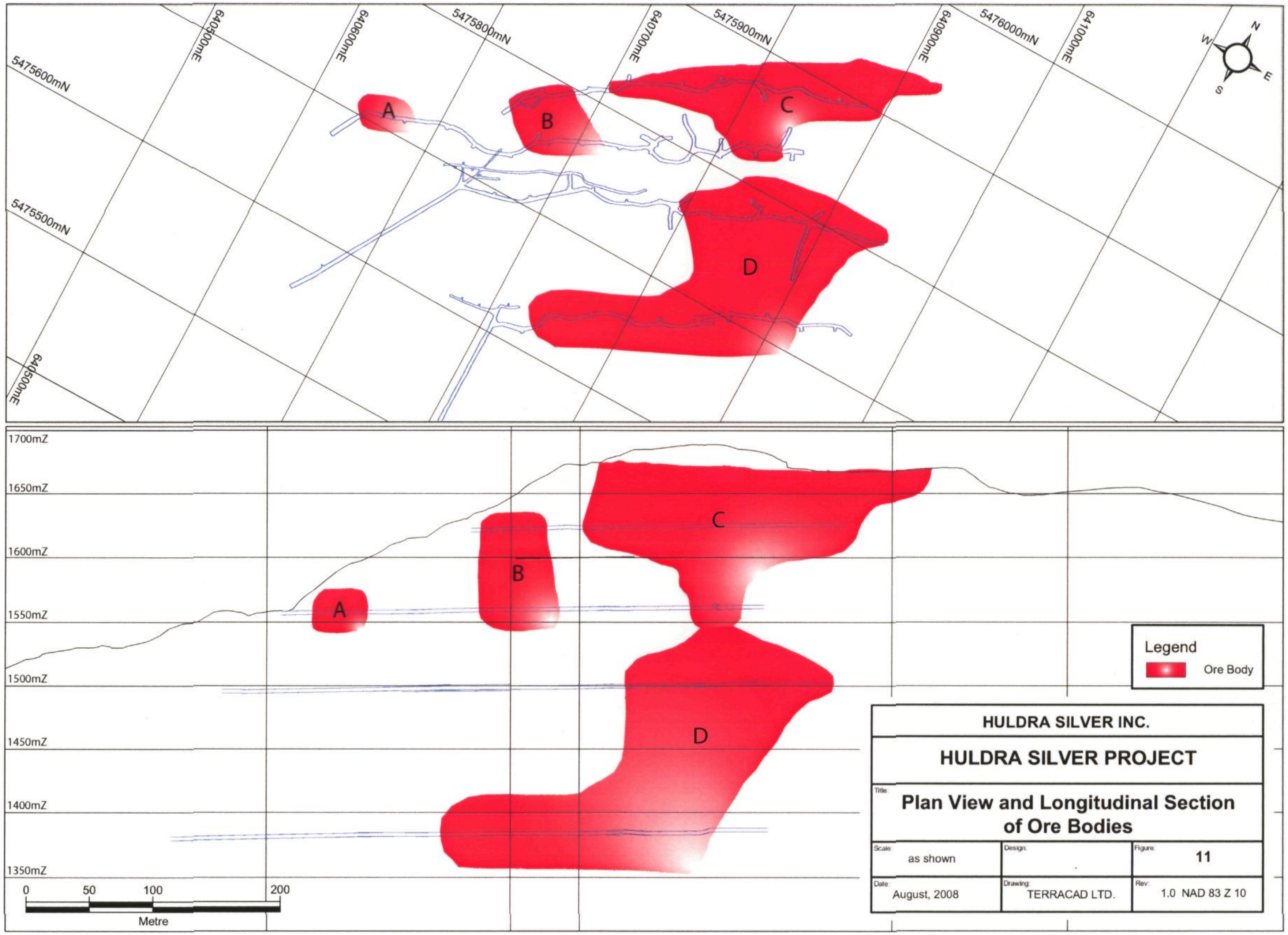


Figure 11: Hangingwall Vein Plan View and Longsection of Ore Bodies

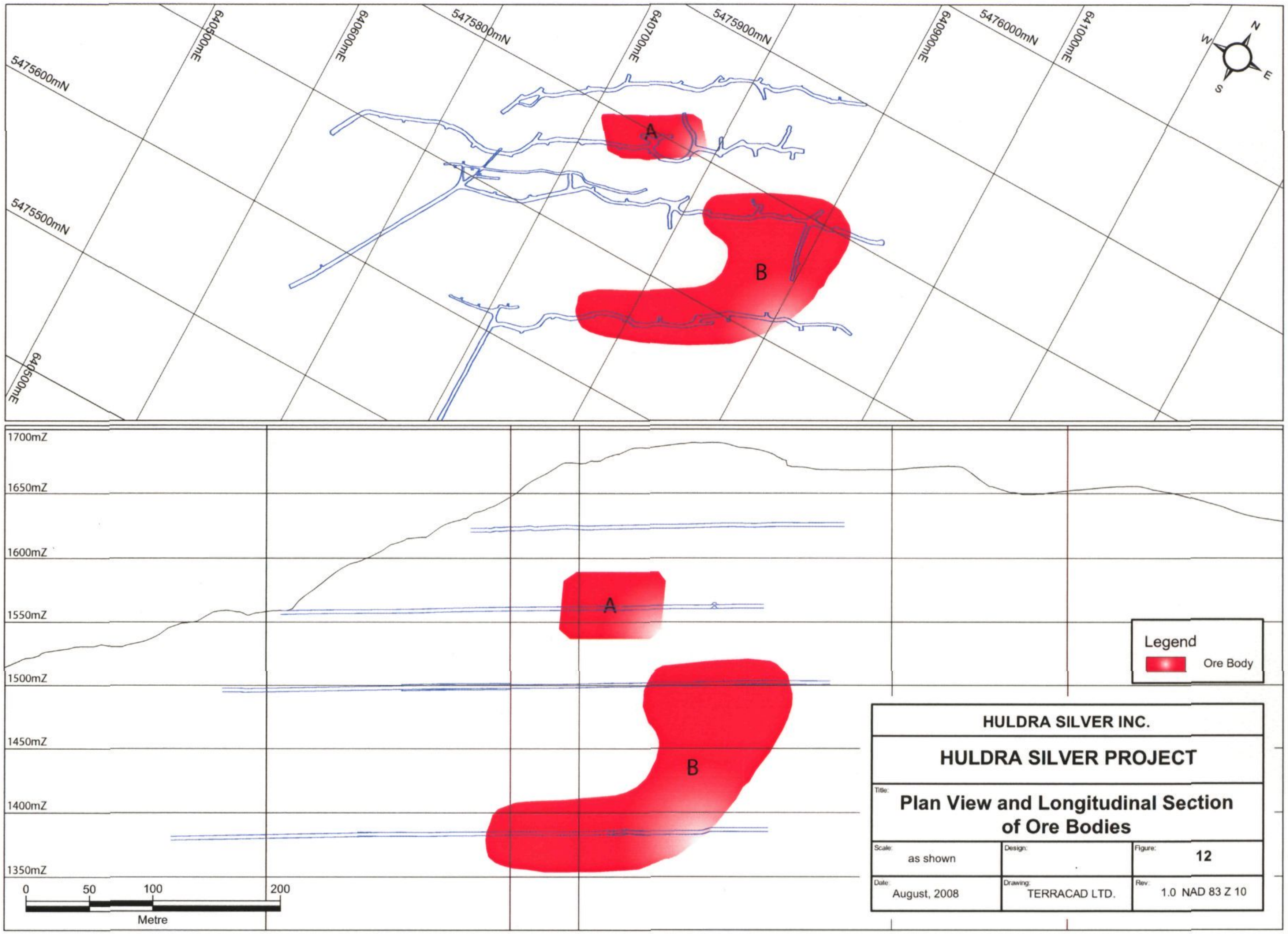


Figure 12: Footwall Vein Plan View and Longsection of Ore Bodies

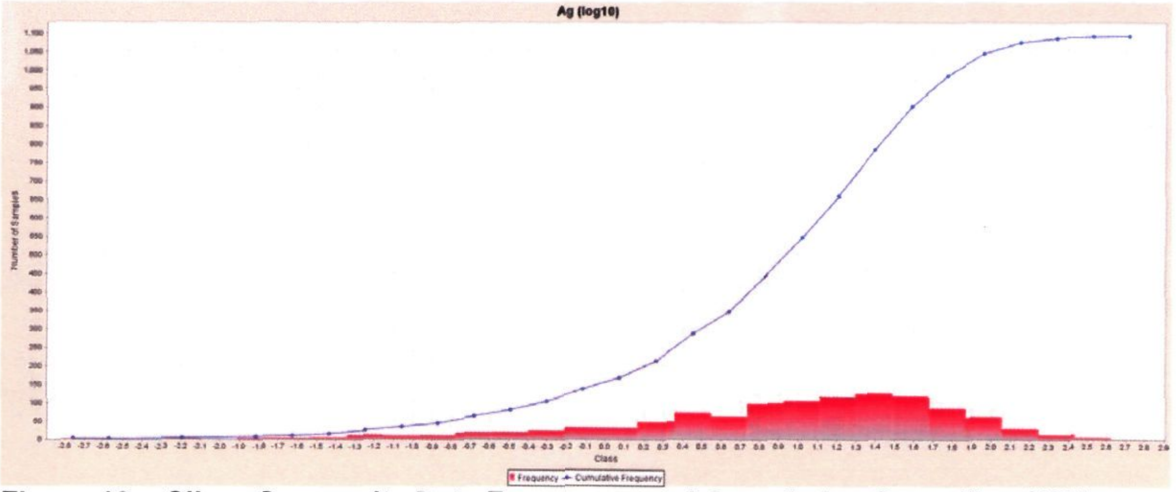


Figure 13a: Silver Composite Data Frequency and Cumulative Curve [log (10)]

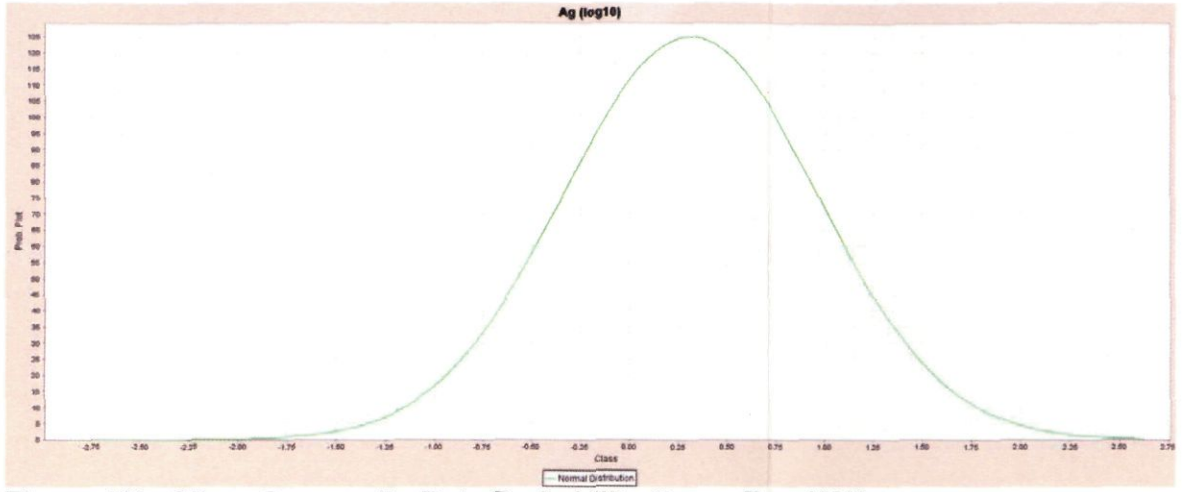


Figure 13b: Silver Composite Data Probability Curve [log (10)]

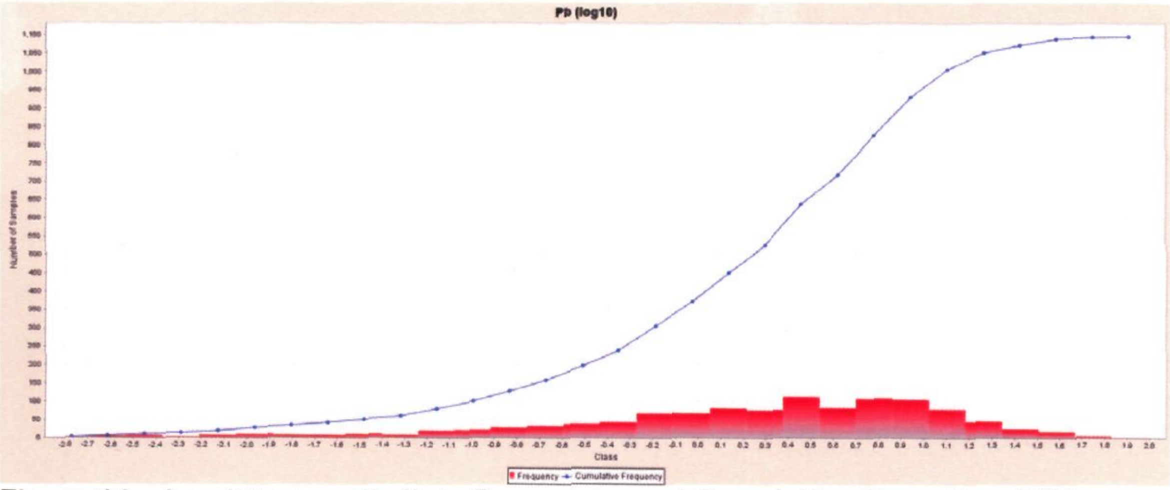


Figure 14a: Lead Composite Data Frequency and Cumulative Curve [log (10)]

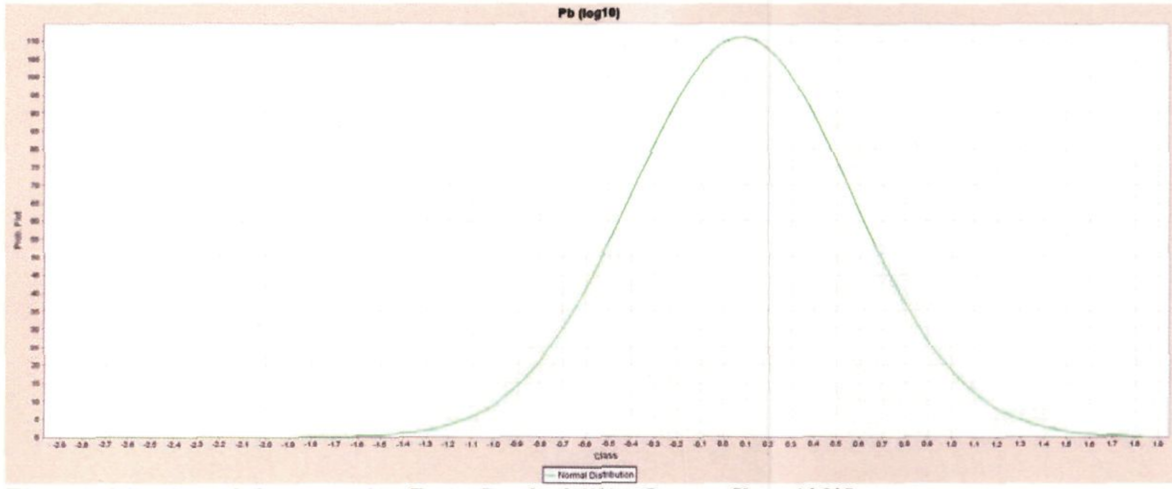


Figure 14b: Lead Composite Data Probability Curve [log (10)]

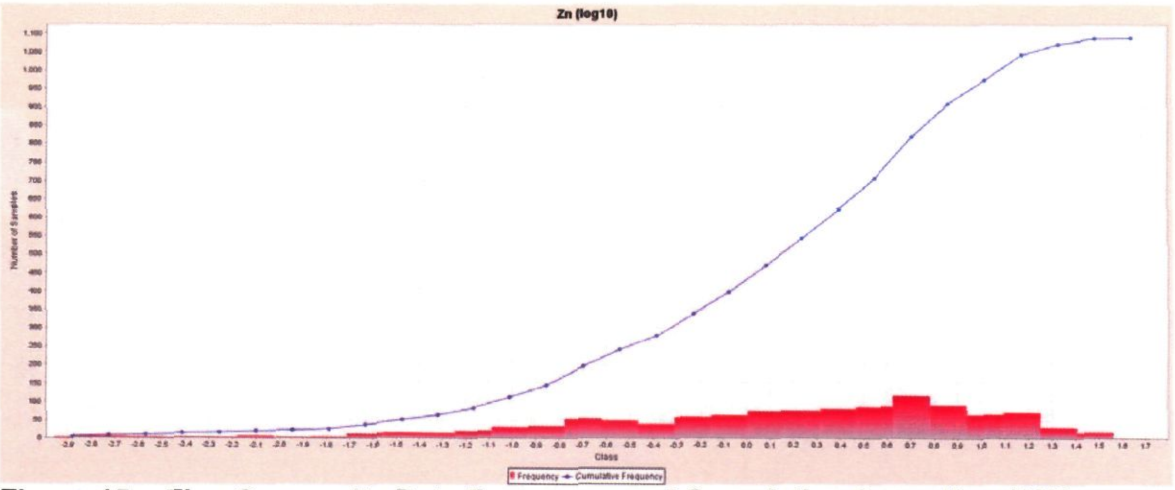


Figure 15a: Zinc Composite Data Frequency and Cumulative Curve [log (10)]

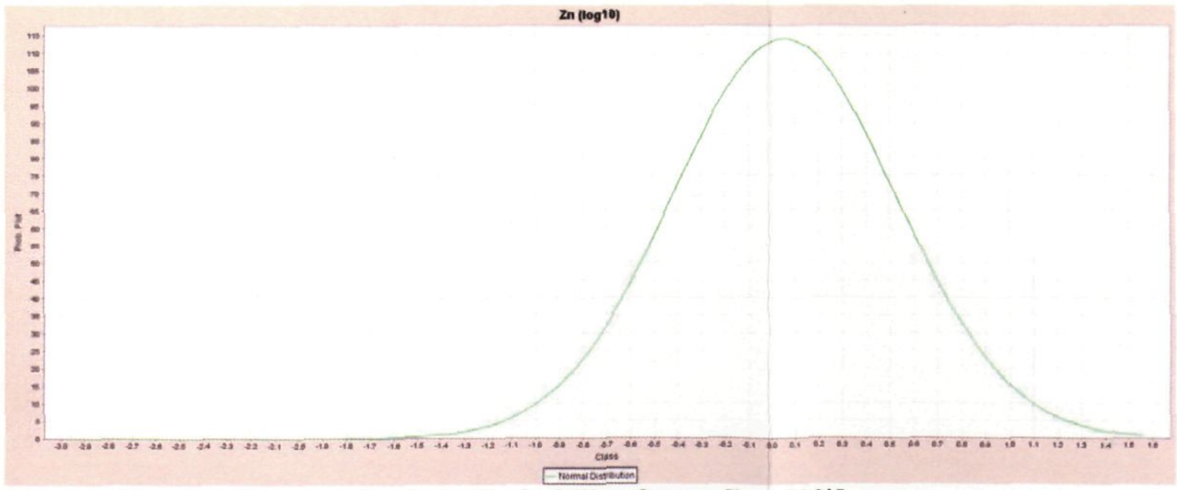


Figure 15b: Zinc Composite Data Probability Curve [log (10)]

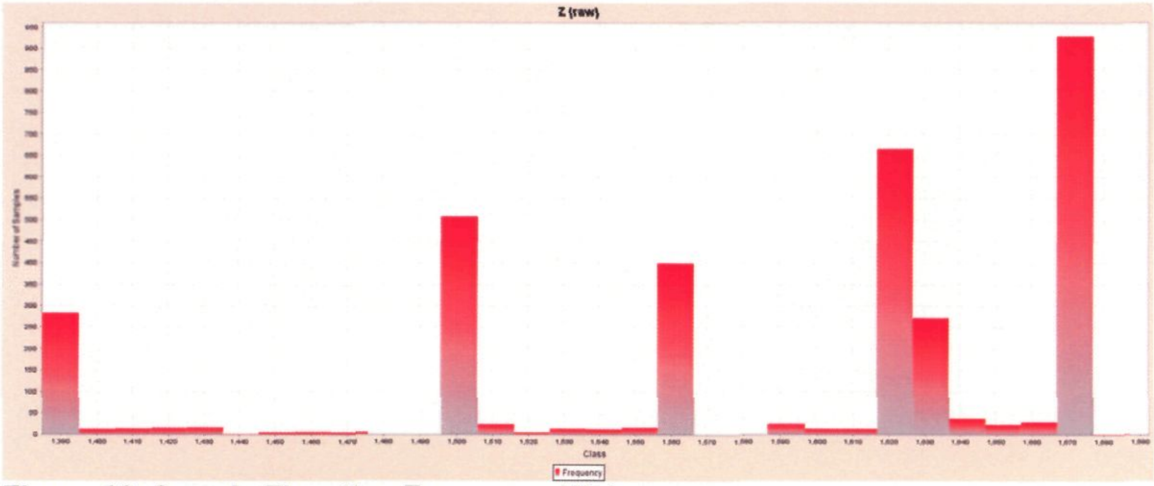


Figure 16: Sample Elevation Frequency Histogram

11.8 Geologic Model

The survey and analytical data enable construction of a conceptual geologic model of the Treasure Mountain structure. It can be characterized as a true vein closely related to a feldspar porphyry dyke and responsive to gentle folds or warps in that dyke. Figure 11 and 12 illustrates the vein in three dimensions. Mineral zones, for purposes of resource estimation, have been defined by the combined value of silver, lead and zinc.

11.9 Resource Calculations

Geologic continuity has been demonstrated at Treasure Mountain by trenching on surface and by drifting and raising in the veins. The principal geologic control of mineralization appears to be proximity to the feldspar porphyry dyke. Elevation influences metal distribution and there is a suggestion that more than one pulse of silver mineralization was involved. A correlation of silver with lead is strongly in evidence (Figure 19 – regression curve silver:lead), but the silver-zinc relationship is, at best, weak (Figure 20 – regression curve silver:zinc).

11.9.1 Variography

The attitude (strike and dip) of the "C" vein was determined by constructing the best-fitting surface on the vein. The azimuth of the vein is 056.3°, dip is -52.8° southeast. Semivariograms were constructed for each element on the basis of that orientation to show the direction and distances of maximum continuity. The spherical model of the major anisotropy was used to calculate the semi-major and minor search ellipsoid parameters for silver, lead and zinc (Table 8). Due to insufficient data, it was not possible to develop separate footwall vein variography and only the hangingwall variography was used in resource studies.

Table 8: Variogram Modelling Parameters

Silver			Lead			Zinc		
Model Type:		Spherical	Model Type:		Spherical	Model Type:		Spherical
Nugget:		0.42	Nugget:		0.34	Nugget:		0.25
Structure	Sill	Range	Structure	Sill	Range	Structure	Sill	Range
1	0.23	8.00	1	0.14	3.40	1	0.07	5.00
2	0.32	25.50	2	0.42	25.50	2	0.62	33.00

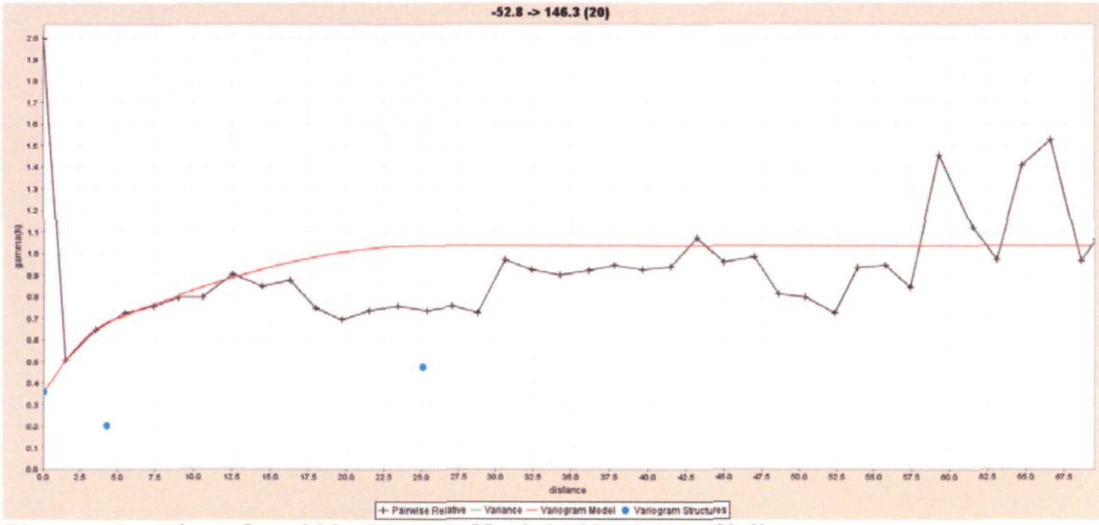


Figure 17: Silver Semi Variogram Model– Hanging Wall

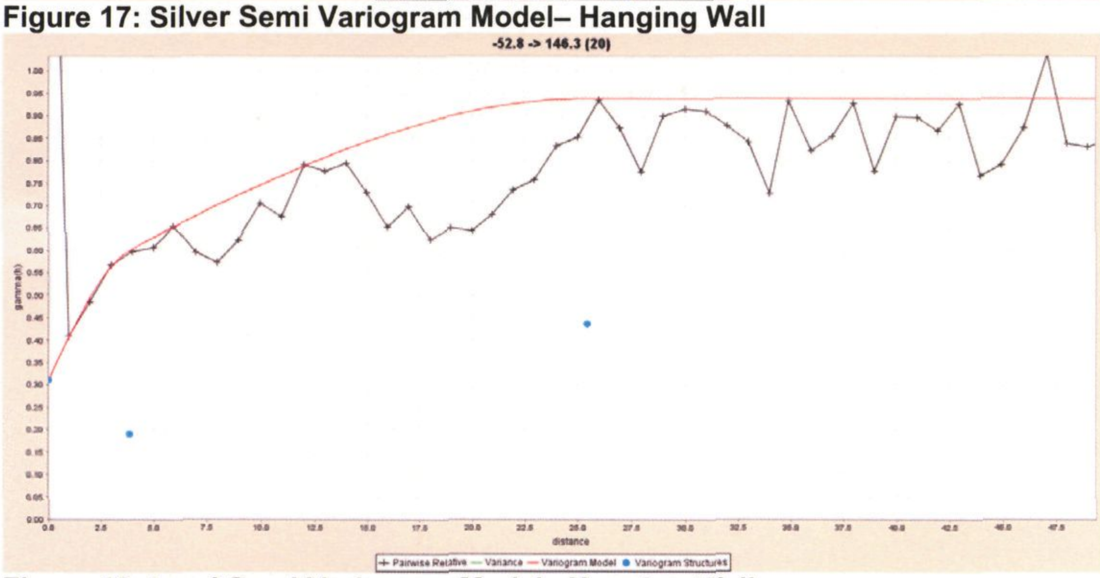


Figure 18: Lead Semi Variogram Model– Hanging Wall

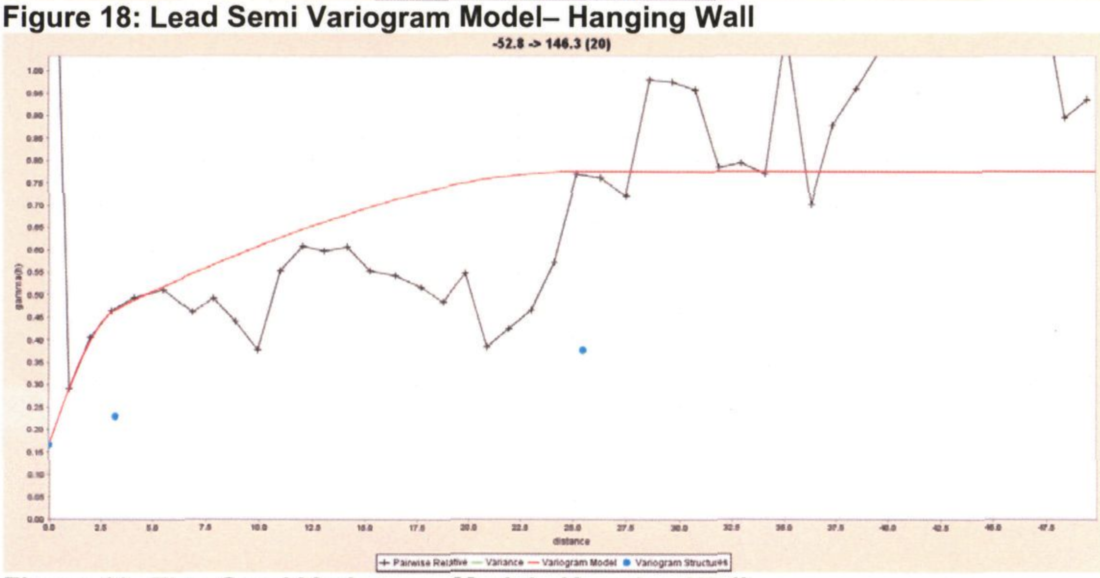


Figure 19: Zinc Semi Variogram Model– Hanging Wall

11.9.2 Grade Interpolation

The Treasure Mountain vein system comprises two domains, hangingwall and footwall domains, which were interpolated separately on the basis of the composited data for each.

For the hangingwall domain, silver, lead and zinc grades were interpolated by ordinary kriging and values were assigned to the blocks. The proportion of the block inside the solid model of the vein was calculated in order to partition waste rock from mineralization. The small number of samples in the footwall vein envelope necessitated using the hangingwall variogram parameters to krige the footwall blocks.

Two kriging passes using the same search ellipsoid were run to estimate the block values. Block values were assigned as follows:

Pass 1 – search radius equal to one-half of the longest axis of continuity, with a minimum 4 samples and maximum of 8 samples used to interpolate the block values. If more than 8 samples were enclosed by the search, the closest 8 samples were used. Blocks that were assigned values in Pass 1 were designated "Indicated" resources.

Pass 2 – search radius equal to the maximum range of the semivariogram, with minimum 3, maximum 8 samples, and the kriged value was assigned to all unassigned blocks located within the range that were then designated "Inferred" resources.

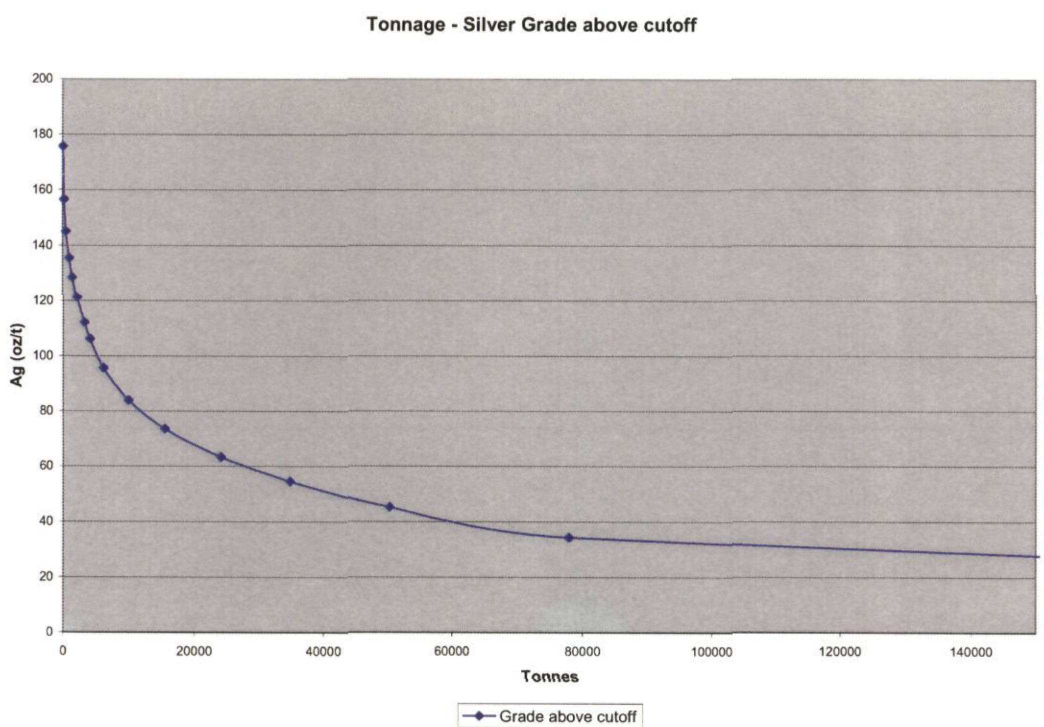


Figure 20: Silver Grade (oz/tonne) – Tonnage Curve

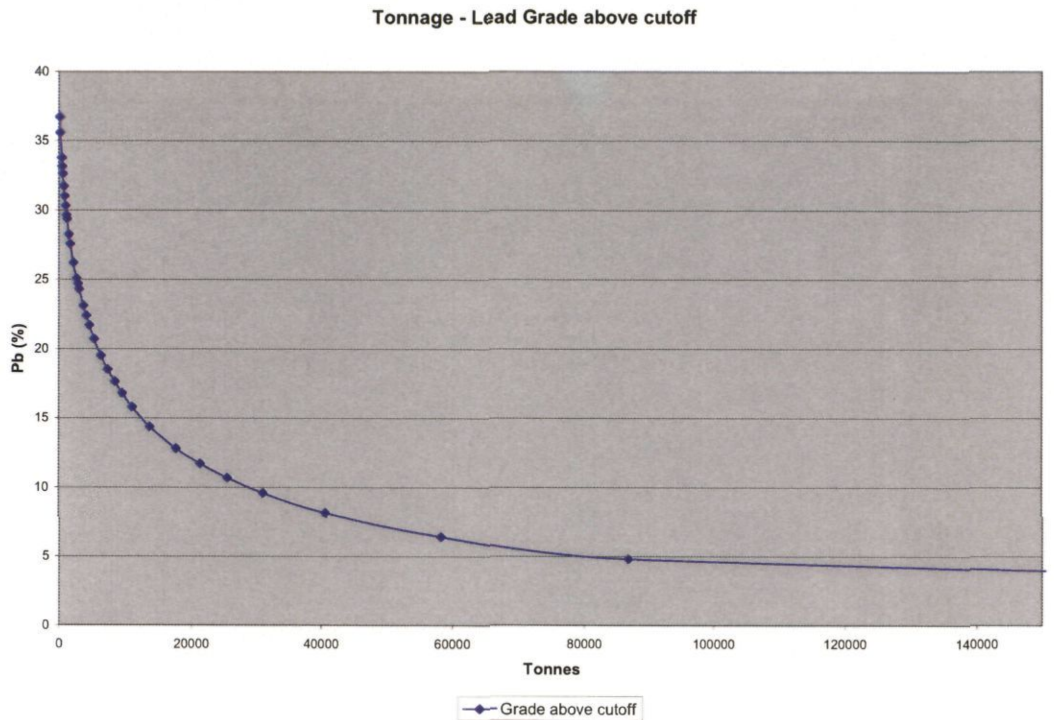


Figure 21: Lead Grade (%) – Tonnage Curve

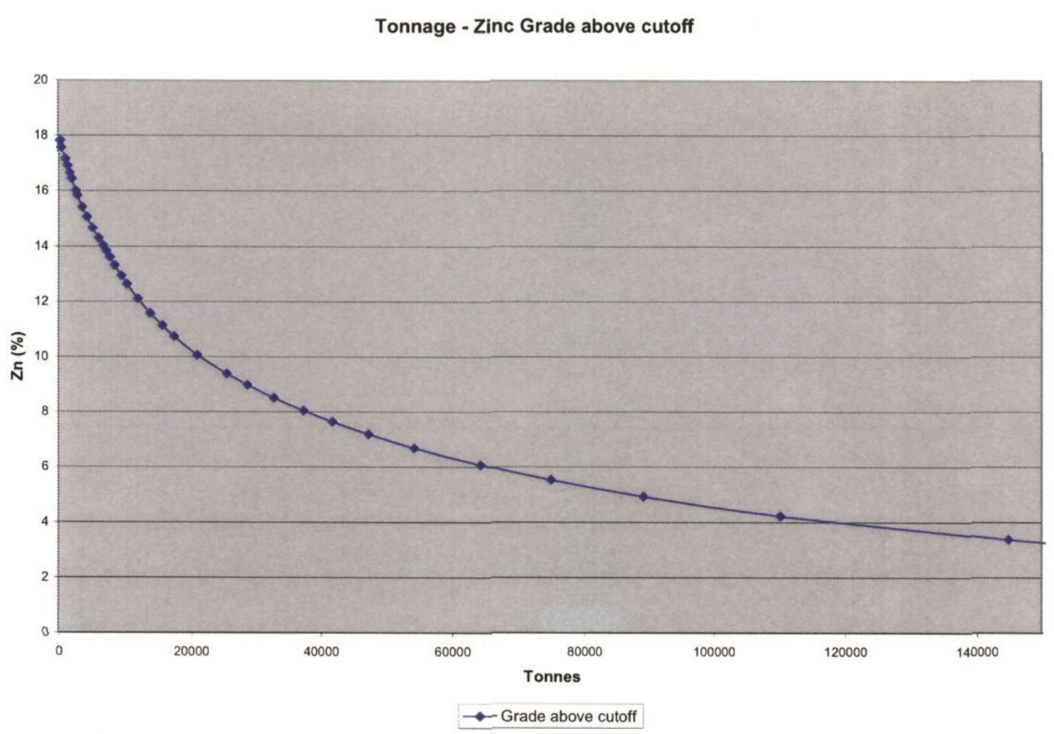


Figure 22: Zinc Grade (%) – Tonnage Curve

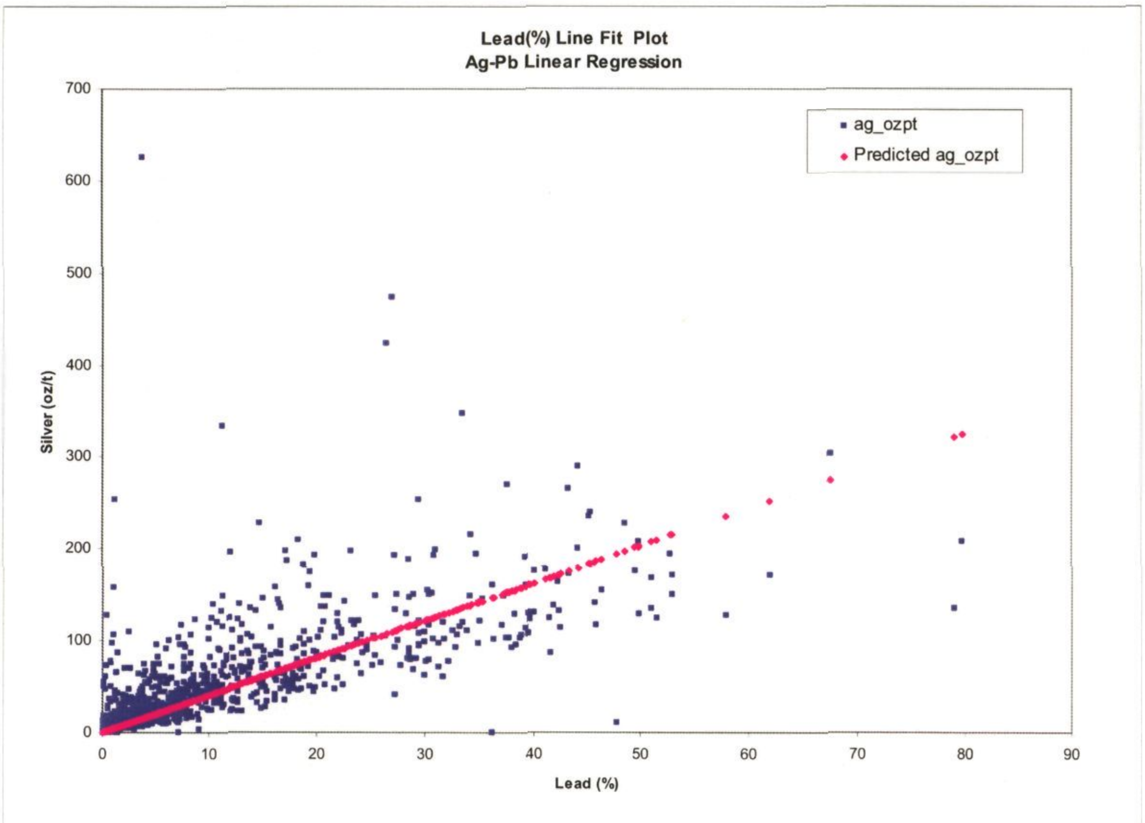


Figure 23: Regression Curve Silver : Lead

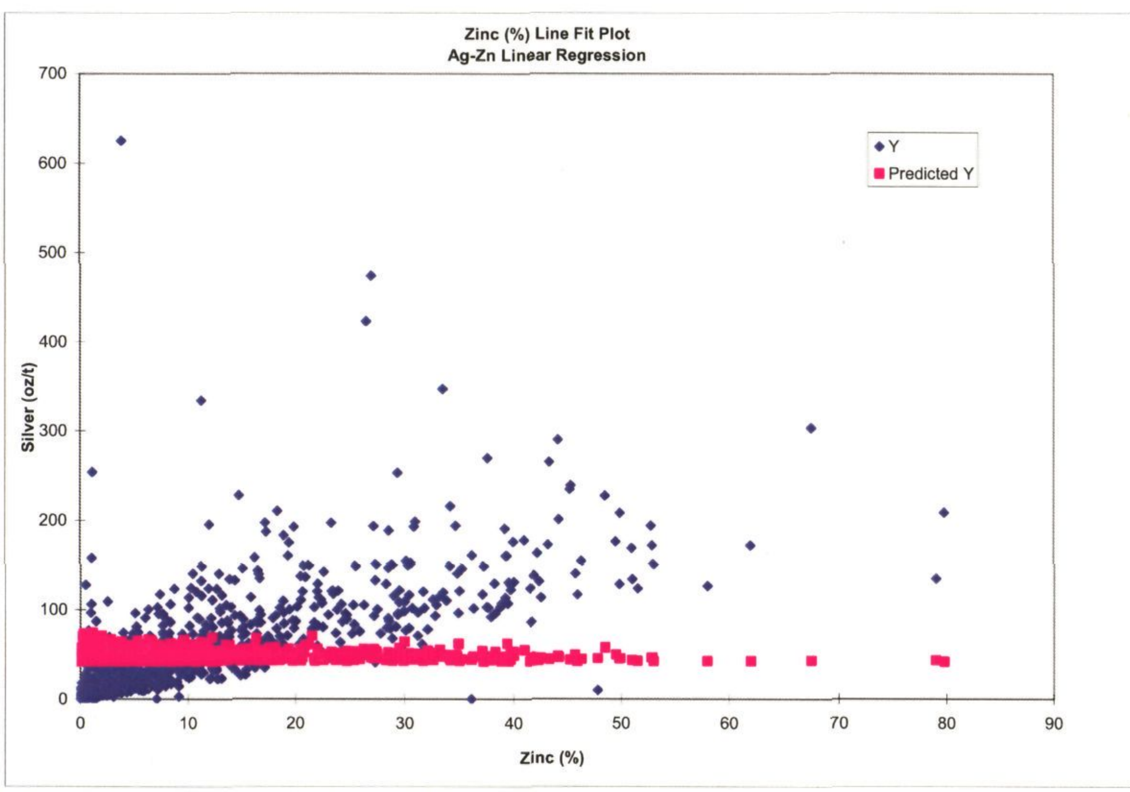


Figure 24: Regression Curve Silver: Zinc

11.9.3 Classification

As was discussed in section 12.4 of this report, no Reserves of any kind were identified on the Treasure Mountain property. Similarly, in part due to the lack of confirmation of metal values, no Measured Resources were calculated. Geologic continuity has been well established by surface mapping and by underground mapping and sampling on four levels and in several raises. Metal values, especially silver, are erratic but have been confirmed in a general way by various sampling initiatives and by at least two laboratories. Smelter shipments and metallurgical test work have further substantiated the metal values. Additional sampling test work is recommended in a later section of this report.

Indicated and Inferred resources have been defined by variography and kriging.

11.9.4 Sensitivity – Value-Tonnage Curves

Figure 25 illustrates the sensitivity of Treasure Mountain resources to the price of silver in the range \$6.00/ounce to USD\$20.00/ounce. Figure 26 and 27 illustrate the sensitivity of Treasure Mountain resources to the price of lead and zinc in the range of USD\$0.50/lbs. to USD\$1.30/lbs.

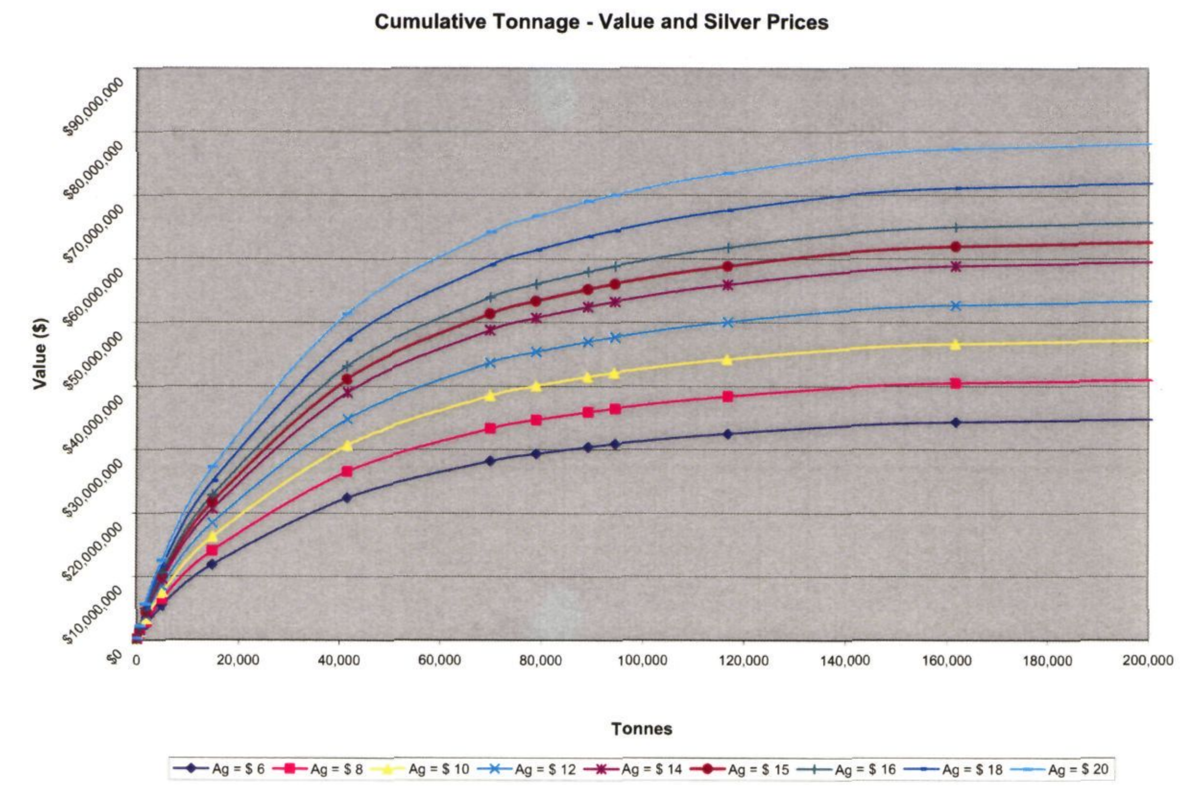


Figure 25: Cumulative Tonnage – Value and Silver Prices

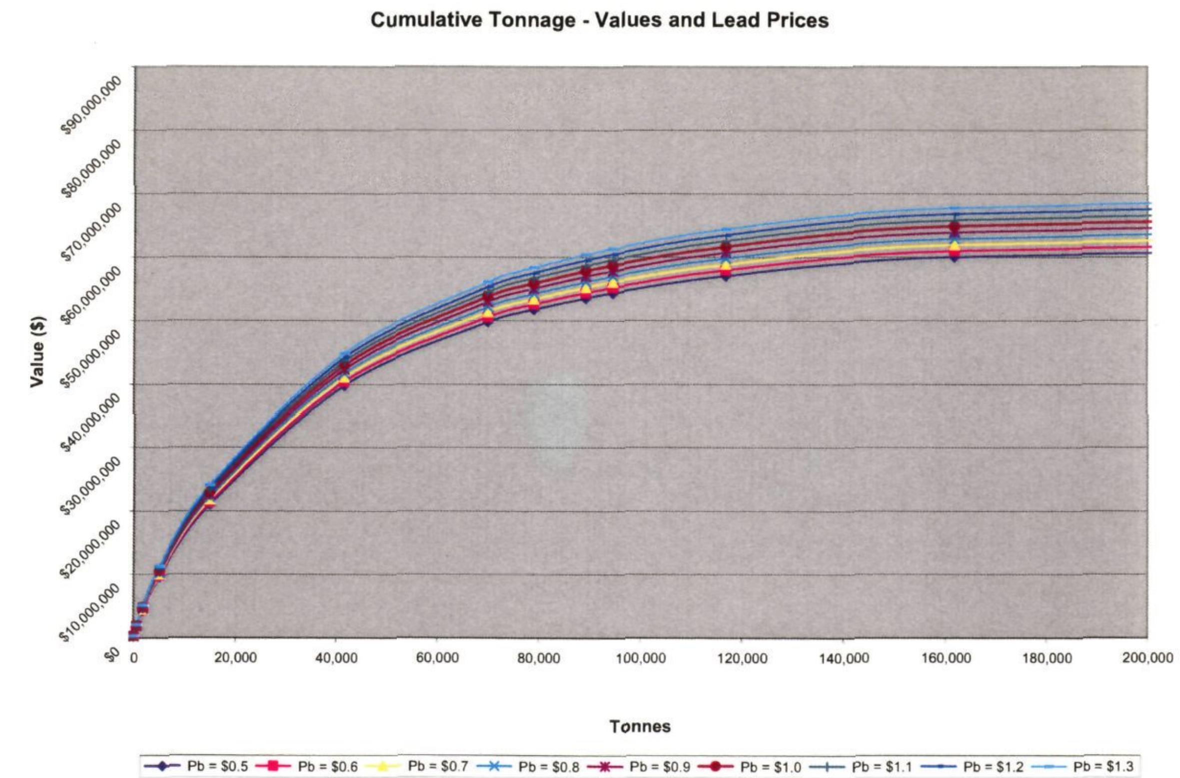


Figure 26: Cumulative Tonnage – Value and Lead Prices

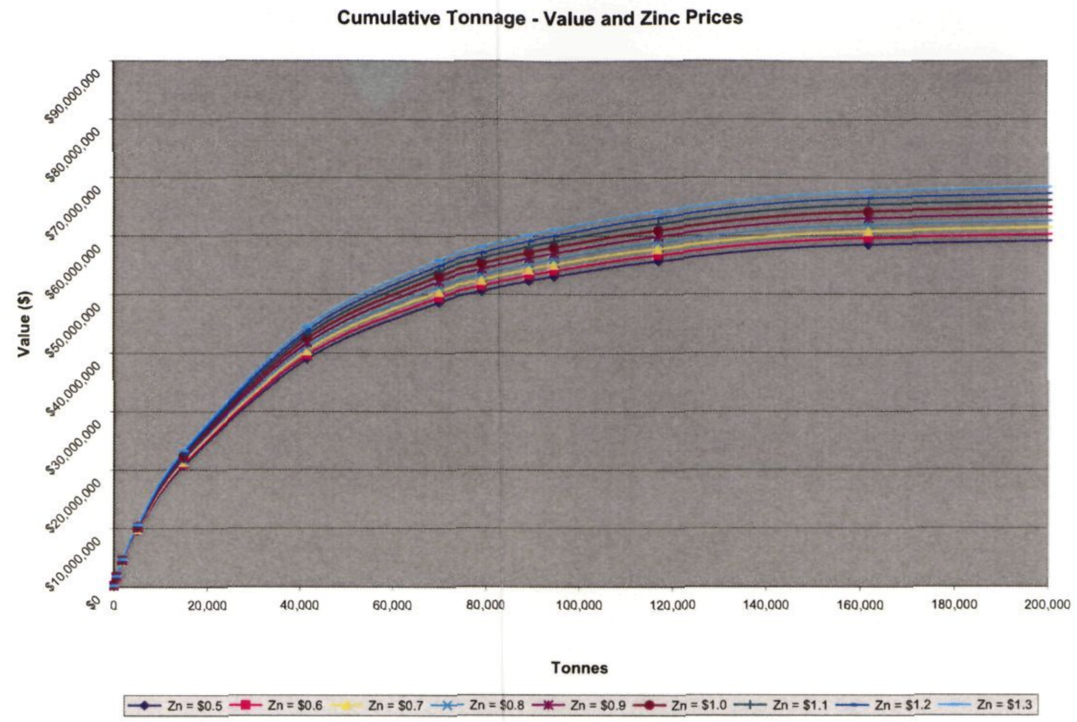


Figure 27: Cumulative Tonnage – Value and Zinc Prices

11.9.5 Block Model Value Calculations [all values in USD]

In order to assign values to blocks in the block model, it was necessary to calculate a composite value that recognized each of silver, lead and zinc. For calculation purposes silver was valued at $15/oz, lead at $0.70/lb, and zinc at $0.80/lb. and the block value comprised [$15*oz/tonne silver + $0.70*%Pb + $0.80*/%Zn]. No allowance was made for losses in milling nor for any additional dilution that might occur in extraction. The assigned metal values were selected on the basis of present and recent values as reported on various websites including Metalprices.com, Bloomberg Financial, and Kitco Metals, and have been discounted in order to reflect longer term uncertainty concerning the volatility of metal prices. Average LME prices in USD for silver in ounces in 2007 was $13.38 and for 2008 to date (July 30th) was $17.48; average lead price in period January 1, 2006 through July 30, 2008 was $1.00 per lb., and zinc, in the same period was $1.37 per lb.

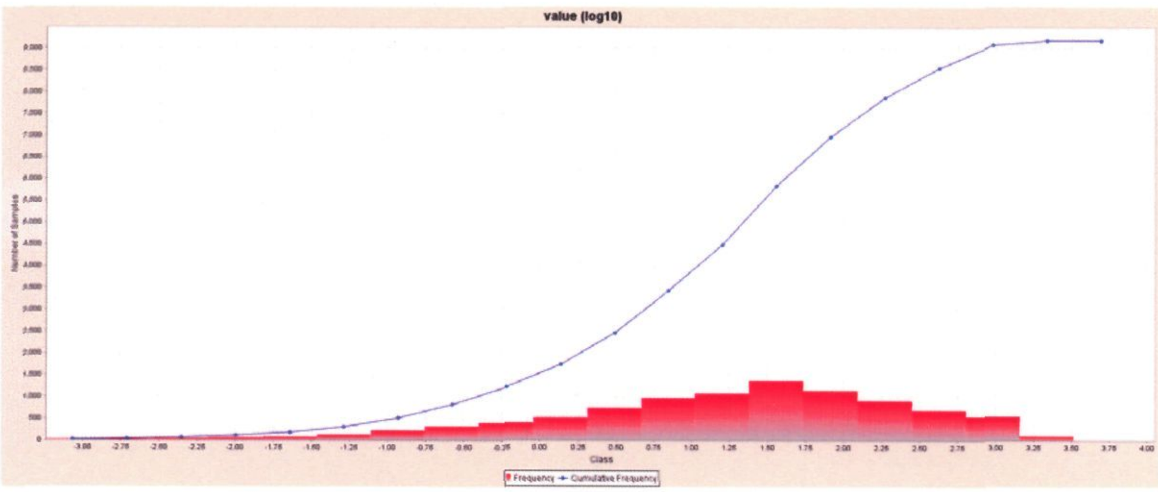
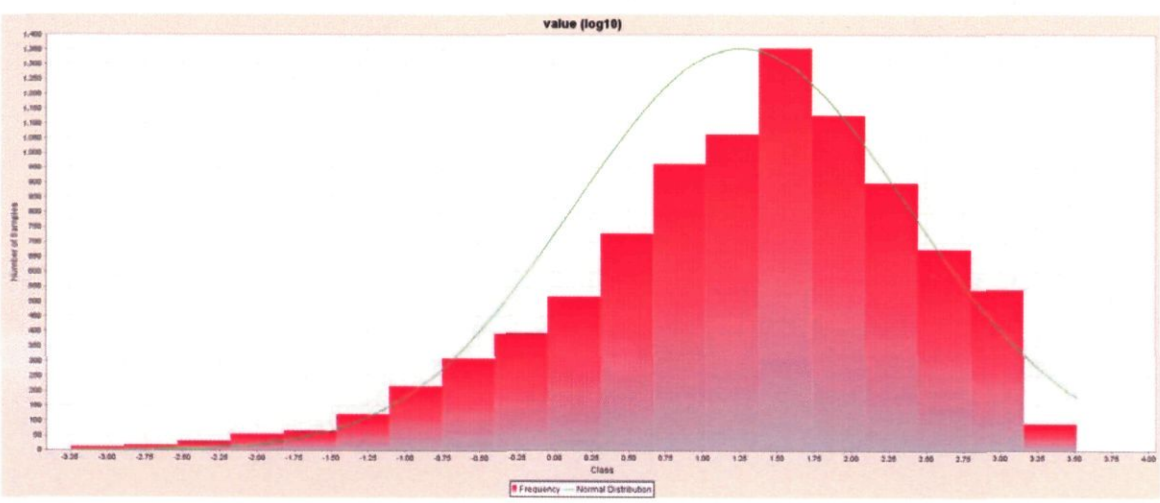


Figure 28: Block Model Calculated USD$ Value Frequency and Cumulative Curve [log (10)]



11.9.6 Indicated and Inferred Resource Calculations

Hangingwall Domain

Table 9 presents combined indicated and inferred resources for hangingwall blocks with dollar values greater than $165 along with calculations for other cutoff values and Table 10 presents the same information for footwall blocks.

Table 9: Hangingwall Resources

Class	Value ($)	Volume	Tonnes	Value ($/tonne)	Ag (oz/t)	Pb (%)	Zn (%)	Sg
	165.0 -> 200.0	2,070	6,113.59	$181.39	9.14	1.19	1.47	2.97
	200.0 -> 250.0	1,545	4,666.55	$223.04	10.77	1.32	2.34	3.04
	250.0 -> 500.0	5,355	16,727.30	$363.73	18.43	1.93	3.26	3.16
	500.0 -> 1000.0	5,415	17,312.95	$722.30	37.27	3.20	6.46	3.23
	1000.0 -> 1500.0	1,230	4,321.00	$1,212.02	60.83	7.23	10.66	3.59
	1500.0 -> 2000.0	300	1,149.33	$1,699.69	88.61	12.72	9.88	3.97
	2000.0 -> 2500.0	90	344.83	$2,174.55	117.98	15.43	9.45	3.98
Inferred	*2500.0 -> 3000.0*	75	355.26	$2,623.78	133.41	30.34	8.75	4.75
Grand Total		16,080	50,990.80	$580.73	29.66	3.20	4.90	3.22

Class	Value ($)	Volume	Tonnes	Value ($/tonne)	Ag (oz/t)	Pb (%)	Zn (%)	Sg
	165.0 -> 200.0	1,380	4,085.78	$181.37	8.37	1.66	1.71	2.98
	200.0 -> 250.0	1,455	4,383.77	$224.97	10.50	2.00	2.07	3.03
	250.0 -> 500.0	3,600	11,544.91	$364.56	16.75	3.42	3.43	3.24
	500.0 -> 1000.0	2,715	9,377.67	$726.62	35.10	6.01	6.09	3.50
	1000.0 -> 1500.0	1,485	5,432.05	$1,189.82	59.87	9.54	8.20	3.71
	1500.0 -> 2000.0	510	2,077.85	$1,724.77	84.39	16.28	11.78	4.12
	2000.0 -> 2500.0	210	911.97	$2,178.01	110.18	20.55	11.80	4.39
	2500.0 -> 3000.0	45	227.13	$2,669.87	140.14	31.72	4.44	5.05
Indicated	*3000.0 -> 3500.0*	15	73.18	$3,259.51	175.65	34.37	5.35	4.88
Grand Total		11,415	38,114.31	$672.41	32.76	5.92	5.09	3.41

The calculated diluted volume of hangingwall domain resources considered to be "indicated" is estimated to be 38,114 tonnes with 32.76 oz/ tonne silver, 5.92% lead and 5.09% zinc and the "inferred" resource is estimated to be 50,991 tonnes with 29.66 oz/tonne silver, 3.20% lead and 4.90% zinc. The calculated diluted volume of footwall domain resources considered to be "inferred" is estimated to be 17,478.22 tonnes with 15.05 oz/tonne silver, 0.29% lead and 4.33% zinc.

Table 10: Footwall Resources

Class	Value Range	Volume	Tonnes	Value ($/ tonne)	Ag (opt)	Pb (%)	Zn (%)	SG
Inferred	165.0 -> 200.0	1,275.00	3,538.11	$180.26	9.82	0.18	1.71	2.78
	200.0 -> 500.0	4,335.00	12,285.78	$308.31	14.82	0.3	4.53	2.83
	500.0 -> 1,000.0	570.00	1,654.33	$563.49	27.23	0.42	8.43	2.9
	Grand Total	6,180.00	17,478.22	$306.54	15.05	0.29	4.33	2.83

12.0 OTHER RELEVANT DATA AND INFORMATION

The foregoing sections of this report contain all relevant data and information pertaining to the Treasure Mountain mine that can be retrieved from sources that include the BC Ministry of Energy, Mines and Petroleum Resources data base (i.e. Minfile, ARIS), technical libraries (particularly Geol. Survey of Canada library), and company files maintained by the president, Mr. Bratlien. The author has personally examined parts of the Treasure Mountain property, including surface areas and Levels 1 and 2 of the mine.

13.0 INTERPRETATION AND CONCLUSIONS

The Treasure Mountain mine comprises a series of lenses of silver-lead-zinc mineralization that lie along the hangingwall and footwall of an andesite porphyry dyke and in proximity to the Treasure Mountain fault zone. Metal values across "mining widths", defined by a mining engineer who is familiar with current operating parameters and requirements, as a 1.2 metre width, current metal prices discounted to a longer term average, and mining costs, details of which are forthcoming, appear to be sufficient to sustain a small underground mining operation for a period of several years but that opinion is not based upon a formal economic assessment of the proposed mine. The likelihood of being able to expand the resource and prolong the mine life by pursuing exploration in and near the present mine is judged by the author and other qualified and knowledgeable explorationists, including Mssrs. MacDougall, P. Eng. and Vulimiri (1987), to be "good".

The 2007 program of check sampling of parts of Levels 1 and 2 of the mine workings affirmed the presence of substantial silver, lead and zinc values similar to those reported by previous samplers. Resource calculations prepared in 1989 by Livgard Consultants Ltd. Using then acceptable methods and definitions, are considered by the authors of this report to be reasonable, but the grade calculations are based on assay data that were not fully supported by the 2007 sampling. Reasons for the discrepancies have been presented in previous sections of this report and are largely speculative pending additional sampling that will undoubtedly form part of any further work on the Treasure Mountain property. Resource Calculations that form parts of this report were based on the original sampling data and despite the lack of correspondence between the data sets, the variance of which could be as large as 20%, there is ample sampling evidence that the Treasure Mountain deposit comprises a high value resource. There should be little difficulty in maintaining production at or above the projected $165 per tonne cut-off grade and ensuring a profitable operation. The existing and planned underground workings on four levels will enable flexibility in "ore" extraction to respond to vagaries of metal prices by blending, as needed, "high grade" and "low grade" feed to the mill.

Note that the above-stated economic opinion is not based on a detailed economic assessment of Treasure Mountain resources and a possible mining operation and is presented as an opinion rather than a forecast. Such a forecast should be prepared by experts that are better able to determine costs and are more knowledgeable of mine operations and productivity.

14.0 RECOMMENDATIONS

Studies currently in progress, including mine planning, mill design, environmental studies, resource calculations and logistics, will be the subject of an economic analysis. Pending receipt of that more definitive study, it is recommended that Huldra Silver Inc. continue exploring the Treasure Mountain mine and property and preparing for production. The tonnage of "indicated" resources that has been outlined by surface and underground work may be determined by the studies to be sufficient to assure a viable seasonal mining operation. Obviously one of the company's immediate objectives should be to promote inferred and indicated resources to a measured category, to further explore the immediate mine area and nearby prospective areas and to prolong the mine life.

It is recommended that Huldra Silver Inc. continue exploration and development work at Treasure Mountain while awaiting completion of economic and other studies. As part of Phase 1 work, the company's mining consultant has recommended that several parts of the present mine should be prepared for stoping by shrinkage and open stoping methods. Present mine levels have to be equipped with ventilation, compressed air and other services and access raises have to be driven and, variously equipped. Because much of the development work will follow the principal vein, added benefits of such preparation will be the opportunity to add to the stockpile of broken "ore" already stored on surface and also to upgrade a portion of the present inferred resource to indicated status.

Much of the resource that at present must be classified as "inferred" due to uncertainties of configuration and contained metal values, particularly silver values, should be further defined to an "indicated" category and with the benefit of more rigorous check sampling, to a "measured" category. Some of that up-grading may be accomplished in the course of preparing the presently defined "indicated" resources for production. Current mine planning envisions placing a number of raises for access, ventilation, manways and ore passes and all or, at least most, of these openings will follow and/or pass through the Treasure Mountain vein and by increasing the density of sample spacing will facilitate reclassification of the material. Additionally, earlier miners removed an unknown volume of a mineral zone located above the western end of Level 3 and thereby rendered the remaining, overlying portion of the zone inaccessible for sampling purposes. None of that area has been included in the current resource calculation but when mining resumes, it should be relatively easy to provide access for sampling and exploration purposes and may as a result add to the life of the mine.

Two areas near the Treasure Mountain mine have been explored by drilling: the Jensen Adit area west of Level 3 portal and the "Ruby Zone", 1.1 km to the east. The former has been explored by historic underground workings and more recently by rotary drilling. An exploration drift westerly from the mine into the Jensen Adit area where rotary drill holes and historic workings intersected silver and base metal mineralization similar to that present in the main mine workings, will resolve uncertainties and may present an opportunity to add substantially to resources . Further study of the mine data will help determine how the area should be accessed. The "Ruby Zone" where work in recent years has found significant occurrences of silver, lead and zinc mineralization has been explored by bulldozer stripping, drilling and sampling but its size and tenor have not been determined.

Both areas, on the basis of geology and mineralogy, may be extensions of the Treasure Mountain mineralized structure and both have produced samples of mineral zones that resemble the main deposit in character and metal content. The Ruby vein area samples contain more gold than do those from other parts of the property.

Other areas that are prospective for mineralization north and west of the present mine that have been insufficiently explored, should be re-evaluated.

All property exploration data should be reviewed prior to commencement of a program of surface diamond drilling. Modeling exercises that were the basis of the resource calculation reveal a significant area lying between Levels 1 and 3 and east of the end of Level 2 that is virtually unexplored. It is recommended that several drill holes from surface should be directed to that area. Previous diamond drilling work employed small diameter coring tools and often experienced poor recovery in the critical vein portion: larger diameter cores, i.e. NQ2 size, may give better results.

The following estimate of work and costs does not include provisions for a mill facility or for engineering and environmental studies that may be conducted separately from underground and surface exploration work. Much of the underground development work will be in mineral zones and is likely to produce mill feed that can be stockpiled and, in due course, processed when a mill is available.

Phase 1. Mine exploration and development, drilling, sampling, miscellaneous

Objective: to develop Levels 1 and 2 for production on the basis of the Indicated mineral resources and to upgrade inferred resources to a higher level of confidence.

Permitting and preparation..	$10,000
Provide services in the present underground workings, including vent tubing, air lines and track - estimated	$200,000
Development raises to access proposed stopes at estimated cost of $2000/metre (Beaton, pers. comm. June, 2008), allow	$400,000
Surface structures – first aid, lunch room, work shop, site office	$75,000
Camp facility (i.e. Atco trailers)	$125,000
Drilling - Allow 2000 metres total in underground and from surface contracted @ $90/metre	$180,000
Supervision, engineering, geology, sampling and analytical, allow	$150,000
Building data base for planning purposes, allow	$50,000
Final report	$10,000
Sub-total..	$1,200,000
Allowance for unscheduled expenditures @ 15%..............................	$181,500
Total estimated cost of Phase 1 underground and surface work	$1,391,500

Phase 2.

Phase 2 work, depending upon the time of the year, may continue almost seamlessly from Phase 1 and will comprise further mine development, exploration and drilling, in part as recommended by the company's mining engineer:

Permitting and planning	$10,000
Allow 300 metres underground development @ $3000/metre	$900,000
Allow 500 metres diamond drilling @ $90/metre	$45,000
Supervision, geology, analyses, reporting, et al.	$25,000
Final report	$5,000
Sub-total...	$985,000
Allowance for unscheduled expenditures @ 15%.............................	$147,750
Total estimated cost of Phase 2 underground and surface work	$1,132,750
Total amount of Phases 1 and 2	**$2,524,250**

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16.0 Certificate - Erik A. Ostensoe, P. Geo.

I, Erik A. Ostensoe, P. Geo., a consulting geoscientist, do hereby certify that:

1. I am a consulting geologist with an office at 1403 - 675 West Hastings Street, Vancouver, British Columbia, Canada, V6B 1N2

2. I am a graduate of the University of British Columbia with the degree of Bachelor of Science in Honours Geology

3. I am registered as a Professional Geoscientist with the Association of Professional Engineers and Geoscientists of the Province of British Columbia, member no. 18,727

4. I have been engaged in mineral exploration for more than forty years and have worked in most regions of western and northern North America, and, to a lesser extent, in overseas countries

5. I, in the period July 13 to 18, 2007, examined in the field parts of the Treasure Mountain property of Huldra Silver Inc. and completed a limited program of chip sampling of mineralized portions of Levels 1 and 2 of the Treasure Mountain mine

6. I am the principal co-author of the accompanying report titled "REPORT OF PROPERTY REVIEW AND SAMPLING PROJECT, TREASURE MOUNTAIN PROPERTY, TULAMEEN RIVER AREA, B. C., CANADA" dated July 30, 2008, and I am not aware of any material fact or material change with respect to the subject matter of the report, the omission to disclose which makes the report misleading

7. I have no ownership interest in the Treasure Mountain property of Huldra Silver Inc., nor in the shares or assets of Huldra Silver Inc. or any related company and I am completely independent of Huldra Silver Inc. in accordance with Section 1.4 of National Instrument 43-101

8. I fulfill the definition of a "Qualified Person" as defined by National Instrument 43-101 and the accompanying report has been prepared in accordance with provisions of that Instrument and related guidelines

9. I have supervised the preparation by Farshad Shirvani, a non-Qualified Person and co-author, of mineral resource calculations presented in this report and I believe that that person possesses experience, competence and qualifications that enable him to prepare such calculations

10. I have read National Instrument 43-101 - Standards of Disclosure for Mineral Projects, the Companion Policy, and Form 43-101F1 - as downloaded from the website of the British Columbia Securities Commission on March 6, 2008 and the accompanying report was prepared in compliance with the Instrument, companion policy and form

11. To the best of my knowledge, information and belief, the accompanying technical report contains all scientific and technical information that is required to be disclosed to make the report not misleading

12. The accompanying report was prepared expressly for Huldra Silver Inc. and, as required, may be filed with regulatory authorities and may be included in literature or public company files by Huldra Silver Inc. provided only that any quotation or abbreviations are accurate and are appropriately attributed.

Dated this 30th day of July, 2008. Erik A. Ostensoe, P. Geo.

17.0 CERTIFICATE – FARSHAD SHIRVANI, M.Sc.

I, Farshad Shirvani, M.Sc., geologist and GIS specialist, do hereby certify that

1. I am a geologist and geographical information systems specialist and I am the proprietor of Terracad Ltd., a company that offers computer assisted data management services to mineral exploration and mining companies, and I have offices at Room 1405 - 675 West Hastings Street, Vancouver, B. C., V6B 1N2

2. I am a graduate of Shiraz University, Shiraz, Iran, and I hold B.Sc. (1983) and M.Sc.(1986) degrees from that Institute

3. I practiced geology in Iran for more than ten years in mineral exploration, engineering geology and hydrogeology and as Project Manager of the Malayer Reservoir Dam and City pipeline to Hamadan. I have lived in Canada since 1996 and was granted Canadian citizenship in 2002.

4. I have worked in Canada since 1996 as a geologist, AutoCAD specialist, 3D modeler, GIS specialist and web designer and I believe that I possess skills, qualifications and experience required to competently prepare, using sophisticated computer-based methods, mineral resource models and estimates

5. I am a co-author of the accompanying technical report titled "REPORT OF PROPERTY REVIEW AND SAMPLING PROJECT, TREASURE MOUNTAIN PROPERTY, TULAMEEN RIVER AREA, B. C., CANADA" dated July 30, 2008, and, using computer-based graphic and mathematical programs, I prepared the three dimensional model of the Treasure Mountain mine workings and the resource estimate that are included in that report on the basis of all pertinent available survey and analytical data, and I am not aware of any omissions from that data the failure to disclose which could make that model and resource estimate misleading

6. I personally supervised the compilation of the accompanying report by Ian Vaughan, B.Sc., geologist with specialization in geophysics, GIS, graphic design and modeling, an employee of Terracad Ltd.

7. I have no ownership interest in the Treasure Mountain property nor in any mineral tenures in the vicinity of that property, nor have I any ownership interest of any type in Huldra Silver Inc. or any related company and I am completely independent of Huldra Silver Inc. in accordance with Section 1.4 of National Instrument 43-101

8. I am not a "Qualified Person" as defined by National Instrument 43-101 and my contributions to the accompanying report have been prepared with the guidance of the co-author Erik Ostensoe, P. Geo., a Qualified Person as defined by that Instrument. I am in the process of applying for registration with the Association of Professional Engineers and Geoscientists of British Columbia and I expect to complete that registration during 2008

9. I have read National Instrument 43-101 and CIM Definition Standards for Mineral Resources and Mineral Reserves adopted by CIM Council on December 11, 2005 and

my contributions to the accompanying report have been prepared in compliance with those Standards

10. To the best of my knowledge, information and belief, my contributions to the accompanying report are based on data that is accurate and complete and that any areas of uncertainty or possible ambiguity have been disclosed and discussed in the text of that report

11. I consent to the inclusion of mineral resource estimates prepared from the geological and mine model that I prepared for Huldra Silver Inc., in literature and public company files, websites and documents of Huldra Silver Inc., provided only that any quotations or abbreviations are accurate and are appropriately attributed.

Dated this 30th day of July, 2008. Farshad Shirvani, M.Sc.

18.0 GLOSSARY OF TECHNICAL TERMS

Many of the following terms and definitions some of which have been taken from Glossary of Geology, Fourth Edition, Julia A. Jackson, editor, American Geological Institute, Alexandria, Virginia, 1997 *appear in the accompanying report:*

Allocthon - (1) an accreted terrane, formed by the juxtaposition of dissimilar geological features as a result of crustal fragmentation and movement
(2) a mass of rock or fault block that has been moved from its place of origin by tectonic processes...many allocthonous rocks have been moved so far from their original sites that they differ greatly in facies and structure from those on which they now lie;
(3) a naturally occurring geological unit that has been moved a large distance by tectonic processes such as continental drift

Andesite - a common volcanic rock type composed of feldspars and Fe-Mg silicate minerals – similar to dacite but contains more ferrous components

Argillic - a form of alteration characterized by formation of clay minerals

Argillite - a fine-grained sedimentary rock, usually exhibits strong banding

Arkose - granular sedimentary material largely comprising feldspar particles

Batholith - a body of crystalline plutonic rock, may be homogeneous or compounded from more than one magmatic source; area in outcrop or subcrop in excess of 100 square kms

Boulangerite - a lead-antimony sulphide mineral

Bournonite - a lead-copper-antimony sulphide mineral, a minor ore that frequently occurs in association with the more common base metal minerals including galena, tetrahedrite, chalcopyrite, sphalerite and pyrite

Braunite - a minor metallic mineral comprising manganese and zinc, may be misidentified as sphalerite

Conglomerate - a coarsely fragmental sedimentary rock in which the clasts are commonly sub-rounded or rounded

Dacite - a common volcanic or intrusive rock type, highly feldspathic but with little free quartz, usually fine grained

Drift - an underground mine working that generally follows the trend of a mineral zone as compared to a cross-cut that crosses the trend

Dyke - an intrusive body with limited thickness relative to other dimensions that penetrates and crosses its host rock(s)

Epigenetic - refers to a mineral deposit that is introduced into a rock formation as opposed to "syngenetic" deposits that are formed contemporaneously with the host formation

Epithermal - refers to the process of near surface ore deposition by fluids from an intrusive source, see also *mesothermal*; said of a mineral deposit formed within about 1 km of the earth's surface and in the temperature range 50 – 200 degrees C, occurring mainly as veins. Also said of that environment

Flotation - a metallurgical process that employs mechanical and chemical methods to separate valuable minerals and metals from closely related but worthless materials by attracting them to froth that can then be skimmed or otherwise captured

Footwall - that portion of a geological structure lying on the underside of that structure See - hangingwall

Freibergite - see *Tetrahedrite*. A silver-rich copper, et al. antimony sulphide mineral

Gabbro - a dark coloured phase of quartz-poor, strongly feldspathic granitic intrusive rocks in which the feldspars are more calcic than those found in granites and syenites

Greenschist facies - a low intensity stage of metamorphism with incipient development of low temperature micaceous minerals

Hornfels- a thermally metamorphosed rock, usually sedimentary, that has been sufficiently heated to cause growth of new silicate minerals, often with micaceous habit

Hydrothermal - refers to a mineral deposit formed by circulating fluids, usually implies elevated temperatures but is without any particular restrictions of temperature or pressure

Induced polarization survey - a ground-based geophysical method employing an electrical transmitter and an array of receivers, measuring the ability of a rock mass to retain an electrical charge

Lahar - a fragmental rock of volcanic origin characterized by chaotic unsorted mixtures of ejecta, may have "welded" textures resulting from rapid accumulation of very hot fragments and gases

Lamprophyre - a dark coloured igneous intrusive rock, commonly porphyritic and tabular

Mesothermal - refers to a mineral deposit formed at moderate depth hence at "moderate" temperature and pressures: said of a hydrothermal mineral deposit formed in the temperature range of 200 – 300 degrees C. Also said of that environment

Muck - a generic term signifying rock that has been broken by blasting or other means, commonly used in reference to underground operations

National Instrument 43-101 and Form 43-101F1 - the written policy statements applicable to publicly-traded mining and mineral exploration companies in most Canadian provinces and territories that govern disclosure of scientific and/or technical information about a mineral project or property material to the issuer

Open cut - a surface working in and around a mineral deposit created for the purpose of exposing and/or extracting materials or to better determine the nature of that deposit

Polymetallic - a mineral deposit with substantial metal values accruing from more than one metal component

Porphyry - a heterogeneous rock characterized by the presence of crystals in a relatively finer-grained matrix

Portal - the entrance to an underground space

Propyllitic alteration - a metamorphic assemblage with sericite, chlorite and carbonate minerals, characteristic of the outer alteration zone related to porphyry-type deposits

Qualified Person - as defined in section 1.1 of National Instrument 43-101, Standards of Disclosure for Mineral Projects, is an engineer or geoscientist with at least five years experience in mineral exploration, mine development or operation or mineral project assessment, or any combination of these; has experience relevant to the subject matter of the mineral project and the technical report; and is in good standing with a professional association

Raise - an internal mine working that follows or gives access to parts of a mine that lie above the principal workings or is extended to surface for access or ventilation purposes

Rhyolite - a silica-rich fine-grained volcanic rock, vaguely analogous chemically to granite

Sericitization - a natural process in which particular colourless or nearly colourless micaceous minerals are formed as part of a metamorphic or mineralizing event, often a useful guide to locating valuable mineral deposits

Shrinkage - a method of mining ores by drilling and blasting followed by "underhand" extraction through draw points

Siderite - FeCO3, a common iron carbonate mineral frequently found in association with metallic ores

Silicification - a natural process in which silica is introduced into and replaces pre-existing natural rock components

Skarn - a metamorphic rock formed in the thermal aureole of an intrusive body, also applied to rocks that have been altered with the addition of components such as calcium, metals and gases. Other definitions are recognized

Sphalerite - zinc sulphide, ZnS, the most common naturally occurring source of zinc

Tetrahedrite - a common copper-iron-zinc antimony sulphide mineral that may contain important amounts of silver and, frequently, arsenic.

Vitrophyre - a volcanic rock that formed without crystallization; glassy textures, may accompany other fragmental volcanic rocks

Volcaniclastic - pertaining to all clastic volcanic materials formed by any process of fragmentation, dispersed by any kind of transporting agent, deposited in any environment or mixed in any significant portion with non-volcanic fragments.

Zinkenite - contrary to its name is a lead-antimony-sulphide mineral, a minor ore of lead

APPENDIX 1: Treasure Mountain Ore Reserves

May 1989

Prepared by: Livgard Consultants Ltd.

For: Huldra Silver Inc.

Note that these reserve calculations pre-date adoption of National Instrument 43-101 reporting standards and CIM Definition Standards for Mineral Resources and Mineral Reserves.

HULDRA SILVER INC.

TREASURE MOUNTAIN ORE RESERVES

May 1989

DEFINITIONS - ASSUMPTIONS

Tonnages are metric based on an assumed specific gravity of 3.5 (3.5 kg per litre (or dm^3))

Grade is troy ounces of silver per metric tonne and percentages of lead and zinc.

Ore is that tonne which contains ounces of silver plus percentage zinc in excess of 15.0.

Hence cut-off grade is ounces Ag + %Zn of 15.0

PROVEN ORE is assumed to extend 12.5m out from a series (more than a few) of ore-grade chip channel samples.

In a few cases where ore has been sampled on two sides, the ore block has been assumed to extend more than 12.5m (max. 23m in one case).

PROBABLE ORE is assumed to extend from 12.5m to 25.0m from a series of ore-grade chip channel samples
and is assumed to extend 12.5 m from an ore grade diamond drill intersection and it is assumed to extend 12.5m from a few sample points (not sufficient samples to be considered proven ore).

POSSIBLE ORE is assumed to extend from 25.0 to 50.0m from a series of ore grade sample points or from 12.5 to 37.5m from a few ore grade sample points or a diamond drill hole ore intersection.

All ore blocks are assumed to extend in "geologically open" directions or in "trend directions" only.

Possible ore has also been assumed to exist in the old (1930-1956) mine area.

The available assays here are from samples taken in the 1950's by consultants.

Extensive sample series are shown to exceed cut-off grade but because of unknown sampling and assaying reliability, this area has been classified as possible ore only. The area in question extends between #3 and #4 Levels 150m horizontally and 140m on dip. If half of this area is ore it amounts to 43,000 tonnes. This figure has been added to possible ore reserves.

	PROVEN				PROBABLE			
Level	Tonnage	Oz Ag/Tonne	% Pb	% Zn	Tonnage	Oz Ag/Tonne	% Pb	% Zn
#1L H.W.	19,238	34.71	6.97	4.14	14,902	36.22	6.95	2.51
#1L F.W.	NIL	-	-	-	NIL	-	-	-
#2L H.W.	11,146	28.40	6.01	5.27	11,973	29.61	5.98	5.05
#2L F.W.	6,888	26.34	2.42	8.68	8,162	26.04	1.96	6.70
#3L H.W.	7,821	28.16	5.85	4.45	7,401	27.49	5.70	4.51
#3L F.W.	6,888	26.35	2.42	8.68	9,146	27.71	2.47	8.25
#4L H.W.	9,654	14.82	3.01	7.77	10,263	17.35	3.14	6.89
#4L F.W.	NIL	-	-	-	9,555	28.02	0.52	0.46
PV + PB	61,635	27.75	5.02	5.97	71,402	28.14	4.11	4.71
TOTAL	133,037	27.96	4.53	5.29				
POSSIBLE	148,000							

METRIC TONNES - GRADE OZ AG/TONNE (M)

#1L H.W.

P R O V E N

Block	Tonnage	Oz Ag/Tonne	% Pb	% Zn
# 6	1,312	15.58	4.04	5.07
# 8	1,260	32.20	10.20	2.19
	3,915	29.17	6.83	6.62
#12	4,852	41.07	6.90	3.01
	1,733	43.73	7.73	4.41
	1,441	35.08	7.17	8.64
#14	1,575	45.78	8.80	1.49
	945	21.46	6.34	1.51
	472	28.76	4.77	1.72
	1,733	35.19	5.78	2.98
	19,238	34.71	6.97	4.14

P R O B A B L E

Block	Tonnage	Oz Ag/Tonne	% Pb	% Zn
# 6	656	15.58	4.04	5.07
# 8	2,314	30.69	8.52	4.41
#12	2,621	44.76	8.27	2.95
#14	3,494	34.62	6.87	1.51
	4,200	40.02	6.81	2.26
#16	1,617	32.24	4.27	0.86
	14,902	36.22	6.95	2.51

#1LEVEL F.W.

PROVEN

Block	Tonnage	Oz Ag/Tonne	% Pb	% Zn
	NIL			

#2L H.W.

P R O V E N

Block	Tonnage	Oz Ag/Tonne	% Pb	% Zn
# 6	1,470	16.51	2.96	9.00
	1,312	15.58	4.04	5.07
# 8	3,308	32.03	7.57	4.76
#12	2,117	37.06	7.21	7.72
#14	945	21.46	6.34	1.51
	472	28.76	4.77	1.72
	1,522	35.19	5.78	2.98
	11,146	28.40	6.01	5.27

P R O B A B L E

Block	Tonnage	Oz Ag/Tonne	% Pb	% Zn
# 4	525	22.87	3.60	5.09
# 6	1,470	16.51	2.96	9.00
	1,312	15.58	4.04	5.07
	3,098	32.03	7.57	4.76
	2,470	45.14	7.46	6.68
	1,838	23.29	5.95	1.60
	1,260	35.19	5.78	2.98
	11,973	29.61	5.98	5.05

#2L F.W.

PROVEN

Block	Tonnage	Oz Ag/Tonne	% Pb	% Zn
#3	2,100	14.08	3.93	6.12
#7	2,440	38.03	1.01	13.78
#9	1,560	27.60	0.84	8.09
	788	20.34	5.88	0.90
	6,888	26.34	2.42	8.68

PROBABLE

#3	2,100	14.08	3.93	6.12
	1,312	14.08	3.93	6.12
#7	2,440	38.03	1.01	13.78
#9	2,310	31.05	0.04	0.07
	8,162	26.04	1.96	6.70

#3L H.W.

P R O V E N

Block	Tonnage	Oz Ag/Tonne	% Pb	% Zn
# 6	1,470	16.51	2.96	9.00
#10	1,260	40.97	8.30	4.06
	1,575	32.82	8.40	3.39
	1,732	30.49	5.86	4.31
	472	17.13	3.37	3.15
	840	23.42	4.63	1.95
	472	25.62	4.46	1.09
	7,821	28.16	5.85	4.45

P R O B A B L E

Block	Tonnage	Oz Ag/Tonne	% Pb	% Zn
# 4	525	22.87	3.60	5.09
# 6	1,470	16.51	2.96	9.00
#10	630	40.97	8.30	4.06
	630	40.97	8.30	4.06
	787	32.88	8.40	3.39
	787	32.88	8.40	3.39
	788	30.49	5.86	4.31
	472	17.13	3.37	3.15
	840	23.42	4.63	1.95
	472	25.62	4.46	1.09
	7,401	27.49	5.70	4.51

#3L F.W.

P R O V E N

Block	Tonnage	Oz Ag/Tonne	% Pb	% Zn
#3	2,100	14.08	3.93	6.12
#7	2,440	38.03	1.01	13.78
#9	1,560	27.60	0.84	8.09
	788	20.34	5.88	0.91
	6,888	26.35	2.42	8.68

P R O B A B L E

Block	Tonnage	Oz Ag/Tonne	% Pb	% Zn
#3	2,100	14.08	3.93	6.12
	1,312	14.08	3.93	6.12
#7	2,440	38.03	1.01	13.78
#9	1,560	27.60	0.84	8.09
	788	20.34	5.88	0.90
	946	27.60	0.84	8.09
	9,146	27.71	2.47	8.25

#4L H.W.

P R O V E N

Block	Tonnage	Oz Ag/Tonne	% Pb	% Zn
# 4	3,439	4.37	1.19	14.15
#10	2,646	12.76	2.61	5.69
	1,785	30.49	5.86	4.31
	472	17.13	3.37	3.15
	840	23.12	4.63	1.95
	472	25.62	4.46	1.09
	9,654	14.82	3.01	7.77

P R O B A B L E

	3,439	4.37	1.19	14.15
	1,785	30.49	5.85	4.31
	472	17.13	3.37	3.15
	840	23.42	4.63	1.95
	472	25.62	4.46	1.09
	1,890	27.17	3.43	1.56
	1,365	12.76	2.61	5.69
	10,263	17.35	3.14	6.89

#4L F.W.

P R O V E N

Block	Tonnage	Oz Ag/Tonne	% Pb	% Zn
	NIL			

P R O B A B L E

Block	Tonnage	Oz Ag/Tonne	% Pb	% Zn
#11	3,675	19.29	1.14	1.19
	5,880	33.48	0.14	0.01
	9,555	28.02	0.52	0.46

APPENDIX 2:Certificates of Analysis

international Plasma Laboratories Ltd.

1. Certificate of Analysis iPL 07G3198
2. Certificate of Analysis iPL 07H3759
3. Certificate of Analysis iPL 08A0325

INTERNATIONAL PLASMA LABS LTD.
ISO 9001:2000 CERTIFIED COMPANY

CERTIFICATE OF ANALYSIS
iPL 07G3198

Richmond, B.C.
Canada V7A 4V5
Phone (604) 879-7878
Fax (604) 272-0851
Website www.ipl.ca

Erik Ostensoe **79 Samples** Print: Aug 09, 2007 In: Jul 26, 2007 [319815:37:03:70080907:001]

Project : None Given
Shipper : Erik Ostensoe
Shipment: PO#: None Given
Comment:

CODE	AMOUNT	TYPE	PREPARATION DESCRIPTION	PULP	REJECT
B21100	79	Rock	crush, split & pulverize to -150 mesh.	12M/Dis	03M/Dis
B84100	5	Repeat	Repeat sample - no Charge	12M/Dis	00M/Dis
B82101	1	Blk iPL	Blank iPL - no charge.	00M/Dis	00M/Dis
B90017	1	Std iPL	Std iPL(Au Certified) - no charge		

NS=No Sample Rep=Replicate M=Month Dis=Discard

Document Distribution

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EN	RT	CC	IN	FX
0	0	0	1	0
DL	3D	EM	BT	BL
0	0	1	0	0

Analytical Summary
Analysis: Au(FA/AAS) / ICP(Multi-Acid)30

##	Code	Method	Units	Description	Element	Limit Low	Limit High
01	0801	Spec	Kg	Weight in Kilogram (1 decimal place)	Wt	0.1	9999.0
02	0368	FA/AAS	g/mt	Au (FA/AAS 30g) g/mt	Gold	0.01	5000.00
03	0354	FAGrav	g/mt	Ag FA/Grav in g/mt	Silver	0.3	9999.0
04	0771	ICPM	ppm	Ag ICP(Multi-Acid)	Silver	0.5	500.0
05	0761	ICPM	ppm	Cu ICP(Multi-Acid)	Copper	1	20000
06	0764	ICPM	ppm	Pb ICP(Multi-Acid) Depressed	Lead	2	10000
07	0780	ICPM	ppm	Zn ICP(Multi-Acid)	Zinc	1	10000
08	0753	ICPM	ppm	As ICP(Multi-Acid) Depressed	Arsenic	5	10000
09	0752	ICPM	ppm	Sb ICP(Multi-Acid) Depressed	Antimony	5	2000
10	0782	ICPM	ppm	Hg ICP(Multi-Acid)	Mercury	3	10000
11	0767	ICPM	ppm	Mo ICP(Multi-Acid)	Molydenum	1	1000
12	0797	ICPM	ppm	Tl ICP(Multi-Acid)	Thallium	2	1000
13	0755	ICPM	ppm	Bi ICP(Multi-Acid)	Bismuth	2	2000
14	0757	ICPM	ppm	Cd ICP(Multi-Acid)	Cadmium	0.2	2000.0
15	0760	ICPM	ppm	Co ICP(Multi-Acid)	Cobalt	1	10000
16	0768	ICPM	ppm	Ni ICP(Multi-Acid)	Nickel	1	10000
17	0754	ICPM	ppm	Ba ICP(Multi-Acid)	Barium	2	10000
18	0777	ICPM	ppm	W ICP(Multi-Acid)	Tungsten	5	1000
19	0759	ICPM	ppm	Cr ICP(Multi-Acid)	Chromium	1	10000
20	0779	ICPM	ppm	V ICP(Multi-Acid)	Vanadium	1	10000
21	0766	ICPM	ppm	Mn ICP(Multi-Acid)	Manganese	1	10000
22	0763	ICPM	ppm	La ICP(Multi-Acid)	Lanthanum	2	10000
23	0773	ICPM	ppm	Sr ICP(Multi-Acid)	Strontium	1	10000
24	0781	ICPM	ppm	Zr ICP(Multi-Acid)	Zirconium	1	10000
25	0786	ICPM	ppm	Sc ICP(Multi-Acid)	Scandium	1	10000
26	0776	ICPM	%	Ti ICP(Multi-Acid)	Titanium	0.01	10.00
27	0751	ICPM	%	Al ICP(Multi-Acid)	Aluminum	0.01	5.00
28	0758	ICPM	%	Ca ICP(Multi-Acid)	Calcium	0.01	10.00
29	0762	ICPM	%	Fe ICP(Multi-Acid)	Iron	0.01	5.00
30	0765	ICPM	%	Mg ICP(Multi-Acid)	Magnesium	0.01	10.00
31	0770	ICPM	%	K ICP(Multi-Acid)	Potassium	0.01	10.00
32	0772	ICPM	%	Na ICP(Multi-Acid)	Sodium	0.01	10.00
33	0769	ICPM	%	P ICP(Multi-Acid)	Phosphorus	0.01	5.00

EN=Envelope # RT=Report Style CC=Copies IN=Invoices Fx=Fax(1=Yes 0=No) Totals: 0=Copy 1=Invoice 0=3½ Disk
DL=Download 3D=3½ Disk EM=E-Mail BT=BBS Type BL=BBS(1=Yes 0=No) ID=C104701
* Our liability is limited solely to the analytical cost of these analyses.

BC Certified Assayers: David Chiu, Ron Williams

Signature:

INTERNATIONAL PLASMA LABS LTD.
ISO 9001:2000 CERTIFIED COMPANY

Certificate of Analysis
iPL 07G3198

Richmond, B.C.
Canada V7A 4V5
Phone (604) 879-7878
Fax (604) 272-0851
Website www.ipl.ca

Intertek

Client : Erik Ostensoe
Project: None Given   Ship#

79 Samples
79=Rock 5=Repeat 1=Blk iPL 1=Std iPL [319815:37:03:70080907:001]

Sample Name	Type	Wt Kg	Au g/mt	Ag g/mt	Ag ppm	Cu ppm	Pb ppm	Zn ppm	As ppm	Sb ppm	Hg ppm	Mo ppm	Tl ppm	Bi ppm	Cd ppm	Co ppm	Ni ppm	Ba ppm	W ppm
588051	Rock	1.6	0.03	—	26.8	104	5784	8771	6	61	<3	9	<2	8	44.9	10	<1	291	39
588052	Rock	1.5	0.07	—	273.6	2095	1.75%	6.69%	<5	1253	<3	<1	<2	33	551.9	25	9	370	<5
588053	Rock	1.6	0.01	—	9.5	41	433	3029	<5	54	<3	5	<2	<2	12.2	11	8	427	17
588054	Rock	1.4	0.07	—	251.2	605	1.36%	11%	<5	711	<3	<1	<2	24	1205.9	26	<1	302	<5
588055	Rock	1.2	0.06	—	6.0	14	832	2775	80	34	<3	7	<2	9	0.9	10	2	375	14
588056	Rock	1.1	0.02	—	6.9	29	849	3789	156	177	<3	6	<2	16	8.9	10	<1	397	15
588057	Rock	1.2	0.05	—	107.4	510	1.21%	6.49%	171	579	<3	14	<2	38	534.4	16	<1	302	440
588058	Rock	1.5	0.03	—	180.0	366	2.56%	9.45%	<5	366	<3	<1	20	45	831.9	21	<1	229	<5
588059	Rock	1.9	0.09	—	0.8m	1228	9.20%	22%	<5	1493	<3	<1	<2	142	2.6m	31	<1	20	<5
588065	Rock	1.7	0.23	1767.7	1.6m	1243	12%	4.96%	337	0.48%	<3	11	195	135	311.1	9	<1	<2	241
588066	Rock	1.4	0.24	—	119.7	353	4283	1.99%	242	380	<3	6	<2	13	107.0	12	<1	223	80
588067	Rock	1.9	0.16	3387.4	1.1m	2571	17%	2.82%	276	3.79%	<3	12	15	20	252.7	5	<1	<2	131
588068	Rock	2.2	0.08	4682.4	2.1m	1182	16%	1.32%	20	1.21%	<3	9	<2	9	128.8	3	<1	<2	54
588069	Rock	1.8	0.43	8370.3	1.1m	7452	17%	15%	<5	1.44%	<3	<1	<2	20	1662.4	11	<1	<2	<5
588070	Rock	1.5	0.15	885.5	0.8m	3909	16%	1.88%	328	0.24%	<3	8	<2	139	106.5	18	<1	<2	38
588071	Rock	1.3	0.10	1831.8	1.7m	1145	16%	2.59%	321	2.65%	<3	6	<2	14	240.0	6	<1	87	129
588072	Rock	1.3	0.32	4241.7	1.6m	4231	19%	6.00%	705	0.94%	<3	11	<2	14	529.9	11	<1	<2	356
588073	Rock	1.4	0.22	506.7	490.7	3374	2.04%	9788	350	0.27%	<3	7	<2	38	57.2	13	<1	153	40
588074	Rock	1.6	0.37	303.4	295.2	410	2.44%	5.72%	4439	0.86%	<3	16	<2	17	449.1	29	3	212	340
588075	Rock	1.7	0.19	1478.6	1.3m	1200	12%	9.03%	76	0.60%	<3	<1	34	24	801.2	20	<1	<2	<5
588076	Rock	2.6	0.08	1341.8	1.3m	4487	11%	2.04%	159	0.26%	<3	9	61	34	149.6	8	<1	<2	88
588077	Rock	2.8	0.03	—	149.5	203	1.97%	9756	156	0.38%	<3	8	<2	<2	66.5	16	8	488	42
588078	Rock	2.1	0.02	—	33.3	79	3330	2929	<5	146	<3	8	<2	11	2.1	16	7	590	13
588079	Rock	1.9	0.12	6491.4	1.0m	6598	16%	7.79%	<5	8.47%	<3	<1	<2	55	696.3	11	3	<2	<5
588080	Rock	1.4	0.03	—	209.6	189	8.83%	6506	269	3.04%	<3	5	<2	15	47.7	6	<1	195	25
588081	Rock	2.2	0.04	—	37.7	57	1898	3163	78	520	<3	7	<2	13	7.2	13	5	336	22
588082	Rock	1.4	0.19	1385.8	1.5m	1404	10%	16%	<5	0.69%	<3	<1	<2	29	1844.2	26	<1	22	<5
588083	Rock	1.9	0.12	1698.7	1.7m	560	14%	4.41%	95	0.35%	<3	11	<2	108	387.1	13	6	185	256
588084	Rock	1.9	0.08	213.9	221.0	296	1.16%	13%	<5	833	<3	<1	<2	17	1339.5	20	<1	234	<5
588085	Rock	1.9	0.04	—	105.8	284	5761	6.42%	<5	312	<3	15	<2	19	535.2	18	6	326	433
588086	Rock	1.8	0.08	—	32.4	65	2459	2.05%	195	87	<3	8	<2	15	148.2	9	<1	333	77
588087	Rock	2.7	0.30	—	183.9	430	5.71%	4.50%	858	1.75%	<3	12	<2	137	336.0	12	<1	77	208
588088	Rock	2.0	0.08	—	169.3	251	1.17%	8.63%	<5	445	<3	<1	<2	<2	769.5	23	7	320	<5
588089	Rock	2.4	0.04	—	11.2	29	950	2337	114	62	<3	6	<2	<2	1.9	9	<1	433	10
588090	Rock	1.4	0.07	317.9	293.8	245	3.46%	5883	<5	412	<3	7	<2	8	32.7	18	11	423	25
588091	Rock	2.0	0.26	—	182.6	308	2.58%	14%	472	449	<3	<1	<2	33	1631.6	24	<1	99	<5
588092	Rock	4.3	0.10	984.2	0.9m	479	15%	3.73%	156	1722	<3	<1	<2	17	316.7	21	<1	145	<5
588093	Rock	2.1	0.02	—	14.7	34	873	868	<5	27	<3	6	<2	<2	<0.2	13	5	550	10
588094	Rock	1.9	0.19	424.1	378.1	509	3.55%	3.57%	389	639	<3	12	<2	13	283.8	15	<1	123	199
Minimum Detection		0.1	0.01	0.3	0.5	1	2	1	5	5	3	1	2	2	0.2	1	1	2	5
Maximum Detection		9999.0	5000.00	9999.0	500.0	20000	10000	10000	10000	2000	10000	1000	1000	2000	2000.0	10000	10000	10000	1000
Method		Spec	FA/AAS	FAGrav	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM

—=No Test Ins=Insufficient Sample Del=Delay Max=No Estimate Rec=ReCheck m=x1000 %=Estimate % NS=No Sample

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CERTIFICATE OF ANALYSIS
iPL 07G3198

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Intertek

Client : Erik Ostensoe
Project: None Given   Ship#

79 Samples
79=Rock   5=Repeat   1=Blk iPL   1=Std iPL   [319815:37:03:70080907:001]

Print: Aug 09, 2007   In: Jul 26, 2007

Sample Name	Cr ppm	V ppm	Mn ppm	La ppm	Sr ppm	Zr ppm	Sc ppm	Ti %	Al %	Ca %	Fe %	Mg %	K %	Na %	P %
588051	75	58	3.49%	8	43	<1	7	0.08	5.81%	0.54	4.60	0.39	2.26	0.20	0.02
588052	62	60	3.75%	5	38	<1	8	0.09	4.97	0.70	6.42%	0.38	2.02	0.18	0.04
588053	84	69	7291	9	78	<1	8	0.12	6.72%	2.39	3.17	0.75	2.60	0.29	0.03
588054	68	58	2.30%	4	53	<1	7	0.12	5.04%	0.51	5.22%	0.27	1.98	0.22	0.04
588055	94	47	2.72%	6	38	<1	6	0.09	5.82%	0.38	4.49	0.28	2.27	0.16	0.02
588056	75	53	4.06%	6	95	<1	6	0.10	5.46%	1.33	4.40	0.45	2.03	0.21	0.03
588057	81	45	4.80%	4	40	<1	6	0.07	4.52	0.37	6.98%	0.27	1.73	0.15	0.02
588058	64	35	6.97%	3	43	<1	6	0.06	3.28	0.66	7.59%	0.38	1.29	0.13	0.03
588059	38	7	6.44%	3	15	<1	2	0.03	1.19	0.54	10%	0.24	0.32	0.10	0.01
588065	<1	<1	9.49%	<2	35	<1	<1	0.01	0.51	3.13	8.75%	0.84	0.20	0.07	<0.01
588066	91	41	3.31%	4	56	<1	4	0.05	4.23	2.61	5.54%	0.77	1.68	0.10	0.01
588067	117	3	5.06%	<2	14	<1	<1	0.01	0.79	0.31	5.25%	0.16	0.31	0.07	<0.01
588068	39	4	1.56%	<2	14	<1	<1	0.01	0.53	0.94	1.88	0.28	0.21	0.10	<0.01
588069	70	3	3.07%	2	13	<1	1	0.01	1.52	0.85	5.48%	0.36	0.50	0.08	0.01
588070	11	<1	9.86%	3	13	<1	<1	0.01	0.75	0.37	8.77%	0.16	0.28	0.08	<0.01
588071	123	28	2.74%	5	31	<1	3	0.05	3.37	0.66	3.57	0.28	1.61	0.12	0.01
588072	121	19	3.55%	2	26	<1	2	0.02	1.74	0.53	6.25%	0.25	0.62	0.09	<0.01
588073	162	19	5.76%	4	34	<1	3	0.04	2.66	0.70	7.76%	0.31	0.96	0.10	<0.01
588074	100	52	3.98%	4	13	<1	6	0.08	4.46	0.41	9.31%	0.36	2.05	0.09	0.03
588075	113	6	6.78%	3	25	<1	2	0.02	0.92	1.20	7.65%	0.35	0.41	0.07	0.02
588076	133	11	7.47%	3	21	<1	2	0.02	1.22	0.93	7.61%	0.35	0.44	0.08	0.01
588077	93	87	3.97%	8	25	<1	11	0.21	6.81%	0.62	4.95	0.43	3.16	0.14	0.04
588078	66	111	3.30%	9	34	<1	13	0.19	8.85%	0.54	4.65	0.51	4.15	0.13	0.06
588079	85	<1	1.66%	<2	23	<1	<1	0.01	0.58	1.49	2.85	0.42	0.25	0.07	0.01
588080	113	24	3.76%	6	20	<1	3	0.06	4.34	0.79	3.57	0.39	2.01	0.10	0.01
588081	86	73	4.83%	7	49	<1	9	0.20	6.09%	0.62	5.68%	0.40	2.44	0.19	0.03
588082	94	10	4.68%	2	29	<1	2	0.04	1.51	1.68	7.41%	0.58	0.55	0.09	0.02
588083	75	57	1.93%	4	41	<1	7	0.13	4.50	0.47	3.97	0.27	1.80	0.14	0.03
588084	78	53	4.65%	4	48	<1	7	0.12	4.03	0.74	7.26%	0.36	1.68	0.15	0.04
588085	63	87	3.39%	7	50	<1	10	0.19	6.28%	0.29	6.29%	0.26	2.57	0.23	0.05
588086	74	75	4.11%	7	63	<1	9	0.14	6.18%	1.59	7.08%	0.53	2.41	0.21	0.06
588087	<1	18	10%	3	51	<1	3	0.04	1.92	1.84	11%	0.59	0.63	0.12	0.01
588088	65	73	2.97%	7	65	<1	9	0.18	5.69%	0.33	6.30%	0.23	2.25	0.27	0.06
588089	83	93	2.20%	9	122	<1	11	0.20	7.65%	0.48	4.45	0.39	3.12	0.23	0.07
588090	67	93	2.39%	8	83	<1	11	0.24	7.21%	1.83	5.22%	0.70	2.77	0.31	0.04
588091	63	19	8.37%	4	33	<1	3	0.05	2.32	0.83	11%	0.42	0.78	0.15	0.02
588092	87	53	3.63%	5	39	<1	8	0.12	4.97	0.34	5.28%	0.31	2.28	0.12	0.06
588093	88	87	2.22%	10	86	<1	11	0.19	7.99%	2.81	4.70	0.82	3.42	0.24	0.06
588094	127	34	4.84%	5	44	<1	5	0.06	3.16	1.73	5.67%	0.58	1.48	0.09	0.02
Minimum Detection	1	1	1	2	1	1	1	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Maximum Detection	10000	10000	10000	10000	10000	10000	10000	10.00	5.00	10.00	5.00	10.00	10.00	10.00	5.00
Method	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM

—=No Test Ins=Insufficient Sample Del=Delay Max=No Estimate Rec=ReCheck m=x1000 %=Estimate % NS=No Sample

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CERTIFICATE OF ANALYSIS
iPL 07G3198

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Website www.ipl.ca

Client : Erik Ostensoe
Project: None Given Ship#

79 Samples
79=Rock 5=Repeat 1=Blk iPL 1=Std iPL [319815:37:03:70080907:001]

Sample Name	Type	Wt Kg	Au g/mt	Ag g/mt	Ag ppm	Cu ppm	Pb ppm	Zn ppm	As ppm	Sb ppm	Hg ppm	Mo ppm	Tl ppm	Bi ppm	Cd ppm	Co ppm	Ni ppm	Ba ppm	W ppm
588095	Rock	1.9	0.12	—	15.1	22	1363	2482	461	72	<3	8	<2	13	<0.2	14	<1	96	17
588096	Rock	2.9	0.09	—	122.1	194	682	2.27%	147	244	<3	9	<2	17	160.1	18	5	219	112
588097	Rock	2.5	0.13	—	20.6	44	2304	3716	<5	59	<3	7	<2	11	7.2	15	7	366	14
588098	Rock	2.4	0.21	—	200.3	414	1.24%	6.24%	135	520	<3	15	<2	20	490.8	14	<1	110	398
588099	Rock	2.8	0.14	950.0	0.8m	640	14%	7.51%	<5	1238	<3	<1	<2	39	652.4	20	<1	108	<5
588100	Rock	2.2	0.39	3626.3	1.3m	5069	17%	21%	<5	0.50%	<3	<1	<2	68	2.4m	24	<1	<2	<5
588111	Rock	2.8	0.23	1618.6	1.4m	2323	16%	19%	<5	0.29%	<3	<1	<2	50	2.1m	23	<1	<2	<5
588112	Rock	1.4	0.07	—	91.6	103	9950	2.17%	<5	584	<3	10	<2	<2	145.2	19	<1	351	92
588113	Rock	2.2	0.15	850.0	0.9m	608	14%	2.11%	78	1662	<3	8	<2	22	171.7	11	6	173	116
588114	Rock	1.1	0.01	—	15.0	27	1482	621	<5	76	<3	4	<2	<2	<0.2	7	4	356	10
588115	Rock	1.2	0.15	633.0	0.7m	2574	6.54%	1.60%	73	0.84%	<3	8	<2	26	122.8	14	2	<2	104
589451	Rock	3.2	0.07	—	423.6	217	8.26%	1.79%	126	0.33%	<3	11	<2	16	137.5	23	29	99	80
589452	Rock	3.7	0.05	—	333.1	233	5.13%	1517	70	0.41%	<3	6	<2	10	<0.2	15	15	307	12
589453	Rock	4.9	0.06	510.0	0.5m	499	2.86%	2.08%	<5	0.46%	<3	8	<2	15	159.5	12	7	239	96
589454	Rock	2.9	0.04	—	80.2	90	9857	7549	<5	369	<3	6	<2	14	51.1	12	6	232	36
589455	Rock	3.8	0.06	—	52.0	60	6957	3671	371	376	<3	5	<2	11	11.8	13	3	180	19
589456	Rock	3.5	0.04	—	19.8	26	2426	844	91	120	<3	4	<2	<2	<0.2	11	8	49	10
589457	Rock	5.5	0.08	—	334.1	582	2.09%	6.46%	111	0.32%	<3	16	<2	21	541.2	11	<1	104	438
589458	Rock	5.4	0.26	2398.1	1.8m	7459	2.24%	14%	<5	0.64%	<3	<1	<2	41	1555.4	21	<1	18	<5
589459	Rock	3.1	0.03	—	72.7	104	4774	6035	<5	348	<3	4	<2	<2	44.7	8	<1	215	29
589460	Rock	3.3	0.21	524.5	436.7	2003	2816	15%	956	1107	<3	<1	<2	126	1615.0	25	<1	20	<5
589461	Rock	3.4	0.16	341.7	331.1	425	5831	1.60%	264	490	<3	8	24	28	113.5	8	<1	83	63
589462	Rock	4.1	0.17	583.3	0.5m	2177	1.59%	16%	15	1714	<3	<1	<2	12	1722.8	24	<1	33	<5
589463	Rock	4.2	0.10	—	29.8	59	392	5.44%	223	106	<3	17	31	36	391.8	9	<1	106	341
589464	Rock	4.5	0.14	971.2	0.9m	1478	876	18%	<5	1342	<3	<1	<2	25	1989.4	23	<1	34	<5
589465	Rock	2.3	0.32	—	31.9	59	1294	4122	423	259	<3	7	<2	23	15.4	10	<1	71	18
589466	Rock	1.8	0.21	—	169.8	622	6059	7.71%	775	486	<3	<1	<2	140	603.8	26	<1	109	<5
589467	Rock	2.9	0.28	—	118.9	943	2125	23%	242	300	<3	<1	<2	13	2.6m	34	<1	16	<5
589468	Rock	3.5	0.20	508.2	475.9	1555	6784	19%	<5	1704	<3	<1	<2	21	2.1m	31	<1	32	<5
589469	Rock	2.6	0.21	1018.0	0.9m	1618	11%	19%	20	1662	<3	<1	<2	55	2.2m	47	<1	<2	<5
589470	Rock	3.2	0.17	—	76.2	265	2342	8.52%	<5	308	<3	<1	<2	31	665.7	23	<1	85	<5
589471	Rock	5.6	0.13	254.0	246.6	1824	7802	23%	22	564	<3	<1	<2	61	2.6m	43	<1	37	<5
589472	Rock	5.1	0.11	—	<0.5	173	1550	8.73%	692	291	<3	<1	120	133	634.4	19	<1	35	<5
589473	Rock	2.0	0.03	—	192.9	734	1.25%	37%	44	205	<3	<1	<2	113	4.5m	48	<1	7	341
589474	Rock	3.5	0.13	—	145.1	601	1.28%	9.10%	661	822	<3	<1	<2	153	681.3	28	<1	56	<5
589475	Rock	3.7	0.26	3009.7	2.3m	6945	2.01%	10%	606	0.48%	<3	<1	<2	235	863.4	30	<1	51	<5
589476	Rock	5.0	0.11	1136.4	1.2m	2663	5.35%	12%	<5	0.26%	<3	<1	<2	31	1314.2	29	<1	35	<5
589477	Rock	4.1	0.05	—	96.9	223	1.77%	6.54%	257	1174	<3	19	<2	135	548.3	7	<1	25	422
589478	Rock	3.8	0.07	—	163.9	618	4.04%	5.08%	302	1469	<3	16	<2	137	434.9	12	<1	91	273
Minimum Detection		0.1	0.01	0.3	0.5	1	2	1	5	5	3	1	2	2	0.2	1	1	2	5
Maximum Detection		9999.0	5000.00	9999.0	500.0	20000	10000	10000	10000	2000	10000	1000	1000	2000	2000.0	10000	10000	10000	1000
Method		Spec	FA/AAS	FAGrav	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM

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Client : Erik Ostensoe
Project: None Given   Ship#

79 Samples   79=Rock   5=Repeat   1=Blk iPL   1=Std iPL   [319815:37:03:70080907:001]

Print: Aug 09, 2007   In: Jul 26, 2007

Sample Name	Cr ppm	V ppm	Mn ppm	La ppm	Sr ppm	Zr ppm	Sc ppm	Ti %	Al %	Ca %	Fe %	Mg %	K %	Na %	P %
588095	69	65	5.07%	11	73	<1	9	0.17	6.53%	0.35	6.83%	0.42	2.89	0.14	0.06
588096	87	75	3.31%	8	33	<1	9	0.20	5.72%	0.44	5.16%	0.38	2.59	0.14	0.06
588097	75	84	3.51%	8	76	<1	11	0.26	7.55%	1.42	5.37%	0.59	3.30	0.20	0.05
588098	64	41	4.57%	4	14	<1	5	0.06	3.70	0.38	6.35%	0.33	1.80	0.10	0.02
588099	97	29	3.11%	3	28	<1	4	0.06	3.52	0.44	5.39%	0.26	1.51	0.16	0.03
588100	32	<1	3.04%	<2	12	<1	<1	0.01	0.45	0.34	6.89%	0.16	0.17	0.09	<0.01
588111	57	5	4.01%	<2	16	<1	1	0.02	1.35	0.38	7.82%	0.20	0.44	0.10	0.01
588112	43	79	2.79%	12	197	<1	7	0.25	6.31%	3.24	5.95%	0.98	1.17	1.37	0.07
588113	97	49	2.91%	5	48	<1	6	0.09	4.64	0.65	4.25	0.34	1.80	0.21	0.03
588114	73	57	7106	8	80	<1	6	0.12	7.17%	3.50	3.23	1.02	2.81	0.16	0.02
588115	128	54	3.18%	2	28	<1	6	0.13	4.52	0.34	5.04%	0.28	2.07	0.13	0.02
589451	185	86	6.25%	6	60	<1	9	0.19	4.29	1.34	7.15%	0.67	1.65	0.09	0.04
589452	126	74	1.75%	8	54	<1	8	0.16	5.48%	1.93	3.93	0.62	2.39	0.14	0.03
589453	103	59	1.80%	8	69	<1	7	0.15	5.08%	3.48	4.21	0.92	2.16	0.12	0.04
589454	133	50	2.51%	4	58	<1	6	0.12	4.06	2.59	4.26	0.66	1.70	0.11	0.03
589455	83	70	3.35%	6	91	<1	8	0.15	5.64%	0.61	5.07%	0.38	2.48	0.14	0.02
589456	87	97	6561	6	105	<1	10	0.21	6.46%	0.62	2.89	0.30	2.73	0.18	0.03
589457	81	22	5.72%	3	63	<1	3	0.05	2.12	4.88	8.18%	1.31	0.76	0.10	0.01
589458	46	4	7.13%	4	60	<1	1	0.03	0.89	3.86	11%	1.09	0.11	0.08	0.01
589459	102	46	9754	4	88	<1	4	0.07	5.78%	2.64	2.43	0.76	2.26	0.29	0.02
589460	<1	37	4.73%	3	77	<1	5	0.08	3.30	1.61	7.64%	0.49	1.56	0.10	0.03
589461	139	19	6.81%	2	34	<1	3	0.04	1.91	1.25	7.58%	0.41	0.60	0.08	0.01
589462	118	11	4.93%	3	14	<1	2	0.04	1.73	1.30	7.63%	0.40	0.65	0.08	0.02
589463	93	20	7.93%	5	26	<1	3	0.05	2.65	3.06	8.96%	1.11	1.16	0.09	0.01
589464	56	13	6.56%	4	22	<1	3	0.04	1.96	2.80	8.55%	0.92	0.75	0.09	0.01
589465	108	24	4.92%	3	24	<1	3	0.05	4.10	0.33	6.01%	0.22	1.67	0.14	0.01
589466	<1	40	4.77%	11	343	<1	4	0.11	4.52	1.76	8.55%	0.79	0.66	0.09	0.06
589467	70	6	2.89%	<2	10	<1	1	0.02	1.65	0.25	8.21%	0.16	0.60	0.09	0.01
589468	83	8	3.23%	<2	16	<1	2	0.03	2.07	0.27	7.06%	0.16	0.70	0.11	0.01
589469	64	<1	4.83%	<2	20	<1	<1	<0.01	0.26	1.68	9.19%	0.48	0.07	0.08	0.01
589470	76	21	5.28%	4	29	<1	3	0.05	3.06	1.80	8.74%	0.63	1.36	0.12	0.03
589471	56	<1	5.64%	<2	12	<1	<1	0.01	0.59	1.26	10%	0.41	0.25	0.09	<0.01
589472	<1	<1	11%	4	32	<1	<1	0.01	0.36	3.88	14%	1.13	0.11	0.10	0.01
589473	22	<1	2.80%	<2	7	<1	<1	<0.01	0.15	0.31	11%	0.09	0.05	0.08	<0.01
589474	<1	<1	16%	2	25	<1	<1	0.01	0.35	1.50	18%	0.50	0.11	0.09	0.01
589475	<1	5	8.44%	2	18	<1	2	0.04	1.28	0.72	10%	0.36	0.48	0.09	0.01
589476	49	41	3.67%	4	54	<1	6	0.12	3.56	2.87	7.16%	1.00	1.53	0.13	0.03
589477	15	5	8.73%	4	12	<1	2	0.02	0.77	0.45	9.24%	0.24	0.34	0.08	<0.01
589478	<1	24	8.43%	6	27	<1	4	0.04	2.50	0.38	10%	0.33	0.97	0.09	<0.01
Minimum Detection	1	1	1	2	1	1	1	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Maximum Detection	10000	10000	10000	10000	10000	10000	10000	10.00	5.00	10.00	5.00	10.00	10.00	10.00	5.00
Method	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM

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Client : Erik Ostensoe
Project: None Given   Ship#

79 Samples
79=Rock   5=Repeat   1=Blk iPL   1=Std iPL   [319815:37:03:70080907:00] In: Jul 26, 2007

Print: Aug 09, 2007

Sample Name	Type	Wt Kg	Au g/mt	Ag g/mt	Ag ppm	Cu ppm	Pb ppm	Zn ppm	As ppm	Sb ppm	Hg ppm	Mo ppm	Tl ppm	Bi ppm	Cd ppm	Co ppm	Ni ppm	Ba ppm	W ppm
NO NAME	Rock	1.7	0.14	1229.5	1.2m	576	15%	4.39%	126	1778	<3	12	<2	166	377.6	12	<1	<2	256
RE 588051	Repeat	—	0.03	—	26.8	107	5901	8766	7	68	<3	6	<2	9	49.9	10	<1	288	36
RE 588075	Repeat	—	0.19	—	1.3m	1201	12%	9.01%	79	0.60%	<3	<1	66	33	800.4	21	<1	<2	<5
RE 588095	Repeat	—	0.12	—	18.7	29	1374	2493	464	96	<3	7	<2	13	<0.2	14	<1	93	18
RE 589459	Repeat	—	0.03	—	71.0	106	4792	6092	<5	352	<3	7	<2	<2	46.0	9	<1	221	22
RE NO NAME	Repeat	—	0.14	—	1.2m	567	15%	4.38%	129	1778	<3	14	<2	176	365.4	11	<1	<2	241
Blank iPL	Blk iPL	—	<0.01	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—
FA_OXG46	Std iPL	—	1.04	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—
FA_OXG46 REF	Std iPL	—	1.04	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—
Minimum Detection		0.1	0.01	0.3	0.5	1	2	1	5	5	3	1	2	2	0.2	1	1	2	5
Maximum Detection		9999.0	5000.00	9999.0	500.0	20000	10000	10000	10000	2000	10000	1000	1000	2000	2000.0	10000	10000	10000	1000
Method		Spec	FA/AAS	FAGrav	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM

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Client : Erik Ostensoe
Project: None Given

Ship#

79 Samples

79=Rock 5=Repeat 1=Blk iPL 1=Std iPL [319815:37:03:70080907:001] Jul 26, 2007

Print: Aug 09, 2007

Sample Name	Cr ppm	V ppm	Mn ppm	La ppm	Sr ppm	Zr ppm	Sc ppm	Ti %	Al %	Ca %	Fe %	Mg %	K %	Na %	P %
NO NAME	62	19	6.17%	<2	38	<1	3	0.03	2.22	0.48	6.44%	0.23	0.80	0.10	0.01
RE 588051	78	60	3.40%	8	44	<1	7	0.08	5.82%	0.54	4.60	0.38	2.21	0.19	0.02
RE 588075	118	6	6.78%	2	25	<1	2	0.02	0.92	1.20	7.62%	0.35	0.41	0.08	0.02
RE 588095	72	67	5.03%	12	74	<1	9	0.17	6.55%	0.35	6.81%	0.43	2.90	0.14	0.06
RE 589459	100	47	9833	4	90	<1	4	0.07	5.79%	2.69	2.40	0.76	2.26	0.29	0.02
RE NO NAME	61	19	6.19%	<2	37	<1	3	0.03	2.24	0.48	6.45%	0.23	0.79	0.10	0.01
Blank iPL	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—
FA_OXG46	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—
FA_OXG46 REF	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—
Minimum Detection	1	1	1	2	1	1	1	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Maximum Detection	10000	10000	10000	10000	10000	10000	10000	10.00	5.00	10.00	5.00	10.00	10.00	10.00	5.00
Method	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM

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Erik Ostensoe
Project : None Given
Shipper : Erik Ostensoe
Shipment: PO#: None Given
Comment:
Re:07G3198

10 Samples Print: Aug 29, 2007 In: Aug 23, 2007 [375911:27:22:70082907:001]

CODE	AMOUNT	TYPE	PREPARATION DESCRIPTION	PULP	REJECT
B217	10	Reject	Split 250g from reject, pulverize to -150 mesh.	12M/Dis	00M/Dis
B84100	1	Repeat	Repeat sample - no Charge	12M/Dis	00M/Dis
B82101	1	Blk iPL	Blank iPL - no charge.	00M/Dis	00M/Dis
B90021	1	STD iPL	Std iPL(Ag Certified) - no charge		

NS=No Sample Rep=Replicate M=Month Dis=Discard

Analytical Summary
Analysis: Ag(FA/Grav)

##	Code	Method	Units	Description	Element	Limit Low	Limit High
01	0354	FAGrav	g/mt	Ag FA/Grav in g/mt	Silver	0.3	9999.0

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Client : Erik Ostensoe
Project: None Given   Ship#

10 Samples
10=Reject 1=Repeat 1=Blk iPL 1=STD iPL [375911:27:22:70082907:001]

Print: Aug 29, 2007   In: Aug 23, 2007

Sample Name	Type	Ag g/mt
588054	Reject	339.0
588068	Reject	5095.3
588075	Reject	1762.2
588083	Reject	1640.6
588091	Reject	196.5
588099	Reject	1235.3
589452	Reject	380.9
589460	Reject	528.0
589468	Reject	595.1
589476	Reject	1292.7
RE 588054	Repeat	350.3
Blank iPL	Blk iPL	<0.3
FA_SE2	STD iPL	352.6
FA_SE2 REF	STD iPL	354.0

Minimum Detection	0.3
Maximum Detection	9999.0
Method	FAGrav

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Erik Ostensoe — **79 Samples** — Print: Jan 21, 2008 In: Jan 10, 2008 [032517:32:30:80012108:001]

Project : None Given
Shipper : Erik Ostensoe
Shipment: PO#: None Given
Comment:
Re:iPL07G3198

CODE	AMOUNT	TYPE	PREPARATION DESCRIPTION	PULP	REJECT
B31100	79	Pulp	Pulp received as it is, no sample prep.	12M/Dis	00M/Dis
B84100	5	Repeat	Repeat sample - no Charge	12M/Dis	00M/Dis

NS=No Sample Rep=Replicate M=Month Dis=Discard

Analytical Summary

Analysis: Ag Pb Zn Assays (over limits)

##	Code	Method	Units	Description	Element	Limit Low	Limit High
01	0354	FAGrav	g/mt	Ag FA/Grav in g/mt	Silver	0.3	9999.0
02	0118	AsyMuA	%	Pb Assay - Multi-Acid by AA/ICP in %	Lead	0.01	20.00
03	0140	MuAICP	%	Zn Assay - Multi-Acid by AA/ICP in %	Zinc	0.01	20.00

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Client : Erik Ostensoe
79 Samples
Print: Jan 21, 2008

Project: None Given Ship# 79=Pulp 5=Repeat [032517:32:30:80012108:001] In: Jan 10, 2008 Section 1 of 1

Sample Name	Type	Ag g/mt	Pb %	Zn %
588051	Pulp	—	—	—
588052	Pulp	—	1.74	6.58
588053	Pulp	—	—	—
588054	Pulp	—	1.38	11.02
588055	Pulp	—	—	—
588056	Pulp	—	—	—
588057	Pulp	—	1.24	6.45
588058	Pulp	—	2.59	9.38
588059	Pulp	768.2	9.15	22%
588065	Pulp	1765.2	11.85	4.86
588066	Pulp	—	—	1.88
588067	Pulp	3390.1	17.02	2.80
588068	Pulp	4675.4	15.64	1.36
588069	Pulp	8368.9	16.52	14.76
588070	Pulp	889.1	16.50	1.74
588071	Pulp	1835.4	16.43	2.62
588072	Pulp	4246.2	18.76	5.91
588073	Pulp	510.4	2.05	—
588074	Pulp	—	2.46	5.68
588075	Pulp	1489.5	11.85	8.86
588076	Pulp	1348.2	11.29	2.11
588077	Pulp	—	2.01	—
588078	Pulp	—	—	—
588079	Pulp	6485.7	16.24	7.81
588080	Pulp	—	8.73	—
588081	Pulp	—	—	—
588082	Pulp	1380.3	10.04	16.18
588083	Pulp	1679.6	13.56	4.35
588084	Pulp	—	1.21	12.48
588085	Pulp	—	—	6.38
588086	Pulp	—	—	1.98
588087	Pulp	—	5.68	4.45
588088	Pulp	—	1.20	8.57
588089	Pulp	—	—	—
588090	Pulp	—	3.48	—
588091	Pulp	—	2.65	14.27
588092	Pulp	986.4	15.02	3.76
588093	Pulp	—	—	—
588094	Pulp	—	3.59	3.52
Minimum Detection		0.3	0.01	0.01
Maximum Detection		9999.0	20.00	20.00
Method		FAGrav	AsyMuA	MuAICP

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Client : Erik Ostensoe
Project: None Given Ship#

79 Samples
79=Pulp 5=Repeat

[032517:32:30:80012108:001] Print: Jan 21, 2008 In: Jan 10, 2008

Sample Name	Type	Ag g/mt	Pb %	Zn %
588095	Pulp	—	—	—
588096	Pulp	—	—	2.25
588097	Pulp	—	—	—
588098	Pulp	—	1.26	6.19
588099	Pulp	959.6	13.75	7.48
588100	Pulp	6329.8	16.66	21%
588111	Pulp	1625.3	16.21	18.74
588112	Pulp	—	—	2.20
588113	Pulp	864.3	13.48	2.16
588114	Pulp	—	—	—
588115	Pulp	623.5	6.49	1.62
589451	Pulp	—	8.17	1.83
589452	Pulp	—	5.02	—
589453	Pulp	516.4	2.88	2.10
589454	Pulp	—	—	—
589455	Pulp	—	—	—
589456	Pulp	—	—	—
589457	Pulp	—	2.10	6.53
589458	Pulp	2400.3	2.29	13.78
589459	Pulp	—	—	—
589460	Pulp	520.7	—	14.64
589461	Pulp	—	—	1.62
589462	Pulp	589.5	1.60	15.77
589463	Pulp	—	—	5.42
589464	Pulp	980.0	—	18.35
589465	Pulp	—	—	—
589466	Pulp	—	—	7.76
589467	Pulp	—	—	23%
589468	Pulp	501.3	—	18.89
589469	Pulp	1015.4	11.29	19.16
589470	Pulp	—	—	8.49
589471	Pulp	—	—	23%
589472	Pulp	—	—	8.66
589473	Pulp	—	1.28	35%
589474	Pulp	—	1.30	9.05
589475	Pulp	3015.8	2.11	10.28
589476	Pulp	1142.5	5.30	11.86
589477	Pulp	—	1.78	6.49
589478	Pulp	—	4.10	5.02
Minimum Detection		0.3	0.01	0.01
Maximum Detection		9999.0	20.00	20.00
Method		FAGrav	AsyMuA	MuAICP

—=No Test Ins=Insufficient Sample Del=Delay Max=No Estimate Rec=ReCheck m=x1000 %=Estimate % NS=No Sample

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79 Samples

Client : Erik Ostensoe
Project: None Given Ship# 79=Pulp 5=Repeat

[032517:32:30:80012108:001] In: Jan 10, 2008 Section 1 of 1

Sample Name	Type	Ag g/mt	Pb %	Zn %
NO NAME	Pulp	1230.6	14.96	4.32
RE 588051	Repeat	—	—	—
RE 588075	Repeat	—	11.77	8.89
RE 588095	Repeat	—	—	—
RE 589459	Repeat	—	—	—
RE NO NAME	Repeat	—	14.88	4.31
Minimum Detection		0.3	0.01	0.01
Maximum Detection		9999.0	20.00	20.00
Method		FAGrav	AsyMuA	MuAICP

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APPENDIX 3: Petrographic Studies

Prepared by J. F. Harris, Ph.D.

REPORT BY VANCOUVER PETROGRAPHICS LTD.

Samples: *Surface C-vein: S-1, S-3, S-4,*
#1 Level C-vein: 1-11, 1-17; C-vein split(?): 1-25
#2 Level F.W. Dike: 2-9; C-vein: 2-18, 2-20, 2-21

Summary:

The samples contain massive to finely banded veins which vary widely in composition between samples and between adjacent bands within samples.

Major "ore" minerals in veins are galena, sphalerite, tetrahedrite, and boulangerite. Minor "ore" minerals are bournonite and chalcopyrite, and a trace of stibnite and native silver.

Major gangue minerals are quartz and ankerite. Quartz/ankerite ratios vary widely between samples. Minor gangue minerals include arsenopyrite and pyrite, with local pyrrhotite and an unknown mineral, Mineral Z, probably and oxide.

Secondary minerals formed from "ore" minerals include anglesite and cerusite after galena, covellite after chalcopyrite, and unidentified non-reflective material, probably secondary after boulangerite.

Variation in distribution of minerals within levels commonly is as large as that between levels, so suggestions regarding variation of minerals with depth are considered to be preliminary. The following variations are suggested:

1) *Tetrahedrite and chalcopyrite decrease in abundance with depth.*
2) *Boulangerite is very variable between samples, and is most abundant in Level 1.*
3) *Bournonite is rare, and is most abundant in Level 1.*
4) *Pyrite generally is most abundant at depth.*
5) *Pyrrhotite occurs only in one sample on Level 2.*
6) *Native silver occurs only in one sample on Level 1.*
7) *Stibnite occurs only in one sample on Level 2.*

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The distribution of silver cannot be readily explained in terms of mineral variations. Silver is present in native silver (one sample) and tetrahedrite, and probably also occurs in significant amounts in galena and boulangerite, particularly significant at lower levels, where the contents of tetrahedrite and native silver are low. The presence of significant silver in boulangerite and galena is suggested because in the sample containing native silver, all of that mineral occurs in exsolution(?) blebs in boulangerite and in galena.

Although the overall silver assays (expanded to 1 m minimum width) decrease with depth, the actual s lver assay of the vein and the Ag/Pb ratio increase from Level 1 to Level 2.

No obvious mineralogical differences exist between the samples at the west end (1-11 and 2-9) and those in the main zone. However, because of the wide variation between samples in both groups, it would require many more samples to determine if any significant differences in minerology were present.

In Sample 2020, weak flow-foliation in the dyke is warped slightly about irregularities in the vein, suggesting that the dyke was intruded into the vein. However, the vein contains fragments of altered rock, some of which are similar to the dyke.

The altered host rock in Sample S-2 is similar to that in the large fragement (?) in the vein in Sample 2-20.

JOHN G. PAYNE

PETROGRAPHIC EXAMINATION OF SAMPLE #4 FROM THE TREASURE MOUNTAIN PROPERTY

Introduction

Sub-surface exploration by drilling on the C Vein Zone encountered a different style of mineralization to the strongly sulfidic Pb/Zn-rich assemblage seen near to surface.

Though low in total sulfides, this material carries good Ag values (in the order of 50 - 70 oz/ton).

The reject portion from assay sample 8165, typifying this mineralization type, was submitted for mineralogical study. It was prepared as a polished thin section (slide 88-309X), using the size fraction -10+18 mesh (particles 1 - 2 mm in size) for mounting.

Description

Estimated mode:

Carbonate	73
Quartz	17
Other silicates	1
Galena	2.5
Sphalerite	1
Pyrite	1.5
Arsenopyrite	trace
Tetrahedrite	1
Chalcopyrite	trace
Ruby silver	trace
Pyrrhotite	trace
Fe-Mn oxides	3

The estimated total sulfide content in the mounted sample is about 6%.

The most prominent sulfide is galena, which occurs mainly as free grains 1 - 2 mm in size, and in simple intergrowth with sphalerite, on the scale 0.5 - 1.0 mm.

Pyrite forms euhedral to irregular grains, 0.1 - 0.5 mm in size. These occur as individuals and small clusters in gangue, generally not associated with the other sulfides. Occasionally pyrite is seen intergrown with, or peripheral to, patches of tetrahedrite. Aresenopyrite, though less abundant, exhibits a similar textural habit to the pyrite.

Tetrahedrite, the principal Ag-carrier in this ore, exhibits a more variable and, in part, very fine-grained habit. The majority of it occurs as segregations, 0.4 - 1.4 mm in size, within which it commonly shows more or less initmate intergrowth with chalcopyrite and/or carbonate. These intergrowths sometimes consist merely of sparse, tiny inclusions of the other phases (sometimes including galena) in dominant tetrahedrite, but also seen as complex, multi-component intergrowths on a scale down to 10 - 20 microns.

Tiny (exsolved?) inclusions of tetrahedrite are also occasionally seen, with fine-grained chalcopyrite, in the peripheral zones of sphalerite segregations.

The tetrahedrite was checked by scanning electron microanalysis and confimed as an argentiferous variety. This analysis does not provide a quanitive measure of the Ag content of the tetrahedrite, but the strength of the Ag peak suggests a figure of 10% or more.

One other Ag mineral was seen. This is a ruby silver - probably pyrargyrite. It occurs sparsely as small inclusions, 10 - 100 microns in size, in tetrahedrite - and in the carbonite gangue immediately adjacent to tetrahedrite.

A prominent textural feature of the ore is the prevalence of disseminations of minute opaque granules, 2 - 25 microns in size, in much of the carbonate gangue. The abundance of these inclusions in different rock fragments ranges from a few percent to some 30%.

They appear, for the most part, to be of oxidic compositon (indicated by SEM analysis as various combinations of Fe and Mn), but occasionally include recognizable sulfides - particularly pyrite, galena and sphalerite.

It seems likely that a little tetrahedrite may also be present in this form - which will not be recovered by conventioanl milling techniques.

In the absence of comparative study material, it is not possible to say whether this mineralization type differs in kind or merely in surface. Nor can any opinion be formed from a single sample re the implications for continuation at depth.

Systematic collection of samples for mineralogical examination during ongoing exploration is recommended. By this means it may be possible to develop a picture of vertical and lateral zonation in the system and to indulge in some extrapolations.

J.F. HARRIS, Ph.D

APPENDIX 4: Confirmatory Metallurgical Testwork Report (Yee, 2008)

Yee, Jasman, 2008, Confirmatory Metallurgical Testwork on Huldra Silver's Treasure Mountain Project, Hope, B. C., report prepared by Jasman Yee & Associates Inc. for AMEC Earth and Environmental

<u>Note that the attached document comprises text only without appendices. The entire report, if required, is available for inspection.</u>

Jasman Yee & Associates Inc.
6698 Lochdale Street
Burnaby, BC, Canada
V5B 2M8
Phone: 604 420-4772

Confirmatory Metallurgical Testwork on Huldra Silver's Treasure Mountain Project Hope, BC

Prepared for

Mr. Peter Lighthall

AMEC Earth & Environmental

2227 Douglas Road

Burnaby, B.C.

Canada V5C 5A9

By

Jasman Yee,
P. Eng

February 15, 2008

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1.0 Introduction

In an email message dated June 18, 2007, Peter Lighthall of Amec, Inc. indicated that Huldra was ready to advance to the next development phase for their Treasure Mountain Project. As part of the new work program, he requested confirmatory metallurgical testwork and developing representative tailings samples to be used in support of permitting.

The objective of the new test program was to:

1. Obtain representative ore sample from the mine
2. Duplicate the bench scale testing that was used as the basis for flow sheet development and to generate samples of tailings for the following:

- Acid drainage potential testing
- Tailings water quality determinations
- Treatability assessment of the tailings water to meet CCME and BC discharge standards
- Solid-liquid separation testing to confirm that the tailings can be filtered for the dry stack

This report deals with the confirmatory metallurgical testwork used in developing the original flowsheet; producing representative tailings solids and liquids for further testing under the supervision of Fred Sverre of Entech Environmental Consultants Ltd.; and to confirm that the tailings produced can be filtered and stacked based on the results of the thickening and filtration testwork.

1.0 Introduction

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2.0 Summary and Conclusions

A summary of the results of the testwork performed at PRA together with the conclusions drawn are provided below:

1. Confirmatory testwork on a new composite of freshly collected samples from level 1 and level 2 adits at the Treasure Mountain Project site concluded that the flowsheet used in the study of 2006 is viable.
2. The assumed work index of 13.0(Imperial) used in the power calculations to size the grinding power requirement was confirmed. Ball mill work index obtained in the test at PRA was 12.6(Imperial) or 13.9(metric).
3. The sphalerite mineral in the new composite tested was not as active as the sample used in the Orocon study.
4. Use of zinc sulfate as a zinc depressant in the lead float was effective and a lower dosage can be used without decreasing the zinc recovery to the zinc concentrate product.
5. Sodium metabisulfite or sulfur dioxide equivalent was also tested and found to effective as well in depressing zinc in the lead flotation circuit.
6. The above two items suggest they are options available in the event soluble zinc concentrations in the tailings pond are unacceptable for direct discharge to the receiving environment.
7. The lead and silver head grades tested are slightly higher than the estimated ore resource grade and can be responsible for the higher lead and silver recoveries obtained in the lock cycle results. These recoveries are higher than the projected recoveries used in the 2006 study. However, the lead concentrate grade is lower than the projected grade.
8. The zinc concentrate grade and recovery are similar to the projected figures used in the 2006 study.
9. Other potential payable elements in the zinc concentrate were assayed. The elements of interest were indium, germanium and cadmium. The assays of these three elements produced from the locked cycle tests were less than 5g/t, less than 5g/t and about 0.65% respectively.
10. Settling testwork on the flotation tailings indicates flocculent is required to produce faster settling rates and clearer overflows than unflocculated tailings. The calculated thickener area requirement with flocculation is 0.56 m^2/tonne-day of solids. Without flocculent, the thickener area requirement would be ten times larger.
11. Vacuum filtration on the settled product produced a cake with about 20% moisture. It would appear based on the cake characteristics that this cake can be dry-stacked. However, optimization tests are required to confirm the finding as the filtration rate was medium to slow probably due to the fabric used. Besides further testwork would be required in sizing the filter for this application.
12. Samples of tailings solids and liquids from the lock cycle series of tests were composited and shipped to Cantest for environmental testing under the direction of Fred Sverre of Entech Environmental Consultants Ltd.
13. In addition, a composite head sample used in the metallurgical test program was sent to Ms. Emily Chastain of Amec Earth and Environmental for ABA testwork.

3.0 Sample Collection

The samples used for the test program was collected by Erik Ostensoe in July of last year. He sampled the entire underground workings of levels 1 and 2 and the East and West drifts. Entire details such as sample locations and the individual assays can be found in his report. It was unfortunate access to level 3 was not possible as those workings were still flooded at the time of sampling and permission to access was not granted by the Ministry of Energy and Mines due to safety reasons.

A list of the sample assay rejects used for this metallurgical test program together with their weights is documented in PRA's receiving log sheet which is provided in Appendix 1 of this report.

4.0 Head Assay Results

The assays of the individual samples collected by Erik Ostensoe were analysed at IPL and are presented in his report. The locations of these samples together with a drawing showing where these samples were taken are also discussed in his report.

5.0 Metallurgical Test Composite Sample

Prior to testing, PRA was instructed to prepare 4 composites from the different working areas for head assays only. The objective was to ratio the weights for these 4 different areas such that an overall ore grade matching the ore resource grade would be met and it would closely resemble a representative sample.

The instructions for preparing these 4 composites with their designated names and the tag numbers used are provided below:

Level 1 East Composite: Sample numbers 588065 to 588079

Level 1 West Composite: Sample numbers 588051 to 588059, 588080 to 588100 and 588111 to 588115

Level 2 East Composite: Sample numbers 589451 to 589464

Level 2 West Composite: Sample numbers 589465 to 589478.

The sample with no name was not used.

Assays for these 4 composite samples are provided in the table below:

Sample Name	SampleType	Ag g/mt	Ag ppm	Pb %	Zn %
L1 East Comp	Pulp	3045.2	2499.0	20.15	4.67
L1 West Comp	Pulp	--	693.2	5.51	6.18
L2 East Comp	Pulp	--	514.0	1.64	7.11
L2 West Comp	Pulp	--	493.0	2.12	14.92

There are two columns for silver assays. The first column is silver by fire assay with a gravimetric finish; the second is silver using acid digestion and ICP. For very high silver assays, the fire assay with a gravimetric finish is more reliable.

Based on the above composite assays, an overall or master composite for metallurgical testing was prepared according to the following instructions:

1 part of level 1 east
1 part of level 2 west
2 parts of level 1 west
2 parts of level 2 east

The objective was to ensure that the test composite would be representative of the grade of ore in the reserve estimation and closely resemble the lead to zinc ratio.

5.1 Master Composite Head Assay

The master composite head assays were performed in duplicate and the main assays are listed in the table below. Additional details on the head assays are provided in Appendix 2.

PRA

HEAD ASSAY REPORT

Client: Huldra **Date:** 5-Nov-07
Sample: as specified **Project:** 0707109

Elements	Units	Sample ID		Detection Limits		Analytical
		Composite	RE Composite	Min.	Max.	Method
Au	g/mt	0.16	0.16	0.01	5000	FA/AAS
Ag	ppm	943.6	952.7	0.50	1000	MuAICP
Pb	%	7.23	7.20	0.01	20	AsyMuA
Ox.Pb	%	0.19	0.18	0.01	100	AsyLeh
Zn	%	7.88	7.88	0.01	20	MuAICP
Ox.Zn	%	0.15	0.14	0.01	100	AsyLeh
S(tot)	%	6.87	6.92	0.01	20	Leco
S(-2)	%	6.77	6.81	0.01	100	AsyWet

The lead and silver grades were slightly higher than expected but would be suitable for meeting the prescribed metallurgical and environmental testing objectives. Oxidized lead and zinc assays were also performed to determine the degree of oxidation that had taken place since the sample was taken. The values in both cases were very low indicating minimal oxidation.

5.2 Whole Rock Analyses

Whole rock analyses was performed on the master composite and the results indicate the major mineral was silica at 41.7% followed by oxides of iron, aluminum, manganese, potassium and calcium in decreasing order. The quantity of the manganese oxide was not as substantial as was originally expected based on visual examination of the ore zones during sampling.

Complete details of the whole rock analyses can be found in Appendix 3.

5.3 Work Index

The Bond ball mill work index was determined on the master composite sample at a closing screen size of 74 microns. The work index for the sample was 13.9kWh/tonne of feed under simulated steady state conditions. The test was performed with six cycles to stabilize the circulating loads. Details of the test and data are provided in Appendix 4.

5.4 Flotation

The base case test was F1 and the procedure used was a grind of 70% passing 200mesh with zinc sulphate, soda ash, potassium ethyl xanthate and DF250 in the lead float. Lime to pH 11.0, copper sulphate to activate zinc and sodium isopropyl xanthate was used in the zinc float.

Procedure for F2 was similar to F1 but without any zinc sulfate and a lower lead float pH of 7 rather than 9.5.

F3 was similar to F2 but 100g/t of sodium metabisulfite was used for depressing zinc in the lead float.

Test F4 was similar to F1 but the zinc sulfate addition was reduced by a half and the lead float was performed at pH 7.5.

Results of the four tests are summarized in the table below followed by comments after each phase of flotation:

Summary of Flotation Tests Results

Combined Lead Rougher/Scavenger Flotation Results

Test #	Weight		Assay				Distribution			
			Ag	Pb	Zn	S	Ag	Pb	Zn	S^{-2}
	g	%	g/t	%	%	%	%	%	%	%
F1	**445.4**	**22.9**	**4024.3**	**26.92**	**6.24**	**13.4**	**98.6**	**97.7**	**19.9**	**46.7**
F2	369.2	19.1	4542.1	31.99	16.81	19.6	97.5	97.4	41.7	56.8
F3	312.5	15.8	5879.8	41.79	10.86	20.7	96.5	97.2	22.5	46.2
F4	**292.5**	**15.0**	**5068.5**	**37.40**	**6.02**	**16.8**	**93.1**	**96.1**	**13.3**	**39.8**

The high lead float pH of 9 in test 1 resulted in more mass pull to concentrate
Zinc sulfate was more effective than sodium metabisulfite in depressing zinc in the lead float.

Combined Zinc Rougher/Scavenger Flotation Results

Test #	Weight		Assay				Distribution			
			Ag	Pb	Zn	S	Ag	Pb	Zn	S^{-2}
	g	%	g/t	%	%	%	%	%	%	%
F1	**296.8**	**15.2**	**44.2**	**0.23**	**37.03**	**21.6**	**0.7**	**0.6**	**78.9**	**50.2**
F2	245.7	12.7	71.1	0.38	34.56	20.6	1.0	0.8	57.1	39.9
F3	290.5	14.7	130.3	0.76	39.94	23.5	2.0	1.7	76.9	48.8
F4	**284.4**	**14.6**	**99.7**	**0.58**	**39.60**	**24.1**	**1.8**	**1.5**	**85.1**	**55.4**

Higher zinc recoveries were achieved using zinc sulfate in the lead float

Combined Lead & Zinc Flotation Concentrate

Test #	Weight		Assay				Distribution			
			Ag	Pb	Zn	S	Ag	Pb	Zn	S^{-2}
	g	%	g/t	%	%	%	%	%	%	%
F1	**742.2**	**38.1**	**2432.7**	**16.25**	**18.55**	**16.7**	**99.3**	**98.2**	**98.8**	**97.0**
F2	614.9	31.8	2755.5	19.36	23.90	20.0	98.5	98.2	98.8	96.8
F3	**603.0**	**30.5**	**3109.8**	**22.03**	**24.87**	**22.1**	**98.5**	**98.9**	**99.5**	**95.0**
F4	576.9	29.6	2618.7	19.24	22.58	20.4	94.9	97.6	98.3	95.1

Use of metabisulfite in test F3 provided the best concentrate products

Final Flotation Tail

Test #	Weight		Assay				Distribution			
			Ag	Pb	Zn	S	Ag	Pb	Zn	S^{-2}
	g	%	g/t	%	%	%	%	%	%	%
F1	**1,204.4**	**61.9**	**10.9**	**0.18**	**0.14**	**0.3**	**0.7**	**1.8**	**1.2**	**3.0**
F2	1,317.1	68.2	19.7	0.17	0.14	0.3	1.5	1.8	1.2	3.2
F3	1,371.3	69.5	21.3	0.11	0.06	0.5	1.5	1.1	0.5	5.0
F4	1,374.1	70.4	59.2	**0.20**	**0.16**	**0.4**	**5.1**	**2.4**	**1.7**	**4.9**

The silver recovery for test 4 appears to be off due to the poor check on the metallurgical balance.
The balance for the other elements such as lead, zinc and sulfur check very well.

Details of the individual test procedures, results and size analysis can be found in Appendix 5.

5.5 Lock Cycle Test

The lock cycle results show good concentrates can be produced at better than expected recoveries by using recycled water from the previous cycle. There does not appear to be any deleterious effect on the metallurgy with using recycled water. Hence, there is the possibility of reducing reagents further during the next phase of the project's development program.

The combined lead concentrate grade for cycles 4, 5 and 6 was 46.76%Pb, 7.554kg Ag/t with recoveries of 95.4% and 96.6% lead and silver respectively. The combined zinc concentrate for cycles 4, 5 and 6 was 54.76% and the recovery was 83.8%

Agreement of the silver, zinc and sulfur back calculated assays with the actual assayed head values are good and the grades and recoveries for these elements can be relied on with confidence. However, the check on the assayed head assay for lead with the back calculated assay is not as good. The lead recovery is therefore slightly biased on the high side and the lead concentrate assay could be biased on the low side. In any case the overall results are slightly better than the projected figures used in the 2006 report.

The lock cycle tests also demonstrated that the volume of the lead and zinc cleaner recycle streams will be very low.

In addition to regular assays used in the metallurgical balance, additional assays for indium, germanium and cadmium in the zinc concentrates were requested as these are potential payable elements in the zinc product. The assays of indium, germanium and cadmium are less than 5ppm, less than 5ppm and about 58ppm respectively.

Complete details of the lock cycle tests are provided in Appendix 6.

5.6 Thickening and Filtration

Two settling tests, one without flocculent and one with 40g/L P351 flocculent, were conducted on a split of the combined locked cycle zinc flotation tails. The pH of the sample tested dropped to 8.3 from 11.0 when left standing for a few days before the settling tests were conducted.

The initial settling rate without flocculent was 0.3m/day whereas with flocculent, the settling was about 10 times faster at 3.3m/day. Also the unit area requirement to produce an underflow density of 50% was 5.18m^2/tpd without flocculent and 0.56m^2/tpd with flocculent.

The thickening tests suggest flocculent would be required in the plant operation but optimization of flocculent usage and pH would be required in future test programs.

Details of the thickening tests and data are provided in Appendix 7

A vacuum filtration test was performed on the flocculated settled sludge from the thickening tests. The filter feed density was 50% solids and the filtration rate for solids was 148.3 kg/m^2/h and 102 L/m^2/h for liquids. The cake moisture was 19.6% and the filtrate was clear. The test demonstrated that the flocculated zinc tailings can be filtered with some difficulty and the characteristics of the filtered product suggests it is stackable. However, the filtration rate is slow and a larger than normal filter would be required for this application. It is recommended that optimization testwork is required in the final sizing of the filter together with a better selection of the filter medium.

The filtration test report is provided in Appendix 8.

6.0 Environmental Testing

Approximately 17 liters of zinc flotation tailings from the locked cycle series of test were sent for environmental testing on January 11th, 2008. The tailings pulps from the six cycles were composited and a summary of the analytical results of the solids are provided in the table below:

	Assays			
Product	Ag	Pb	Zn	S(T)
Flotation Tails	g/t	%	%	%
Zn Tails, Cycle 1	16.5	0.22	0.22	0.45
Zn Tails, Cycle 2	18.6	0.27	0.16	0.47
Zn Tails, Cycle 3	29.1	0.24	0.28	0.48
Zn Tails, Cycle 4	25.9	0.24	0.17	0.38
Zn Tails, Cycle 5	29.6	0.30	0.20	0.53
Zn Tails, Cycle 6	15.9	0.26	0.22	0.45
Total Zn Flotation Tails	22.6	0.26	0.21	0.46

These tailings solids would closely resemble the tailings product from the operating plant. The solids and the liquids were sent to:

CANTEST LTD
4606 Canada Way
Burnaby, BC V5G 1K5
Tel: 604 734 7276
Fax: 604 731 2386

Attention: Mr. Tim O'Hearn

These were the instructions of Fred Sverre of Entech Environmental Consultants Ltd.

In addition to the tailings sample, a cut of the head sample used in the locked cycle test was also sent for environmental ABA testing. This sample was sent to:

Amec Earth and Environmental
2227 Douglas Road
Burnaby, BC
V5C 5A9

Attn: Ms. Emily Chastain.